



Grey Fox Project Prefeasibility Study NI 43-101 Technical Report

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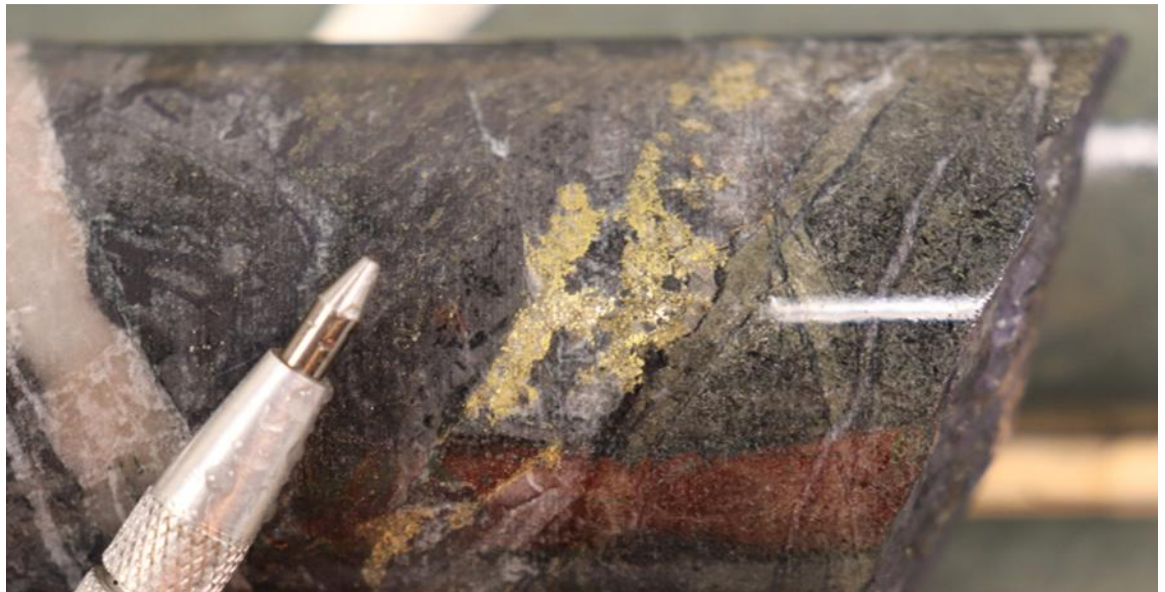
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Acronyms / Abbreviations

Acronym / Abbreviation	Full Name
ABA	Acid-Base Accounting
AEP	annual exceedance probability
ALS	ALS Metallurgy
AP	Acid Potential
ARD/ML	Acid Rock Drainage / Metallic Leaching
Au	gold
BGS	below ground surface
Bwi	Bond Ball Mill Work Index
Ca	calcium
CaO	calcium oxide (quicklime)
CaCO ₃	calcium carbonate
CIL	carbon-in-leach
CIP	carbon-in-pulp
CLDZ	Cadillac-Larder Lake Deformation Zone
CN, CN ⁻	cyanide ion
CN _{WAD}	Weak-Acid-Dissociable Cyanide
CO ₂	carbon dioxide
COOP	Central Ontario Orthophotography Project
Cu	copper
CZ	Contact Zone
DEM	digital elevation model
Dwi	Drop-Weight Index
EA	Environmental Assessment
ECA	Environmental Compliance Approval
EM	electromagnetic
ERD	External Reference Distribution
FeS ₂	Pyrite (Iron Disulfide)
FW	footwall
FZ	Fault Zone
GKDZ	Gibson-Kelore Deformation Zone
GRG	Gravity Recovery Grade
HCT	Humidity Cell Test
HG	High Grade



Acronym / Abbreviation	Full Name
HPGR	High Pressure Grinding Rolls
HR	Hydraulic Radius
HW	hangingwall
ICP	Inductively Coupled Plasma
JK Tech	Julius Kruttschnitt Technology
LG	Low Grade
LGC	Low-grade, low-pyrite
LGP	Low-grade, high-pyrite
LOM	life-of-mine
MC	Master Composite
MDMER	Metal and Diamond Mining Effluent Regulations
MECP	Ministry of the Environment, Conservation and Parks
MEM	Ministry of Environment and Mines
MIBC	methyl isobutyl carbinol
MMER	Metal Mining Effluent Regulations
Mo	molybdenum
MSO	Mineable Shape Optimizer
NaCN	sodium cyanide
NE	northeast
NHLFZ	Nighthawk Lake Fault Zone
NNP	Net Neutralization Potential
NP	Neutralizing Potential
NPAG	Non-Potentially Acid Generating
NPI	Net Profits Interest
NPR	Neutralization Potential Ratio
NPV	Net Present Value
NSG	non-sulphide gangue
NSR	Net Smelter Return
OPEX	Operating Expenditure
PAG	Potentially Acid Generating
PAX	potassium amyl xanthate
PCS	Process Control System
PDDZ	Porcupine-Destor Deformation Zone
PEA	Preliminary Economic Assessment
PFS	Prefeasibility Study
PMV	Pillowed Mafic Volcanic



Acronym / Abbreviation	Full Name
ppm	parts per million
PVD	prefabricated vertical drain
PWQO	Provincial Water Quality Objectives
P ₈₀	Particle Size (80th percentile)
PAX	Potassium Amyl Xanthate
PSD	Particle Size Distribution
ppm	parts per million
PTTW	Permit to Take Water
ROM	Run-of-Mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RSD	Relative Standard Deviation
SAG	Semi-Autogenous Grinding
SAGB	Southern Abitibi Greenstone Belt
SABC	SAG mill–Pebble Crusher–Ball mill Circuit
SAC	strong acid cation
SBA	strong base anion
SGH	Spatiotemporal Geochemical Hydrocarbon
SGS	SGS Minerals Services
SEDS	Siltstone/Epidote/Diopside Skarn
SFE	Shake Flask Extraction
SO ₂	sulphur dioxide
S (or S%)	Sulphur Content
SMC	Steve Morrell Comminution
SP	Set Point
T ₈₀	Transfer Size (80th Percentile)
TDH	Total Dynamic Head
TOC	Total Organic Carbon
TW	True-Width
UAV	Unmanned Aerial Vehicle
UMW	underground mine workings
US\$	United States dollar
VTEM	Versatile Time Domain Electromagnetic
XPS	XPS Minerals Services (now ALS Minerals)
µm	micrometers
w/w	weight-by-weight
WAD	Weak Acid Dissociable



Acronym / Abbreviation	Full Name
WEA	Western Expansion Area

Units of Measure

Unit	Definition
d	day
g	gram
g/L	gram(s) per litre
g/t	gram(s) per tonne
gpm	gallon(s) per minute
h	hour
km	kilometre
HP	horsepower
kWh/t	Kilowatt-hours per tonne
kWh/m ³	Kilowatt-hours per cubic metre
kg/t	kilogram(s) per tonne
L	litre
m	metre
m ²	square metre(s)
m ² /t/d	square metre(s) per tonne per day
m ³	cubic metre(s)
mg/L or mg·L ⁻¹	milligram(s) per litre
ppm	parts per million
t	Metric tonne
T	Imperial Ton
t/d	tonne(s) per day
t/m ² /h	tonne(s) per square metre per hour
t/m ³ /h	tonne(s) per cubic metre per hour



Glossary

Term	Definition
Eastern Properties	The land tenure forming the eastern portion of properties of the Fox Complex, which includes Grey Fox, Froome, and Black Fox.
Fox Complex	The entire package of properties owned by McEwen Inc. in Ontario.
Stock Properties	The land tenure forming the central portion of properties of the Fox Complex, located in Stock Township.
Western Properties	The land tenure forming the western portion of properties of the Fox Complex, known by the names Fuller, Buffalo-Ankerite, Davidson-Tisdale, and Paymaster.



1 Summary

1.1 Introduction

The Grey Fox project (Grey Fox) is a gold mine project located near Matheson, Ontario, adjacent to the existing Black Fox Mine, and is being developed by McEwen Inc. (McEwen). This Technical Report (Report) presents the results of a prefeasibility study (PFS) for Grey Fox.

The project consists of two separate underground mines, accessed via dedicated portals and declines, utilizing longhole mining methods for ore extraction at a nominal rate of 1,500 tonnes per day (t/d) for the initial seven years, then ramping up to a nominal 2,700 t/d for the subsequent six years. Ore is expected to be processed at the existing Stock Mill, approximately 30 km away.

This Report describes several aspects of the Grey Fox project, including:

- A Mineral Resource estimate.
- A life-of-mine (LOM) plan based on Probable Mineral Reserves, and the resultant Mineral Reserve estimate.
- Capital and operating cost estimates.
- A summary financial model.

Preparation of this Technical Report was completed with contributions from the following firms.

- Stantec Consulting Ltd.
- McEwen Inc.
- DRA
- Arcadis

All costs are in Q1 2026 United States dollars (US\$), unless otherwise stated.

All units of measurement are metric, unless otherwise stated.

1.2 Property Description and Location

Grey Fox is part of the Fox Complex, a portfolio of gold properties located in the Timmins–Matheson region of northeastern Ontario, comprising multiple past-producing, operating, and development-stage assets with access to established regional infrastructure and processing capacity. The Grey Fox deposit is made up of six principal mineralized zones: 147, 147NE, Contact, Grey Fox South, Gibson and Whiskey Jack. The Complex is organized into three principal groups—Eastern, Stock, and Western Properties—and includes the operating Froome Mine and Stock Mill, the former Black Fox and Stock mines, and several development-stage and historical deposits.

McEwen holds a majority interest in the mineral tenure across the Complex within a well-established mining jurisdiction. Certain properties are subject to royalties and a gold stream, and Indigenous agreements are in place for the Eastern and Stock Properties, with additional agreements required for the Western Properties.



Environmental obligations are managed under approved closure plans, and no material issues have been identified that would preclude project development.

Further details regarding property ownership, agreements, and permitting considerations are provided in Section 4 of the main report.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Fox Complex is located within the established Timmins mining district of northeastern Ontario and benefits from excellent year-round accessibility via provincial highways, secondary roads, and nearby commercial air service. The project is situated close to the regional centres of Timmins, Matheson, and Kirkland Lake, where mining services, skilled labour, and supplies are readily available. It is supported by existing infrastructure including electrical power, transportation corridors, and nearby processing facilities.

The region experiences a typical northern Ontario climate with cold winters and warm summers, which permit year-round exploration and mining activities. Physiography across the project area is characterized by low to moderate relief, glacial overburden, wetlands, lakes, and boreal forest vegetation, with elevations generally ranging from approximately 250 m to 330 m above sea level.

Further details on accessibility, climate, local resources, infrastructure, and physiography for the Grey Fox Project and the surrounding area are provided in Section 5 of the main report.

1.4 History

Grey Fox is the primary focus of this PFS and represents a key growth and development asset within McEwen's Fox Complex in the Timmins–Matheson mining district of Ontario, a historically significant gold-producing region. The deposit comprises several principal mineralized zones, including Gibson, Contact, Grey Fox South, and Whiskey Jack.

The project has been advanced through successive phases of exploration and development by prior operators and McEwen, including drilling, underground development, geophysical surveys, trenching, and resource definition programs. Historical work at Gibson included underground development and limited production, while more recent exploration campaigns have refined the geological model and supported Mineral Resource estimation.

Since acquiring the property in 2017, McEwen has completed systematic drilling programs to de-risk known zones, test extensions, and identify additional mineralized lenses. A key refinement in 2019 improved the understanding of mineralization orientation and supported enhanced confidence in geological interpretation and modelling.

The 2025 Mineral Resource estimate forms the basis for this PFS, and previous estimates are considered historical and not relied upon. Further details on the exploration history and development of the Grey Fox deposit are provided in Section 6 of the main report.



1.5 Geological Setting and Mineralization

The Grey Fox deposit is interpreted to be a multi-domain, structurally controlled gold system hosted within mafic metavolcanic rocks, metasedimentary units, and intrusive rocks known as the Gibson Syenite. The deposit is divided into two principal geological domains, or blocks, separated by the Contact Fault: the 147–Grey Fox South Block and the Gibson Block. Gold mineralization is primarily associated with steeply dipping, northeast-southwest-striking sheeted vein arrays (C-veins), with the principal mineralized zones including 147, 147NE, Grey Fox South, and Gibson. Mineralization is controlled by vein geometry, structural preparation, lithological and rheological contrasts, and host-rock chemistry, with higher-grade zones typically developed in favourable metavolcanic and metasedimentary units.

The geological model is considered appropriate for the current PFS level of study and provides a reasonable basis for Mineral Resource estimation and mine planning, with additional work expected to refine structural controls and mineralization continuity. Further details on the Grey Fox deposit and its geological interpretation are provided in Section 7.

1.6 Deposit Types

The Grey Fox vein systems exhibit characteristics consistent with a shallow-crustal, epizonal orogenic gold environment, characterized by quartz–carbonate vein-hosted mineralization. Mineralization occurs within a network of structurally controlled vein arrays developed under brittle upper-crustal conditions.

This deposit model supports continuity of mineralized structures and provides a framework for ongoing exploration targeting within similar structural settings across the project area. Further details on deposit characteristics and geological interpretation are provided in Section 8 of the main report.

1.7 Exploration

Grey Fox represents the principal exploration focus within the Eastern Properties and a key growth component of the Fox Complex. Since acquiring the property in 2017, McEwen has advanced the Grey Fox Project through extensive drilling, geophysical surveys, geological interpretation, UAV imagery, trenching, mechanical stripping, mapping, channel sampling, and surface validation programs designed to define and expand mineralized zones and improve the geological model.

Exploration activities have improved confidence in the interpretation of mineralization and supported the current Mineral Resource estimate. The Grey Fox deposit comprises multiple zones within a broader structural and lithological corridor that remains partially tested along strike and at depth, with surface exploration constrained in part by widespread overburden.

Recent programs have enhanced the definition of structural corridors and identified additional potential beyond the currently defined resources. As a result, the Grey Fox deposit provides a strong opportunity for further resource growth within the PFS framework.

Further details on exploration activities and results for the Grey Fox deposit and surrounding area are provided in Section 9 in the main report.



1.8 Drilling

Drilling at Grey Fox has been completed over multiple campaigns by historical operators and, more recently, by McEwen, providing the geological, assay, geotechnical, hydrogeological, and metallurgical information required to support the PFS. Historical drilling from the late 1970s through the 1990s included sludge sampling, BQ- and NQ-diameter core drilling, although documentation of some early sampling procedures is limited.

Since 2018, McEwen has advanced the Grey Fox drill database through systematic surface diamond drilling, primarily using NQ core, with work completed by established drilling contractors and logged at controlled company facilities. Further details on drilling programs are provided in Section 10 in the main report.

1.9 Sample Preparation, Analyses and Security

Sampling, sample preparation, analytical procedures, quality assurance/quality control (QA/QC), and security protocols for Grey Fox and the Eastern Properties have been implemented in accordance with industry best practices to support Mineral Resource estimation. Current programs utilize systematic, geologically controlled sampling, supported by documented chain-of-custody procedures to maintain sample integrity.

Analytical work has been carried out primarily by ISO/IEC 17025-accredited laboratories using fire assay and, more recently, photon assay techniques, with appropriate multi-element methods applied as required. The QA/QC program, including the insertion of control samples and external verification, demonstrates acceptable accuracy, precision, and absence of material bias. Historical data incorporated into the Mineral Resource estimate has been reviewed and deemed acceptable, with no identified issues that would materially impact data reliability.

The Qualified Person (QP) considers the sampling, analytical methodologies, QA/QC procedures, and sample security measures to be adequate and appropriate, and concludes that the resulting dataset is reliable and suitable for use in Mineral Resource estimation. Further details on sampling, sample preparation and security are provided in Section 11 in the main report.

1.10 Data Verification

Data verification for the Grey Fox Mineral Resource has been conducted by the QP through comprehensive review and validation procedures, incorporating drilling data collected from 2014 through 2026 together with relevant historical datasets. Verification activities included database audits against original assay certificates, collar and survey validation, statistical review of QA/QC data, site inspections of core handling and logging practices, and independent umpire laboratory testing.

Historical data from prior operators were selectively validated through certificate reconciliation, confirmatory drilling, and comparison with previous technical studies, with greater reliance placed on modern datasets where documentation is more complete. Minor limitations typical of legacy datasets were identified;



however, these have been considered in Mineral Resource classification and do not materially affect data reliability.

Overall, verification results demonstrate acceptable accuracy, precision, and consistency of the geological, assay, and survey data, and the QP considers the dataset to be reliable and suitable for use in Mineral Resource estimation. Further details on data verification procedures and limitations are provided in Section 12 in the main report.

1.11 Mineral Processing and Metallurgical Testing

Metallurgical testwork conducted on Grey Fox mineralization indicates that the ore is generally amenable to conventional gold recovery methods, with performance varying by domain and sulphide content. Section 13 of the main report provides detailed information on past and recent processing and testing of Fox Complex ores.

1.11.1 Comminution

Comminution testing shows that the ore is relatively competent and requires high grinding energy, with Bond Ball Mill Work Index values ranging from approximately 18 kWh/t to 27 kWh/t. The majority of the orebody corresponds to the higher end of this range, and process design has been based on conservative hardness assumptions. Additional variability testing is planned to further refine the hardness distribution.

1.11.2 Gravity Testing

Gravity recovery testwork conducted on open pit samples confirmed the presence of coarse free gold, with gravity recoverable gold (GRG) test values ranging between 4% and 32%. Extended gravity recoverable gold (EGRG) tests achieved mass pulls below 0.2% of new feed mass, with high gold grades recovered to a concentrate between 200 g/t and 1,200 g/t Au, confirming the presence of free coarse gold in the samples tested.

Underground samples tested in 2025 contained significantly less GRG compared to the open pit samples tested. For the underground ore, a gravity concentration step is not essential and can be omitted. Inclusion of a gravity recovery step is recommended only for treatment of open pit ores.

1.11.3 Flotation

Bulk sulphide flotation has been shown to be highly effective on Grey Fox ore, particularly for pyritic and high-sulphur ore types. Testing on open pit samples yielded gold recoveries ranging from 91% to 97% into an 8% to 12% mass pull concentrate. The highest recoveries exceeding 95% were achieved at finer grind sizes ($P_{80} = 60 \mu\text{m}$); however, similar recoveries were also achieved at coarser grind sizes (up to $P_{80} = 120 \mu\text{m}$). The flotation concentrates produced are primarily pyrite with lower amounts of base metals that are amenable to direct cyanidation. A carbon pre-float was ineffective but did not significantly impact overall flotation performance. Underground samples yielded marginally lower recoveries, with an average gold recovery of 87.2%.



For all samples tested to date, a gravity-flotation flowsheet outperformed a gravity-leach flowsheet by approximately 7% to 14% in terms of overall gold recovery. However, the rougher flotation concentrates did not achieve saleable concentrate grades as a standalone product and additional upgrading through cleaning stages would be required.

1.11.4 Cyanidation of Flotation Products

Cyanidation of flotation concentrates resulted in variable gold extraction, dependent on pyrite content and grain size. As-floated concentrates ($P_{80} \sim 60 \mu\text{m}$) yielded gold recoveries via cyanidation of approximately 77% to 82% for open pit samples. Regrinding to a P_{80} of approximately $25 \mu\text{m}$ improved gold extraction by approximately 2% to 4%, with no additional benefit observed at finer grind sizes, indicating that a portion of the gold is not accessible through cyanidation.

Underground samples showed greater variability and generally lower performance compared to open pit samples. Cyanidation of flotation tailings consistently yielded high extractions, with open pit samples achieving recoveries exceeding 74% and final tailings grades below 0.1 g/t Au, while underground samples yielded an average extraction of 72% and an average final tailings grade of 0.19 g/t Au.

A split-leach approach, comprising separate treatment of flotation concentrates and tailings, is supported by the testwork and has the potential to match or exceed whole-ore leach recoveries while providing flexibility in balancing recovery and grinding requirements.

1.11.5 Whole-Ore Cyanidation

Whole-ore leach testwork demonstrated a wide range of recoveries related to head grade and sulphur content. Higher-grade, lower sulphide mineralization achieved gold recoveries of approximately 88% to 94%, whereas lower-grade or higher sulphide material yielded reduced recoveries in the range of approximately 71% to 80%. Recent open pit testwork showed similar variability, with recoveries ranging from as low as 64% to as high as 92% and averaging approximately 80%.

Preg-robbing effects are minimal. Whole-ore cyanidation is considered viable for higher-grade, lower sulphide ores but is less suitable for high-pyrite or low-grade domains due to lower recoveries and high reagent consumption.

1.11.6 Cyanide Detoxification

Cyanide destruction using the SO_2 /air process is effective and reduces Weak Acid Dissociable (WAD) cyanide concentrations to below applicable regulatory criteria (Ontario Provincial Water Quality Objectives). Treated solutions generally meet water quality guidelines for cyanide and most metals, with minor molybdenum exceedances noted. The process is considered robust and scalable.

Underground samples have not yet been tested but are expected to respond similarly; additional testing is recommended to confirm this assumption.



1.11.7 Sedimentation

Settling tests show that underflow densities of approximately 65% solids (w/w) can be achieved using non-ionic flocculants. Dynamic settling and rheology testing indicate that conventional thickening and deposition strategies are applicable. The sedimentation behavior of the Whiskey Jack and Gibson Zone underground samples has not been tested. Future sedimentation testwork should include flotation concentrate samples to confirm the sizing of a concentrate thickener for the proposed flowsheet.

1.11.8 Environmental

Static geochemical testing of waste rock and tailings materials suggests that most samples are non-potentially acid generating (NPAG), with the remainder considered uncertain and subject to ongoing kinetic testing. Tailings residues from cyanidation and detoxification were classified as NPAG. Decant water chemistry met applicable discharge criteria, aside from minor molybdenum exceedances. No environmental testwork was conducted on underground samples as part of the 2025 program.

1.12 Mineral Resource Estimates

The Grey Fox Mineral Resource estimate, reported in accordance with the 2014 CIM Definition Standards and exclusive of Mineral Reserves reported in Section 15, has an effective date of July 11, 2025. The Mineral Resource estimate is presented in Table 1-1.

Table 1-1: Grey Fox Mineral Resource Estimate, 11 July 2025

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Indicated	9,681	2.25	701
Inferred	4,696	2.57	388

Notes:

- The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t Au, assuming underground extraction methods, and a metallurgical recovery of 90%, mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, variable royalty NSR from 0.15% to 3.0%, dilution of 15%, and a gold price of US\$3,000/oz.
- Figures may not sum due to rounding.

Inclusive of Mineral Reserves, the Grey Fox Mineral Resource estimate includes the totals summarized in Table 1-2.

Table 1-2: Grey Fox Mineral Resource, Inclusive of Mineral Reserves, 11 July 2025

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Indicated	18,819	3.28	1,986
Inferred	5,056	2.69	438

Notes:

- The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.



- Indicated resources are reported inclusive of reserves reported in Section 15.
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t gold assuming underground extraction methods and based on a mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, metallurgical recovery of 90%, variable royalty NSR from 0.15% to 3.0%, dilution of 15%, and a gold price of US\$3,000/oz.
- Figures may not sum due to rounding.

This report also includes a maiden Mineral Resource estimate for the nearby Stroud deposit, with an effective date of March 26, 2026. The Stroud estimate includes 1,866 kt at 2.80 g/t Au for 168 koz Au in the Inferred category. No Indicated Mineral Resources are reported for Stroud. Additional deposits within the Fox Complex have also been updated and are included in this report. In addition to Grey Fox and Stroud, Fox East includes Black Fox and Froome. Stock is divided into the Stock West, Stock Main, and Stock East, while Fox West includes Fuller and Davidson-Tisdale. Mineral Resource statements for these deposits are presented in Section 14.

1.13 Mineral Reserve Estimates

The Grey Fox Mineral Reserve estimate, prepared by Stantec and dated March 27, 2026, is based on Indicated Mineral Resources from the block model provided by McEwen. Material not classified as Measured or Indicated was excluded prior to Mineral Reserve estimation. CIM (2014) definitions were followed.

The Mineral Reserve includes longitudinal stopes, overhand cut-and-fill stoping, ore development, marginal longitudinal stopes, and marginal development, with allowances for external dilution, internal dilution, and mining recovery. Cut-off grades vary by mining method, with undiluted cut-off values of 2.40 g/t Au for longitudinal stopes, 2.00 g/t Au for marginal longitudinal stopes, and 2.70 g/t Au for cut-and-fill stopes. A minimum mining width of 3.0 m was applied for all stoping.

Total dilution is estimated at 46.5%, consisting of 11.6% external dilution and 34.9% internal dilution. Total mineable recovery is estimated at 91.2%.

Table 1-3 outlines the estimated Grey Fox Mineral Reserve.

Table 1-3: Grey Fox Mineral Reserve Estimate, March 27, 2026

Classification	Ore (kt)	Au Grade (g/t)	Contained Au (koz)
Proven	0	0.00	0
Probable	9,406.8	3.24	980.3
Total Reserve	9,405.8	3.24	980.3

Note: Some numbers may not sum due to rounding.

1.14 Mining Methods

Grey Fox will be developed as two underground operations: Grey Fox North, comprising the Whiskey Jack, Gibson, Contact, and 147NE zones, and Grey Fox South, comprising the 147 and South zones. The zones are illustrated in Figure 16-1. Ore will be mined using a combination of longitudinal retreat longhole



mining, supplemented by overhand cut-and-fill methods, and hauled to the existing Stock Mill at the Stock Property.

Mining will occur in two phases. Phase 1 (Grey Fox North) begins with development to access the Whiskey Jack Zone, with operations ramping up to approximately 1,500 t/d by 2030. Phase 2 (Grey Fox South) commences in 2035, with combined production reaching approximately 2,700 t/d before ramping down through 2041.

1.14.1 Geomechanical

Geomechanical assessments indicate that most stopes are expected to be stable under unsupported conditions, with local support or modified sequencing required at depth. Pillar design and crown pillar requirements are considered adequate to maintain stability, with conventional ground support systems appropriate for the rock mass conditions. More detailed information is provided in Section 16.1.

1.14.2 Hydrogeological Considerations

A hydrogeological model prepared by Wood (Wood 2022) was updated to support an original business case that included an open pit, for conservatism. This model was subsequently used to assess the all-underground mining scenario. Model sensitivity results indicate that long-term groundwater inflow to the underground workings may be 1,700 m³/d, which can be managed using conventional mine dewatering systems.

1.14.3 Mine Design and Mining Methods

Development will generally utilize drift dimensions of approximately 4.0 m W x 4.3 m H with 1 m arch corners for capital development and 3.5 m W x 4.0 m H with 1 m arch corners for operating development. Production mining will comprise a combination of longitudinal longhole stoping and overhand cut-and-fill methods. Longhole stopes are typically 25 m to 30 m high and 10 m to 40 m in length (25 m average), while cut-and-fill stopes are typically 3 m to 5 m in height. A minimum mining width of 3 m will be applied for both methods. Cemented rock fill or unconsolidated rock fill will be used as required to support adjacent stoping.

Stoping sequences have been designed to minimize interaction between stopes and maintain overall ground stability. Most of the mineral resource will be extracted using the longitudinal longhole mining method, as summarized in Table 1-4.

Table 1-4: Ore Tonnes by Mining Method

Mining Method	Grey Fox North Recovered Ore Tonnes	Grey Fox South Recovered Ore Tonnes	Recovered Ore Tonnes
Longitudinal Longhole	4,713,651	3,009,295	7,722,946
Cut and Fill	55,165	62,819	117,984
Development	921,239	643,660	1,564,899
Total	5,690,055	3,715,774	9,405,829



1.14.4 Mine Production Schedule

The initial focus at Grey Fox will be the extraction of higher-grade material from the Whiskey Jack Zone. Mining will ramp up to 1,500 t/d within Grey Fox North, and subsequently to 2,700 t/d with the addition of Grey Fox South in 2035. See Figure 1-1 for the production schedule.

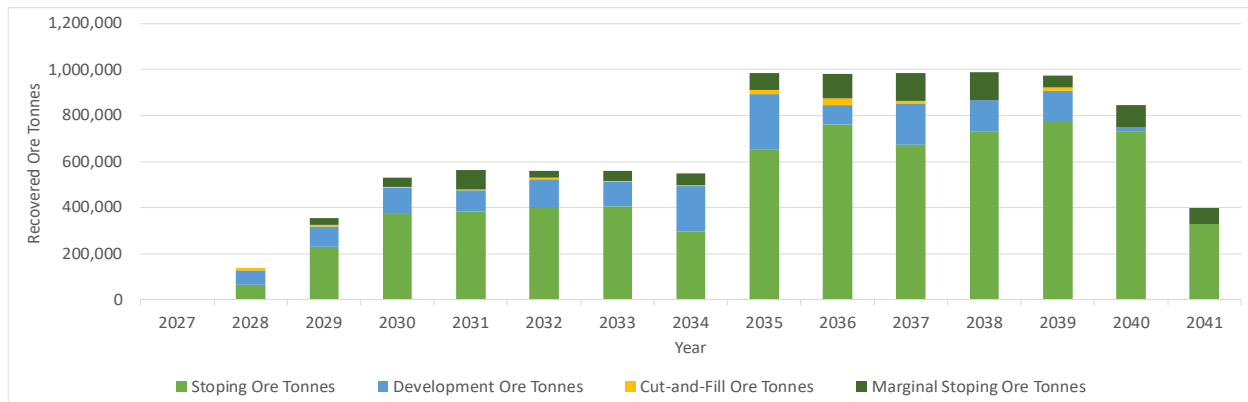


Figure 1-1: Production Profile by Mining Method

1.14.5 Mobile Equipment Selection

Conventional trackless underground mining equipment will be used for lateral development required to access the zones. Longhole stopes will be drilled using a combination of In-the-Hole (ITH) and top hammer drills, blasted using ANFO and emulsion explosives, and mucked using diesel load haul dump (LHD) vehicles. Ore will be hauled to surface using trucks, and waste rock will be hauled either to surface or to an active production level for use as stope backfill.

1.14.6 Ventilation

The underground mine ventilation system is designed to be a “push-pull” system, with fresh air supplied through raises and portals and exhaust air removed through return air raises. Surface heaters will be utilized at the fresh air raise as required for winter operation.

1.14.7 Mine Dewatering

Mine dewatering will be managed using a staged, multi-level sump and pump system designed to collect groundwater inflows and service water and transfer it to surface. Main sump and pump stations will be placed approximately every 100 m. An additional sump will be placed at the portal entrances to collect surface run-off water that enters the mine via the ramp or portal. A network of borehole sumps and level sumps will be fed to these main sumps. All sumps will collect any rock mass inflow water and any service water used during mining activities.



1.14.8 Backfill Distribution

Stopes will be backfilled with a combination of cemented rock fill and unconsolidated rock fill to reduce the use of cement while still minimizing backfill dilution in adjacent stopes. For upper stopes, backfilling is not planned and rib pillars may be left in place during mining to support the voids. A cement slurry will be hopper-fed from surface through boreholes to an underground slurry receiving station. The slurry receiving station will be used to distribute the slurry to each level, and spray bars will be used to add the slurry to waste rock in LHD buckets for placement in open stopes. When possible, waste rock will be sourced directly from active development underground.

1.14.9 General Underground Infrastructure

All personnel and materials will be transferred to and from the underground workings via ramp, with emergency egress supported by a ladder system located in the fresh air raise to surface. A series of dedicated internal escapeways will lead to this emergency egress. Refuge stations will be placed approximately every 100 m for access during an emergency.

Underground storage magazines will be strategically placed near the bottom of each mining block to support operations.

Underground workshops will be strategically placed within the underground development: one in Whiskey Jack Zone, two in the Gibson Zone, and two in Grey Fox South.

1.14.10 General Underground Services

Underground services will include compressed air, service water, potable water, and fire protection systems, supported by surface infrastructure and distributed throughout the mine.

1.14.11 Underground Electrical and Power Distribution

Electrical power will be provided via four 5 kV surface feeders and distributed underground through a network of electrical rooms. Peak power demand is estimated at approximately 20,589 kVA.

1.14.12 Underground Communication and IT

The underground communication and IT system will consist of a fibre optic backbone, a 5G wireless network, a leaky feeder radio system, and analog telephones, routed via ramps, drifts, or boreholes to optimize coverage and maintain separation from power infrastructure for safety and reliability.

The fibre optic network will serve as the primary high-speed data backbone supporting business, process control, and other critical systems. Building on this backbone, the 5G wireless network will provide the main communication platform underground, enabling voice and data connectivity for mobile devices, equipment telemetry, personnel tracking, and tele-remote operations.

A leaky feeder system installed along ramps and level access drifts will provide supplementary radio coverage, particularly during early development. In addition, analog telephones will be installed in electrical



rooms and refuge stations to provide a simple, reliable backup communication system for safety and operational redundancy.

1.15 Recovery Methods

1.15.1 Process Design Basis and Criteria

Metallurgical testwork demonstrated variable ore characteristics, with whole-ore cyanidation recoveries ranging from approximately 70% to 95% at a grind size of 80% passing 75 µm. Comminution testing indicated that the material is hard and competent, available grinding power at the Stock Mill represents the primary constraint on overall throughput.

Testwork also indicted that flotation with concentrate regrinding could improve recoveries for some domains; however, this option was not incorporated into the flowsheet due to cost and operational complexity.

The proposed flowsheet therefore utilizes a new modular primary crusher at the Grey Fox portals to produce a sub-120 mm product suitable for haulage to the Stock Mill, where processing will occur using the existing crushing, grinding, and leaching circuits. The operation is designed for a nominal throughput of approximately 1,470 t/d, with overall gold recovery of approximately 88%.

1.15.2 Process Description

The process flowsheet comprises primary crushing at Grey Fox, followed by transport to the Stock Mill for secondary and tertiary crushing, two-stage grinding and classification, and cyanidation in the existing conventional carbon-in-lead circuit.

Gold will be recovered onto activated carbon, followed by elution, electrowinning, and smelting to produce doré using existing facilities.

1.16 Project Infrastructure

1.16.1 Surface

The Grey Fox Project will consist of a compact, centralized surface infrastructure supporting underground operations via the North and South portals. Core infrastructure includes ore and waste stockpiles, a primary crushing facility, water management systems, ventilation plants, and key support buildings such as administration offices, mine dry, maintenance shop, warehouses, and a security gatehouse.

Site access is provided via Tamarack Road connecting to Highway 101, with ore hauled approximately 34 km to the Stock Mill. Electrical power will be supplied through a 27.6 kV grid connection along Tamarack Road.



1.16.1.1 Integration with Existing Infrastructure

The project benefits from proximity (~3 km) to the operating Black Fox Mine site, enabling use of existing infrastructure for early development, including power supply, maintenance facilities, stockpiles, water management systems, and administrative services. This integration reduces initial capital requirements and supports a phased development approach.

1.16.1.2 Site Preparation and Access

Initial site preparation includes clearing, grading, and stockpiling of soils for future reclamation, along with establishment of construction staging and laydown areas. Tamarack Road will serve as the primary access route and requires minimal upgrades; however, safety improvements at the Highway 101 intersection such as turn lanes and lighting are recommended. The existing Tamarack Road bridge is structurally adequate for standard traffic but requires deck strengthening or replacement to support heavy mining vehicles.

1.16.1.3 Surface Facilities

Key surface facilities include a maintenance shop, warehouses, administration and mine dry complexes, fuel storage, and site security and waste management systems.

Additional infrastructure includes laydown areas, a pumphouse for water supply and transfer, and domestic water and sewage services.

1.16.1.4 Ore and Waste Management

Separate ore and waste areas will be developed. Ore will be crushed on site and hauled to the Stock Mill, while waste rock will be stockpiled and later used as underground backfill. Additional waste from the Black Fox mine site may be used to supplement backfill requirements.

1.16.1.5 Water Management

Water management systems are designed to collect and manage contact water from stockpiles, underground operations, and site facilities, with transfer to the Black Fox site for treatment. Stormwater will be managed using pond facilities designed for major events, along with drainage ditches, sumps, pumps, and pipelines to control runoff and minimize environmental impacts.

A site-wide water balance has been developed to support system design, with underground dewatering expected to reach approximately 6,540 m³/d at full production.

1.17 Market Studies and Contracts

Gold is a freely traded commodity with widely established pricing, and the sale of production is considered assured. As such, a detailed market study was not undertaken.

Based on monthly closing data from April 2024 to April 2026, the approximate trailing average gold price is **US\$3,000 per ounce**.



No sales contracts are currently in place for Grey Fox. Future doré sales are expected to follow industry terms.

Refining and transportation costs have been estimated based on comparable operations in the region, with typical refining charges, high gold payability (approximately 99.95%), and nominal shipping costs applied in the economic analysis. Further details regarding market assumptions, pricing, and sales terms are provided in Section 19.

1.18 Environmental Studies, Permitting, and Social or Community Impact

The Grey Fox Project is located near Matheson, Ontario, in an established mining region of the Abitibi Greenstone Belt. The Project benefits from existing mining infrastructure, a well-defined regulatory framework, and extensive environmental baseline data from current and historical operations.

Baseline environmental studies indicate a typical northern boreal forest environment with mixed woodland, wetlands, and glaciolacustrine soils. Surface water and groundwater systems are well defined and monitored, with conditions generally meeting applicable regulatory standards. Ecological surveys indicate that plant and wildlife communities are common to the region, and no federally listed Species at Risk identified within the Project area.

A site-wide water balance model has been developed to assess water management requirements associated with the integration of Grey Fox operations with existing Black Fox facilities over the Project life. Contact water from Grey Fox will be transferred to the Black Fox site for centralized treatment, with the open pit providing additional storage capacity to buffer seasonal fluctuations in water volumes. While certain metals, including arsenic, are expected to exceed applicable discharge criteria without treatment, established treatment methods are expected to effectively manage water quality.

The Project is subject to federal and provincial regulatory requirements. Based on current design, a federal Impact Assessment is not required. Key provincial approvals include Environmental Compliance Approvals (air, noise, and water), a Permit to Take Water (PTTW), and a Closure Plan Amendment. The Grey Fox PTTW, classified as a Category 3 permit, and the Closure Plan Amendment represent the critical path items for permitting, with typical approval timelines expected.

Environmental management will focus primarily on water collection, diversion, and treatment, with infrastructure designed to control runoff and minimize impacts to natural waterbodies and fish habitat.

The Project is located within areas traditionally used by Indigenous communities, and engagement has been ongoing with local First Nations and Métis organizations. An Impact Benefit Agreement is in place with Apitipi Anicinapek Nation, and continued engagement with Indigenous communities, regulators, and local stakeholders will remain a key component of Project development.

Closure and reclamation activities have been defined in a 2026 Closure Plan Amendment and include flooding and sealing of underground workings, stabilization of open pits, regrading and revegetation of waste rock storage areas, and removal of infrastructure. Closure objectives are to achieve long-term site stability and return the land to a condition suitable for forest and wildlife habitat.



Overall, environmental baseline conditions, existing infrastructure, and permitting experience at the Black Fox mine indicate that the Grey Fox Project is environmentally and socially viable. Key risks related to water treatment, permitting timelines, and stakeholder engagement are well understood and considered manageable through established mitigation measures and ongoing consultation.

Further details regarding environmental studies, permitting, and social or community impacts, are provided in Section 20.

1.19 Capital and Operating Costs

1.19.1 Capital Costs

The capital cost estimate meets the requirements for a Class 3 estimate (AACE), with an expected accuracy of $\pm 15\%$, and is reported in Q1 2026 US dollars.

Total capital costs are estimated at \$350.3 million, comprising \$176.6 million in project period capital and \$173.7 million in sustaining capital. Including pre-production and closure costs, total capital is estimated at approximately **\$361.9 million**.

Capital costs are primarily associated with underground mine development and infrastructure, with additional costs related to surface facilities, water treatment systems, contingency, and closure.

1.19.2 Operating Costs

LOM operating costs are estimated at approximately **\$1,397.7 million**, or **\$148.6 per tonne processed**. Operating costs are dominated by underground mining, with additional contributions from processing, tailings management, and water treatment.

Further details regarding capital and operating costs are provided in Section 21.

1.20 Economic Analysis

An economic model was prepared to estimate annual project cash flows based on the engineering, cost, and revenue assumptions for the Project. The economic analysis results are based on forward-looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented herein.

Using a base case discount rate of 5%, the Project returns an after-tax net present value (NPV) of \$282.1 million, an after-tax internal rate of return (IRR) of 24.8%, and after-tax cumulative cash flow of \$493.2 million.

Sensitivity analysis evaluated the impact of changes in key parameters, including gold price, head grade, operating costs, and capital costs. The Project is most sensitive to variations in head grade and gold price.

Further details regarding the economic analysis are provided in Section 22.



1.21 Adjacent Properties

The Grey Fox deposit is located proximal to the Porcupine-Destor fault zone in the Abitibi Greenstone Belt, an area well known for historical and current gold production. The Eastern Properties, which include Grey Fox, are surrounded by claims and leases held by exploration companies and operators, including Onyx Gold Inc., STLLR Gold Inc., Mayfair Gold Corp., and Agnico Eagle Mines Ltd.

Further details regarding adjacent properties are provided in Section 23.

1.22 Interpretation and Conclusions

The QPs conclude that the Grey Fox Project is technically robust and appropriately defined at the PFS level. McEwen holds a 100% interest in the mineral tenure, with surface rights, royalties, and stakeholder agreements considered typical and sufficient to support development, with no material constraints identified.

Geologically, Grey Fox is interpreted as a structurally controlled epizonal gold system hosted in metavolcanic, metasedimentary, and intrusive rocks. Mineralization is primarily controlled by vein arrays, major fault structures, and lithological contrasts, with multiple domains requiring consideration in modelling and mine planning. Drilling density is generally sufficient, with additional work recommended to improve confidence and support resource conversion.

Metallurgical testwork indicates relatively hard ore with variable recoveries (71% to 94%), and modest gravity recovery potential, highlighting the importance of comminution design and geometallurgical domaining. Mineral Resources are considered reliable and support the defined Mineral Reserve of 9.4 Mt at 3.24 g/t Au (approximately 980 koz), based on a gold price of \$2,200/oz.

Mining methods, infrastructure, and water management strategies are conventional and supported by operating experience in the region, with optimization opportunities identified in areas such as tailings management and water treatment.

Economically, the Project demonstrates positive results, including an after-tax NPV (5%) of \$282.1 million and IRR of 24.8%, with sensitivity primarily to gold price and head grade. Overall, the Project is considered viable at the PFS level, with further work focused on reducing geological and metallurgical uncertainties and optimizing design assumptions.

Further details regarding interpretation and conclusions are provided in Section 25.

1.23 Recommendations

The Grey Fox Project is recommended to advance to the next stage of development, supported by targeted technical programs to further de-risk and optimize the Project.



Key recommended programs include:

- **Early Works:** Surface geotechnical investigations, confirmation of overburden conditions, and refinement of portal and ventilation raise designs.
- **Exploration:** Continued diamond drilling to expand and upgrade resources, particularly in the Gibson, Whiskey Jack (at depth), Grey Fox South, and 147 zones, supported by geochemical and structural analysis.
- **Mining:** Increased geotechnical characterization, optimization of stope design and sequencing, and evaluation of alternative development strategies and automation.
- **Metallurgy:** Additional testwork to optimize recovery and confirm processing assumptions.
- **Tailings:** Assessment of alternative tailings storage options aligned with GISTM best practices, including potential use of the Black Fox open pit.
- **Infrastructure:** Site investigations and confirmation of integration with existing Black Fox infrastructure.
- **Hydrogeology:** Updating of the hydrogeological model to reflect an underground-only mining scenario and validate groundwater inflow assumptions.

The estimated cost of these programs is approximately **\$13.5 million**.

Further details regarding recommendations are provided in Section 26.



2 Introduction

2.1 Terms of Reference

At the request of McEwen, this Technical Report was prepared to support the development of an underground mining operation at the Grey Fox property near Matheson, Ontario. McEwen is headquartered in Toronto, Ontario, is legally domiciled and incorporated in the State of Colorado, and trades on the Toronto and New York Stock Exchanges under the symbol MUX. McEwen holds a 100% interest in the mineral claims and mining leases associated with the Grey Fox project (Grey Fox).

The purpose of this report is to summarize the findings of the PFS completed for the project as of the effective date given in Section 2.3.

The user of this report is encouraged to confirm that this is the most recent Technical Report for the Grey Fox project, as it is superseded upon issuance of a newer Technical Report.

The quality of information contained within this report is considered consistent with the level of effort associated with the services provided and is based on the following:

- Data supplied by third-party sources.
- Information available at the time of preparation.
- Qualifications and assumptions presented within this report.

This Technical Report includes the following key activities:

- Review and acceptance of the current resource model for the Grey Fox project.
- Preparation of an underground mine design and production schedule using previous hydrogeology modelling and geotechnical information prepared by Stantec.
- Preparation of a Mineral Reserve estimate using appropriate modifying factors.
- Development of a metallurgical flowsheet for processing Grey Fox material at the existing Stock Mill.
- Preparation of specifications for supporting project infrastructure, including access, power supply, water treatment, ventilation, buildings, and tailings storage facilities.
- Summary of the environmental baseline work and permitting requirements.
- Development of capital and operating cost estimates.
- Preparation of an economic model, including discounted cash flow and sensitivity analyses.
- Development of recommendations for further work required for the continued development of the Project.

The presentation of this information requires qualified persons (QPs) to derive subtotals, totals and weighted averages that may inherently involve rounding, and consequently, introduce a margin of error. These are not considered material.

This report provides Mineral Resource and Mineral Reserve estimates prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources



and Mineral Reserves (May 10, 2014). Inferred Mineral Resources are considered too speculative geologically to be included in Mineral Reserves. The Mineral Reserve estimate presented herein represents the first public disclosure of Mineral Reserves for the Project.

2.2 Site Visits and Scope of Personal Inspection

By virtue of their education, experience, and professional qualifications, the authors of this Technical Report are QPs in the relevant disciplines, including resource estimation, mining, geology, metallurgy, engineering, cost estimation, and mineral economics.

Table 2-1 lists the individual QPs responsible for preparation of this report, their titles, areas of responsibility, and the dates of site visits. QP certificates are provided in Section 28 of this report.

Table 2-1: Details of QPs and Site Visits

Qualified Person	Title	Area of Responsibility	Site Visit Date
Mark Hatton, P.Eng. Stantec	Senior Mining Engineer	Sections 1.1, 1.13, 1.14, 1.16, 1.17, 1.19, 1.20, 1.22, 1.23, 2, 3, 15, 16, 18.1 to 18.7, 19, 21.1, 21.2, 21.2.1, 21.2.4 to 21.3.1, 22, 24, 25.7 to 25.9, 25.12 to 25.15, 26.1.1, 26.1.3, 26.1.6, 26.2, 27, 28	Oct 7, 2025
Carson Cybolsky, P.Geo McEwen	Senior Resource Geologist	Sections 1.8 to 1.10, 1.12, 6.4.1, 10.11, 10.12, 11, 12.1 to 12.2, 14.1 to 14.8, 25.6	Actively Works On-Site
Sean Farrell, P.Geo. McEwen	Exploration Manager	Sections 1.2 to 1.7, 1.21, 4, 5, 6.1 to 6.4, 6.5, 7 to 9, 10.1 to 10.10, 23, 25.1 to 25.4, 26.1.2	Actively Works On-Site
Luke Willis, P. Geo. McEwen	Director of Resource Modelling	Sections 14.9, 14.10	Jan 17, 2025
Gerd Wiatzka, P. Eng. Arcadis	National Expert, VP, Director of Mining	Sections 12.3 to 12.5, 18.10 to 18.12, 21.2.3, 21.3.3, 21.3.4, 25.10, 25.11, 26.1.5	Oct 7, 2025
Niel Morrison, P.Eng. DRA	Principal Process Engineer	Sections 1.11, 1.15, 12.6, 13, 17, 18.8, 21.2.2, 21.3.2, 25.5, 26.1.4	Feb 26 2026
Sheldon Smith, MES P.Geo. Stantec	Senior Hydrologist	Sections 1.18, 18.9, 20, 26.1.7	N/A



2.3 Effective Dates

The effective date of the Grey Fox Mineral Resource estimate is July 11, 2025.

The effective date of the Mineral Reserve estimate is March 27, 2026.

The effective date of this Technical Report is June 8, 2026.

2.4 Information Sources and References

Key sources of information for this report include:

- Referenced documents listed in Section 27.
- Information provided by the McEwen management team, as described in Section 21 and 22.

2.5 Units of Measure

All currency units and references to “dollars” or “\$” are in United States dollars unless otherwise noted.

The metric system is used throughout. References to “t” or “tonnes” refer to dry metric tonnes unless otherwise noted.

2.6 Previous Technical Reports

- “Technical Report for the Black Fox Complex, Canada” April 2018 by SRK for McEwen Mining
- “NI 43-101 Technical Report on the Preliminary Economic Assessment of the Fox Complex, Ontario, Canada.” January 2022 by Wood for McEwen Mining
- “NI 43-101 Technical Report on the Mineral Resources Estimate of the Fox Complex, Ontario, Canada.” October 2025 by McEwen Mining



3 Reliance on Other Experts

3.1 Mineral Tenure, Surface Rights, and Encumbrances

In this Technical Report, information regarding mineral tenure, surface rights, royalties, permitting, taxation, and environmental matters is based on material provided by McEwen. The QPs are not qualified to comment on such matters and have relied on the representations and documentation provided by McEwen.

All data used in this report were initially provided by McEwen. The QPs have reviewed and analyzed these data and have drawn independent conclusions based on their professional judgment.

The QPs offer no legal opinion as to the validity of title to the mineral claims and mining leases and have relied on information provided by McEwen in that regard.

3.2 Taxation

The QPs have relied upon, and disclaim responsibility for, taxation information provided by McEwen and its retained experts. Invoices were compiled, examined and cross-referenced for payment confirmation or account statements were requested by the governing body to ensure payment compliance.

This information supports the economic analysis presented in Section 22.

3.3 Royalties

The QPs have relied upon, and disclaim responsibility for, information provided by McEwen regarding royalties applicable to the Project, including:

- Parsons Ginn Royalty (5% net profits interest (NPI))
- Triple Flag Royalty (formerly Newmont Royalty) (2.5% net smelter return (NSR))
- Frederick W. Schumacher Royalty (3% NSR)
- Gray Royalty (0.15% NSR)

This information supports the economic analysis in Section 22 and the Mineral Reserve estimate presented in Section 15.



4 Property Description and Location

The Fox Complex comprises three groups of properties, including past-producing and currently operating mines, as well as a processing facility, located within and around the City of Timmins, in northeastern Ontario, Canada (Figure 4-1).

The Eastern Properties include the currently operating Froome Mine (Froome and Froome West), the Grey Fox property, the Stroud property, Black Fox North, and the formerly operating Black Fox Mine. These properties are located in Beatty and Hislop Townships, approximately 60 km east of Timmins.

The approximate geographic centre of the Black Fox Mine is 48° 32' 2" N and 80° 20' 2" W (UTM coordinates: 549170 E and 5375871 N, NAD83, Zone 17). Black Fox North is located approximately 4 km north of the Black Fox Mine. The Grey Fox property is located approximately 3 km southeast of Black Fox, and the Stroud property is contiguous with the southwestern portion of Grey Fox.

The approximate geographic centre of the Grey Fox property is 48°30'20.0" N and 80°18'20.0" W (UTM coordinates: 551100 E and 5372750 N, NAD83, Zone 17). Surface elevations in the area range from approximately 275 m to 325 m above sea level.

The Stock Properties include the historical Stock Mine, the Stock West and Stock East deposits, and the currently operating Stock Mill. These are centrally located within Stock Township, north of Highway 101, approximately 20 km west of Matheson and 43 km east of Timmins.

In 2024 McEwen started infrastructure-related operations at the historical underground Stock Mine. The approximate geographic centre of Stock is 48° 33' 0" N and 80° 45' 1" W (UTM coordinates: 518421 E and 5377476 N, NAD83, Zone 17).

The Western Properties include the Buffalo Ankerite, Paymaster, Fuller, and Davidson-Tisdale deposits, located in Tisdale and Deloro Townships within the municipal boundaries of Timmins. These deposits are past producers and form part of the historic Porcupine Gold Camp, which has produced more than 70 Moz of gold over the past 110 years.

The approximate geographic centres are as follows:

- Buffalo Ankerite: 48° 26' 31" N and 81° 16' 14" W (UTM: 479994 E and 5365448 N).
- Paymaster: 48° 27' 28" N and 81° 15' 42" W (UTM: 479415 E and 5366605 N).
- Fuller: 48° 27' 8" N and 81° 16' 42" W (UTM: 479415 E and 5366605 N).
- Davidson-Tisdale: 48° 30' 46" N and 81° 13' 40" W (UTM: 483186 E and 5373326 N).

(All coordinates are in NAD83, Zone 17.)



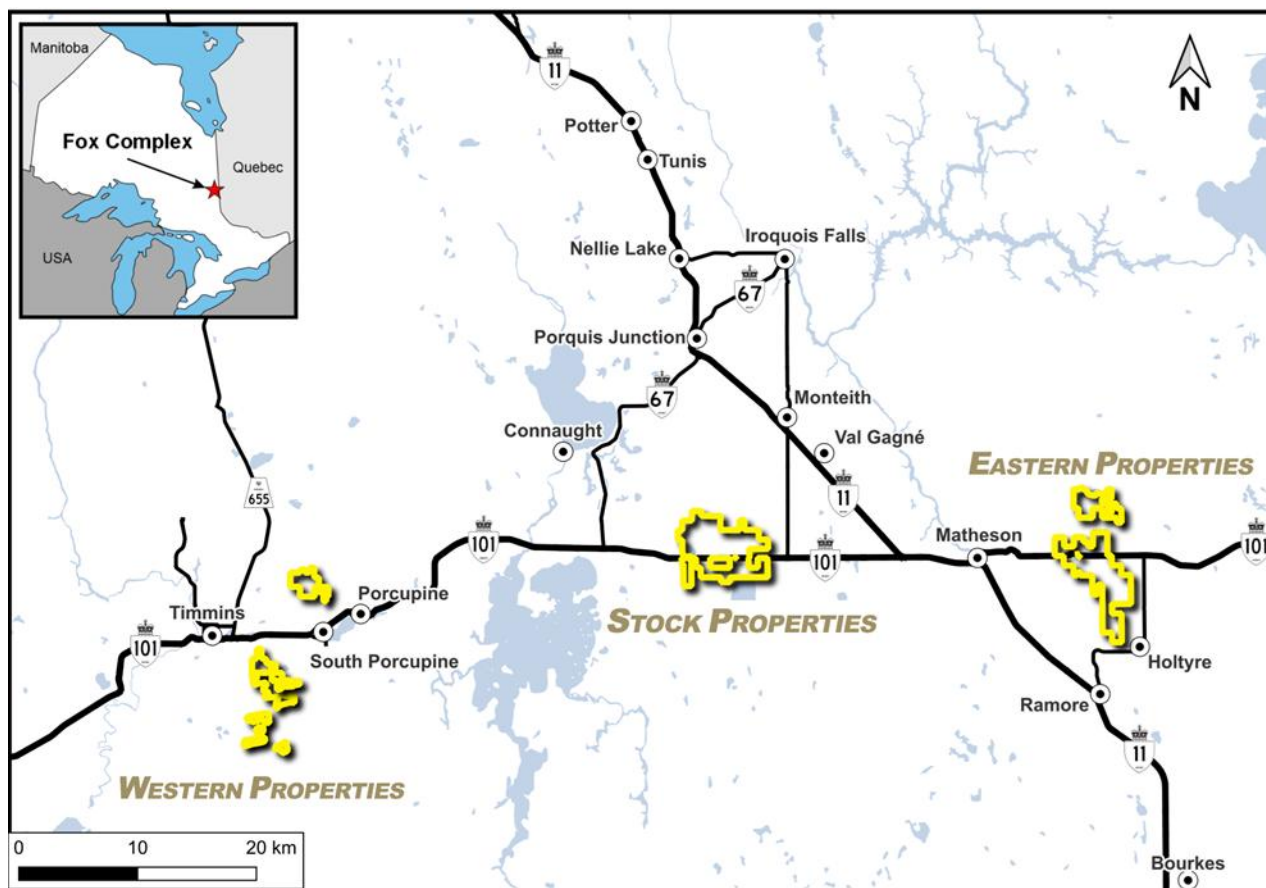


Figure 4-1: Fox Complex Properties

4.1 Mineral Tenure and Surface Rights

The properties of the Fox Complex consist of several blocks of land totaling 135 parcels, representing either patented mining claims, mining leases, and overlapping surface rights, covering approximately 7,234 ha of mining land and 3,461 ha of surface land.

All surface rights and mining rights parcels are located in the Tisdale, Deloro, Stock, Bond, Beatty, and Hislop townships, with boundaries defined by the Cochrane (06) Land Registry Office. The parcels are one of the following: a freehold mining land (mining patent), an unpatented mining claim, a mining lease, a freehold surface land, or a surface lease.

In the figures that follow, parcels with surface holdings without mining rights are shown in green; parcels with mining rights without surface rights are shown in gray, and parcels with both surface and mining rights are shown in orange.

All required fees and duties have been paid, and the claims are in good standing.

The Property Identification Number (PIN) is a numeric identifier issued by the Land Registry Office that references registered transactions associated with the land.



4.1.1 Eastern Properties

McEwen holds a 100% ownership interest in the Eastern Properties, located in Hislop and Beatty townships, covering Grey Fox, Stroud, Froome, Black Fox, and Black Fox North deposits (Figure 4-2).

The Eastern Properties comprise 52 parcels with surface and/or mining rights, including 10 mining leases and 3 leased patents, covering approximately 2,184 ha (5,396 ac). The Black Fox North property consists of 50 unpatented claims totaling approximately 650 ha (1,606 ac).

4.1.1.1 Grey Fox

The entire mineralized zone for the Grey Fox Project, including Gibson, is situated within the boundaries of the following PINs: 65380-0489, 65380-0556, 65380-0636, 65380-0490, 65380-0491 and 65380-0498. The property is located approximately 3 km southeast of the Black Fox Mine.

4.1.1.2 Stroud

The Stroud property is located south of, and contiguous with the Grey Fox property and lies within the boundaries of the following PINs: 65380-0422, 65380-0443, 65380-0444, 65380-0445, 65380-0465 and 65380-0466. These parcels are patented mining claims for which McEwen owns both surface and mining rights, with the exception of surface rights for PINs 65380-0441 and 65380-0442.

4.1.1.3 Froome Mine (Main Zone & West Zone)

The Froome Main Zone is situated within the boundaries of the following PINs: 65366-0143, 65380-0552 and 65380-0553. The Main Zone is situated approximately 700 m west of the Black Fox Mine open pit. The Froome West Zone is located within PIN 65366-0143, approximately 200 m west of the Main Zone.

4.1.1.4 Black Fox Mine

The Black Fox Mine open pit and underground mine are located within the boundaries of PIN 65380-0556, with the associated mine complex spanning PINs 65380-0676, 65380-0556 and 65380-0670.

4.1.1.5 Black Fox North

The Black Fox North property consists of 50 unpatented mining claims, totaling approximately 650 ha (1,606 acres), located approximately 4 km north of the Black Fox Mine. McEwen does not hold surface rights over this property (Figure 4-2).

4.1.2 Stock Property

McEwen holds a 100% ownership interest in the Stock Property, which comprises 31 parcels, including 14 patented parcels with surface and/or mining rights and 17 mining leases, covering approximately 1,633 ha in Stock Township (Figure 4-3).



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The Stock Mill and related facilities are situated within the boundaries of the following PINs: 65363-0060, 65363-0061, 65363-0085, 65363-0086, 65363-0087, 65363-0088, 65363-0232 and 65363-0241.

The entire Stock West mineralized zone is situated within the following PINs: 65363-0061, 65363-0237, and 65363-0240, approximately 1 km southwest of the Stock Mill.

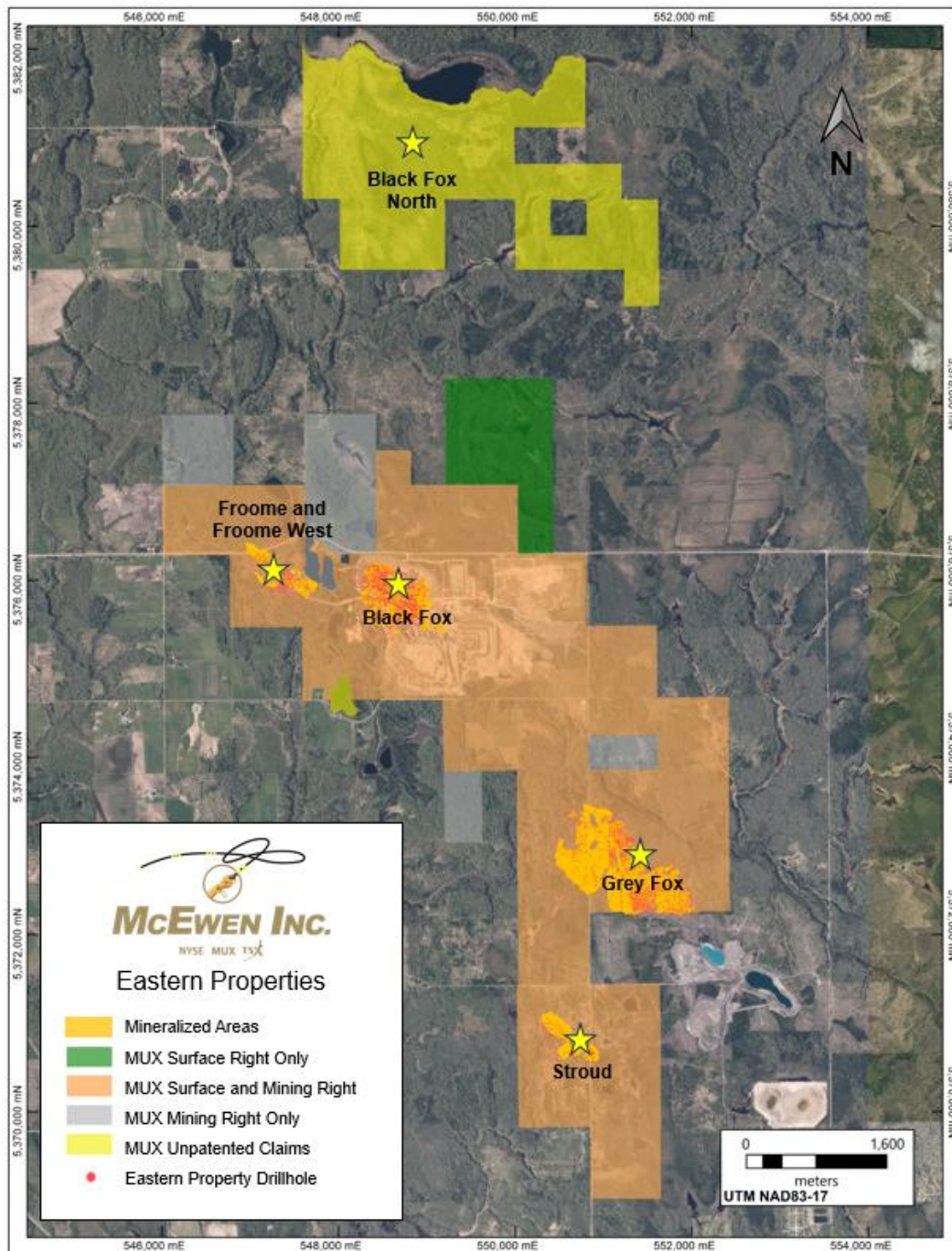


Figure 4-2: Eastern Properties



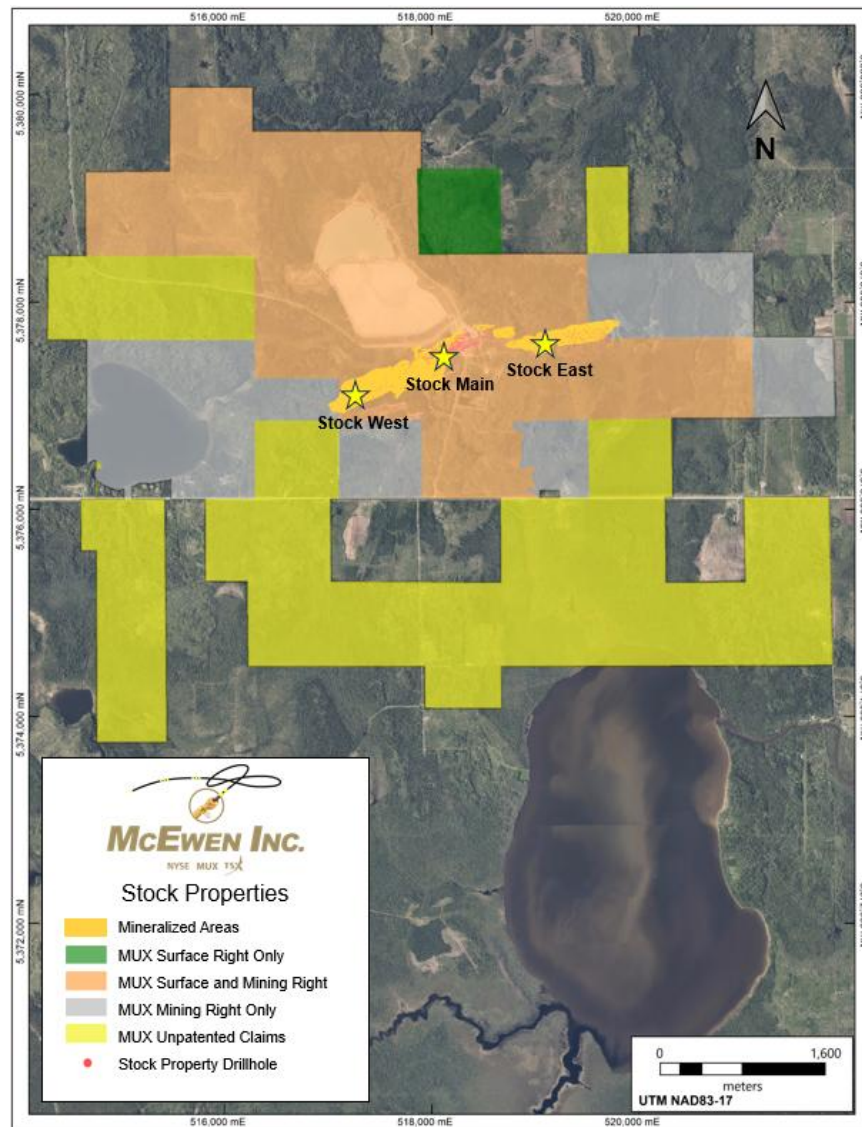


Figure 4-3: Stock Properties

4.1.3 Western Properties

The Western Properties are located in several parcel blocks, including the Davidson-Tisdale property in the northern part of Tisdale Township and a second block of contiguous parcels comprising the Buffalo Ankerite, Fuller, and Paymaster properties to the south in Tisdale and Deloro townships. Surface and mining rights for the northern parcel blocks are shown in Figure 4-4, and those for the southern parcel blocks are shown in Figure 4-5.

4.1.3.1 Buffalo Ankerite

McEwen holds a 100% interest in the Buffalo Ankerite property, which consists of 17 parcels with surface and/or mining rights, covering approximately 654 ha in Deloro Township. The property boundaries are



based on field surveys, historical records, and parcel mapping from the Ministry of Mines' Mining Lands Administration System (MLAS).

The mineralized zone for the Buffalo Ankerite deposit is located within the following PINs: 65442-0714, 65442-0717, 65442-0718, and 65442-0860.

4.1.3.2 Fuller

McEwen holds a 100% interest in the Fuller property, consisting of four parcels with surface and/or mining rights, covering an area of approximately 210 ha in Tisdale Township within the Porcupine Mining Division.

Access to the underground workings was established via a ramp constructed in the 1980s, collared on a single patented claim P13189 (PIN 65410-0069), for which McEwen holds both surface and mineral rights. In 2008, McEwen acquired the mineral rights for the Chisholm Property (PIN 65410-0085) from Goldcorp Inc. (Goldcorp). In exchange, McEwen transferred surface rights for five Fuller claims (P13099, P13100, P13313, P13314, and P13084) to Goldcorp.

The entire mineralized zone is situated within the boundaries of PIN 65410-0069 and 65410-0071.

4.1.3.3 Paymaster

McEwen holds a 61% interest in the Paymaster property, with the remaining 39% held by Discovery Mining Limited. Paymaster consists of 15 contiguous mining rights parcels covering approximately 179.2 ha, along with one surface rights lease.

Two mining right parcels are located in the south-central part of Tisdale Township, with the remaining parcels located in the north-central part of Deloro Township. The joint venture interest applies to mineral rights above the 4,750 level below surface.

The mineralized zone is situated within the boundaries of the following PINs: 65398-0284, 65398-286, 65442-0580, 65442-0793, 65442-0795, 65442-0799, 65442-0801, 65442-0803, 65442-0805, 65442-0807, 65442-0809, 65442-0811, 65442-0813, and 65442-0815.

4.1.3.4 Davidson-Tisdale

McEwen holds a 100% interest in the Davidson-Tisdale property, which consists of 14 parcels with surface and/or mining rights, covering approximately 448 ha in Tisdale Township.

The mineralized zone is situated within the boundaries of the following PINs: 65399-0129, 65399-0130, and 65399-0133.



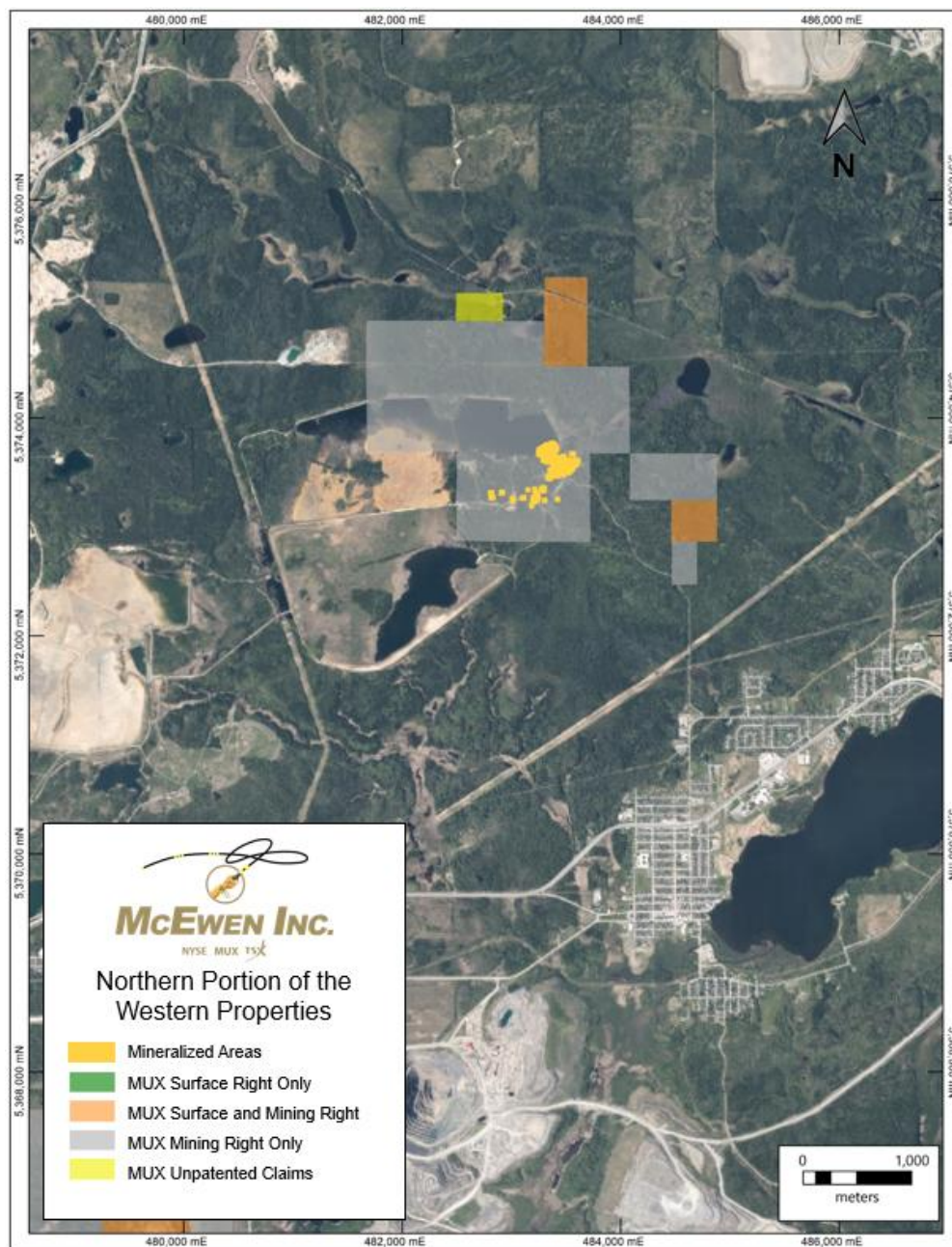


Figure 4-4: Western Properties North Parcel Blocks

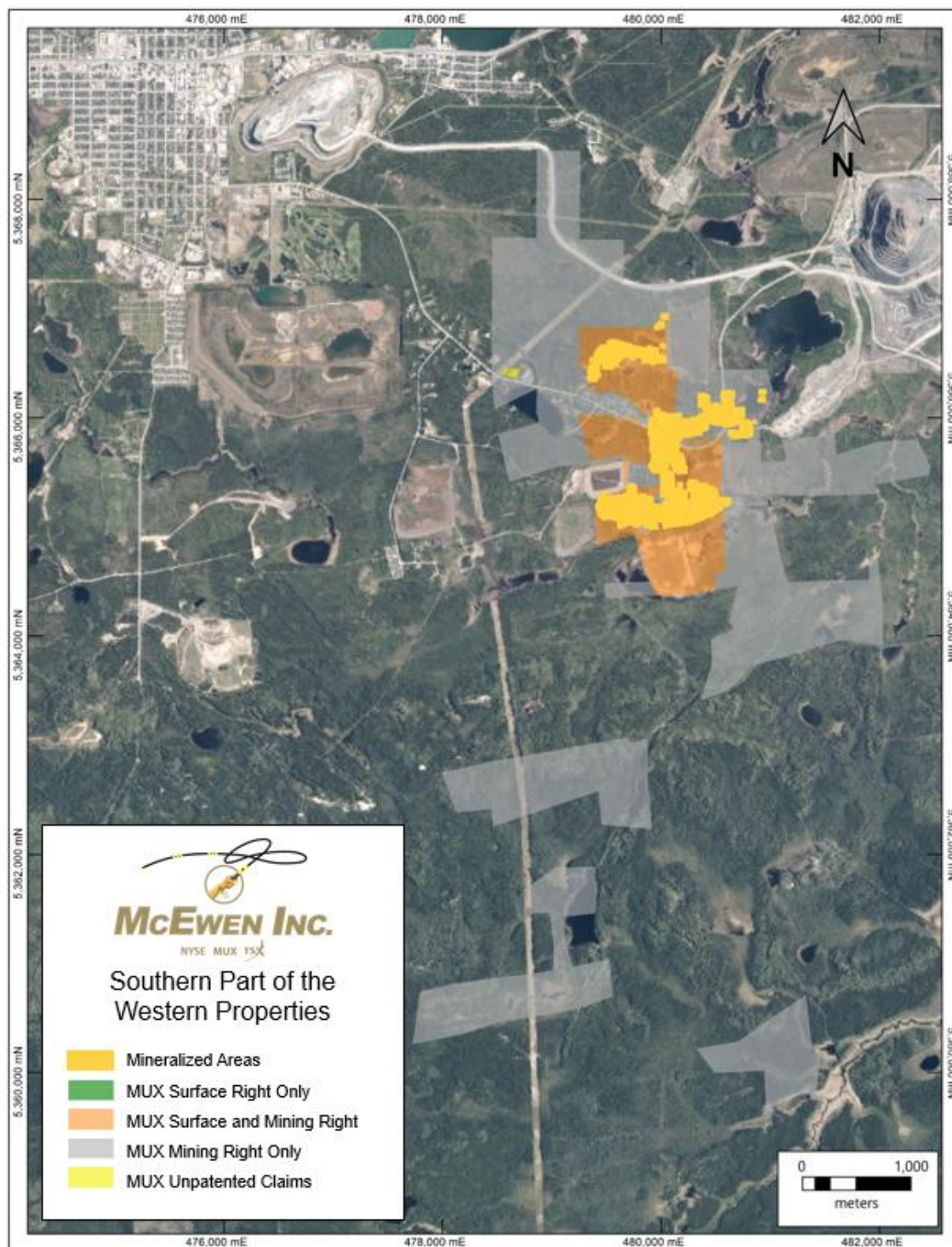


Figure 4-5: Western Properties South Parcel Blocks

4.2 Royalties and Encumbrances

4.2.1 Royalties

The Eastern Properties are subject to royalties shown in Figure 4-6.

Portions of the Grey Fox project are subject to the following royalties:

- Schumacher royalty (3% NSR)
- Triple Flag Royalty (formerly Newmont royalty) (2.5% NSR)
- Parsons-Ginn royalty (5% NPI or sliding-scale NSR)

In addition, a portion of the Grey Fox Project is subject to the Gray royalty (0.15% NSR).

The Froome Mine is subject to two royalties: a 3% NSR on the Steinman parcel and a 1.5% NSR on the Durham parcel. The Durham royalty is not applicable to the current mine plan.

The Black Fox Mine is not subject to royalties; however, both the Black Fox and Froome mines are subject to a gold streaming agreement originally with Sandstorm Resources Ltd. (Sandstorm), now held by Royal Gold Inc. (see Section 4.3.1)

Portions of the Stock Properties are subject to royalties as shown in Figure 4-7. The historical Stock Mine is subject to a 1% NSR payable to Franco-Nevada. The Stock West and Stock East deposits are not subject to these royalties.

The Western Properties are subject to royalties as shown in Figure 4-8. The Fuller property is subject to a 10% NPI royalty held by Summit Organization Inc. The Davidson-Tisdale property is not subject to royalties in the area of the current Mineral Resources.



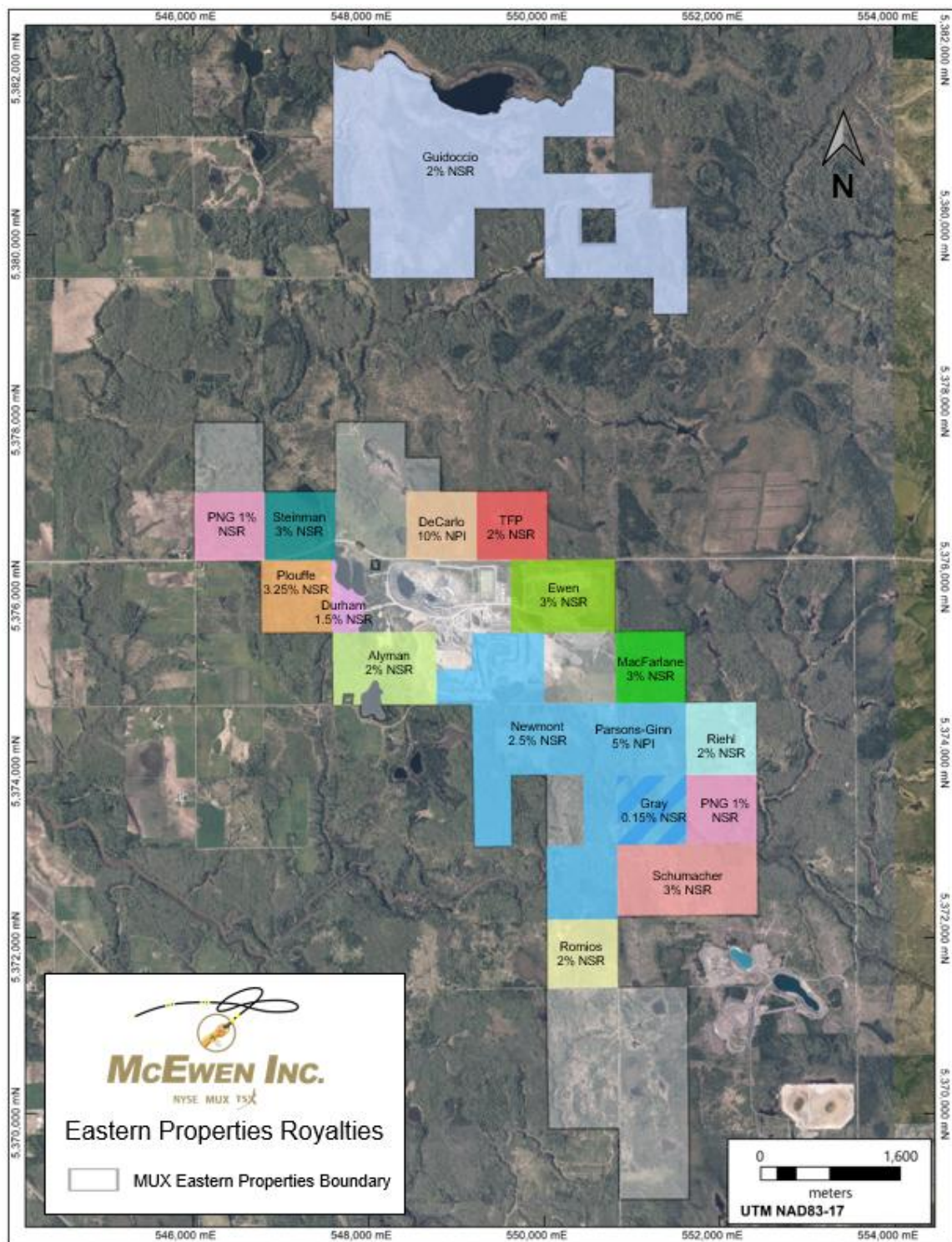


Figure 4-6: Eastern Properties Royalties

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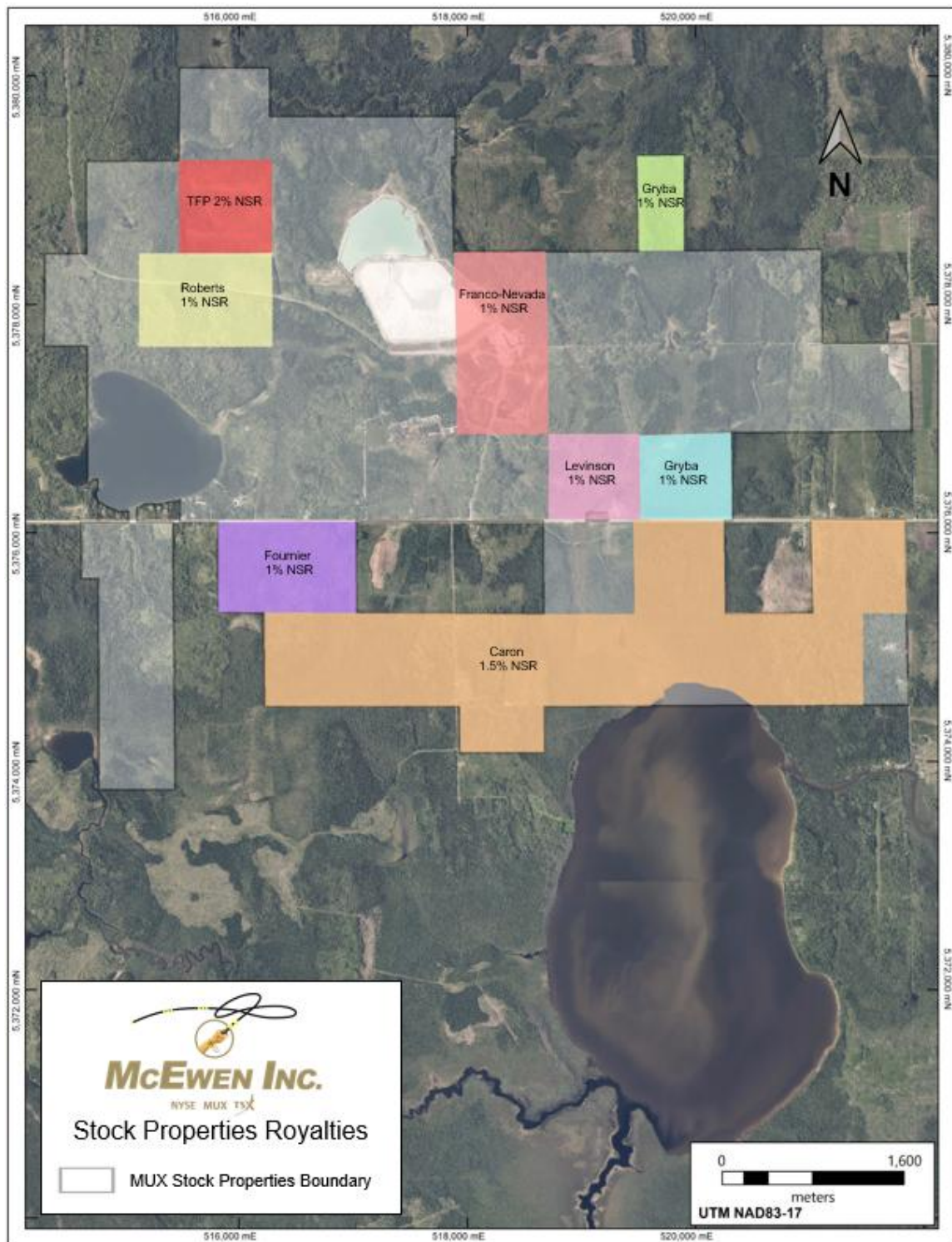


Figure 4-7: Stock Properties Royalties



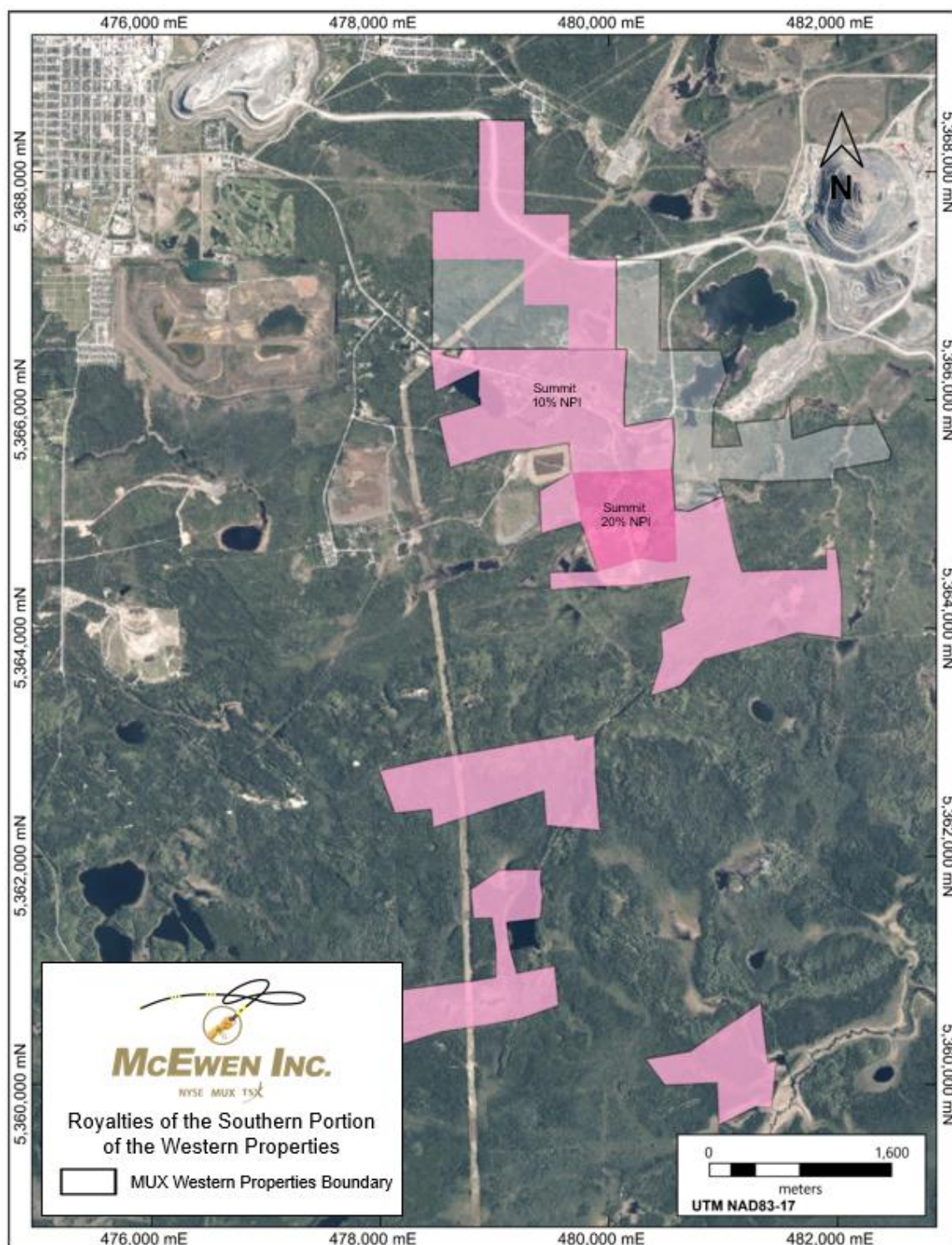


Figure 4-8: Western Properties Royalties

4.2.2 Stakeholders and Interested Parties

Stakeholders include the First Nations associated with the Abitibi Indian Reserve 70, which is jointly used by the Abitibiwiinn (Québec) and Apatipi Anicinapek (Ontario) First Nations, as well as local private landowners within Hislop and Beatty townships. The Abitibi Indian Reserve 70 is located approximately 25 km east of the Black Fox Mine site.

McEwen has undertaken ongoing consultation with Indigenous communities, the public, and government regulators regarding operations, environmental commitments, and planned activities.

4.3 Agreements

4.3.1 Sandstorm Streaming Agreement

On November 9, 2010, Brigus Gold entered into a gold streaming agreement with Sandstorm, under which Sandstorm agreed to purchase 12% of gold production from the Black Fox Mine beginning in January 2011 and 10% of future production from the Black Fox Extension, at a fixed price of \$500/oz. Sandstorm made an upfront payment of \$56.3 million in connection with this agreement.

On November 5, 2012, Brigus Gold exercised an option to repurchase a portion of the stream for \$24.4 million, reducing Sandstorm's interest to 8% of production from the Black Fox Mine and 6.3% from the Black Fox Extension (Figure 4-9).

On October 20, 2025, Sandstorm transferred its interest in the streaming agreement to Royal Gold Inc.

Currently, McEwen sells 8% of gold production from the Froome Mine to Royal Gold Inc. at a base price of \$624.86/oz (2026), subject to annual adjustment based on the consumer price index, capped at 2%. Any future production from remaining Black Fox areas would also be subject to this agreement.

The streaming agreement applies to gold produced within the defined agreement boundaries, including Froome and any future mining at Black Fox.



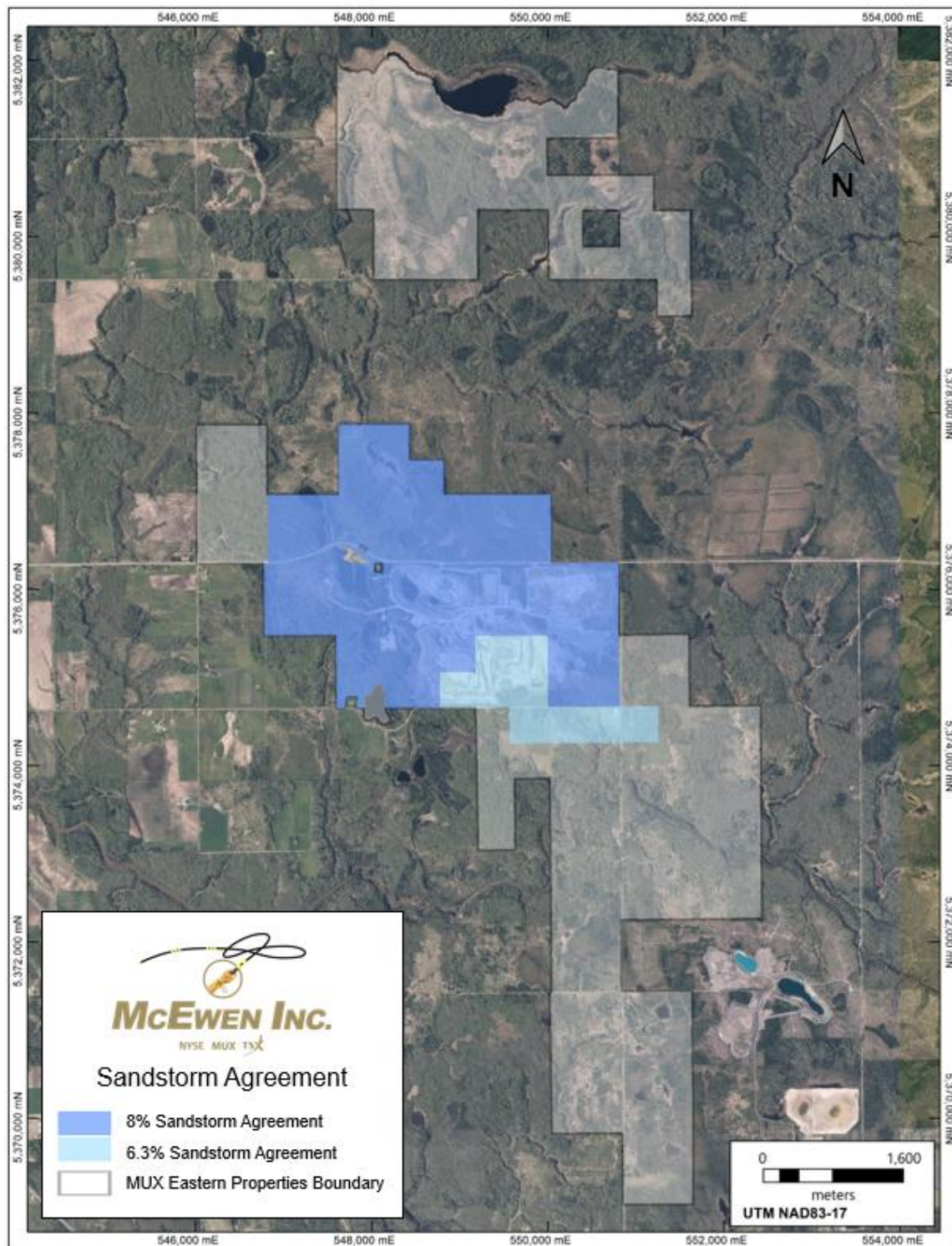


Figure 4-9: Sandstorm Gold Stream Agreement



4.3.2 Indigenous Communities

McEwen has entered into agreements with First Nations that assert treaty and Aboriginal rights within the areas of the Eastern and Stock properties. These agreements provide a framework for collaboration in the development and operation of the Grey Fox project and include provisions for financial participation, employment and skills training opportunities, business development, and consultation and dispute resolution.

In addition, McEwen engages with Indigenous communities in connection with permitting applications and ongoing project activities.

An Impact Benefit Agreement has been in place with the Apitipi Anicinapek Nation for the Eastern properties since 2011 and updated in 2026. The Métis Nation of Ontario has also been consulted with respect to the Eastern Properties.

McEwen does not currently have agreements in place with Indigenous communities for the Western Properties. Agreements will be required for those projects to move forward. Nearby operations have agreements in place with Flying Post First Nation, Matachewan First Nation, and Mattagami First Nation through the Wabun Tribal Council, as well as Apitipi Anicinapek Nation.

4.3.3 Newmont Agreement

The Newmont agreement (now held by Triple Flag Precious Metals Corp.) was originally executed with Apollo Gold in 2009. Under the terms of the agreement, if McEwen intends to dispose of any of any interest in property located near the Black Fox Mine (excluding corporate transactions), McEwen must notify the counterparty in writing, which then has the opportunity to consider acquiring all or part of McEwen's interest in the named property.

On February 4, 2026, a one-time payment of C\$1.0 million was made to Maverix Metals Inc. (a subsidiary of Triple Flag Precious Metals Corp.) to satisfy a provision of the agreement related to total gold resources (M+I+I) meeting or exceeding 500,000 oz on specified claims listed in the original agreement with Newmont.



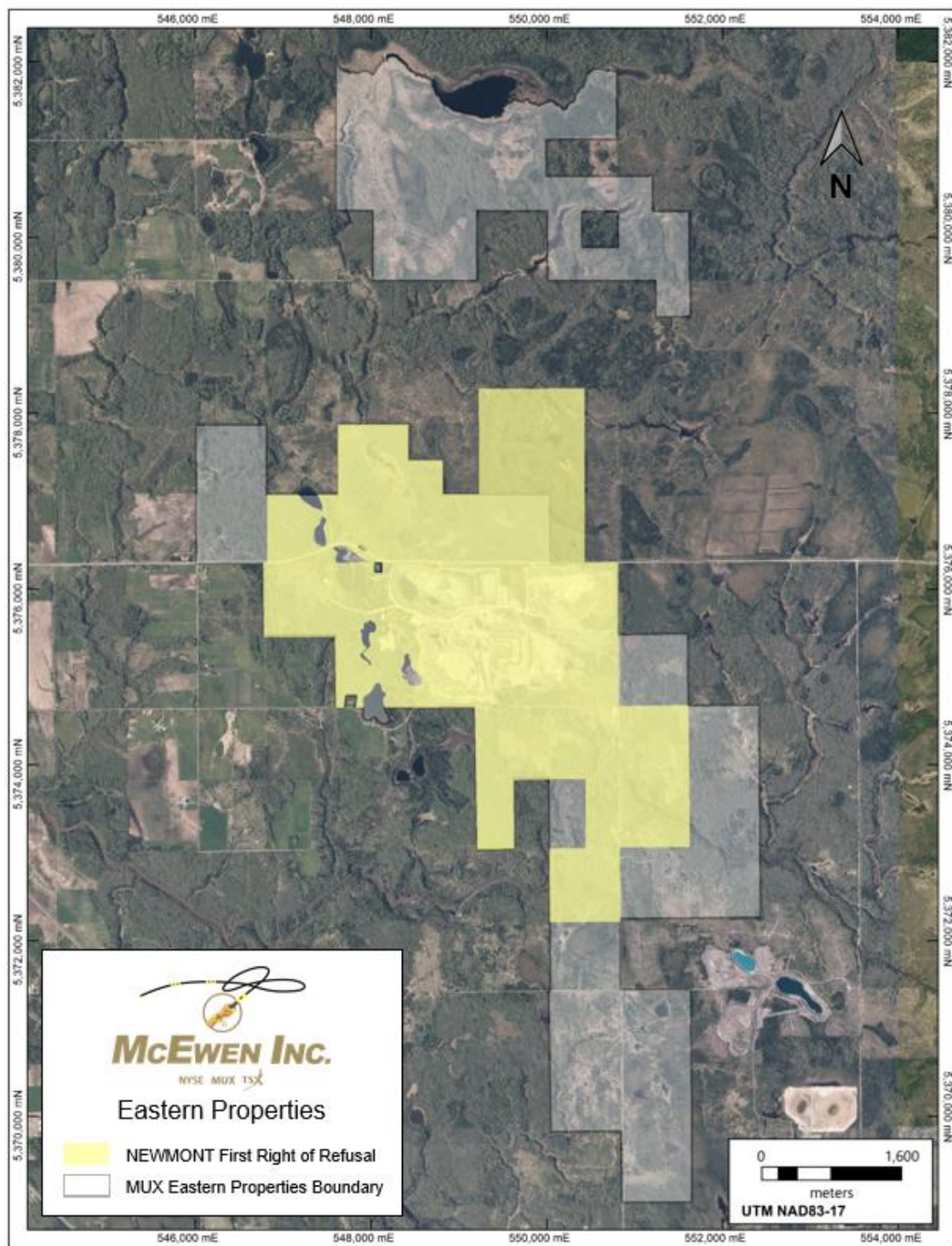


Figure 4-10: Newmont First Right of Refusal



4.4 Paymaster Option and Joint Venture Agreement

The Paymaster joint venture agreement with Newmont (now held by Discovery Silver Corp.) established the framework for McEwen (through Lexam VG Gold) to earn a 61% interest in the Paymaster property. The interest was earned through a combination of cash payments, stock, drilling, and exploration expenditures, and was fully vested upon expiry of the option term in 2012, with an additional 1% interest earned through McEwen's subsequent investment in the Paymaster property.

The agreement provides for adjustments in ownership based on participation in funding, with non-participating parties subject to dilution. The agreement remains in effect as long as products are produced from the Paymaster property or until 2062.

McEwen manages exploration, development, and mining activities on the Paymaster property to the 4,075 ft level, below which Discovery holds 100% interest.

As majority interest holder and operator, McEwen leads the management committee and retains decision-making authority for project activities.

4.5 Environmental Liabilities

Environmental liabilities related to historical and current mining activities at the Fox Complex properties are addressed through approved closure plans. These plans have been prepared in accordance with regulatory requirements and approved by the Ministry of Mines and the Ministry of Environment, Conservation and Parks.

4.6 Significant Risk Factors

The Fraser Institute Annual Survey of Mining Companies (Mejía and Aliakbari 2024) provides an independent assessment of the overall political risk facing an exploration or mining project across various global jurisdictions. Overall, Ontario ranked 10 out of 86 jurisdictions in the survey on the investment attractiveness index which combines the policy perception index in which Ontario ranked 13th, with results from the best practices mineral potential index where it ranked 14th.

The QP has identified the following risk factors associated with operating in Ontario:

- **Mineral title risk:** Uncertainties may exist with respect to mineral tenure, including issues related to boundary definition, assessment work, and overlapping claims.
- **Environmental regulation:** The introduction of stricter standards and enforcement, increased fines and penalties for noncompliance, more stringent environmental assessments of proposed projects, and a heightened degree of responsibility could have an adverse effect on project development. Unknown environmental conditions arising from historical or natural factors may also present risk.



5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility

The Grey Fox Project area is accessible from Toronto via Highways 400 and 69 to Sudbury, Highway 144 to Timmins, Highway 11 from Barrie to Matheson, and Highway 101 westward to Timmins. The City of Timmins is also serviced by regularly scheduled commercial flights from Toronto and other regional centres.

The Grey Fox property is accessible from Highway 101 via Tamarack Road (Hislop 2 Road), located approximately 10 km east of Matheson. The Black River-Matheson Township, which includes the communities of Holtre, Matheson, Ramore, Shillington, Val Gagné, and Wavell, has a population of approximately 2,500. Access within the property is achieved by various drill roads and all-terrain vehicle trails.

The Eastern Properties are located approximately 10 km east of Matheson, 60 km east of Timmins (population approximately 45,000), and 55 km north-northwest of Kirkland Lake (population approximately 8,000). Highway 101 provides direct access through the Black Fox property, with site infrastructure located south of the highway.

All claims pertaining to the Western Properties are located under the municipality of the City of Timmins and are accessible by either provincial or municipal roads. The Buffalo Ankerite, Paymaster, and Fuller properties are all near, or intersected by, secondary highways. The Davidson-Tisdale property is accessible by an all-weather gravel road north of Crawford Street in South Porcupine.

The Stock Property is located in Stock Township on the north side of Highway 101, approximately 20 km west of the community of Matheson and 43 km east of the City of Timmins. The area encompassing the mine property is not noted as a major tourist attraction or its outdoor recreational esteem. However, there are permanent residences, cottages, and tourist cabins located at Reid Lake (west and upstream of the property) and at Moose Lake (south and downstream of the mine property). The Stock Mine site is easily accessible via an access road from Provincial Highway 101 located approximately 1.5 km to the south. Highway 101 (along with numerous all-weather secondary, concession, and lot roads) provides excellent access to all of the Stock claim groups.

See Figure 4-1 for locations of the properties making up the Fox Complex.

5.2 Local Resources and Infrastructure

Supplies and services are available in Kirkland Lake, Matheson, Sudbury or Timmins, and materials can be delivered with a 12-hour turnaround time. Forestry and mining are the primary industries in the region, and the properties are located within well-established mining camps. Therefore, mining and exploration personnel as well as equipment can be locally sourced.



A 500 kV power line and transformer station are within 2 km of the Fuller, Buffalo-Ankerite, and Paymaster properties. Numerous operational gold processing facilities (as well as facilities on care and maintenance) are located in the Timmins area. The closest is the Dome Mill Complex owned and operated by Discovery. This complex is located approximately 2 km northeast of the Paymaster pit and has a rated capacity between 12,000 t/d and 14,000 t/d dependent on rock hardness.

Electrical power is available at Stock and Froome and readily available at the Grey Fox Project site via power lines along Tamarack Road.

5.3 Climate

The minimum mean annual temperatures in the Timmins region range from -22°C in January to +11°C in July. The maximum mean annual temperatures in the Timmins region range from -10°C in January to +24°C in July. The mean annual rainfall for the region is 543 mm. The mean annual snowfall over the winter is 308 cm (<https://climate.weather.gc.ca/>).

Rapid melting of accumulated snowfall can produce local flooding on the property for short periods during the spring months. Average monthly wind speeds for the region are 11 km/h to 15 km/h (Dyck, 2007). It is possible to conduct mining and exploration activities year-round.

Operations on all the properties can continue year-round.

5.4 Physiography

5.4.1 Eastern Properties

The Eastern Properties are predominantly agricultural land with a mature willow shrub, poplar, black spruce, and white birch forest along the southern and eastern edges of the property. The region is characterized by outwash deposits from continental glaciation, including raised beaches, flat clay pans and eskers. The low to moderate topography is marked by rock knobs and ridges (Dyck, 2007). The elevation around the Eastern Properties ranges from 295 m to 330 m above sea level. There is limited bedrock outcrop exposure for the Eastern Properties.

Surface waters around the eastern property include lakes, rivers, and their associated habitats. Lakes include Froome Lake located 0.25 km west of the Black Fox mine; Leach Lake located 1.4 km northwest of the mine; and Lawler Lake located 1.7 km to the south. Two other lakes, Salve Lake and Nickel Lake, are respectively located 5.2 km and 5.9 km north of the mine and form the headwaters of Salve Creek.

The Eastern Properties are located within the Salve Creek and Pike River watersheds, which are both tributaries of the Black River. Black River flows north into the Abitibi River, which in turn flows into Moose River. Moose River ultimately flows into James Bay (Dyck, 2007).

5.4.2 Stock Property

The Stock Property consists of an active mill and tailings facility. The historical Stock Mine is in redevelopment with anticipated production to commence late 2026.



The site is contained in an area of stratified silts and clays and includes wetland depressions. The topography in the vicinity of the property is controlled by a level lacustrine plain of sand and clay bounded by esker/outwash units to the east and west. The ridges are considered to be bedrock-controlled, although available information indicates that overburden thickness over the bedrock is significant.

Local topographic variations range from 267 m to 276 m above sea level. The gentle undulating terrain is characterized by dry-to-moist clay surfaces with shallow wet organics occurring in the depressions. Local areas of moderate relief are generally well drained. In contrast, topographical lows are frequently occupied by organic wetland deposits and are poorly drained. There are no known bedrock outcrop exposures at the Stock Property.

5.4.3 Western Properties

The Western Properties are historical mining sites with residential areas adjacent to them, except at Davidson-Tisdale. The area in the vicinity of the properties is typical of glacial regions with low to moderate topographic relief and numerous rivers and lakes. Elevations range from approximately 250 m to 300 m above sea level. Drainages are characterized by creeks and rivers which comprise part of the Arctic watershed. Bedrock outcrop exposure is limited on the properties.

The Timmins area supports boreal forest tree species and an active timber, pulp, and paper industry. Local tree species include the American Mountain Ash, Balsam Fir, Black Spruce, Eastern White Cedar, Eastern White Pine, Jack Pine, Pin Cherry, Red, Tamarack, Trembling Aspen, White Birch, White Spruce, and Speckled Alder.



6 History

Exploration and mining activities have occurred in the Timmins and Matheson areas since the early 1900s with numerous operators, varying ownership structures, partnerships, agreements, and joint ventures with varying degrees of success. The properties that now make up the present-day Fox Complex are the result of an amalgamation of several of these ownership histories culminating in the purchase of both the Lexam VG and Primero Gold Corp. properties and assets in 2017 through separate transactions.

The Fox Complex comprises a group of gold properties in Ontario currently owned by McEwen, including the Eastern Properties, Stock Property, and Western Properties. The Eastern Properties include the Grey Fox, Froome, and Black Fox properties; the Stock Property forms the central portion of the Fox Complex located in Stock Township; and the Western Properties comprise the Fuller, Buffalo Ankerite, Davidson-Tisdale, and Paymaster properties. See Figure 4-1 for the locations of the McEwen-owned properties.

This section focuses on the Grey Fox Project, which forms the basis of this Technical Report, followed by relevant and supporting historical context from adjacent properties within the Fox Complex.

6.1 Grey Fox Project

The Grey Fox Project is located within the Eastern Properties of the Fox Complex and represents the primary focus of this Technical Report. The project comprises several zones of mineralization, including the Gibson, Contact, Grey Fox South and Whiskey Jack zones, developed through successive exploration programs.

Drilling history at Stock, including Stock Main, Stock East and Stock West are tabulated in Table 6-6.

The Grey Fox property was first staked by Frederick Schumacher in the early 1900s. In 1936, the area was mapped by the Ontario Department of Mines. Eventually the claims were patented and worked as farmland until 1992. From 1937 to 1989, there was no reported exploration activity. In 1989, Goldpost Resources drilled an unknown number of holes in the Contact breccia zone, with unspecified results (Atherton 1989).

In the area of the Gibson deposit, originally called Hislop West, two shafts were sunk between 1933 and 1939. Both were mined to 122 m, with underground and surface diamond drilling, drifting, and crosscutting. In 1946, S.J. Bird (later Martin-Bird) diamond drilled 11 holes totaling 2,972 m. During the period between 1979 and 1980, A.P. Ginn and G.E. Parsons (Parsons) completed geological and magnetic surveys, and diamond drilling. Additional diamond drilling and a VLF-EM survey took place in 1981 by Parsons and Armco Minerals Exploration Ltd. (Armco) in the north half of lot 4, concession 4, which were reported to have intersected high-grade gold tenors (Atherton 1981).

In 1983, Geddes Resources Ltd. (Geddes Resources) and Armco completed geophysical surveys, trenching, and drilling. In 1986, subsequent to a 2,133 m surface diamond drilling program, Goldpost Resources Inc. (Goldpost Resources) initiated an exploration decline ramp to explore anomalous gold tenors associated with the north-northeast striking Gibson Fault. In 1987, Goldpost Resources mined a 61 m crosscut; diamond drilled 56 holes totaling 7,798 m from underground; and extended the decline ramp an additional 579 m to the 122 m level. In the following year, Goldpost Resources completed a magnetic



survey, extended the decline ramp by 610 m, drifted 168 m, and diamond drilled more than 7,620 m from underground.

Goldpost Resources reported drill defined material at Gibson, above the 122 m level in 1989.

Mining Corporation of Canada Ltd. (Mining Corp) purchased Gibson from Goldpost Resources and began mining in 1989 (Fenwick et al., 1990; Atherton, 1989). According to Hemlo maps for the Pike River project, about 8,000 t were mined at a grade of 27.4 g/t Au (Ontario Ministry of Mines n.d.).

According to Buss (2010), Noranda acquired the property in the early 1990s and developed a north-south grid along the Contact Zone in 1993 (Garber, 1997) and conducted a magnetometer and induced polarization resistivity survey on the property in 1994. This was followed up with three exploration holes spaced 200 m apart along the north end of the Contact Zone, for a total of 919 m. Whole rock geochemistry was also performed on one of the drill holes (Garber 1997).

Noranda optioned the property in 1995 to Hemlo (Buss 2010). Hemlo and Battle Mountain Gold Company (Battle Mountain) developed an east-west grid over the property and drilled holes on the south end of the zone. Battle Mountain also calculated a resource estimate on the Contact Zone based on results from the previous year's drilling program.

In July 1996, Hemlo merged with Battle Mountain And (in conjunction with Cameco Gold) drilled holes on the central portion of the Contact Zone at a 200 m spacing and submitted samples for mineralogical examination (Garber 1997). Additional drilling was conducted on the south end of the Contact Zone in 1997, after which a resource calculation was completed. The property was then transferred back to the Schumacher Estate.

Apollo Gold acquired the property in November 2007 and began drilling the southern extension of the Contact Zone in 2008.

The 2010 and 2011 Brigus Gold exploration drilling programs led to the completion of a Mineral Resource estimate supporting open pit and underground mining in October 2012 (Daigle, 2012).

To upgrade the classification, an updated Mineral Resource estimate followed the 2012 and part of the 2013 drilling programs completed by Brigus Gold (Pelletier et al., 2013).

The 2014 and 2015 Primero drilling programs focused on infill drilling and expanding the Mineral Resources on the 147 Zone, Contact, and Grey Fox South zones.

McEwen began drilling both infill and step-out drill holes at Grey Fox in 2018. That same year, SRK produced an updated Mineral Resource estimate for a Technical Report for McEwen.

The 2019 drill programs at Grey Fox focused on deep exploration at Gibson, the discovery of Whiskey Jack and de-risk drilling at 147, 147NE, and Grey Fox South. Successive drilling campaigns from 2020 to 2025 were concentrated on further delineating Whiskey Jack and Gibson with additional drilling at 147 and Grey Fox South. This drilling was designed to identify new lenses of mineralization and continue growing the resource.



Importantly, up until 2018, the majority of the drill holes at Grey Fox were drilled perpendicular to stratigraphy, a practice that reflected the understanding at the time of the orientation of the mineralization. In early 2019, much of the mineralization (excluding the Contact Zone) was determined to be oriented oblique to stratigraphy (mineralization trending NE-SW). Therefore, many of the subsequent drill holes were reoriented in a SE direction to achieve more optimal drill angles for the mineralization.

McEwen updated the resource estimates for Grey Fox in 2019, 2020, 2021, 2024 and 2025. The 2025 Mineral Resource estimate that forms the basis for this PFS is the most recent and current resource estimate. All previous resource estimates are now considered historical and should not be relied upon.

Drilling history at Grey Fox is tabulated in Table 6-1.

Table 6-1: Summary of Grey Fox Drilling

Company	Year	No. Drill Holes	Metres Drilled	Target Zone
Abuy Gold Mines	1939-1945	16	1,277.0	Gibson
Martin-Bird Gold Mines	1946	11	2,972.0	Gibson
Nevada Exploration	1973-1974	11	610.0	
A.P. Ginn and G.E. Parsons	1979-1980	38	3,910.0	Gibson
Armco	1981	18	884.0	Gibson
Geddes Resources and Armco	1983-1984	28	2,170.0	Gibson
Goldpost Resources	1986-1989	At least 56	17,549.0	Gibson surface and underground drilling
Noranda	1993	21	5,533.0	Contact
Noranda	1994	6	1,367.0	Contact
Battle Mountain/Hemlo	1995	8	2,109.0	Grey Fox
Battle Mountain and Cameco Gold	1996	16	5,872.0	Contact, 147, South, Gibson, and Hislop North
Battle Mountain and Cameco Gold	1997	13	5,367.5	Contact, 147, South, Gibson, and Hislop North
Glimmer/Exall	2001	4	1,667.0	Romios
Apollo Gold	2008	16	3,715.0	Southern extension of Contact
Apollo Gold	2009	53	9,960.0	Contact
Brigus Gold	2010	76	29,084.0	Contact, Gibson, South, Hislop North, and 147
Brigus Gold	2011	274	101,893.0	Contact, Gibson, South, 147
Brigus Gold	2012	282	87,971.0	Contact, 147, Whiskey Jack, South
Brigus Gold	2013	148	65,417.0	Contact, 147, South
Primero	2014	199	81,933.0	Contact, 147, South



Company	Year	No. Drill Holes	Metres Drilled	Target Zone
Primero	2015	57	26,094.0	Contact Zone, Gibson, South
McEwen	2018	64	25,181.3	Gibson, 147, 147NE
McEwen	2019	219	89,266.0	Gibson, 147, 147NE, South, Whiskey Jack, and Romios
McEwen	2020	13	4,438.0	Whiskey Jack
McEwen	2021	29	11,396.0	Gibson and Whiskey Jack
McEwen	2022	14	6,135	Gibson and Whiskey Jack
McEwen	2023	21	10,703	Gibson and Whiskey Jack
McEwen	2024	128	42,419	Gibson, 147, 147NE, South, and Whiskey Jack Zone
McEwen	2025	179	64,001	Gibson, 147, South, Contact, Whiskey Jack, Stroud
McEwen	2026 to date	12	4,700	Gibson, Whiskey Jack, Stroud

6.2 Stroud Property

The Stroud property is contiguous with the Grey Fox property and forms part of the Eastern Properties of the Fox Complex. The acquisition of the Stroud property by McEwen was completed in late 2024. The first meaningful drilling took place on the property in the early 1970s by Gunnex Ltd. on the Main Zone. Subsequent drilling programs by different operators demonstrated the presence of additional zones of mineralization on the property. An internal and undisclosed historical resource estimate prepared by Atticus Geoscience Consulting in 2021 reported approximately 271 koz of gold (M+I+I). After McEwen's purchase of Stroud, about 6,500 m of drilling was executed during the 2025 drilling season with the principal purpose of verifying this 2021 historical resource.

6.3 Eastern Properties

The following sections summarize the history of other properties within the Eastern Properties of the Fox Complex, which provide regional and operational context to the Grey Fox Project.

McEwen acquired all assets of the Eastern and Stock properties in 2017 from Primero Mining Corp (Primero). All assets associated with the Stroud deposit were acquired in 2024 from Stroud Resources Ltd. Refer to Figure 4-1 for the position of McEwen's Ontario properties.

6.3.1 Black Fox Mine

Drilling on the Black Fox properties was first conducted by Dominion Gulf in 1952, followed by Hollinger Consolidated Gold Mines Ltd. (Hollinger) in 1962. The drill holes were drilled near diabase dykes located in the easternmost part of the properties. In 1988, Glimmer Resources, Inc. (Glimmer) put together the property package using a combination of crown and private lands. In 1989, Noranda Exploration Company Ltd. (Noranda) entered into a joint venture agreement with Glimmer to earn a 60% interest in the properties.



Between 1989 and 1994, Noranda, and later Hemlo Gold Mines Inc. (Hemlo), completed eight drill programs. In all, 27,800 m of drilling was completed in 142 holes. In addition to diamond drilling, exploration was conducted by way of geological, magnetic and gradiometer surveys, a UTEM survey, and a limited induced polarization survey. In 1996, a final feasibility study on the Glimmer Gold Project was based on probable reserves outlined to a depth of 250 m.

The joint venture advanced an access ramp from surface to 55 m depth in 1996. Exall Resources Ltd. (Exall) purchased the property from Hemlo in April 1996, obtaining approximately 60% interest with Glimmer holding the remaining portion. A bulk sample was taken in 1997 from underground following development of a spiral decline ramp to a depth of 120 m, and 3,800 m of drifting. Commercial production from the Glimmer Mine was achieved in 1998. The sample and production were custom milled at St Andrew Goldfields Ltd.'s (St Andrew's) Stock Mill from 1997 through 2001, after mineral tests carried out by Lakefield Research and others.

In September 2002, Apollo Gold Corporation (Apollo Gold) completed the acquisition of the assets of the Glimmer Mine from Exall and Glimmer. The project was renamed the Black Fox property. Between 2003 and 2007, Apollo Gold completed five drill programs. In addition to diamond drilling, exploration was conducted by way of induced polarization surveys.

In 2008, Apollo Gold produced a feasibility study declaring Mineral Reserves. On 28 July 2008, Apollo Gold completed the acquisition from St Andrew of its Stock Mill and related equipment, infrastructure, property rights, laboratory, and tailings facilities.

In October 2008, Apollo Gold awarded a contract for the removal of the glacial till material over the open pit site, and work commenced on 23 October 2008. During the same year, Apollo Gold received all necessary permits and approvals required to commence mining activities of the open pit. Apollo Gold received a Certified Closure Plan Approval, an Amended Certificate of Approval for Industrial Sewage Works, and a surface and groundwater Permit to Take Water (PTTW).

Apollo Gold commenced open pit mining at the Black Fox Mine in March 2009. The 2009 drilling program focused on the Pike River property to test the northern extension of mineralization from the adjoining Grey Fox Project.

Apollo Gold and Linear Gold Corporation (Linear Gold) merged to form Brigus Gold Corporation (Brigus Gold) in June 2010. The drilling surface program completed 14 condemnation drill holes (3,468 m) around the Black Fox Mine. A helicopter-borne, high-resolution magnetometer survey was completed in September 2010, covering the 17 km² Black Fox mine.

Apollo Gold and Brigus Gold continued drilling at the Black Fox Mine from January 1, 2008 to December 31, 2013.

Details of the drilling history at Black Fox is tabulated in Table 6-2.

On March 5, 2014, Primero acquired all issued and outstanding common shares of Brigus Gold.



Table 6-2: Summary of Black Fox Drilling

Company	Year	No. Drill Holes	Metres Drilled
Dominion Gulf	1952	Unknown	Unknown
Hollinger	1962	Unknown	Unknown
Noranda	1989-1994	142	27,800
Glimmer/Exall	2000-2002	1,088	96,053
Apollo Gold	2003-2007	889	224,162
Apollo Gold/Brigus Gold	2010-2013	1,318	135,800
Primero	2014-2017	2,605	387,419
McEwen	2018-2026	1,793	247,528
Total		7,835	1,118,762

6.3.1.1 Past Production

Historical production from the Glimmer Mine era is shown in Table 6-3. The Black Fox Mine production between 2009 through December 2024 is summarized in Table 6-4. Note that the Stock Mill production includes material mined from the Froome Mine and from stockpiles.

*Table 6-3: Production History (mined) of the Glimmer Mine from 1997 to 2001**

Year	Tonnes	Au Grade (g/t)**	Au (oz)	Recovery (%)
1997	194,460	6.62	39,884	96.4
1998	308,734	6.69	64,319	96.9
1999	258,699	5.93	48,266	97.8
2000	255,234	5.83	46,418	97.0
2001	81,700	4.61	11,895	98.2
Total	1,098,827	6.14	210,782	97.1

Notes:

* From historical records. Some numbers may not sum due to rounding and conversion factors. Grades and recoveries are straight averages

** Grades are calculated from reported production ounces, tonnes and recovery. From 2008 Apollo Gold Corporation Technical Report.

Table 6-4: Production History of the Black Fox & Froome Mine from 2009 to December 2025

Year	Open Pit Tonnes	UG Tonnes	Total Tonnes Mined	Tonnes Milled	Mill Au Head Grade (g/t)	Au Produced (oz)	Recovery (%)
2009	631,000	-	631,000	531,000	3.28	52,152	93%
2010	792,482	-	792,482	718,400	3.17	67,499	92%
2011	433,267	170,889	604,156	725,541	2.54	55,756	94%



Year	Open Pit Tonnes	UG Tonnes	Total Tonnes Mined	Tonnes Milled	Mill Au Head Grade (g/t)	Au Produced (oz)	Recovery (%)
2012	907,077	164,926	1,072,003	735,573	3.43	77,374	95%
2013	663,428	297,110	960,538	752,959	4.34	98,710	94%
2014	775,403	122,249	897,652	695,131	3.00	64,018	95%
2015	849,668	140,836	990,504	875,833	2.58	69,733	96%
2016	-	234,518	234,518	913,235	2.22	62,171	96%
2017	-	263,549	263,549	685,293	3.14	66,733	96%
2018	-	255,982	255,982	268,288	5.57	46,672	97%
2019	-	213,887	213,887	243,677	4.88	37,288	98%
2020	-	200,326	200,326	234,744	3.19	23,129	96%
2021	-	306,962	306,962	303,377	3.24	29,290	93%
2022	-	419,496	419,496	344,587	3.77	37,315	89%
2023	-	395,713	395,713	456,831	3.35	44,236	90%
2024	-	303,979	303,979	403,669	2.54	29,430	89%
2025	-	285,054	285,054	290,365	2.89	23,249	86%
Total	5,052,325	3,775,476	8,827,801	9,178,503	3.19	884,755	94%

Notes:

- UG = underground.
- Milling tonne values (for 2016 & 2017) are higher than the mining tonnes due to processing of residual open pit low grade stockpile.
- Two milling tonne values (for 2023 & 2024) are higher than the mining tonnes due to re-claiming material because of the higher gold price (originally sent as waste from the open pit waste dump).
- 2025 Milled Tonnes, Ounces and Grade calculated from 2025 KPI report, removing as a percentage, material processed from Stock Mine.
- Total Head Grade and Recovery calculated from weighted average of 2009-2025 values.

6.3.2 Froome Mine (Main Zone and West Zone)

Noranda began exploration on the Froome Lake claims in 1991 using a ground total field magnetometer and a very low frequency electromagnetic survey (EM), finding results consistent with the geological terrain of the Porcupine-Destor Deformation Zone (PDDZ). This work was followed by geological mapping in both the Froome and Glimmer claims.

The Froome deposit is located on the Durham, Plouffe, and Steinman properties acquired by Apollo Gold. The Durham and Plouffe properties were acquired in 2003 and the Steinman property in 2007.

In 2014, Primero completed a diamond drilling program which discovered the Froome (Main) deposit. This program was targeting an induced polarization anomaly, intercepting 23 m (core length) of silicified rock with disseminated pyrite. Through the latter part of 2015 and early 2016, subsequent diamond drilling defined mineralization over a strike length of approximately 150 m and dip length of approximately 300 m. In addition, drilling in 2016 identified a second zone of mineralization 25 m below the main deposit.



By the end of 2025 most of the mineralized material from the main zone at Froome had been mined out and McEwen had transitioned to mining at the West zone which is described in more detail below.

The Froome West zone is located approximately 300 m west of the main Froome zone and saw limited drilling until 2024. The West zone is characterized by generally narrow, orogenic quartz-carbonate veining hosted in mafic volcanic rocks. The mineralization is interpreted to occur in a tight, northwest-southeast trending fold with the fold hinge plunging steeply to the southwest. Currently the zone has been traced approximately 175 m along strike and has a dip component of about 350 m. Most of the higher grade and wider mineralization occurs within the fold hinge.

Drilling history at Froome Mine (through December 2025) is summarized in Table 6-5.

The initial Mineral Resource estimate for the Froome (Main) deposit was calculated in 2017 by Primero. The Primero Mineral Resource estimate is now considered historic as it has been updated numerous times since then, including in this report.

Table 6-5: Summary of Froome Drilling

Company	Year	No. Drill Holes	Metres Drilled
Noranda	1991-1994	2	551
Glimmer/Exall	2000-2002	1	200
Apollo Gold	2003-2007	1	693
Primero	2014-2017	219	65,051
McEwen	2018-2025	915	164,594
Total		1,138	231,089

6.3.3 Black Fox North

The claims pertaining to Black Fox North were acquired by McEwen in 2017 along with the remaining portions of the Eastern Properties. All assessment work performed on these claims, including work completed by McEwen, is publicly available through the Mining Land Administration System (Ontario Ministry of Mines, n.d.).

6.4 Stock Property

The Stock Property forms the central portion of the Fox Complex and hosts the Stock Mine and processing facilities, including the Stock Mill, which provide infrastructure support to the Eastern Properties.

The Stock Property was originally staked and drilled by Hollinger Gold Mines Ltd. (Hollinger) from 1959 to 1964 with mineralization being discovered in 1961. This work located auriferous carbonate rocks, volcanic rocks, and porphyries in proximity to the PDDZ, but limited exploration funds and changing exploration priorities postponed further evaluation. The claims were allowed to lapse.



Quebec Sturgeon River Mines Ltd. (Quebec Sturgeon) re-staked the property and discovered substantial gold mineralization in 1973. Development began in 1974 and included construction of a headframe with a shaft being collared and driven through 15 m of overburden and 4 m of bedrock.

From 1980 to 1981, Quebec Sturgeon deepened the three-compartment shaft to 82 m below surface, and a shaft station was established on the 61 m level. Some 413 m of drifting and crosscutting were completed on this level along with a total of 4,481 m of underground drilling from 113 holes. Quebec Sturgeon ceased all work in early 1982 due to the drop in gold price.

In 1983, a \$14 million equity financing was completed by turning the property over to St Andrew Goldfields Ltd. (St Andrew), a 68% owned subsidiary of Quebec Sturgeon. Development resumed on the property in mid-1983 and the shaft had been deepened to 269 m and underground development had been conducted on four levels by the end of 1985.

In early 1986, St Andrew acquired additional property along the PDDZ by purchasing Labrador Mining and Exploration Company Ltd.'s (Labrador Mining and successor of Hollinger) interest in properties in three townships and by entering into joint venture agreements with Esso Resources Canada Ltd. (Esso) and Quebec Sturgeon. In mid-1987, St Andrew announced that the Stock Township deposit would be brought into production. Financing was raised in Europe in 1988, and an announcement was made that a 454 t/d mill would be constructed by late 1988. Production and milling of stockpiled development material at the Stock Mine began in 1989.

In 1989, St Andrew bought out the remaining interests of Esso. By the end of 1989, five levels in the Stock Mine had been developed and 91,500 t of material had been milled to produce 17,999 oz of gold. The shaft could not be used below the fourth level due to bad ground conditions, and the fifth, sixth and seventh levels had to be accessed by a ramp. Mining methods included cut-and-fill, room-and-pillar, and longhole.

Between 1988 to 2000, production of 130,000 oz of gold from 733,000 t of material at an average grade of 5.5 g/t Au was mined from the N2, West and Central chutes. Mining and milling were suspended in December 2000 to focus on a financial restructuring and to deal with a working capital deficit and debt burden.

In 2004, St Andrew dewatered and rehabilitated the underground workings, and the Stock head frame and hoist were refurbished. An underground development, mining and drilling program was then undertaken. During 2004, 30,500 t grading 2.56 g/t Au were mined and treated at the Stock Mill.

In 2005, underground mining operations were again suspended at the Stock Mine because of the decision to extend the period of the Advanced Exploration Program at the nearby, St Andrew's-owned Clavos Mine. Daily production levels at the Stock Mine were insufficient for the Stock Mill to operate at profitable levels. Underground development and drilling in the deeper portions of the Stock Mine did not generate sufficient gold resources to sustain future economic mining operations and the mine was placed on care and maintenance pending further exploration.

In 2006, RPA prepared a technical report on behalf of St Andrew on their properties in and around the Timmins area. No Mineral Resources or Mineral Reserves were reported at Stock (Gow and Roscoe, 2006).



The Stock Mine and mill facility were sold in 2008 to Apollo Gold (later Brigus Gold). Brigus Gold purchased and upgraded the Stock Mill to provide milling facilities for its Black Fox project. Refurbishment of the mill was completed in April 2009, and it began operating on 01 May 2009.

McEwen acquired the property and all its assets in 2017.

Stock East was discovered in 2018 by McEwen; Stock West was discovered in 2019. In 2019 a drilling program was laid out and designed to identify mineralization below the Main Zone. By mid 2021 mineralization below the N2 shoot at the Main Zone had been intersected. Successive drilling campaigns at Stock up until early 2024 were primarily designed to expand & delineate additional mineralization at all three of these zones (Stock West, Stock Main/Mine, and Stock East).

Since 2025, McEwen has been developing a new access ramp into the historical Stock Mine workings and is establishing the underground and surface infrastructure needed to support initial production later in 2026. Production is intended to ramp up to 1,500 t/d in 2027. The evaluation of the potential mine life is ongoing.

Drilling history at Stock, including Stock Main, Stock East and Stock West are tabulated in Table 6-6.

Table 6-6: Summary of Stock Drilling

Company	Year	No. Drill Holes	Metres Drilled	Target Zone
Various companies	1959-1982	315	31,950	
St Andrew	1983-2008	1,124	91,282	Stock Main, Stock East
Brigus Gold	2008-2017	6	2,164	
Primero	2014-2015	35	10,541	
McEwen	2018	91	28,389	Stock East
McEwen	2019	108	43,219	Stock East, Stock Main N-2 Chute Extension, Stock West
McEwen	2020	33	17,813	Stock West
McEwen	2021	136	61,046	Stock West, Stock Main N-2 Chute Extension
McEwen	2022	125	45,612	Stock West, Stock Main
McEwen	2023	209	65,505	Stock West, Stock Main, Stock East
McEwen	2024	52	11,402	Stock East, Stock West
McEwen	2025	0	0	
McEwen	2026 to date	25	3,064	Stock East
Total		2,259	411,987	

6.4.1 Past Production

Production commenced in July 1989 and ceased in mid-1994 due to a lack of working capital. The central chute was discovered in 1994 by underground drilling, and the 8-12 decline had almost reached the central shoot when the mine closed.



The overall historical production of the Stock Mine to date is shown in Table 6-7. Over half of the production came from the N2 chute where 405,298 t gold was mined, with an average grade of 6.2 g/t Au. The average grade increased in 1992, when production from the higher-grade west chute commenced. From 1992 to 1994, 103,857 tonnes at an average grade of 9.1 g/t Au were mined and processed from the west shoot, recovering 30,425 oz of gold.

Table 6-7: Stock Mine Production History, St Andrew

Year	Tonnes Milled	Average Mined Au Grade (g/t)	Contained Au (koz)	Recovered Au (koz)
1989	90,883	3.87	11,320	10,657
1990	163,129	5.59	29,310	27,295
1991	132,191	5.62	23,897	21,961
1992	173,413	6.79	37,848	36,301
1993	151,187	5.14	24,998	23,788
1994	20,677	5.38	3,578	3,403
2000	68,723	5.88	12,983	12,395
2004	23,015	1.83	1,355	1,355
2005	7,527	4.78	172	172
Total	830,745	5.48	145,461	137,327

Table 6-8 summarizes the gold ounces recovered in 2025 by McEwen during the development of the new Stock Mine ramp:

Table 6-8: Production History of the Stock Mine from 2025

Year	Underground Tonnes	Total Tonnes Mined	Tonnes Milled	Mill Au Head Grade (g/t)	Au Produced (oz)	Recovery (%)
2025	16,746	16,746	15,331	0.96	408	86%
Total	16,746	16,746	15,331	0.96	408	86%

6.5 Western Properties

The Western Properties form the western portion of the Fox Complex and include the Fuller, Buffalo Ankerite, Davidson-Tisdale, and Paymaster properties. These properties are not directly material to the Grey Fox Project but provide regional context within the Fox Complex.

McEwen acquired the mineral rights to the Western Properties through the 2017 acquisition of Lexam VG Explorations Inc. The history of the individual property mineral rights ownership is discussed in the history for each property.



6.5.1 Buffalo Ankerite Property

Prior to 1935, Buffalo Ankerite was developed by two independent owners. The operations were distinct and covered two different mineralized bodies, the South Zone (Buffalo Ankerite South) and the North Zone (Buffalo Ankerite North). In 1935, the operator of Buffalo Ankerite North, Buffalo Ankerite Holdings Ltd. (BAH, previously Buffalo Ankerite Gold Mines Ltd.) consolidated both properties under its ownership. Since 1935 the property has been held by numerous operators who performed various exploration and mining activities.

Details of the drilling history at Buffalo Ankerite is tabulated in Table 6-9.

Table 6-9: Summary of Buffalo Ankerite Drilling

Company	Year	No. Drill Holes	Metres Drilled	Target Zone
Various companies	Prior 1953	Unknown	Unknown	Historical records limited
Vedron and Pamour	1982	15	1,245	
Placer Dome	2001	15	2,728	North
Placer Dome	2002	59	6,097	South
Lexam VG	2005-2012	66	26,806.1	North
Lexam VG	2005-2012	159	54,419.4	South
Total		314	91,295.5	

6.5.1.1 Past Production

Production from the Buffalo Ankerite and the March operations between 1926 and 1953 totaled 4.8 Mt with an average grade of 6.5 g/t Au. Total metal produced was 1.02 Moz Au.

Between 1978 and 1982, production came from Pamour extracting the Buffalo Ankerite South crown pillars and milled at their adjacent Timmins Mill. The amount of material mined was not consistently reported by Pamour and the total is unknown. In 1978, Pamour reported milling 38,160 t. The Regional Geologist for Timmins reported 16,300 t in 1982 (Kustra1983). A later three-dimensional model created by Dome Mines suggests that a total of approximately 320,000 t of material was removed.

6.5.2 Fuller Property

Periodic surface exploration has been performed on Fuller since 1910 and limited production from the Fuller claim has occurred. The claim was originally held by W. S. Edwards of Chicago and was purchased from his estate by A. S. Fuller of Toronto.

Fuller was optioned to Placer Dome in 2002. Placer Dome carried out both field exploration and office database management activities on Fuller in early 2002 and drilled several holes.

In 2007 Wardrop Engineering Inc. prepared a Mineral Resource estimate on behalf of Vedron.



Lexam VG and predecessor companies completed significant surface diamond drilling on the Fuller property during the period 2004 through 2012. Most of the drilling was conducted on the Contact Zone/Edwards porphyry area. A significant amount of both surface and underground drilling completed by various operators during the period 1986 through 1997.

A diamond drilling program was performed by Lexam VG between 2004 to 2012.

In 2014, RPA prepared a Mineral Resource estimate for the Fuller Property on behalf of Lexam VG (Altman et al., 2014). An updated Mineral Resource estimate was performed for the 2022 Fox Complex Preliminary Economic Assessment (PEA) and this has been updated for this PFS and is now considered as the current Mineral Resource estimate.

Details of the drilling history at Fuller are tabulated in Table 6-10. In addition to the drilling, 691 chip samples were taken during the Belmoral work, totaling 2,586.5 m. Bazooka drilling (short probe core holes) during the same period totaled 184 holes totaling 1,487 m.

Table 6-10: Summary of Fuller Drilling

Company	Year	No. Drill Holes	Metres Drilled
Various companies	Prior 1974	92	7,572
Pamour	1974	1	162
Vedron	1983	15	1,219
Belmoral	1986-1989	545	37,196
Vedron	1996-1998	83	39,636
Placer Dome	2002	9	3,618
Lexam VG	2004-2012	71	22,858
Total		816	112,261

6.5.2.1 Past Production

A production report in 1942 showed 39,941 t with a recovered grade of 5.14 g/t Au was mined and milled at the Faymar Mill, producing 6,566 oz Au and 586 oz Ag (Ferguson 1968).

6.5.3 Paymaster Property

Placer Dome acquired the property in 1989 and conducted surface mapping, a lithogeochemical survey, a magnetic survey, power stripping, and channel sampling.

In June 2008, Lexam entered into a four-year option agreement with Goldcorp (now Newmont) to acquire a 60% interest, by spending \$6.0 million over four years in the 16 patented mining claims that represent the Paymaster property. In June 2012, Lexam completed the earn-in requirements and elected to exercise the option to acquire 60% interest in the Paymaster property. Following the acquisition, Lexam proposed a work program that Goldcorp declined to participate in. In accordance with the joint venture agreement,



Goldcorp's 40% ownership was then diluted, and, by the end of 2016, Lexam had increased its interest to 61%.

A Mineral Resource estimate was prepared by P&E and RPA (Armstrong et al., 2013) for the Paymaster property on behalf of Lexam. The QP has not verified the information relating to the preparation of the estimate to endorse it.

Details of the drilling history at Paymaster is tabulated in Table 6-11.

Table 6-11: Summary of Paymaster Drilling

Company	Year	No. Drill Holes	Metres Drilled	Target Zone
Various companies	Prior 1995	Unknown	Unknown	Historical records limited
Placer Dome	1995-1996	75	16,154	No. 2/3 shaft area
Placer Dome	1999-2000	17	12,008	No. 4 shaft area
Lexam VG	2008-2012	163	48,729	
Total		255	76,891	

Past Production

Past production from the Paymaster mine totaled 5.1 Mt at an average grade of 7.2 g/t Au. In total, 1.19 Moz of gold were produced between 1915 and 1966 (Atkinson et al., 1999). In the mid 1990s, Placer Dome produced 25,737 t of development ore at an average grade of 6.15 g/t Au.

6.5.4 Davidson-Tisdale Property

The property was incorporated as Davidson Gold Mines Ltd. in 1911 and was succeeded in 1919 by Davidson Consolidated Gold Mines Ltd. Little work was done from 1945 to the early 1980s.

In 1994, a study commissioned by Placer Dome provided an estimate of an open pit mineable resource.

From 1999 to 2003, Northcott Gold Inc. (Northcott, formerly DTM) drilled several holes. Northcott subsequently optioned the property to Vedron in 2003, allowing Vedron to earn a 50% undivided interest. From 2003 to 2006, Vedron drilled a significant number of holes. In 2004, the company amended the option agreement to allow earning 75%. In 2005, the option agreement was further amended, resulting in Vedron acquiring 54% of the interest in the property.

A Mineral Resource was estimated by Guy and Puritch in 2007 (Guy and Puritch, 2007). Vedron expenditures on the property had increased the interest to 66% by the filing of their 2007 annual information form in June. Later in 2007, a preliminary economic assessment was filed on Davidson-Tisdale. In 2008, the option expired with Vedron earning a 68.5% interest in the Tisdale property by 2014, with Laurion Mineral Exploration Inc. (Laurion, previously Northcott) holding the remaining 31.5%.

In 2010, Vedron purchased the remaining 31.5% minority stake in Davidson-Tisdale, held by SGX Resources, Inc. (previously Laurion) to fully control the property.



From 2006 to 2010, Vedron continued to drill diamond drill holes from surface.

In 2013, a Mineral Resource estimate and technical report was prepared by P&E and RPA (Armstrong et al., 2013). An updated Mineral Resource estimate was performed for the 2021 Fox Complex PEA and this has been updated for this PFS and is now considered as the current Mineral Resource estimate.

Details of the drilling history at Davidson-Tisdale is tabulated in Table 6-12.

Table 6-12: Summary of Davidson-Tisdale Drilling

Company	Year	No. Drill Holes	Metres Drilled
Various companies	Prior 1980	24	5,267.0
Dome Mines	1981	11	1,118.0
DTM	1983-1984	35	4,576.8
Getty	1984-1987	537	49,165.9
Midas Minerals	1989	5	1,486.2
Northcott	1999-2003	17	399.3
Lexam VG	2003-2015	97	23,397.5
Total		726	86,410.7

6.5.4.1 Past Production

Production from 1918 to 1921 was processed on site with a 10-stamp mill until it burned down in 1924. A reported total of 8,501 t at 8.9 g/t Au was milled, and 2,438 oz gold recovered using mercury amalgamation, with about 20% of the gold content was lost using this process.

From April to November 1988, custom milling of the mined material at Davidson-Tisdale occurred at the nearby Giant Yellowknife Go-Mill. A total of 39,800 t was mined and milled during the Getty period, producing a total of 7,300 oz of gold and 5,665 oz silver.



7 Geological Setting and Mineralization

Property geology described herein for Davidson-Tisdale, Fuller, Paymaster and Buffalo Ankerite properties was derived from technical reports prepared by P&E (Mineral Resource estimate) and RPA (Armstrong et al. 2013, Ehasoo et al. 2014).

Figure 7-1 is a geological map showing the locations of the Eastern, Stock and Western Properties.

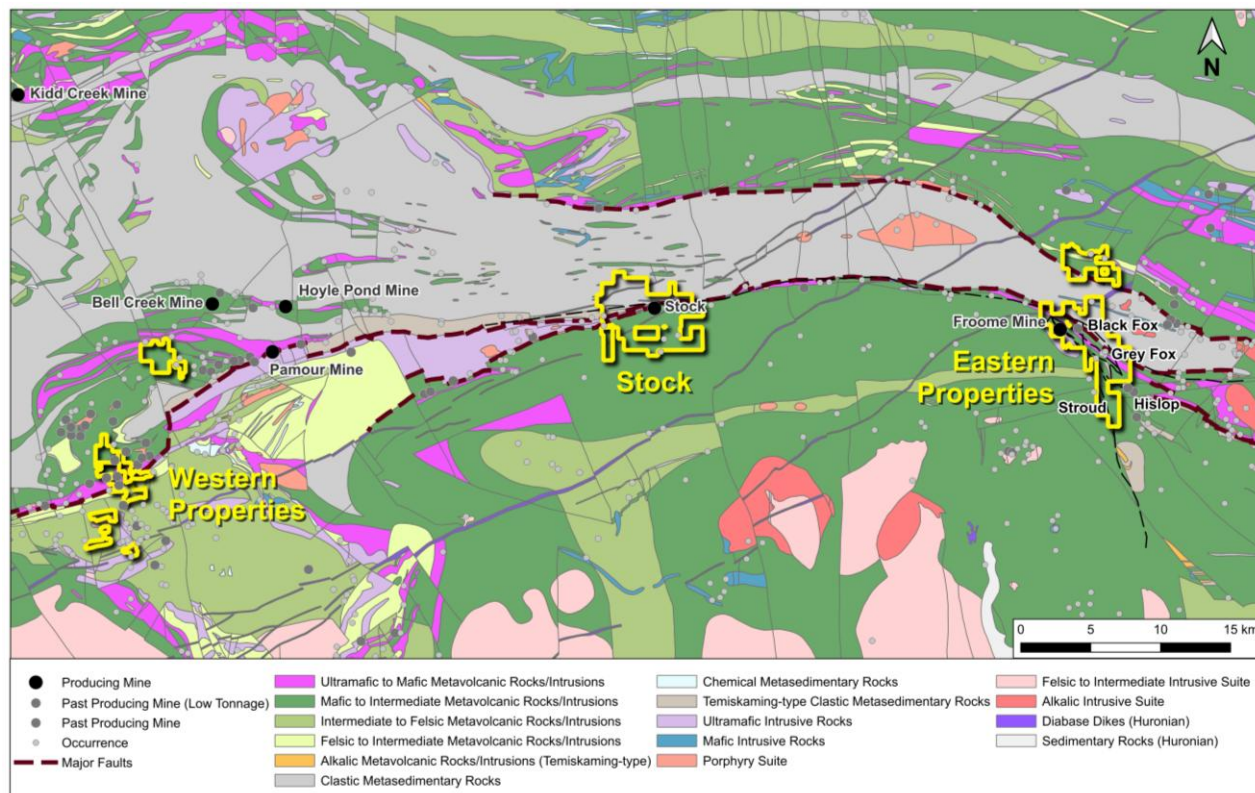
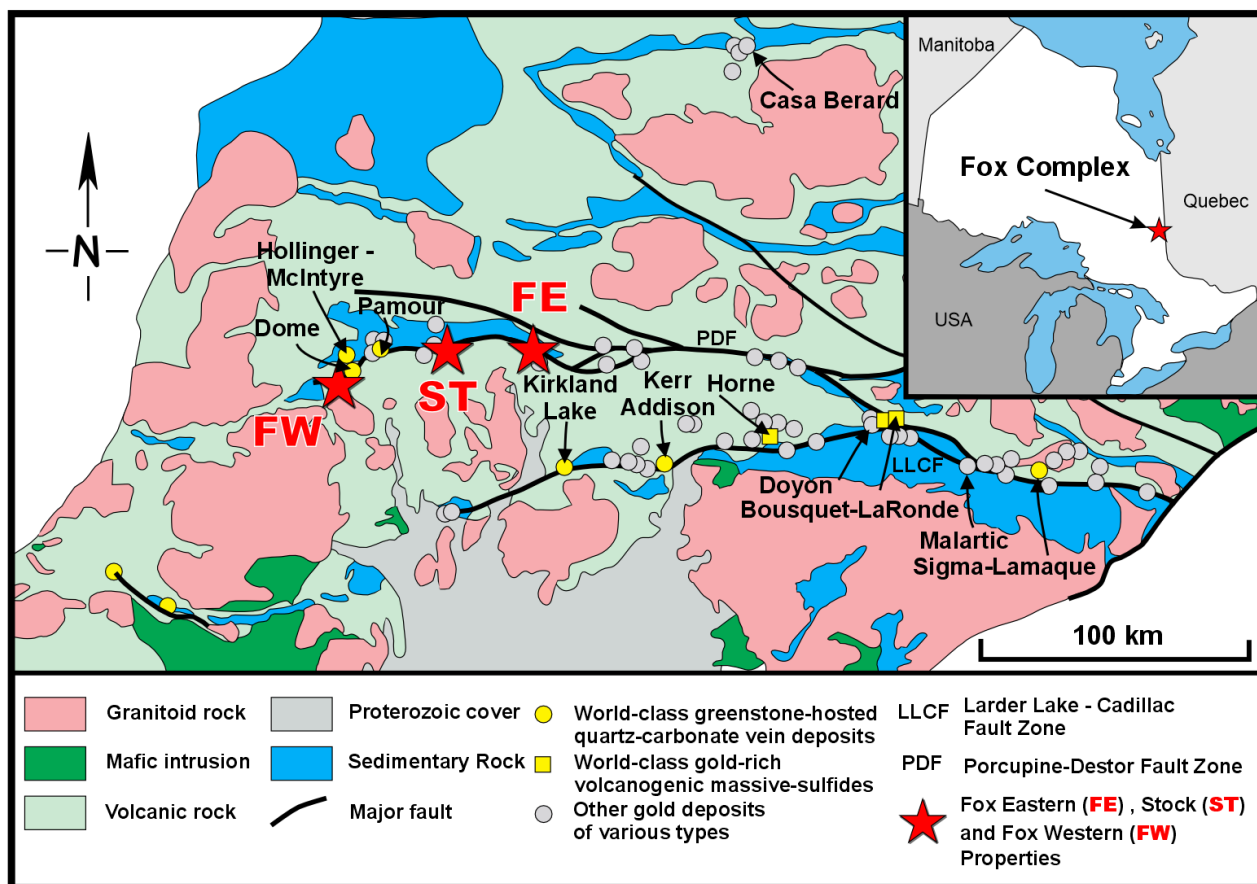


Figure 7-1: Geological Map of the Timmins-Matheson Sector within the Abitibi Subprovince showing Distribution of Gold Mining Operations

7.1 Regional Stratigraphy

The Fox Complex properties are underlain by Precambrian rocks of the Southern Abitibi Greenstone Belt (SAGB), located in the central part of the Wawa-Abitibi Subprovince, southeastern Superior Province, of northeastern Ontario (Figure 7-2). The SAGB consists of numerous intercalated assemblages of 2,750 Ma (million years) to 2,695 Ma metavolcanic rocks and their intrusive equivalents, which are unconformably overlain by the younger 2,690 Ma to 2,685 Ma Porcupine and 2,677 Ma to 2,670 Ma Timiskaming metasedimentary assemblages and alkalic intrusive rocks.



Source: (Modified after Poulsen et al. 2000, and Dube and Gosselin 2007)

Figure 7-2: Map of the Southern Abitibi Greenstone Belt

7.2 Mineral Occurrences & Mineralization Types

Gold mineralization at both the Eastern and Stock Properties is part of a metallogenetic domain, and shares similarities with ultramafic-hosted and associated deposits that occur along the PDDZ corridor between Nighthawk Lake and the Froome and Black Fox Mines to the East. This domain includes deposits such as those at Nighthawk Lake (Porcupine Peninsular Mine, Hopson Zone, Ronnoco deposit), the Aquarius Deposit, the Taylor Mine (West Porphyry, Shoot and Shaft Zones), and the Black Fox Mine. Similar ultramafic-hosted styles are also present along the Cadillac-Larder Lake corridor to the south, as is exemplified by the Kerr Addison Mine. All these deposits occur in association with sets of reverse quartz shear veins and associated sets of gently to moderately dipping quartz-carbonate-albite extension vein arrays in areas where strong complex rheological control influences the position of mineralization in areas of high strain. Specifically, mineralization at Black Fox, Stock and Froome West is interpreted to be orogenic lode gold deposits. The Grey Fox deposit, located about 3 km Southeast of Black Fox, appears to differ in mineralization style, with studies suggesting an epizonal to high level orogenic deposit based on current lithological, structural, and mineralogical observations.



7.2.1 Grey Fox

The Grey Fox deposit is hosted within mafic metavolcanic, metasedimentary, and syenite units, likely from the Lower Blake River Formation. The project is divided into two main blocks with distinct lithological packages. The south-eastern block (147-Grey Fox South Block) is a layered mafic metavolcanic sequence comprised of massive, pillowed, and variolitic metavolcanic units with minor interflow metasediment horizons. The north-western block (Gibson Block) is comprised of altered metasediments intruded by a syenitic stock (the Gibson Syenite). The metasedimentary and metavolcanic units in both blocks strike ~350° and are steeply dipping at approximately 70° to the east. They are angularly unconformable to the Lower Tisdale mafic and ultramafic metavolcanics to the north, which strike ~140° and dip 45° to the SW. The units are generally weakly foliated, although the foliation can increase locally in the presence of shearing. Some lithologies (including sedimentary horizons and contacts) may have localized displacement, as suggested by cataclastic breccias and narrow semi-brittle shear zones associated with mineralization (Ross and Rhys 2011).

The two blocks are separated by the Contact Fault which runs N-S along the contact between the metasediments and the mafic metavolcanics. The deposit is constrained to the NE and at depth by the A-1 thrust fault, which is likely part of an imbricate fan of the PDDZ and separates the Grey Fox units from the Lower Tisdale rocks, and to the SW by the sub-vertical Gibson-Kelore Shear Zone.

Most of the mineralization at Grey Fox occurs within numerous sub-parallel NE-SW striking sheeted vein sets referred to as C-1 veins (Figure 7-4) that dip steeply to the NW. These are accompanied by extensional conjugate C-2 veins, which are mainly E-W trending and south-dipping. Collectively known as C-veins, these are the dominant host of the mineralization in the Gibson, 147, 147NE and Grey Fox South zones. A third, flat lying set of veins (C-3) may also exist, but likely play a minor role in mineralization.

The orientation of the C-1 veins varies slightly across the property, striking ~230° at 147 Zone and Grey Fox South, rotating to 245° at 147NE. They are steeply dipping at ~68-70° to the NW. On the west side of the Contact fault, there is a noticeable difference in the orientation of the veining hosted within the Gibson syenite and metasediments, with the C-1 set with a strike/dip of ~223°/-67° in the south end of the Gibson zone, rotating to 210°/75° near the Gibson portal, and rotating back to ~240°/88° at the north end of the Gibson zone. The differences in the strike/dips of the C-1 veins between the two sides of the Contact Fault is likely due to the rheological contrast between the mafic metavolcanic rocks on the east side of the fault versus the metasediments on the west side.

The “C veins” are quartz+albite+ferromagnesian dolomite veins. They commonly display crustiform textures with molybdenite±graphite stylolites and can be brecciated in the larger vein sets. They range in scale from 1 cm-2 cm to 2 m-3 m wide and can occur individually or as anastomosing lenses with strike lengths between 100 m-700 m. The veining appears to have textures associated with epizonal orogenic gold deposits, and this is supported by fluid inclusion studies (Tuba, 2019). Gold occurs as free gold or associated with fine grained sulphides and is found around quartz grains and in the stylolites and along vein margins. The distribution of the gold within the veins tends to be nuggety/inhomogeneous.

The host lithology also plays an important role in gold deposition. Mineralization at the 147, 147NE, and Grey Fox South zones is strongly concentrated where the NE-SW trending C-veins intersect the N-S striking variolitic iron-tholeiite flows. These intersections create strong, steep mineralized plunges to the



north hosted within the flows. This is likely due to the chemical interaction between the silica+carbonate rich gold bearing fluids in the C-veins and the iron rich metavolcanics creating favorable conditions for gold deposition. Mineralization, especially at the Grey Fox South zone, is associated with hematization which occurs in albite-carbonate dominant alteration assemblages often peripheral to mineralized zones, and also as outer envelopes to some veins. Pyritic carbonate-albite-sericite alteration generally overprints the hematite, suggesting that much of the pervasive hematite is early, although later structurally controlled hematite is suggested in vein envelopes as well. The presence of hydrothermal hematite and carbon in vein envelopes suggest that alternating redox states, potentially in response to fluid mixing or evolution, may have contributed to gold deposition.

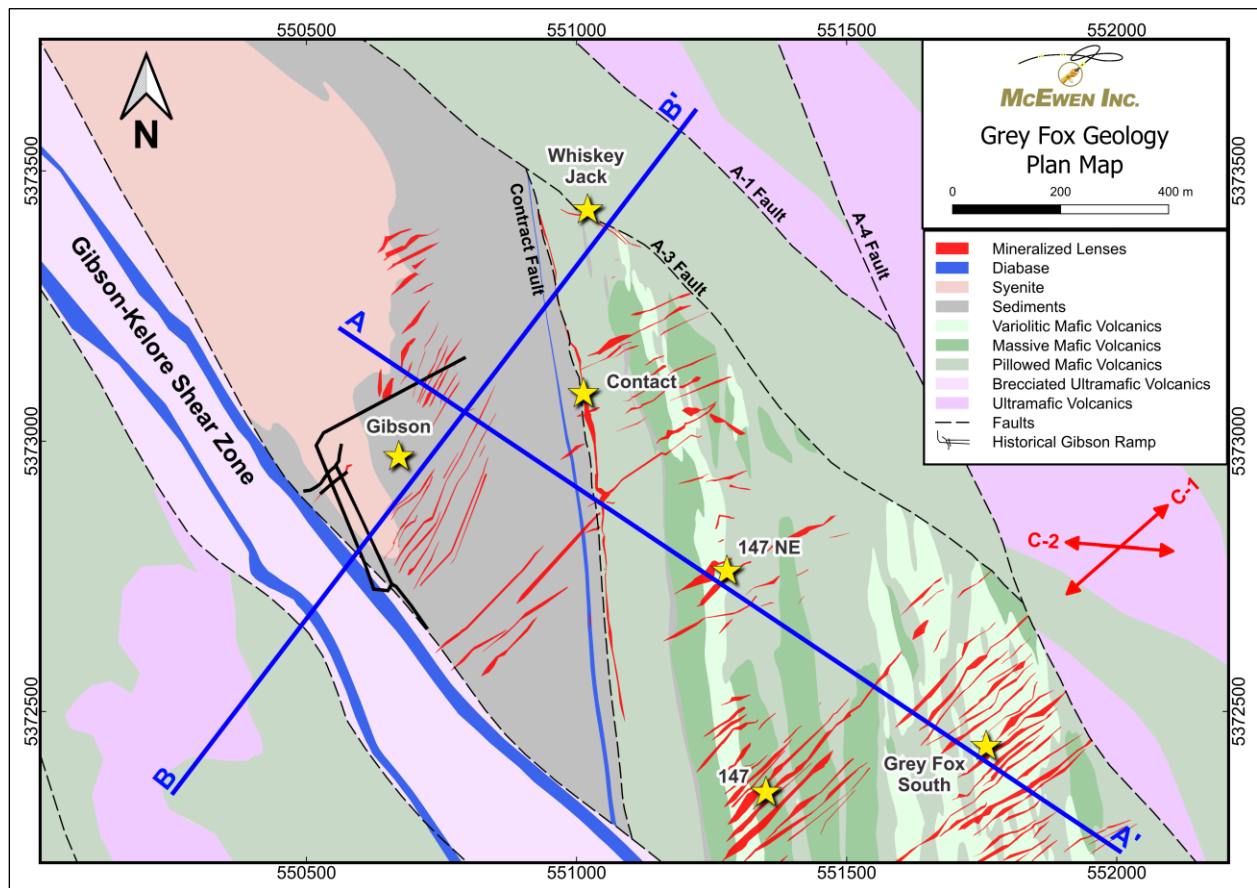
Mineralization at the Gibson zone is mainly hosted within C-veins that crosscut the metasediment unit (Gibson metasediments) and into the Gibson Syenite with the highest concentrations within the C-veins in the metasediments and proximal to the intrusion itself. Gold mineralization in the veins within the Gibson metasediments is typically associated with quartz+sericite+sulphide alteration, along with visually significant but unquantified molybdenite±graphite mineralization in stylolites within the veins and along fractures in the host rock.

The concentration of gold in the C-veins within the Gibson Syenite tends to be lower than in the metasediments with a few exceptions such as the historic bulk sample at the Gibson ramp, and the deeper breccia zone to the southeast of the intrusion. The deeper breccia zone is associated with alternating intense biotite+chlorite+albite alteration and amphibole+sericite+potassium feldspar+dolomite alteration. Drilling has suggested that high grade zones exist below 500 m from surface, although further work is needed to confirm the depth extent of these zones. Minor chalcopyrite-covellite-bornite mineralization is also disseminated throughout the intrusion.

In addition to the sheeted C-veins, gold is also concentrated along the N-S Contact Fault (Contact Zone) and the Whiskey Jack shear. At the Contact Zone, the rheological contrast between the metasediments and the mafic metavolcanics provides a pathway for the gold-bearing fluids. The gold in this area is associated with an increase in quartz+sericite+pyrite alteration in the wall rock, with much of the gold depositing in the metasediment side of the fault.

At Whiskey Jack, the gold mineralization is associated with ferromagnesian dolomite+sericite rich shear veining which occurs extensionally between the A-1 and A-3 shears with a strike/dip of ~125°/65°, parallel to the A-3 fault in that location. Veining consists of multiple sets/pulses of cataclasite textured material over widths of up to approximately 7 m. This veining and associated shearing appear more brittle-ductile suggesting a deeper depth of formation than the C-veins. Compared to the nuggety C-veins, the mineralization in the Whiskey Jack shear veins tends to be more continuous with higher average grades, but lower overall contained ounces when compared to other zones at Grey Fox.





Source: (McEwen 2024)

Figure 7-3: Grey Fox Area Schematic Geological Plan Map with section lines (A-A' and B-B')



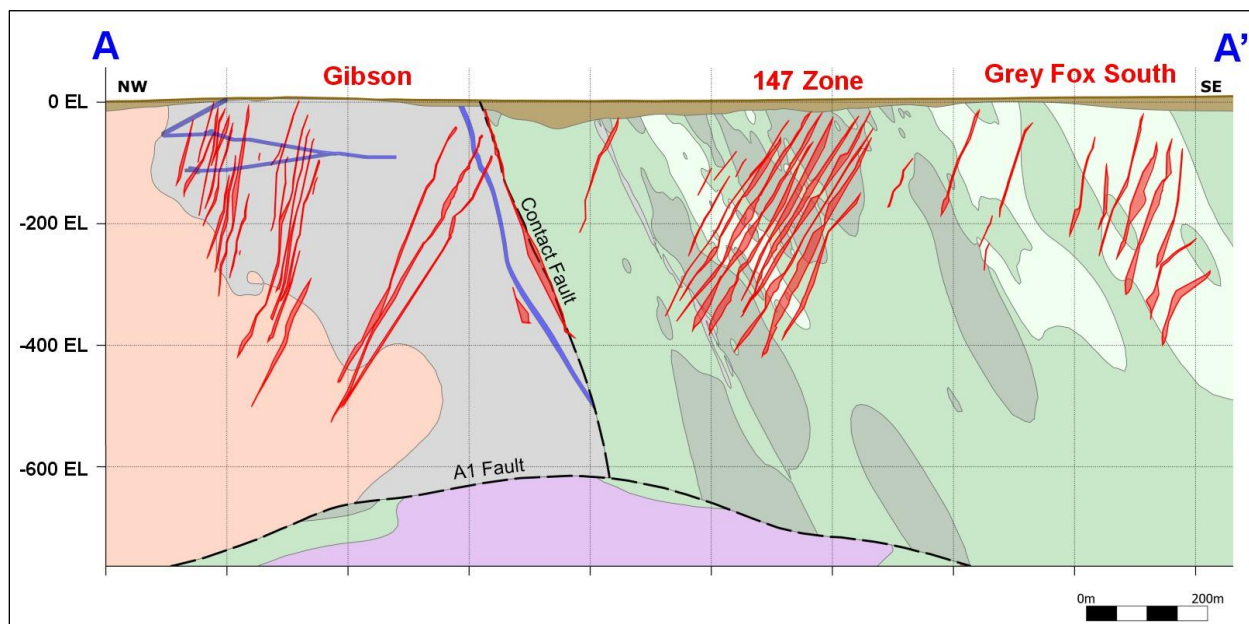


Figure 7-4: Section A-A' cut perpendicular to the principal C1 vein orientation at Grey Fox

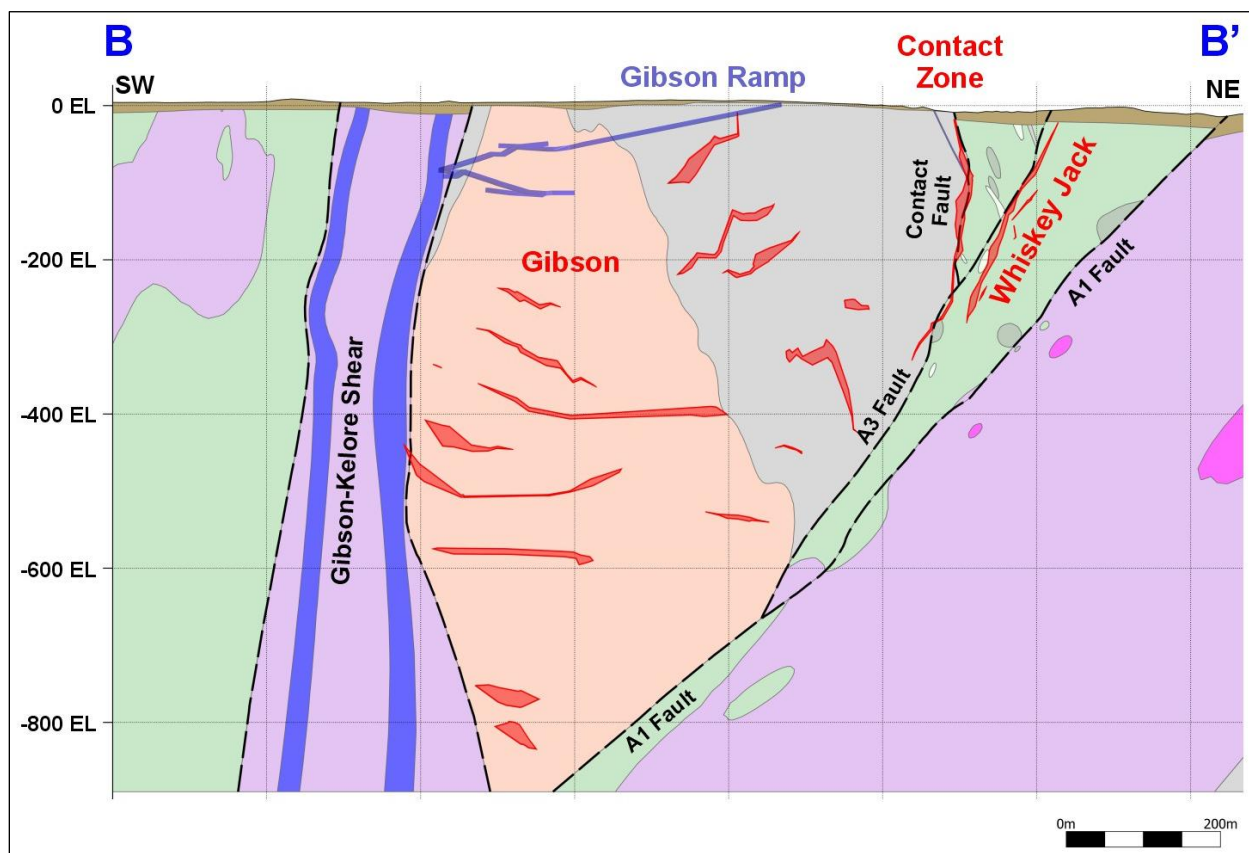
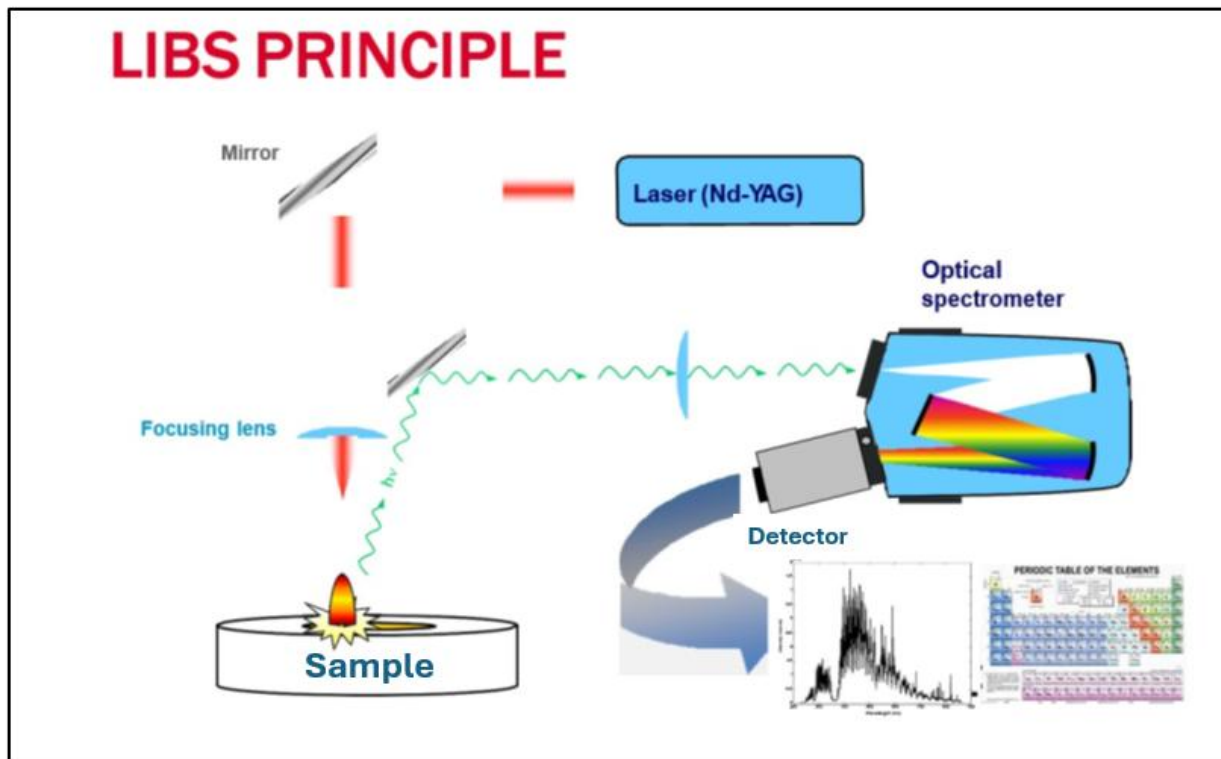


Figure 7-5: Section B-B' cut perpendicular to the Whiskey Jack Zone, the A1 and A3 faults and the Gibson-Kelore Shear



In July of 2024 a deportment study by Jacques Marchand Engineering Consulting on the nature of mineralization at the Whiskey Jack and Gibson zones was commissioned by McEwen. This study included state of the art LIBS (Laser Induced Breakdown Spectroscopy) scanning on select drill core from those two zones. Scanning was performed by an Elemission E CORE LIBS scanning machine in Montreal, QC, Canada. Refer to Figure 7-6 for a diagram on LIBS technology.



Source: (Marchand, 2026)

Figure 7-6: Basic principles for LIBS scanning (Modified after Marchand, 2026).

Some of the key observations from this deportment study include:

- Gold primarily exists mostly as finely plated particles in pyrite cleavage surfaces or intergranular.
- The gold grains identified are predominantly less than 30 μm in size and even as fine as 5 μm .
- The presence of magnetite is likely to lower the gold grade in the mineralized host rocks regardless of lithology.
- Sulphide content is typically low, less than 1%, and the gangue is composed of a significant amount of carbonate.
- A portion of the Grey Fox deposit may be a hybrid type of gold deposit between porphyry copper system and an orogenic gold system.

7.3 Stroud Property

Mineralization on the Stroud property is interpreted to result from a series of continuous structural and hydrothermal events. Following the accumulation of the volcano-sedimentary pile, folding and subsidence,



accompanied by the regional and local development of stress couples, set up a tensional regime whereby dilatant zones were developed, generally preparing the ground for the subsequent series of hydrothermal events, culminating in gold deposition. The development of the regional and local fabric elements, accompanying the folding and fracturing/shearing formed the channel-ways for the auriferous siliceous fluids, resulting in the present gold-bearing siliceous mineralized structures (Wetmore, 2004).

The structure (or deposit) at Stroud is subdivided into three zones, from NW to SE: the Creek Zone, the Creek Zone Extension and the Main (Gunnex) Zone. The Creek Zone Extension is located in the vicinity of late, unmineralized diabase dykes. Within the mineralized structure, shear intensities and rock types vary, and over a general thickness of approximately 20 m to 45 m in width, the gold mineralization associated with “gray veins” (injection of grey quartz-carbonate veins) and siliceous systems is interpreted to occur in shoots in the form of anastomosing networks in plan and section (Wetmore, 2004).

The 2025 drill program at Stroud confirmed the presence of previously reported mineralization and was also consistent with overall structural models for the deposit. Refer to Figure 7-7 for an updated geological plan view map of the Stroud deposit and Figure 7-8 for a longitudinal section of the updated resource block model.



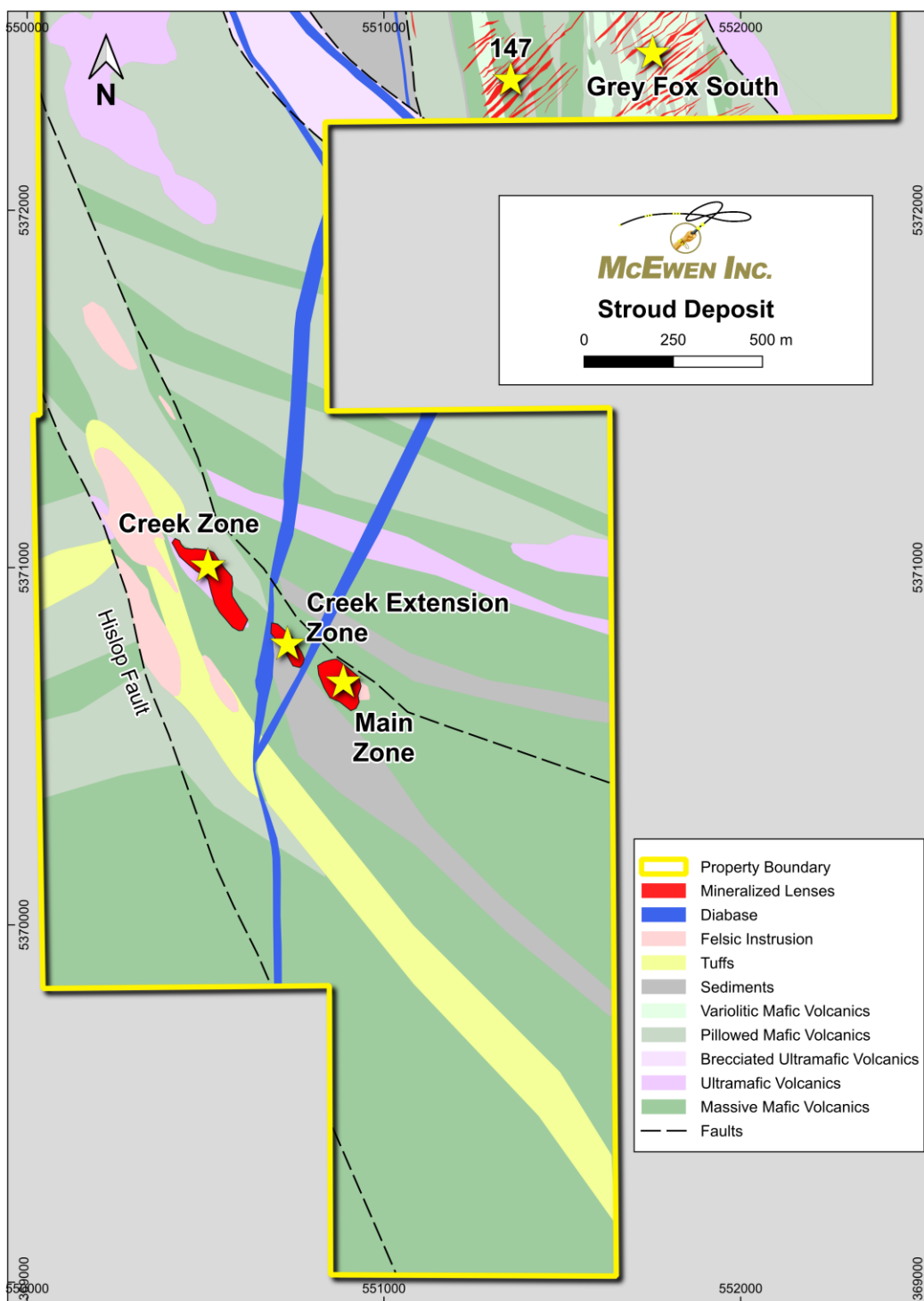


Figure 7-7: Plan View Geology Map of the Stroud Deposit



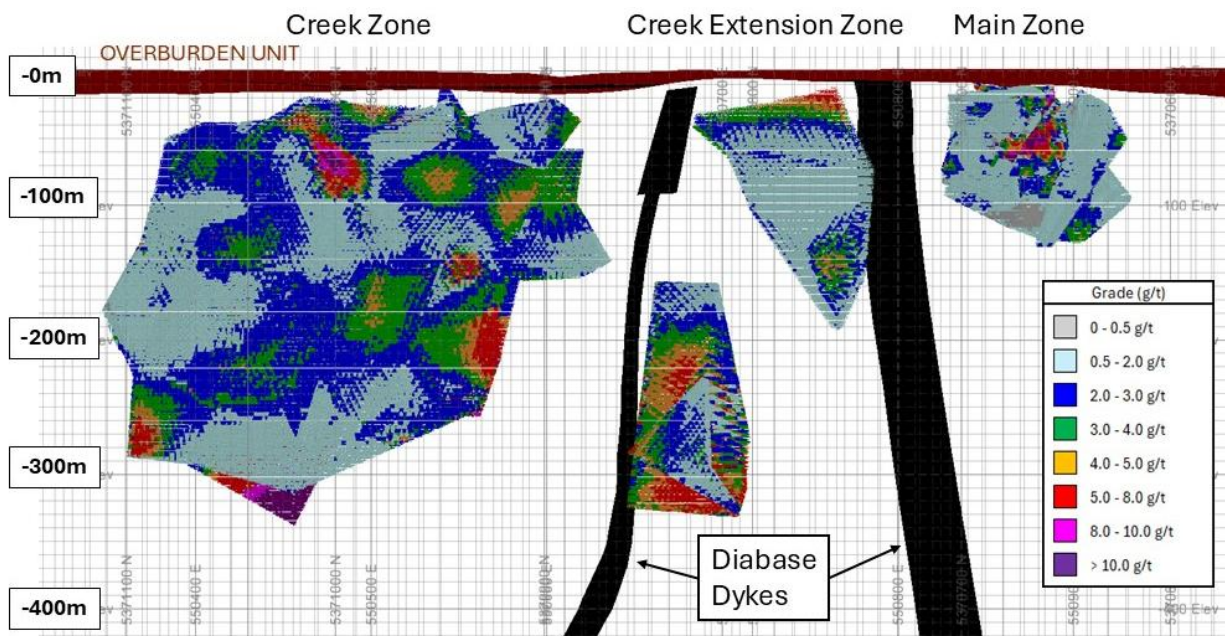
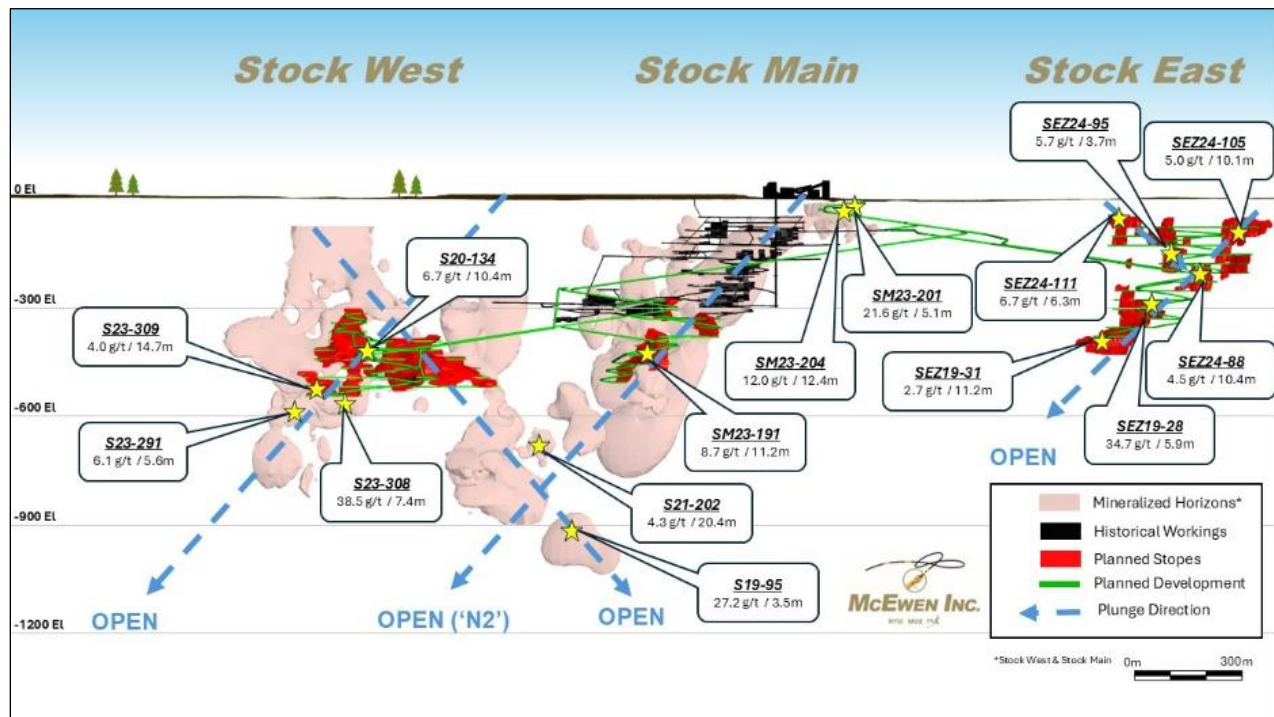


Figure 7-8: Longitudinal Section (looking NE) of the Updated Stroud Resource Block Model

7.3.1 Stock Property

The prospective PDDZ corridor tracks across approximately 7 km through the Stock property. To date, most of the exploration has been focused on assessing a 3 km segment of this and includes the area of the former Stock Mine, as well as at Stock East and Stock West (Figure 7-9). The (historical) Stock Mine deposits (N and M Zones; West and Central zones), have a moderate west plunge defined by the lenticular to lobe-like shapes of hosting mafic volcanic rocks and surrounding carbonate alteration envelope. These are surrounded by lenses of highly strained, talc-chlorite ultramafic rocks (Siragusa, 1994). Limited historical documentation suggests that much of the mined material is comprised of swarms/zones of white quartz veining (both shear and extensional) hosted within sericite-carbonate altered mafic volcanic bands and along mafic-ultramafic contacts. The quartz shear veins were often accompanied by disseminated pyritic mineralization in shear zones. Pyrite altered, albitized dykes formed additional areas of mineralization sometimes associated with the shear veins (Siragusa, 1994).



Note: Image facing north. All drill results are presented as TWs.

Figure 7-9: Project-wide Longitudinal Section of Stock

Stock East is directly associated with the Nighthawk Lake Fault Zone (NHLFZ) which splays off the PDDZ in proximity to that deposit. Gold mineralization in the Stock East area appears to be related to enhanced gold-pyrite accumulation (often exceeding 10%) and is often located within well-developed quartz breccias. In addition, the deposit is in the hangingwall (HW) of the NHLFZ whereas the Stock West deposit is located between the PDDZ and NHLFZ. See Figure 7-10 for a typical cross-section through the Stock East deposit.

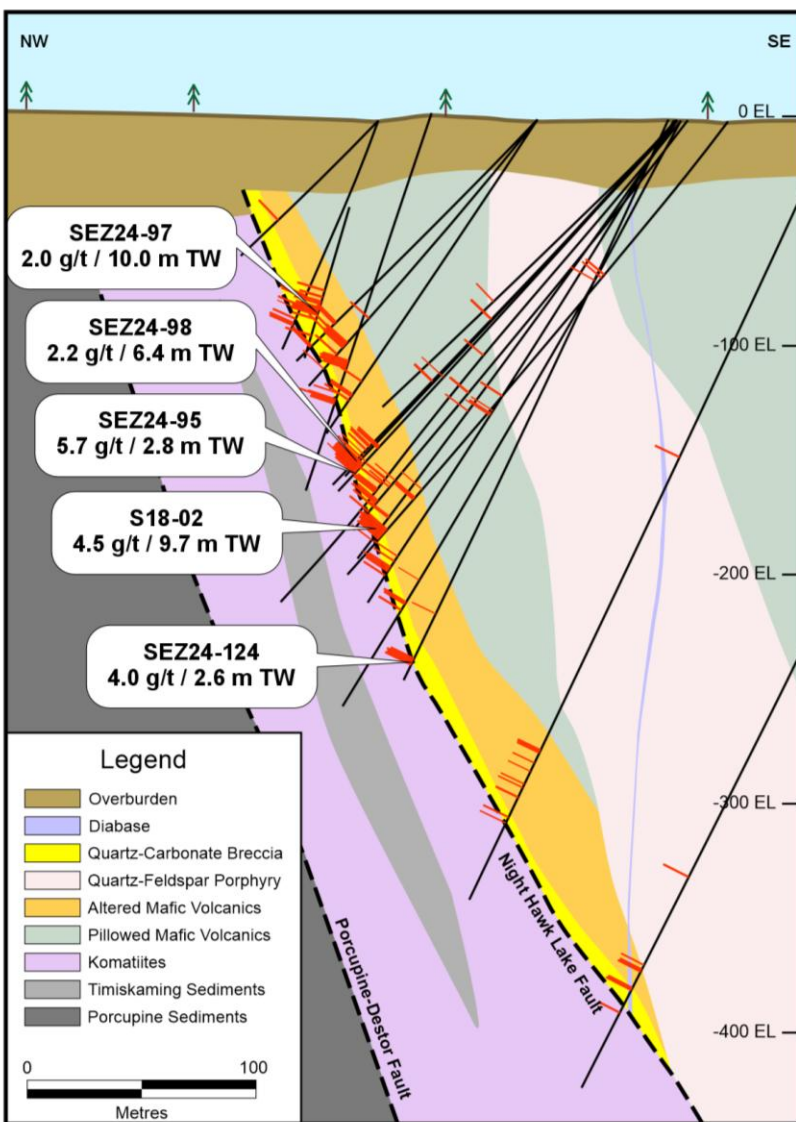
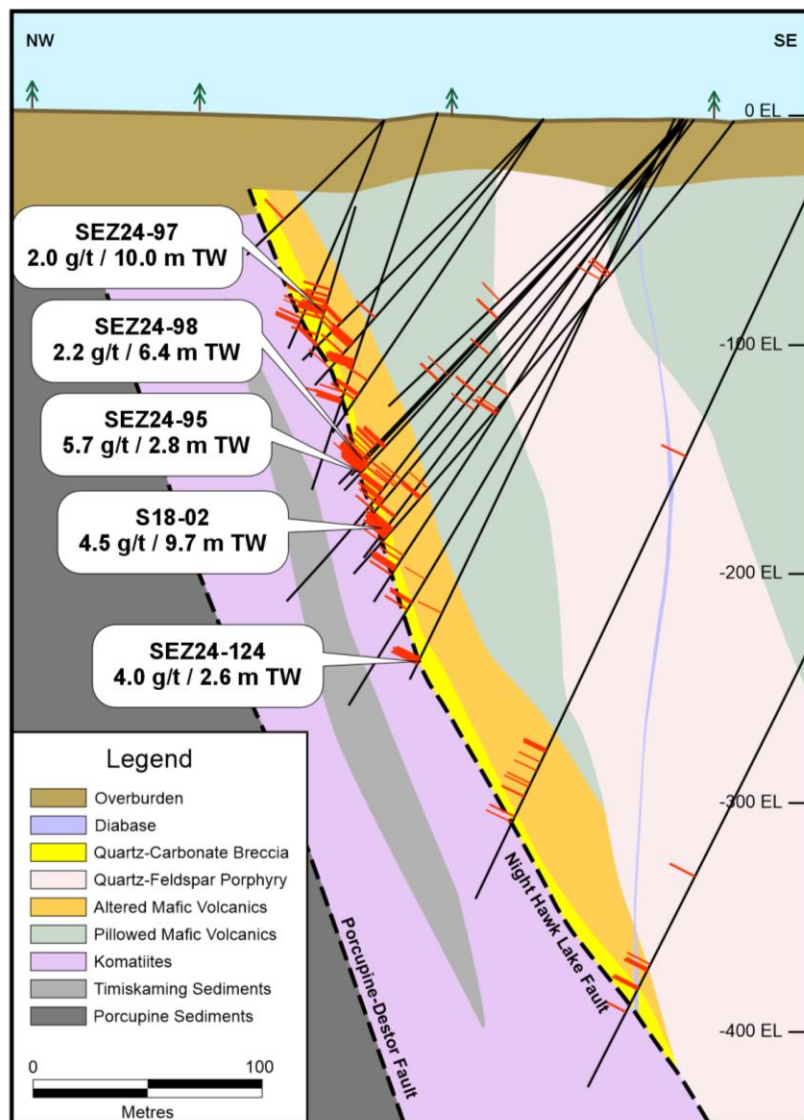


Figure 7-10: Exploration Drilling at Stock East, Schematic Cross-Section Looking to Northeast

The Stock West deposit was discovered in mid-2019 and is associated with the PDDZ, approximately 1.1 km west of the historical Stock Mine shaft, in an area where the NHLFZ has bifurcated/splayed away from the main PDDZ. The section of the Stock West deposit (which steeply dips to the south) is characterized by the following:

- Chlorite (talc ± serpentine schists) with variable green carbonate ultramafics Ankerite alteration
- Minor mafic volcanic rocks
 - In the footwall (FW) to the north, these rocks may be silica-sericite altered, pyritized, and quartz veined bound by Porcupine sedimentary rocks.
 - In the HW to the south, these rocks may be a variably altered feldspar porphyry or relatively undeformed mafic volcanic rocks, likely of the Tisdale Assemblage.

In the Stock West drilling program, principal intercepts obtained in 2019 occur over an approximately 300 m by 200 m area within which multiple >5 g/t Au intercepts have been obtained over intervals up to several tens of metres in green carbonate ultramafics. Adjacent holes to the higher-grade areas show that this body thins laterally and is probably lenticular in form (Figure 7-11).



Source: (McEwen 2024)

Figure 7-11: Exploration Drilling at Stock West, Schematic Cross-Section Looking to Northeast

The overall texture is consistent with a possible origin as a potentially pyroxene-olivine porphyritic ultramafic intrusion that is now completely altered, which may have been coeval with the finer-grained, probable komatiite flow sequence that is host to the intrusion. Local abrupt variations in its grain size from medium to coarse grained suggest potential for relict textures of pegmatitic phases (Rhys and Ross, 2020).



7.3.2 Western Properties

Many deposits spread over the Porcupine Mining Camp have contributed to approximately 70 Moz of gold production since 1910. Common characteristics of this significant gold deposition are listed below:

- Dominant source of gold is within quartz vein lodes containing locally coarse free gold.
- Greater than 50% of the major Hollinger deposit's gold is associated with pyrite formation.
- Quartz vein lode deposits are structurally controlled areas of dilatancy which permitted the development of vein zones.
- The majority of gold production in the camp is hosted by rocks of the Tisdale Assemblage.

Minor production came from 1) pyrite-bearing pyroclastics within the mafic volcanic rocks of the Tisdale Group, 2) vein sets intruding local felsic porphyry bodies (Pearl Lake and Miller porphyries), 3) quartz veins within the sediments of the Porcupine and Timiskaming Assemblage (Dome Mine) where they unconformably overlie productive portions of the Tisdale Assemblage.

The main minerals of these gold-bearing zones are quartz, carbonates, alkali feldspar (most commonly albite), sericite, pyrite, tourmaline, arsenopyrite, scheelite, and molybdenite. Pyrrhotite is common in the deep parts of deposits, as well as in deposits hosted in banded iron formation. Arsenopyrite seems to be common in deposits hosted in sedimentary rocks.

The concentration of gold may be considered a product of the alteration process, as well as the concentrations of barite, tungsten, antimony, tellurium, molybdenum, and arsenic. Although gold in quartz veins is the most distinctive occurrence, the gold in some deposits is found predominantly within the wall rock.

7.4 Structural Geology

Major crustal-scale faults, such as the PDDZ and Cadillac-Larder Lake Deformation Zone (CLDZ) commonly occur at assemblage boundaries and are spatially associated with east-west trending belts of Porcupine and Timiskaming assemblage metasedimentary rocks. These major faults record multiple generations of deformation, including normal, strike-slip, and reverse movements. The PDDZ and CLDZ define a corridor of gold deposits, generally known as the Timmins-Val D'Or camp (Robert et al., 2005), which accounts for the bulk of historical and current gold production from the Superior Province.

The PDDZ extends across the Highway 101 area, continuing westward to the Kapuskasing Structural Zone (Ayer et al., 1999a) and eastward through Québec to the Grenville Front area (Mueller et al., 1996) for a total distance of more than 600 km. The PDDZ strikes southeast in Hislop Township and generally becomes more east striking along the rest of Highway 101 (Siragusa, 1994). The deformation zone is complex, with different structural styles restricted to specific segments. Each segment is bound, to a first order approximation, by prominent north-northwest-striking faults that transect the PDDZ (Figure 7-12Figure 7-12).

Several important secondary structures occur in the vicinity of the Black Fox Mine. The Gibson-Kelore Deformation Zone (GKDZ) is one of the most recognizable structural features on the Black Fox and Grey Fox properties and is defined by a strongly fractured-broken band of talcose-chloritic schist. Regional



airborne geophysical surveys indicate that this splay departs the regional track of the PDDZ and trends south-eastwards for approximately 4 km to 5 km, passing off the southern property limits. Studies suggest the steeply southwest dipping GKDZ brittle-ductile structure contains schist, fault gouge and extensive fracturing. West of the Fault, the stratigraphy is an east-striking, south-facing homoclinal sequence. Structural fabrics are commonly nonpenetrative fracture cleavages.

The GKDZ is one of the five regionally bounding cross-faults that separate different segments along the PDDZ from Timmins to Québec. The structural style and setting of gold mineralization in each segment is different and knowledge of these differences can be used to tailor exploration programs specific to each segment (Berger, 2001).

Jensen (1985) identified the Ross Fault as the northwest-striking lineament immediately east of the Ross Mine. Berger (2002) has modified the extent and strike of the fault based on detailed airborne geophysical data. The fault is located near or on the inferred axis of an anticline that closes in the vicinity of the Ross mine.

The Arrow Fault is a local name applied to a shear zone striking 085° located near the southern limit of the Grey Fox cluster of exploration targets. The fault is defined by a prominent linear disruption in airborne magnetic patterns and corresponds to sheared rock on the ground (Berger, 2002).



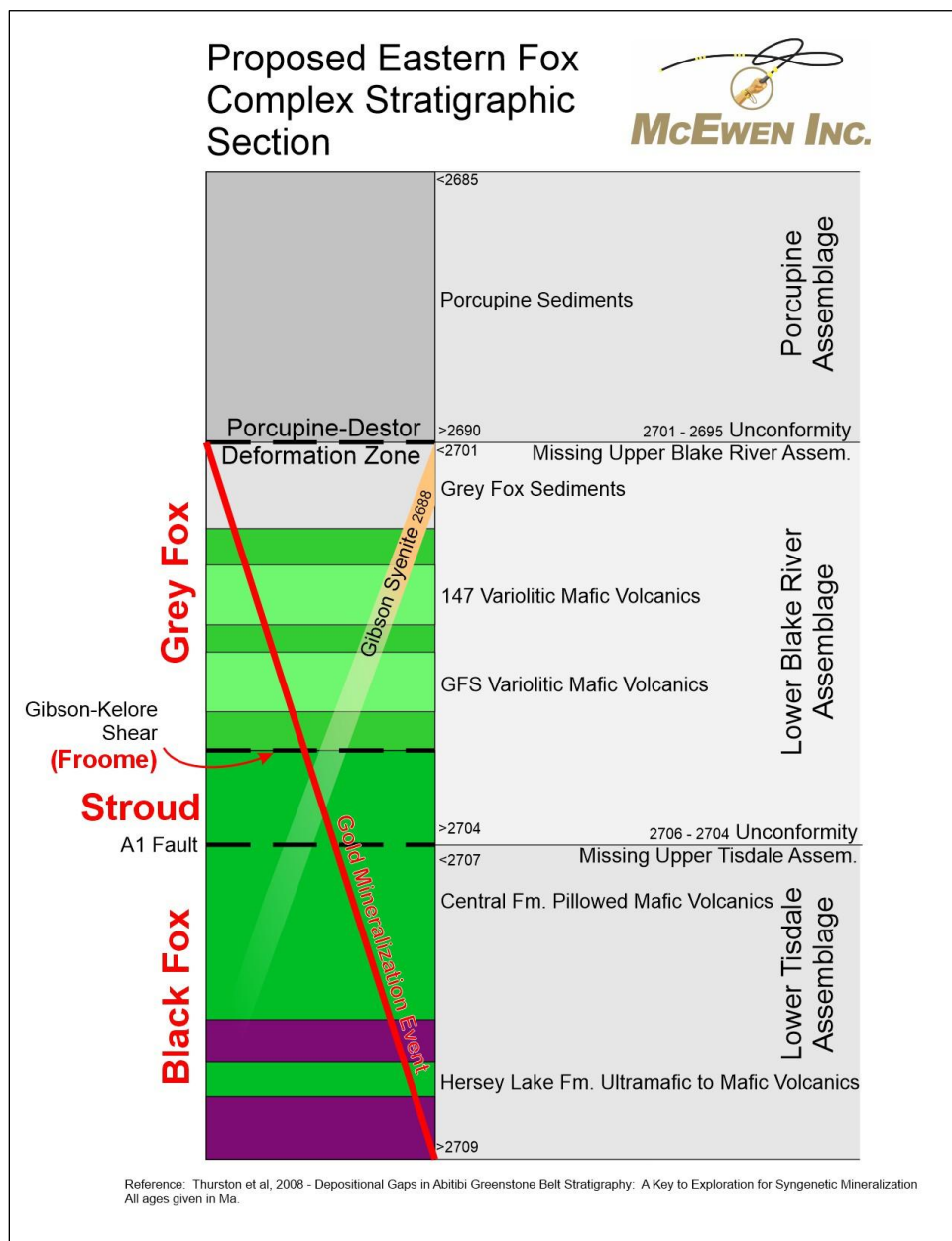


Figure 7-12: Stratigraphic Column for the General Black Fox – Hislop Area

West of the Hislop Fault, the PDDZ strikes southeast to east and dips 45° to 65° to the south. The PDDZ marks the contact between the Porcupine and Tisdale Assemblages and is characterized by mafic and ultramafic schist in zones that range from 250 m to 800 m wide, as well as numerous foliation-parallel and crosscutting brittle faults. The main trace of the PDDZ is accurate between the Hislop and Garrison faults (Berger 2002). Clastic and chemical metasedimentary rocks of the Timiskaming Assemblage occur within the deformation zone that varies between 100 m and 1,500 m wide. Talc-chlorite schist occurs along the north margin of the deformation zone in the Tisdale Assemblage and is indicative of ductile strain. The southern limit of the deformation zone is marked by brittle-ductile faulting accompanied by diabase dyke intrusions and abrupt contacts between the Lower Blake River and Timiskaming Assemblages. The



deformation zone is near vertical and kinematics are poorly constrained. North-northeast and north-northwest brittle and brittle-ductile faults transect and offset the PDDZ.

One splay structure is also important to the Stock Mine property setting. The southwest-trending NHLFZ bifurcates/splays away from the main PDDZ in the vicinity of the Stock East deposit and has been traced by regional geophysical survey responses for approximately 15 km into the historically mined and explored Nighthawk peninsula area.

7.5 Property Geology

The Eastern Property straddles a 5 km segment of the regional PDDZ, between the GKDZ to the southwest and the Porcupine sediments to the northeast (Figure 7-13). The Black Fox and Froom mines are underlain by a complex series of faulted-thrust slabs and wedges structurally emplaced into the 500 to 1,000 m belt that separates the PDDZ from the semi-parallel GKDZ splay structure. These intercalated, nonconformable units are generally composed of Lower Tisdale Assemblage, localized felsic intrusive rocks, and localized wedges of Porcupine and Temiskaming clastic sedimentary rocks. Grey Fox appears to lie stratigraphically above Black Fox and Froom and is possibly part of the Blake River assemblage. Figure 7-9



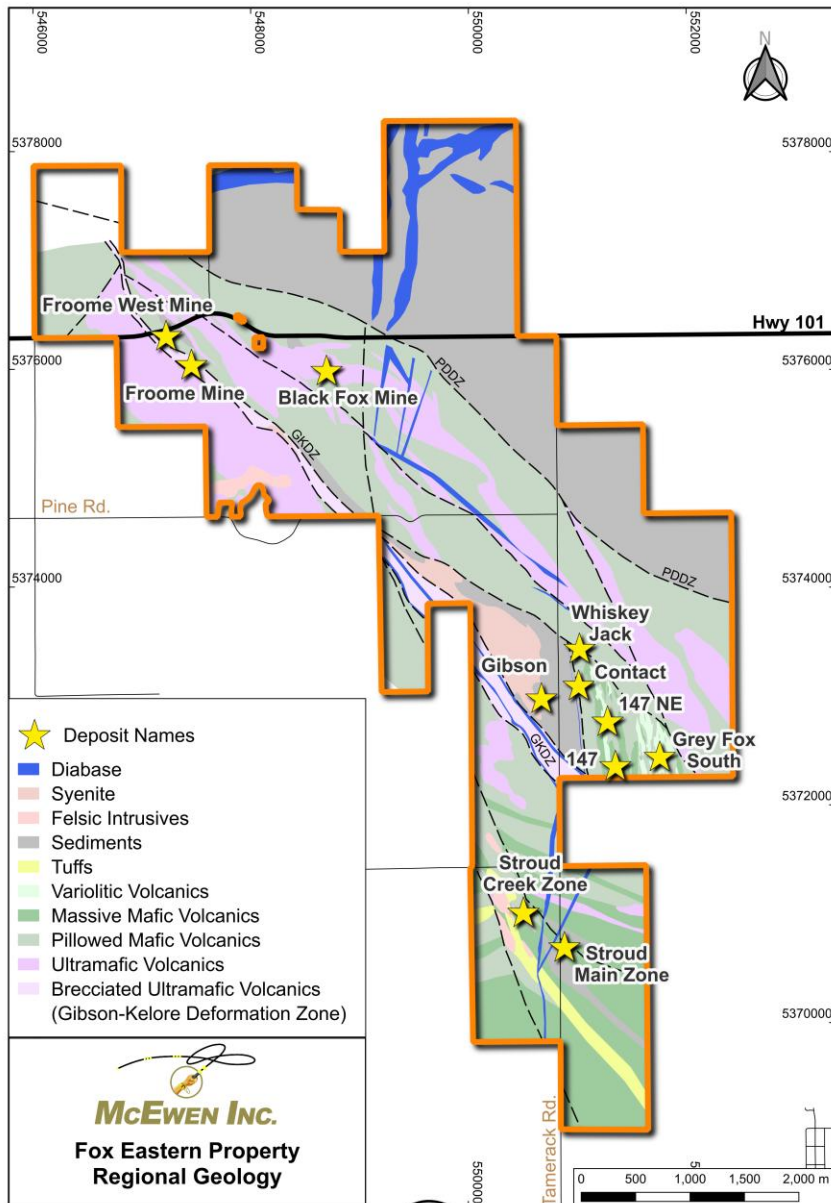


Figure 7-13: Interpreted Geology for the Fox Eastern Property

7.5.1 Grey Fox

The Grey Fox deposit is underlain by an overturned, steeply east dipping assemblage of metasedimentary and massive, pillowed, and variolitic mafic metavolcanics with minor interflow sedimentary rocks. In the western part of the area, a syenitic to dioritic feldspar porphyry (locally referred to as the Gibson Syenite) intrudes the metasedimentary unit.



The deposit area is bound to the West by the steeply southwest dipping GKDZ (which juxtaposes a chlorite – talc schist, typical of the GKDZ at the Black Fox Complex) and a locally brecciated feldspar porphyry. This fault contact is locally crosscut by diabase dykes, likely of the Matachewan dyke swarm. To the east, the Grey Fox deposit is bound by a moderately- to steeply-dipping to the west/southwest fault. The HW rocks are comprised of mafic metavolcanic rocks, metasediments and the syenite package of the Grey Fox area, while the FW rocks are made up of moderately to steeply southwest dipping ultramafic and mafic metavolcanic rocks correlated with those hosting the Black Fox deposit. A penetrative foliation is developed with the bounding faults, and locally in narrow intra-package shear zones associated with mineralization and along contacts. No tectonic fabric corresponding to the regional fabric has been identified within the metasedimentary-metavolcanic-feldspar porphyry package hosting the deposit.

The metasedimentary package, previously assigned to the Timiskaming Assemblage by Berger (2002), is up to 350 m true width and consists of graded sandstone to mudstone layers, with bedding tops to the west. However, geochronological sampling by McEwen in 2021 of the Gibson syenite, which is observed to crosscut the sediments, returned a date of $\sim 2,688.3 \pm 0.9$ Ma, suggesting the sediments (and possibly the adjacent volcanics) are part of the older lower Blake River formation.

At the Contact Zone, the metasedimentary-mafic metavolcanic contact is steeply dipping to the east from surface to a depth of approximately 300 m, below which the contact rolls to a steep dip to the west. West of the 147 Zone, the metasedimentary-metavolcanic contact steeply dips to the east from surface to the extent of drilling at 500 m below surface.

The mafic metavolcanic package, originally assigned to the Tisdale Assemblage by Berger (2002) but is now thought be part of the Lower Blake River Assemblage, is situated between the east dipping metasedimentary package to the west and west dipping ultramafic metavolcanic package to the east. The metavolcanic package ranges from 300 m true width in the north to at least 800 m at the southern boundary of the property. The mafic metavolcanic package consists of interleaved massive to pillowed mafic flows and massive to pillowed variolitic mafic interflows, with true widths ranging from 10 m to 130 m true width. The mafic metavolcanic flows dip steeply to the east and are truncated by the A-1 fault at the contact with the ultramafic metavolcanic rocks to the northeast. In the central part of the deposit, an outcrop of pillowed variolitic metavolcanic rock suggest flow tops are to the west.

7.5.2 Stroud Property

The following description of the Stroud property is taken directly from the *Hislop Township Gold Project, Matheson Area, Ontario, Technical Qualifying Report* (Wetmore, 2004):

“Except where disturbed by faulting and minor folding, the predominantly metavolcanic rocks within the mineralized zone trend northwest and dip some 70 degrees to the southwest. The overlying argillites, arkose and arkosic conglomerates appear to conformably overlie the pillowed and massive metavolcanics. The ultramafic rocks appear to be semi-conformable within the Creek Zone and, in part, somewhat discordant in the Main Zone. The feldspar porphyry, likewise, shows both conformable and cross-cutting relationships to the metavolcanics. The steeply dipping, north to north-west trending diabase dykes cross-cut all rock units.



The hydrothermal alteration accompanying mineralization is typified by increasing carbonatization, sericitization, silicification and (plus or minus) hematite approaching the mineralization. Where hematized, the rocks have a pink cast. Altered metavolcanics, metasediments, and feldspar porphyry sections are subsequently bleached, appearing waxy yellow to buff-coloured, due to sericitization and carbonatization. Ultramafic rocks are generally dark waxy green where serpentinized or buff-coloured where carbonatized.

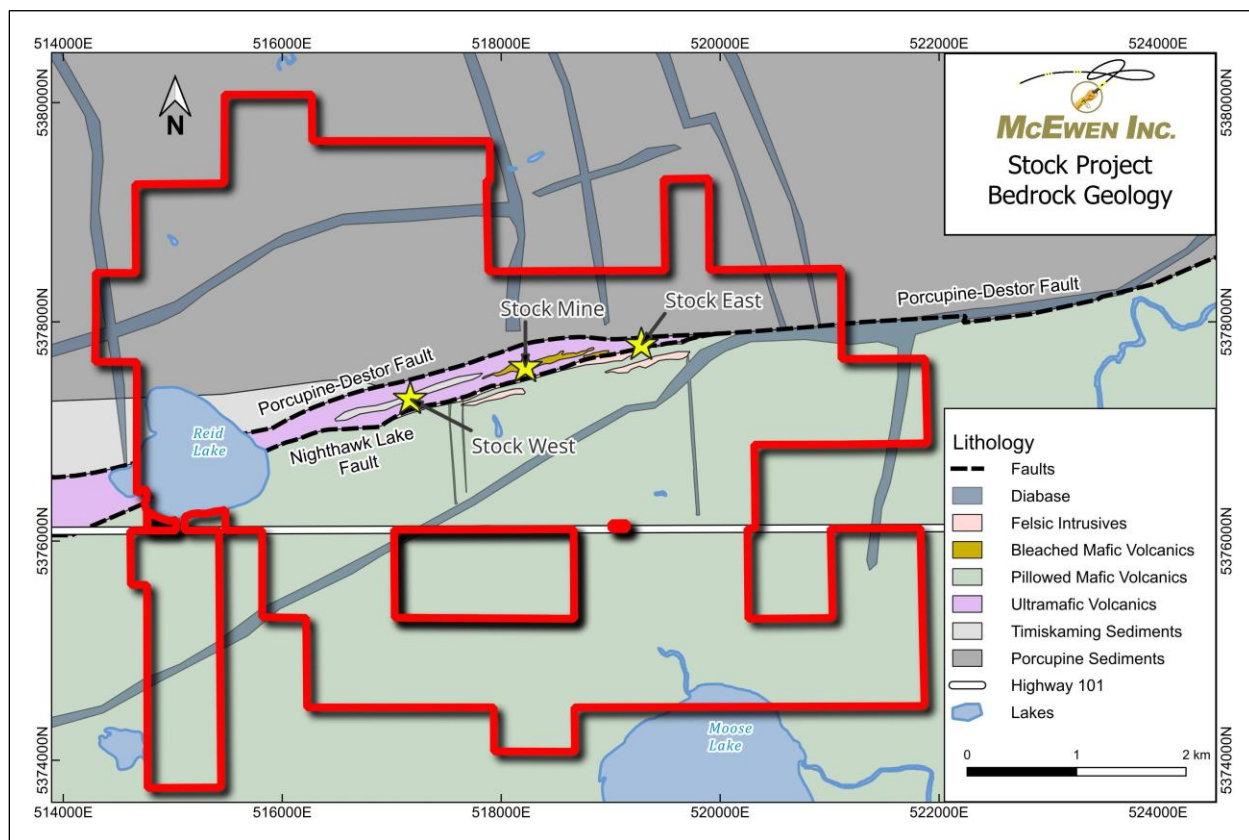
The indicated faults are generally cross-cutting to sub-parallel and exhibit ductile to brittle deformational characteristics. The most prominent of these is the steeply dipping Hislop Fault, and where encountered in drill core, is characterized by intense hydrothermal alteration, (hematization, sericitization), schistosity and /or fault gouge. The effect that it has on mineralization has not been determined.”

In 2025, McEwen drilled approximately 6,300 m at Stroud which was mainly designed to verify a historical resource. The geology and structures seen in the drill core were consistent with observations made from previous drilling campaigns performed on the property.

7.5.3 Stock Property

The Stock property occurs along the PDDZ corridor, which separates moderately- to steeply -south-dipping Porcupine Assemblage turbiditic metasedimentary rocks to the north from ultramafic and mafic metavolcanic rocks of the Tisdale Assemblage to the south (Figure 7-14). Both the Porcupine and Tisdale Assemblages young to the south, based on graded bedding, pillow facing, and stratigraphic distribution patterns both on the property and in drilling on adjacent properties, although folding in the Porcupine Assemblage results in local inversions. The juxtaposition of South-younging (2,680 Ma to 2,690 Ma) Porcupine Assemblage to the north of the fault towards older structurally overlying and south younging (>2,700 Ma) Tisdale Assemblage rocks to the south indicates substantial reverse displacement along the contact area that forms the main strand of the PDDZ to emplace the older volcanic rocks onto the younger sedimentary units.





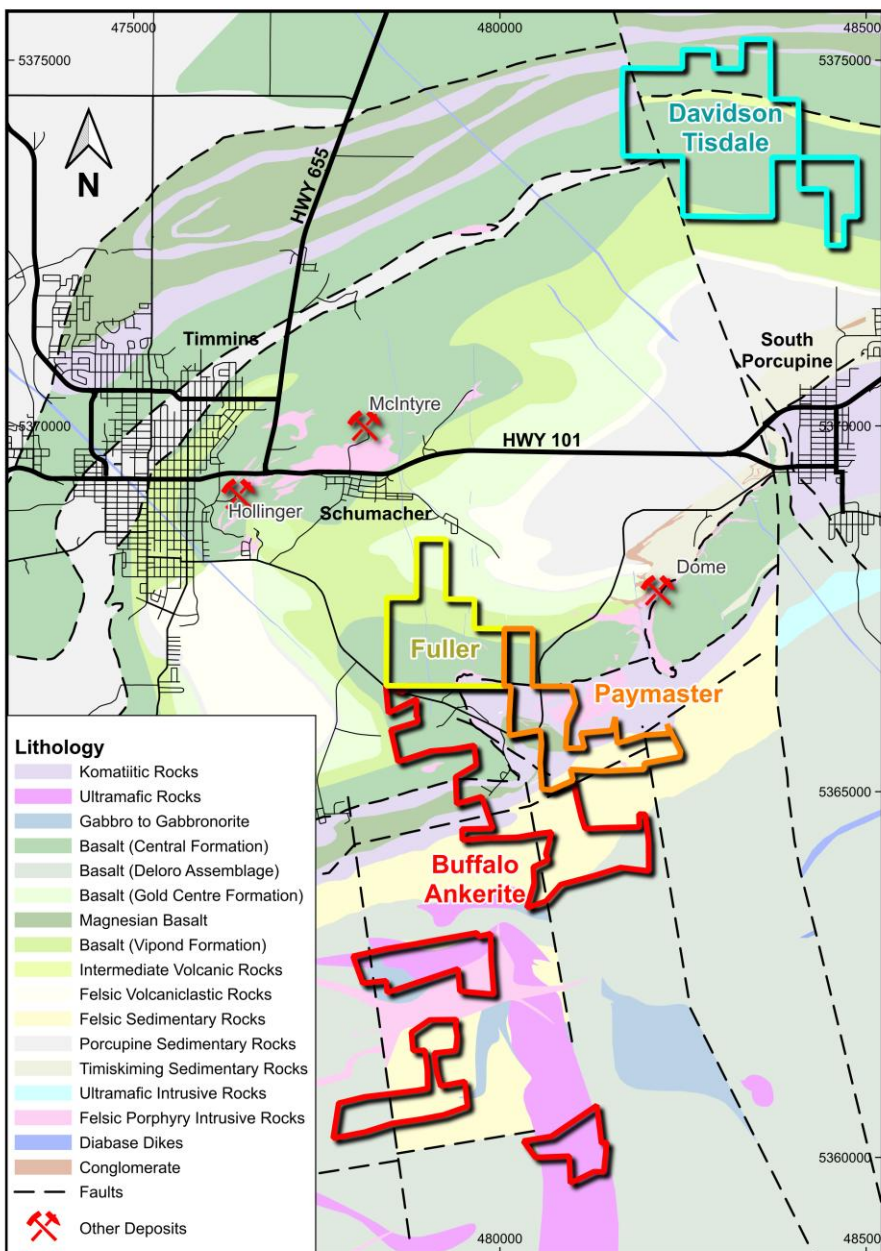
Source: (McEwen 2026)

Figure 7-14: Interpreted Geology of the Stock Property and Locations of the Significant Gold Mineralized Zones (yellow stars)

7.5.4 Western Properties

Generally, thick accumulations of glacial overburden cover more than 90% of the Timmins area. Only a limited number of bedrock exposures occur in the four Western Properties.

The Northern sector of the Porcupine Mining Camp is dominated by an irregular accumulation of Tisdale Assemblage volcanic flows, intercalated with several belts and irregular wedges of Porcupine sediments. This stratigraphic sequence is apparently focused along an irregular lineament (known locally as the New Mines Trend) diverging westwards off a southwest-trending flexure of the PDDZ at the Pamour Mine splay-point (Figure 7-15).



Source: (McEwen, 2026)

Figure 7-15: Schematic Geological Map of the Timmins Area, showing the Complex Stratigraphy-Structure Underlying the Western Properties

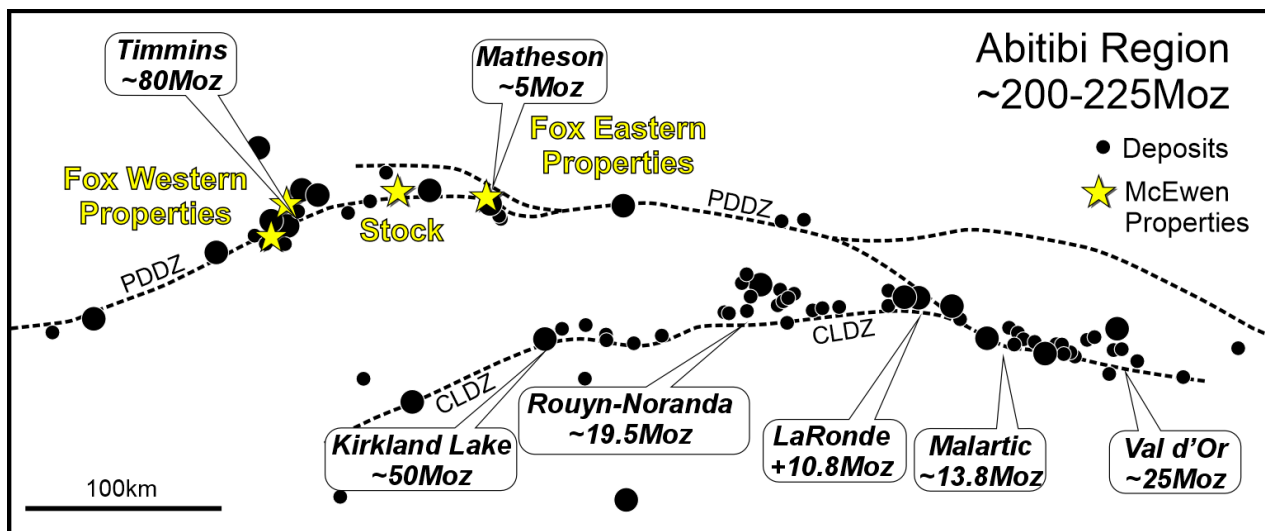
The southern sector of the central Porcupine Mining Camp is focused on a belt-sequence of mineralized bodies which extend from the Dome Mine (to east), through Buffalo Ankerite, Paymaster, and Fuller claims, to the western Delnite Mine property. They occur within the South Tisdale Anticline sector and are underlain by a sequence of ultramafic and mafic flows of the Hershey Lake Formation, and locally subdivided mafic flows (C-series) of the Central Formation. The discordant contact/delineation between these two formations is locally referred to as the Paymaster Shear (Pope, 2000).



8 Deposit Types

Numerous studies and exploration results over the past 50 years have focused on gold deposition styles and mineralization controls within the Abitibi Greenstone Belt. Most deposits are unique, with minor differences in geochemical characteristics and/or formational environments (Figure 8-1). Many of these gold deposit types can be grouped into “clans,” or families, of deposits that formed either through related processes or represent distinct products of large-scale hydrothermal systems (Robert et al., 1997; Poulsen et al., 2000).

The Eastern Properties, Stock Property, and Western Properties are all located proximal to an approximately 200 km central segment of the PDDZ, which has generated approximately 100 Moz of gold since 1910. More than 100 gold deposits are distributed along this major crustal-scale fault zone, which exhibits compressional to trans-extensional characteristics. Studies suggest a long-lived, multi-staged orogenic system, with repeated cycles of opening and closing resulting in the emplacement of auriferous quartz-carbonate veins, a major component of the Greenstone deposit clan (Dubé and Gosselin, 2007).



Source: (McEwen, 2026)

Note: Estimated gold production as of December 2025

Figure 8-1: Schematic Map of Abitibi Greenstone Belt showing Relationship of Gold Deposition to Major Structural Conduits

At the Black Fox and Froome mines and the Stock and Western Properties, gold-bearing veins are predominantly associated with:

- Structurally controlled dilatant zones (shearing, brecciation, offsets)
- Greenschist facies metavolcanic host rocks
- Crustal-scale faults (i.e., the PDDZ)
- Syn-tectonic intrusive bodies

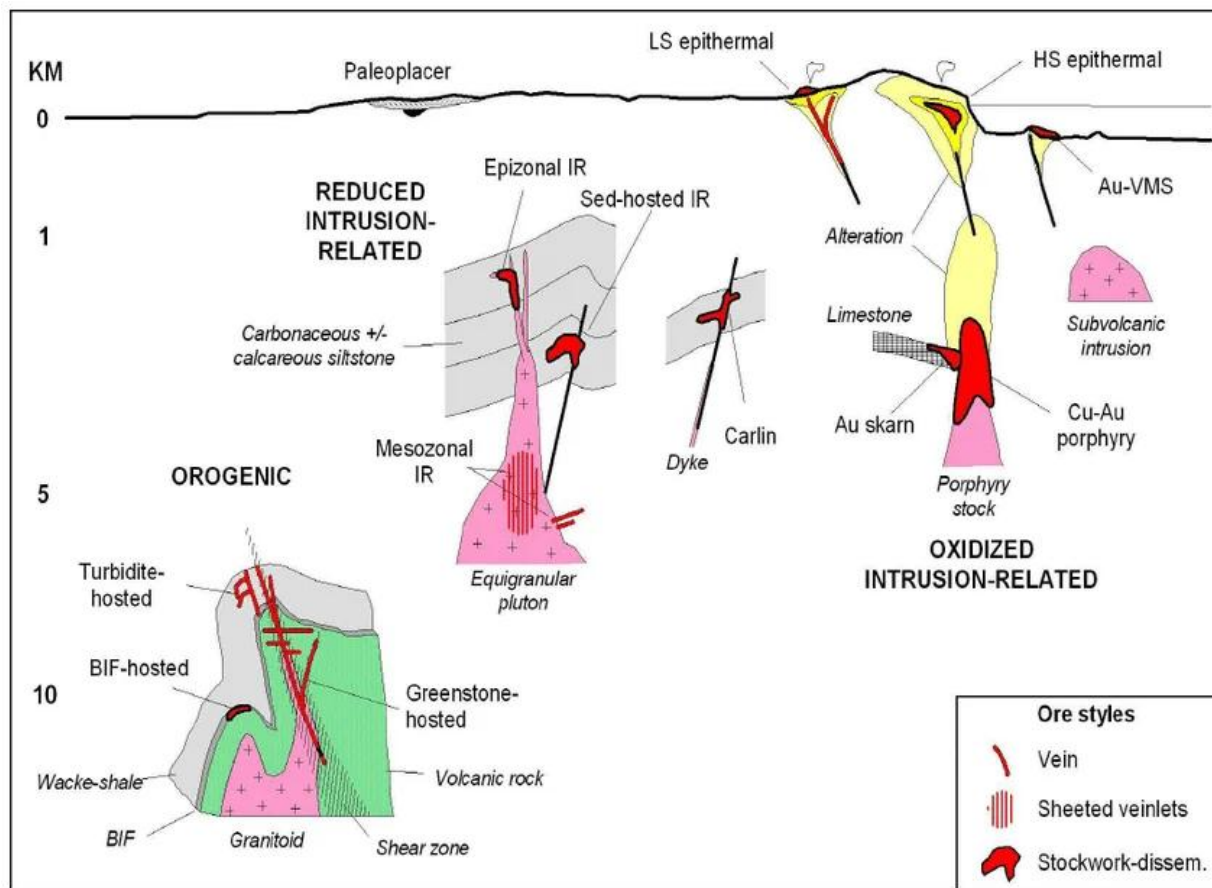
These characteristics are consistently seen at the Western Properties, Stock Property, and Froome sites and are classified as orogenic gold occurrences related to longitudinal shear zones. These greenstone-



hosted quartz-carbonate vein deposits represent a sub-type of lode gold deposits (Poulsen et al., 2000) and correspond to structurally controlled, complex epigenetic deposits hosted in deformed, metamorphosed terranes (Dubé and Gosselin, 2007) (Figure 8-2).

Gold mineralization at the Grey Fox deposit, located approximately 3 km southeast of the Black Fox Mine, exhibits the following characteristics:

- Spatial association with syenite, melanosyenite, and quartz-feldspar porphyritic intrusive rocks.
- Strong association with molybdenum, tungsten, and arsenic geochemistry (Kelly, 2018).
- Occurrence within narrow quartz-carbonate veins exhibiting crustiform and colloform textures.
- Spatial association with sericite-albite-carbonate alteration.



Source: (modified after Poulsen et al. 2000, and Robert, 2004)

Figure 8-2: Schematic Cross-Section of Main Gold Systems Showing Depth of Emplacement

The current interpretation of the Fox Complex Exploration team is that the Grey Fox deposit could be a combination of epizonal and orogenic gold deposits.



8.1 Greenstone-Hosted Quartz Carbonate-Vein Deposits

Greenstone-hosted quartz-carbonate vein deposits typically occur in deformed greenstone terranes of all ages (Dubé and Gosselin, 2007), especially those containing commonly variolitic tholeiitic basalts and ultramafic komatiitic flows intruded by intermediate to felsic porphyry intrusions, sometimes with swarms of albitite or lamprophyre dykes (for example, Timmins and Red Lake districts). Deposits are associated with collisional or accretionary orogenic events and are typically distributed along reverse-oblique, crustal-scale major fault zones, commonly marking convergent margins between major lithological boundaries such as volcano-plutonic and sedimentary domains (for example, the PDDZ and the CLDZ). These major structures are characterized by varying increments of strain, resulting in multiple generations of steeply dipping foliations and folds within a complex collisional geological setting.

Crustal-scale faults are thought to represent the main hydrothermal pathways toward higher crustal levels. However, deposits are spatially and genetically associated with higher-order compressional reverse-oblique to oblique brittle-ductile high-angle shear zones, commonly located less than 5 km away and best developed in the HW of the major fault (Robert, 1990). Brittle faults may also host mineralization, as illustrated by the Kirkland Lake Main Break, a brittle structure hosting the 25 Moz Kirkland Lake gold deposit. The deposits typically formed late in the tectonic-metamorphic history of the greenstone belts (Groves et al., 2000), with mineralization occurring syn- to late-deformation, typically post-peak greenschist facies and syn-peak amphibolite facies metamorphism (Kerrick and Cassidy, 1994; Hagemann and Cassidy, 2000).

The greenstone-hosted quartz-carbonate vein deposits are also commonly spatially associated with Timiskaming-type regional unconformities. Several deposits are hosted by a Timiskaming-like regional unconformity (e.g., the Pamour and Dome deposits in Timmins) or located next to one (e.g., the Campbell-Red Lake deposit in Red Lake) (Dubé et al., 2003), suggesting an empirical time and space relationship between large-scale greenstone quartz-carbonate gold deposits and regional unconformities (Hodgson, 1993; Robert, 2000; Dubé et al., 2003).

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 in Canada) to complex geometries with multiple orientations of anastomosing and/or conjugate sets of veins, breccias, stockworks and associated structures (Dubé et al., 1989; Hodgson, 1989; Robert et al., 1994; Robert and Poulsen, 2001).

Economic-grade mineralization also occurs as disseminated sulphides in altered (carbonatized) rocks along vein selvages. In this document, the term ore is used in its common industry sense (for example, ore shoots, ore mineral, ore pass) and does not imply Mineral Resources or Mineral Reserves.

Ore shoots are commonly controlled by either:

- Intersections between different veins or host structures, or between an auriferous structure and a reactive and/or competent rock type such as iron-rich gabbro (geometric ore shoot).



- The slip vector of the controlling structure(s) (kinematic ore shoot). For laminated fault-fill veins, the kinematic ore shoot is oriented at a high angle to the slip vector (Robert et al., 1994; Robert and Poulsen, 2001).

At the district scale, greenstone-hosted quartz-carbonate vein deposits are associated with large-scale carbonate alteration, commonly distributed along major fault zones and subsidiary structures (Dubé and 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, alteration haloes are zoned and characterized, at greenschist facies, by iron carbonatization and sericitization, with sulphidation of immediate vein selvages (mainly pyrite, less commonly arsenopyrite).

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

8.2 Low-Sulphidation Epithermal Gold Deposits

At the district scale or larger, the tectonic setting of epithermal gold deposits is characterized by extension, which localizes and facilitates the emplacement of magma and, at higher levels, hydrothermal fluids (Taylor, 2007). Regionally extensive rift zones can also provide the extensional framework. Pull-apart basins formed between regional strike-slip faults, or at transitions between these faults, provide favourable sites for intrusions and epithermal deposits. Synchronous tectonic and hydrothermal activity is indicated in some deposits by the fact that many of the vein-bearing faults were active during and after vein filling, resulting in tectonic vein breccias and displacement of mineralized and altered rocks.

Low-sulphidation epithermal gold deposits are harder to recognize in ancient terranes because they are often not as intimately associated with igneous rocks, and their commonly occurring alteration mineral assemblages may not be unique or may no longer be preserved, depending on the grade of subsequent metamorphism (Taylor, 2007).

Low-sulphidation epithermal gold deposits are distinguished from high-sulphidation deposits primarily by their sulphide mineralogy (pyrite, sphalerite, galena, chalcopyrite), typically hosted within quartz veins with local carbonate, and by associated near-neutral wall-rock alteration (illite clays) deposited from dilute hydrothermal fluids (Corbett and Leach, 1998).

Nearly any rock type (even metamorphic rocks) may host epithermal gold deposits, although volcanic, volcanoclastic, and sedimentary rocks tend to be more common (Taylor, 2007). Typically, epithermal deposits are younger than their enclosing rocks, except where deposits form in active volcanic settings and hot springs. In these cases, the host rocks and epithermal deposits can be essentially synchronous with spatially associated intrusive or extrusive rocks. Lithological control mainly reflects competent or brittle host rocks that develop throughgoing fractures that act as vein hosts, although permeability is locally important. In interlayered volcanic sequences, epithermal veins may be confined to competent rocks, while intervening less competent sequences preferentially host fault structures (Corbett, 2007).



Low-sulphidation gold deposits that occur further removed from active magmatic vents may be controlled by structural features, with zones of fluid mixing and emplacement of smaller magmatic bodies (e.g., dykes) (Taylor, 2007). Meteoric waters dominate the hydrothermal systems, which are nearly pH neutral in character. Low-sulphidation gold deposit related geothermal systems are more closely linked to passive rather than active magmatic degassing (if at all) and are sustained by the energy provided by cooling subvolcanic intrusions or deeper magma chambers.

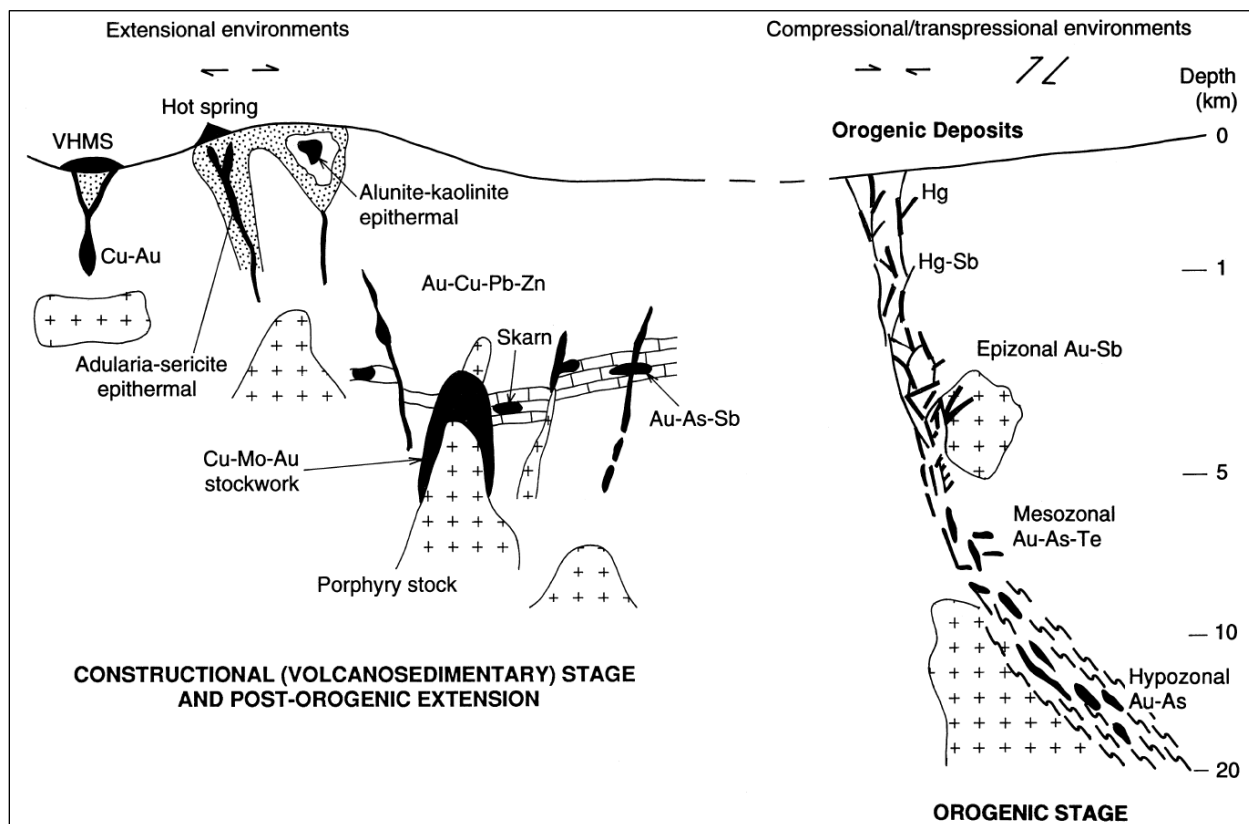
The morphology of epithermal vein-style deposits can be quite variable. Deposits may consist of roughly tabular lodes controlled by the geometry of the principal faults they occupy, or comprise interconnected fracture fillings in stockworks, breccias, and lesser fractures, or form by replacement of rock or void space. These deposits may take on the morphology of the lithological unit or porous rock body replaced (e.g., irregular breccia pipes and lenses). Volumes of replacement mineralization may be discordant and irregular, or concordant and tabular, depending on porosity, permeability, and water-rock interaction. In very near-surface deposits, an upward enlargement of altered and mineralized zones may be found centred around hydrothermal conduits. Brecciation of previously emplaced veins can form permeable zones along irregularities in fault planes, resulting in vertically plunging mineralized zones in strike-slip faults and more horizontal mineralized zones in dip-slip faults.

Structures act as fluid pathways, and dilational portions of these structures may represent sites of enhanced fluid flow, promoting the development of ore shoots that host most mineralization in many low-sulphidation vein systems (Corbett, 2002). Elsewhere, fault intersections host ore shoots at sites of fluid mixing. Several structural settings produce ore shoots of varying orientations. Steeply dipping strike-slip structures develop vertical ore shoots in flexures and fault jogs, where tension veins and dilatant sheeted veins are common. Normal and listric faults in extensional settings host wider and higher-grade veins as flatter ore shoots within steeply dipping vein systems. In compressional settings, reverse faults host flat-plunging ore shoots.

8.3 Epizonal-Type Gold Deposits

Epizonal gold deposits represent a sub-class of orogenic gold systems formed at shallow crustal levels, typically at depths of less than 6 km and under relatively low temperature conditions of approximately 200°C to 350°C (Gebre-Mariam et al., 1995; Groves et al., 1998). These conditions give rise to a suite of geological and structural characteristics that distinguish epizonal deposits from their mesozonal and hypozonal counterparts. Epizonal deposits are further distinguished from epithermal deposits by the depth of formation, and by the former being derived from orogenic fluids utilizing deep-seated structures as fluid + pathways, whereas the latter are largely formed by circulating meteoric fluids due to heating from a proximal igneous source (Figure 8-3).





Source: (Groves et al., 1998)

Figure 8-3: Schematic Comparison of Epithermal and Orogenic Environments

Epizonal deposits form under low-pressure conditions characteristic of upper crustal environments and are associated with brittle deformation regimes. Mineralization is typically hosted within fault zones, brittle shear zones, vein arrays, and fracture networks. Ore-bearing vein systems are dominated by quartz–carbonate–sulfide assemblages, occurring as stockworks and sheeted vein sets. Diagnostic vein textures include crustiform banding, comb quartz, open-space filling, and breccias, which collectively reflect the brittle deformation conditions and episodic, high-flux fluid flow events characteristic of the epizonal environment (Groves et al., 1998; Goldfarb and Groves, 2015).

Past and ongoing studies at Grey Fox including fluid inclusion studies, mapping, core logging, and modelling suggest that the Gibson, 147, Contact and Grey Fox South zones may have formed in an epizonal gold setting. The absence of alteration consistent with circulating meteoric fluids suggest an epizonal orogenic environment as opposed to a near-surface epithermal one. Fluid inclusion work at Grey Fox commissioned by Primero (Tuba, 2019, internal report) suggests a formational depth for the gold veining at Grey Fox at ~1.5 to 2.0 kbar which equates to ~5-7 km depth, deeper than typical epithermal deposits and placing it within the lower epizonal to upper mesozonal regimes (Groves et al., 1998), possibly transitioning from one to the other as vein textures appear to show characteristics of both epizonal (Gibson, 147, Grey Fox South) and mesozonal (Whiskey Jack) environments.



9 Exploration

9.1 Eastern Properties

Following acquisition of the Eastern Properties in 2017, McEwen conducted a range of exploration activities beginning in 2018, including geophysical surveys, drilling, and trenching. Several exploration programs were carried out concurrently across the claim group, allowing evaluation of several targets through property-scale surveys and drilling campaigns.

Figure 9-1 presents a plan view geology map of the Eastern Properties (excluding Black Fox North) and illustrates the locations of the principal zones that make up the Grey Fox deposit (Whiskey Jack, Gibson, Contact, 147NE, 147, and Grey Fox South).



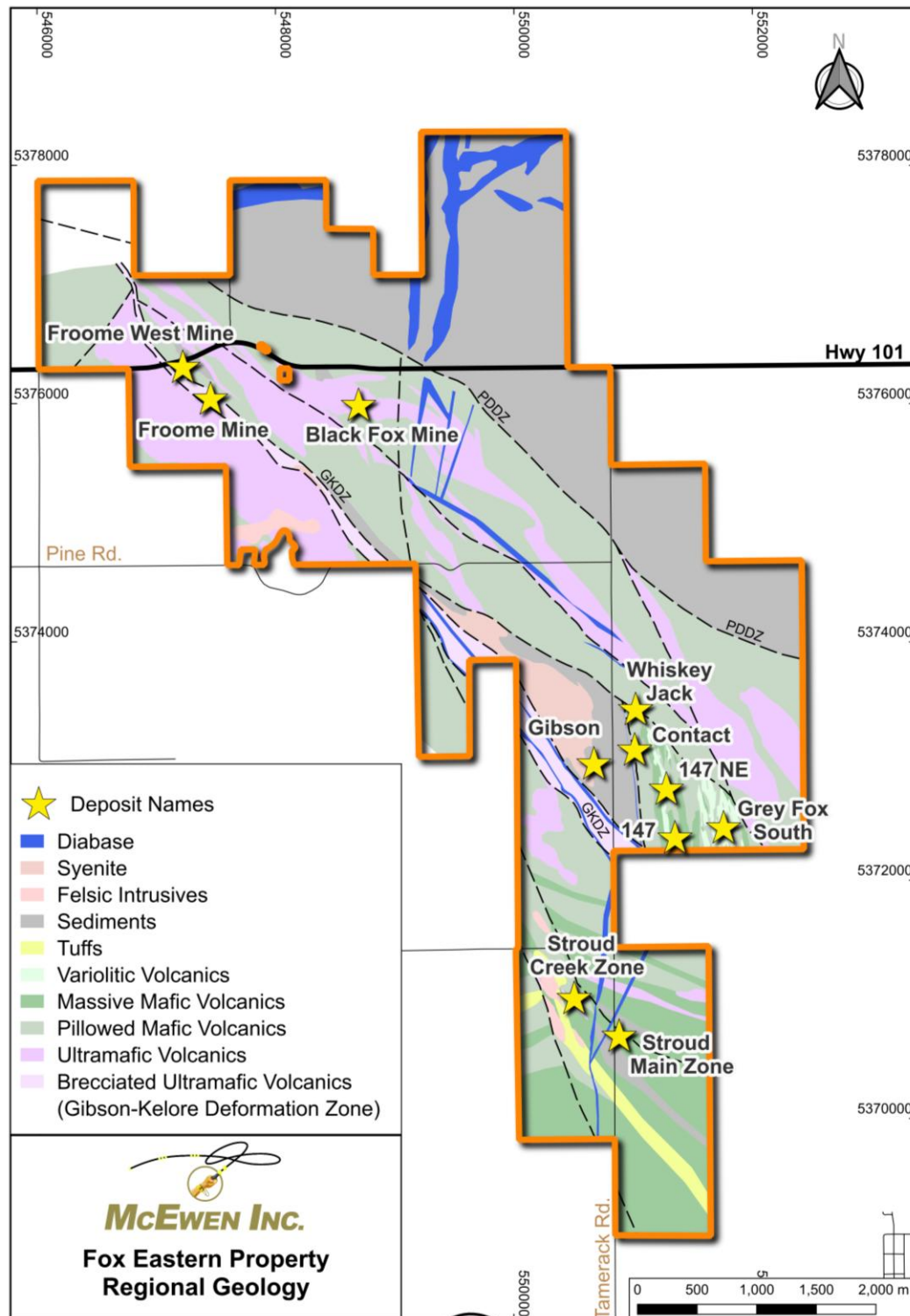


Figure 9-1: Plan View Geology Map for the Eastern Property



9.1.1 Grids and Surveys

The most recent traditional grid work was completed between 2002 and 2003 to facilitate an EM TITAN-24 geophysical survey over the Grey Fox and Froome areas. In September 2019, a UAV photographic survey was completed over the Grey Fox target area to support geological mapping and target evaluation. Several low-altitude UAV surveys were also conducted over exposed bedrock areas within the 147 Zone.

9.1.2 Geological Mapping

The Eastern property is generally overlain by 5 m to 40 m of clay and till overburden, which significantly limits bedrock exposure. Known bedrock exposures are limited to localized areas south of the Black Fox mining operations and at a road cut along Highway 101 west of the property boundary, where mafic volcanic rocks and a Matachewan diabase dyke are exposed. No natural bedrock exposures have been identified within the Grey Fox or Froome areas.

9.1.3 Geochemical Sampling

The presence of extensive glacial tills and dense clay overburden, locally up to 40 m thick, limits the effectiveness of conventional soil geochemistry surveys across the Eastern Properties. As a result, such surveys have not been widely utilized.

9.1.4 Geophysics

Several vintages of regional airborne geophysical surveys have been conducted across the Eastern Properties, including the Grey Fox, Froome, and Black Fox areas. These surveys have supported regional geological interpretation, including identification of structural features and lithological boundaries.

In 2010, a DCIP TITAN-24 geophysical survey was completed over the majority of the Eastern Properties, with anomalies identified from this program contributing directly to the discovery of the Froome deposit. In 2025, an additional TITAN-24 DCIP and MT survey was completed over the southern portion of the Eastern Properties, specifically Grey Fox (Figure 9-2). Several anomalies were identified, including a broad resistivity high in the Gibson area, which is interpreted to correspond to known mineralization.

Airborne magnetic surveys, including a property-wide survey completed in 2010 and higher-resolution drone-based surveys completed in 2025 over the Grey Fox/Stroud and Froome areas, have helped define the lithological and structural boundaries across the property.



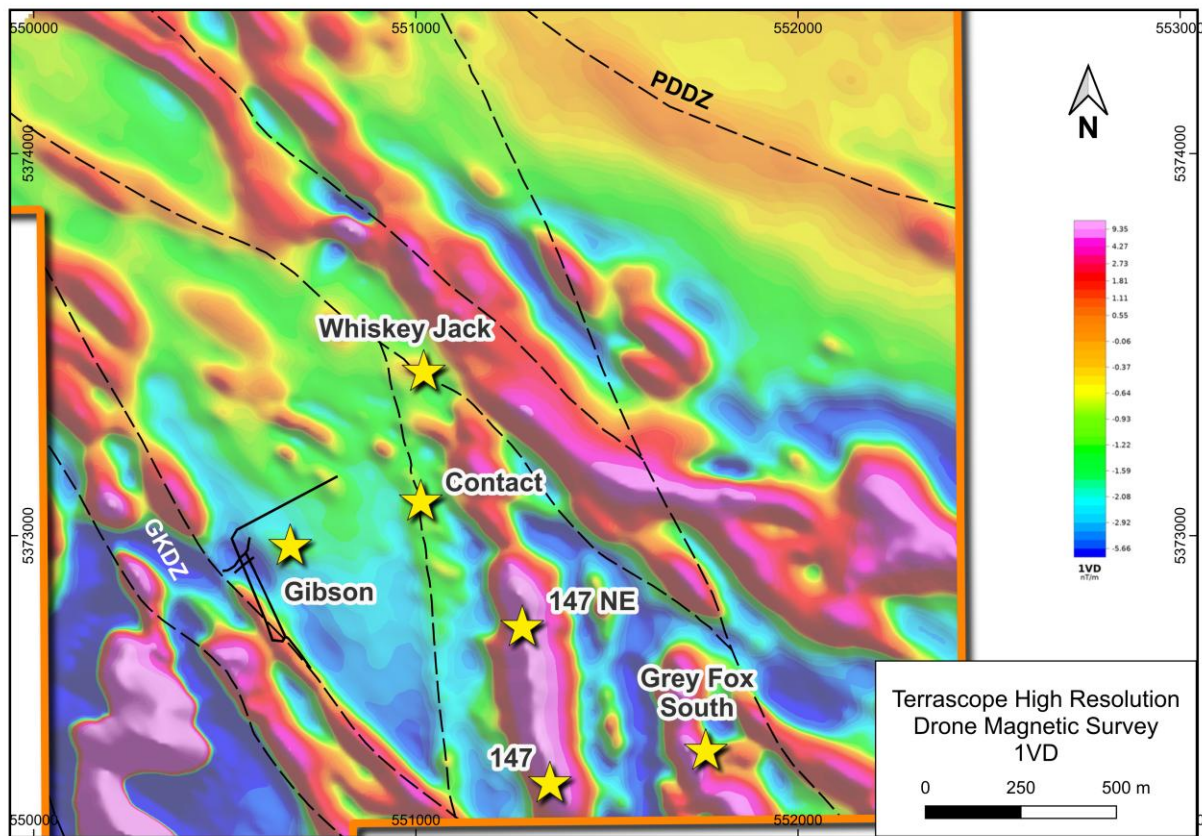


Figure 9-2: 2025 High-Res Drone Magnetic Survey over Grey Fox

Completed geophysical surveys are summarized in Table 9-1.

Table 9-1: Eastern Properties Geophysical Surveys

Date / Location	Service Provider	Survey Type	Survey Basis	Note
1991	Noranda	Total Field Magnetic & VLF Electromagnetic survey	-	Froome Lake Claim
1993	Exscis Exploration Ltd.	Total Field Magnetic VLF Electromagnetic survey	Survey grid totalling 15.4 line-km	-
2003 Black Fox Complex	Quantec	Electromagnetic TITAN 24 geophysical survey	Deep penetrating	-
2010 Black Fox	Scott Hogg & Associates Ltd.	Airborne magnetics	-	-
2010	Quantec	TITAN 24 survey	DCIP & MT	Coverage of the majority of the



Date / Location	Service Provider	Survey Type	Survey Basis	Note
				Eastern properties.
2016 Froome	Exsics Exploration Ltd.	Down-hole mise-à-la-masse electromagnetic survey	Two holes within Froome mineralization	Sedimentary rock dips to the south-southwest and strike length of 125 m-175 m
2018 Black Fox & Stock Mill	Geotech Ltd.	Helicopter-borne electromagnetic survey, high-resolution VTEM	Two survey grids totalling 1,164 line-km split between Black Fox Complex and Stock Mill	Several conductors delineated; no follow up drilling has occurred
2019	CGG Canada Services Ltd. TERRA Resources	Airborne gravity gradiometry employing the HeliFALCON platform	100 m spaced grid lines flown at a height of 35 m. Filtering was completed to reduce terrain effects, line corrections and potential impact from underground voids	Four target areas outlined for exploration follow-up, two of which have been previously drilled and explained.
2019 Grey Fox	Quantec Geoscience	Orion 3D plus electromagnetic survey	3D depth connection between responses collected at surface and in situ bedrock. Electromagnetic injections within a 1.5 km x 1.8 km quadrant – 176 from surface, 256 subsurface from 7 holes.	Favourable resistivity likely indicating silica-albite enrichment at 500 m level occurring below current drill limits. Follow-up with 750 m holes in 2021 and 2022.
2025	Quantec Geoscience	TITAN24 DCIP & MT Geophysics Survey	To support mineral exploration by using subsurface resistivity and chargeability data to identify key geological features, delineate potential mineralized zones, and map lithological variations. Specifically, the survey aims to detect structures such as faults, fractures, and mineralized areas, distinguish mineralized from non-mineralized rock based on electrical contrasts, and differentiate rock types to improve geological mapping and targeting.	
2025	Terrascope Inc.	Drone Magnetic Survey	Drone magnetic survey over Grey Fox including Stroud	



9.1.5 Pits and Trenches

In September 2019, McEwen exposed a part of the 147NE structure near-surface. High-pressure washing followed by detailed geological mapping confirmed the presence of northwest-dipping veinlets and breccias hosted within massive mafic volcanic rocks. This work provided additional insight into the continuity and relationship between hydrothermal breccias and veinlets within the 147 area that define mineralized structures.

Channel samples were then cut-collected across the exposed mineralized structure. Sample channels were cut with walls 8 cm to 10 cm apart and to depths of approximately 10 cm to 13 cm. Gold assays returned low-grade values.

9.1.6 Additional Surveys

In September 2019, Zen GeoMap conducted several low-altitude (<30 m above ground level) UAV survey passes on a grid pattern over the 147-target area, which had recently been exposed by mechanical stripping. This work generated high-resolution imagery of the brecciated vein system and confirmed the intensity, orientation, and structural limits of the breccia-hosted vein model within the Grey Fox area.

9.1.7 Exploration Potential

Considerable exploration potential exists for the remainder of the Eastern Properties, including:

- Untested volcanic stratigraphy within the FW of the Black Fox-Grey Fox trend.
- Assessment of the actual track of the PDDZ structure extending approximately 4 km southeast of the Black Fox mining operations.
- Northwest of the current Gibson Zone resource at Grey Fox where there is evidence of repeating veins sets associated with contact between the Gibson Syenite and metasediments to the east.
- Below 300 m el. at the Whiskey Jack Zone at Grey Fox where the mineralization remains open.
- Deeper (>300 m el.) drilling at Grey Fox South where the drilling density is lower when compared to other parts of the deposit.

An ongoing exploration program at Froome is targeting potential extensions of mineralization at depth and along strike.

Continued drilling at Black Fox North could potentially identify 'Croesus-style' veins that may extend along strike from the historic Croesus Mine, located approximately 5 km southeast of the Black Fox North claims.

9.2 Stock Property

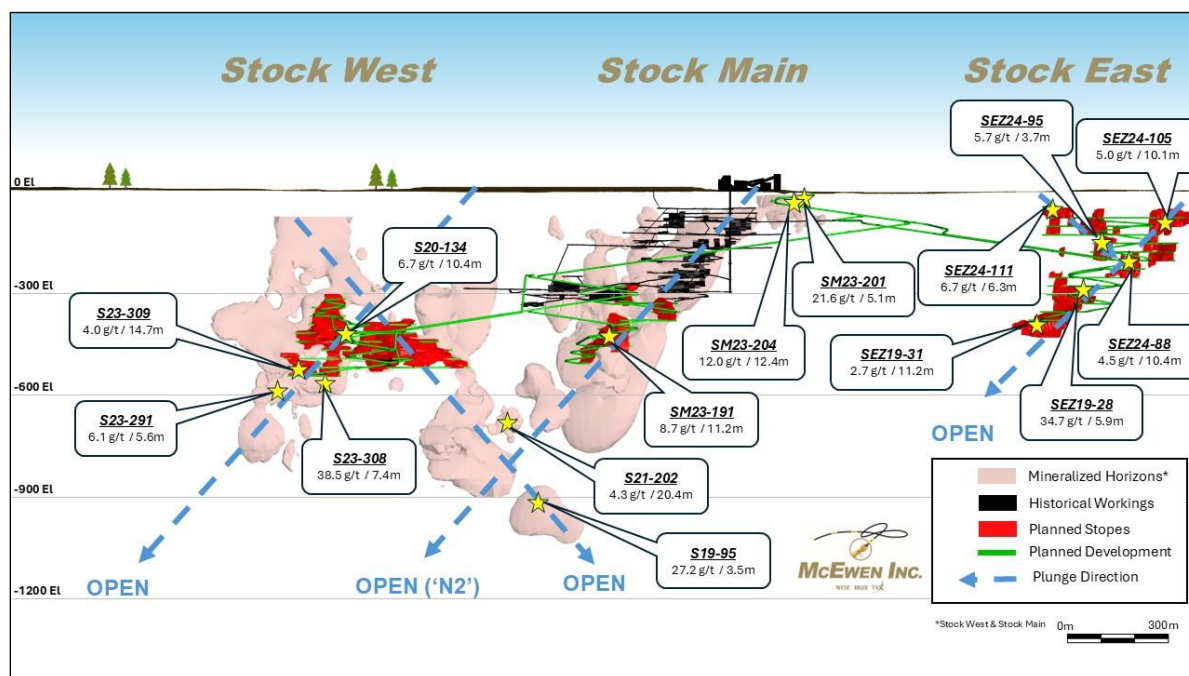
Historically, the lack of sustained exploration funding and limited gold production within the Shillington-Stock area have resulted in exploration efforts being focused primarily on "quick-test" drilling along the east-west track of the PDDZ. Mine infrastructure and hazards commonly limit access into prospective staging points for proposed exploration drilling. A summary of the historical exploration activities at Stock can be found in NI 43-101 Technical Report on the Mineral Resources Estimate of the Fox Complex, Ontario, Canada (McEwen, 2025).



Since McEwen took ownership of the Stock property in 2017, exploration activities have included aerial and geophysical surveys, structural interpretation and analysis, and drilling. Significant drilling programs have been completed at Stock West, Stock Main, and Stock East over the past nine years.

9.2.1 Exploration Potential

The majority of exploration at the Stock property since the 1990s has been concentrated within three discrete areas, each approximately 600 m x 400 m in size, or “windows,” that straddle the prominent PDDZ corridor as it tracks across the property. Escalating success at Stock East between 2018 and 2024, followed by success at Stock West from 2019 to 2023, has dominated exploration planning, culminating in updated Mineral Resource estimates, as well as initiation of studies such as a PEA, an internal economic study and consultations. Future Exploration drilling should focus on the down plunge components of Stock West, Stock Main (down the ‘N2’ chute) and Stock East (Figure 9-3).



Note: Looking north. All drill hole results reported as TWs.

Figure 9-3: Longitudinal Section View of Stock Property

Drilling at Stock Main from 2021 to 2023 has targeted potential additions to mineralization near surface, at depth, and in proximity to the proposed Stock West decline and portal area.

9.3 Western Properties

A summary of exploration activities for the Western Properties can be found in a technical report prepared by Roscoe Postle & Associates (Altman et al. 2014), supplemented by internal documentation generated between 2012 and 2015. In addition, all active Mineral Resource estimates for the Western Properties are not material to the Grey Fox Mineral Resource estimate contained in this report.



10 Drilling

A significant amount of drilling has been completed at the Fox Complex over the past century, much of which is historical in nature. Drilling described herein includes holes drilled for geological exploration, metallurgical testwork and geotechnical investigations. A complete drilling history is summarized in Section 6. Drill holes used in the Mineral Resource estimations are summarized below.

10.1 Eastern Properties

The Eastern Properties comprise Grey Fox, Stroud, Froome, and Black Fox properties.

Approximately 1,713 drill holes totaling 633,384.5 m have been completed in the Grey Fox-Hislop-Stroud area by several operators since 1993. Drill holes used in the Grey Fox and Stroud Mineral Resource estimations are summarized in Table 10-1. Figure 10-1 shows the locations of surface drill hole collars for the Eastern Properties. Drilling at Froome from 2000 to the present consists of 1,111 drill holes (221,282 m). The drill holes used in the Froome Mineral Resource Estimate are summarized in Table 10-2.

A total of 7,731 core drill holes, totaling approximately 1,097,796 m, have been completed on the Black Fox Mine property since 1989. Drill holes in the Black Fox Mineral Resource estimation are summarized in Table 10-3.

Table 10-1: Summary of Grey Fox and Stroud Drilling Used in Mineral Resource Estimation

Company	Year	No. Drill Holes	Metres Drilled	Target Zone
Gunnex Ltd.	1974	32	2,573	Stroud
Pancontinental Mining Ltd.	1980	13	1,691	Stroud
Pancontinental Mining Ltd.	1981	6	1,292	Stroud
Chevron Minerals Ltd.	1986	9	1,437	Stroud
Chevron Minerals Ltd.	1987	23	3,679	Stroud
Chevron Minerals Ltd.	1988	38	10,128	Stroud
Chevron Minerals Ltd.	1989	5	1,806	Stroud
Noranda, St Andrew Goldfields Ltd.	1993	10	3,118	Stroud, Grey Fox
Noranda	1994	4	762.0	Grey Fox
Battle Mountain/ Hemlo Gold	1995	8	2,109.0	Grey Fox
Battle Mountain/ Cameco Gold	1996	16	5,872.0	Contact, 147 and South Zone, Gibson and Hislop North
Battle Mountain/ Cameco Gold	1997	13	5,367.5	Contact, 147, South Zone, Gibson and Hislop North
Glimmer/Exall	2001	4	1,667.0	Romios
St Andrew Goldfields Ltd.	2006	6	2,058	Stroud



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10 Drilling

Company	Year	No. Drill Holes	Metres Drilled	Target Zone
St Andrew Goldfields Ltd.	2007	16	5,337	Stroud
Apollo Gold	2008	16	3,715.0	Southern extension of Contact Zone
Apollo Gold	2009	53	9,960.0	Contact Zone
Brigus Gold	2010	69	26,766.2	Contact Zone, Gibson, South Zone, Hislop North, 147 Zone
Brigus Gold	2011	274	101,893.2	Contact Zone, Gibson, South Zone, 147 Zone
Brigus Gold	2012	268	81,444.4	Contact Zone, 147 Zone, South Zone
Brigus Gold	2013	144	63,816.4	Contact Zone, 147 Zone, South Zone
Primero	2014	187	76,027.2	Contact Zone, 147 Zone, South Zone
Primero	2015	56	25,716.7	Contact Zone, Gibson, South Zone
McEwen	2018	46	17,896.5	Gibson, 147 Zone, 147NE Zone
McEwen	2019	222	90,341.7	Gibson, 147 Zone, 147NE Zone, South Zone, Whiskey Jack
McEwen	2020	15	5,061.7	Whiskey Jack
McEwen	2021	29	11,404.8	Contact Zone, Gibson, 147 Zone, South Zone, Whiskey Jack
McEwen	2022	14	6,561.2	Contact Zone, Whiskey Jack
McEwen	2023	21	11,047.8	Contact Zone, Gibson, Whiskey Jack
McEwen	2024	126	41,872.8	Contact Zone, Gibson, 147 Zone, South Zone, Whiskey Jack
McEwen	2025	93	32,900.4	Stroud, Grey Fox
McEwen	2026	3	668	Stroud
Total		1,839	655,990.5	



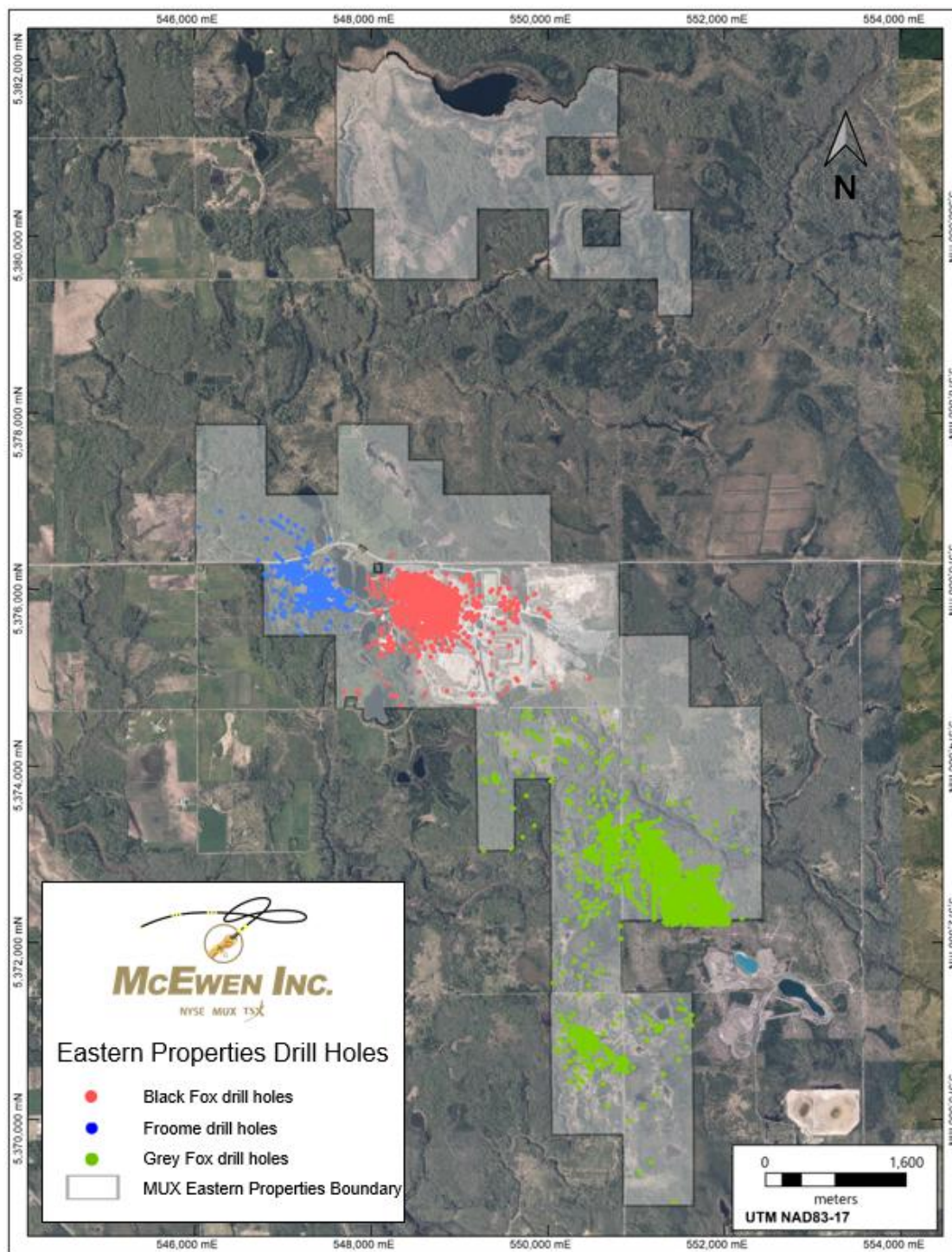


Figure 10-1: Distribution of Drilling at the Eastern Properties



Table 10-2: Summary of Froome Drilling Used in Mineral Resource Estimation

Company	Year(s)	No. Drill Holes	Metres Drilled
Noranda	1989-1994	1	197
Glimmer/Exall	2000-2002	1	200
Apollo Gold	2003-2007	1	693
Primero	2014-2017	219	64,997
McEwen	2018-2025	890	155,392
Total		1,112	221,479

Table 10-3: Summary of Black Fox Drilling Used in Mineral Resource Estimation

Company	Year	No. Drill Holes	Metres Drilled
Noranda	1989-1994	119	23,987
Glimmer/Exall	1998-2002	876	84,828
Apollo Gold	2003-2009	898	222,904
Apollo Gold/Brigus Gold	2010-2013	1,456	146,049
Primero	2014-2017	2,610	374,368
McEwen	2018-October 2024	1,772	245,660
Total		7,731	1,097,796

10.2 Stock Property

Drilling at the Stock Property has been completed across multiple target zones, including Stock East, Stock West, and Stock Main, to support Mineral Resource delineation. Drill holes used in the Stock Mineral Resource estimations are summarized in Table 10-4, with drill hole collar locations shown in Figure 10-2.

Table 10-4: Summary of Stock Drilling Used in Mineral Resource Estimation

Company	Year(s)	No. Drill Holes	Metres Drilled	Target Zone
St Andrew	1983-2008	47	13,320	Stock East
Brigus Gold	2011	6	2,166	Stock West
Brigus Gold, Primero	2014	15	5,663	Stock Main
Primero	2015	19	4,699	Stock Main
McEwen	2018	90	26,704	Stock East, Main
McEwen	2019	106	46,461	Stock East, West
McEwen	2020	30	17,727	Stock West
McEwen	2021	142	68,298	Stock West, Main
McEwen	2022	126	50,288	Stock West, Main



Company	Year(s)	No. Drill Holes	Metres Drilled	Target Zone
McEwen	2023	185	62,928	Stock West, Main, East
McEwen	2024	49	9,627	Stock East
Total		815	307,881	

All drilling data for Grey Fox, Froome, Black Fox and Stock properties are current to the individual drilling database cut-off dates applied in the Mineral Resource estimates, as summarized below:

- Grey Fox – 11 July 2025
- Froome – 31 December 2025
- Black Fox – 31 December 2025
- Stock – West and Main Zones – 23 October 2023
- Stock – East Zone – 20 May 2024



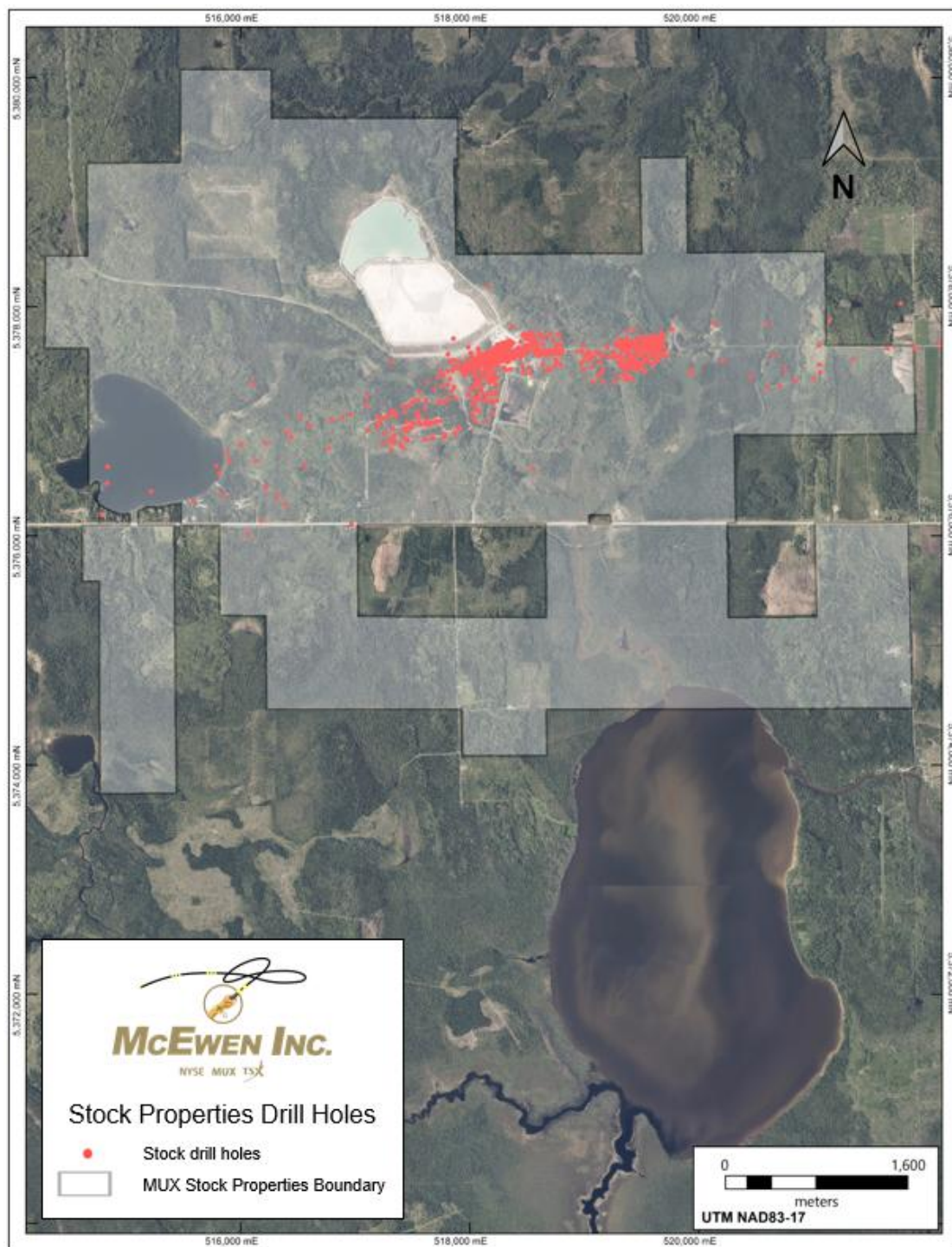


Figure 10-2: Distribution of Surface Drilling at the Stock Property



10.3 Western Properties

The Western Properties include the Fuller, Davidson-Tisdale, Buffalo Ankerite, and Paymaster properties. Multiple phases of exploration and mining activity have occurred on these properties since 1910. Prior to the early 2000s, exploration drilling was sporadic and driven by mineralization boundaries. Historical drilling records are available in paper format and require additional organization to support future work.

Lexam conducted systematic, results-driven exploration programs between 2003 and 2015. Details of drilling procedures during this period are assumed based on review of several available drill logs, maps and cross & longitudinal sections from that era.

A summary of drilling completed on the Fuller and Davidson-Tisdale properties is provided in Table 10-5, with drill hole collar locations shown in Table 10-3. In addition to the drilling at Fuller, 598 chip samples collected during the Belmoral program were used, representing a total sampled length of 2,279.9 m. Bazooka drilling at Fuller during the same period consisted of 184 holes totaling 1,487 m.

Drilling completed in the Buffalo Ankerite and Paymaster deposits does not support a current Mineral Resource estimate and is therefore not considered relevant to this report.

Table 10-5: Summary of the Western Properties Drilling Used in the Mineral Resource Estimate

Company	Year(s)	No. Drill Holes	Metres Drilled
Fuller Property			
Various companies	Prior to 1983	75	7,060.20
Belmoral	1986-1989	458	30,026.20
Vedron	1996-1998	58	27,626.00
Lexam	2004-2012	41	10,403.90
Total		632	75,116.30
Davidson-Tisdale Property			
DTM	1983-1984	35	4,576.8
Getty	1984-1987	537	49,165.9
Midas Minerals	1989	5	1,486.2
Northcott	1999-2003	17	399.3
Lexam	2003-2015	97	24,397.5
Total		691	80,025.7



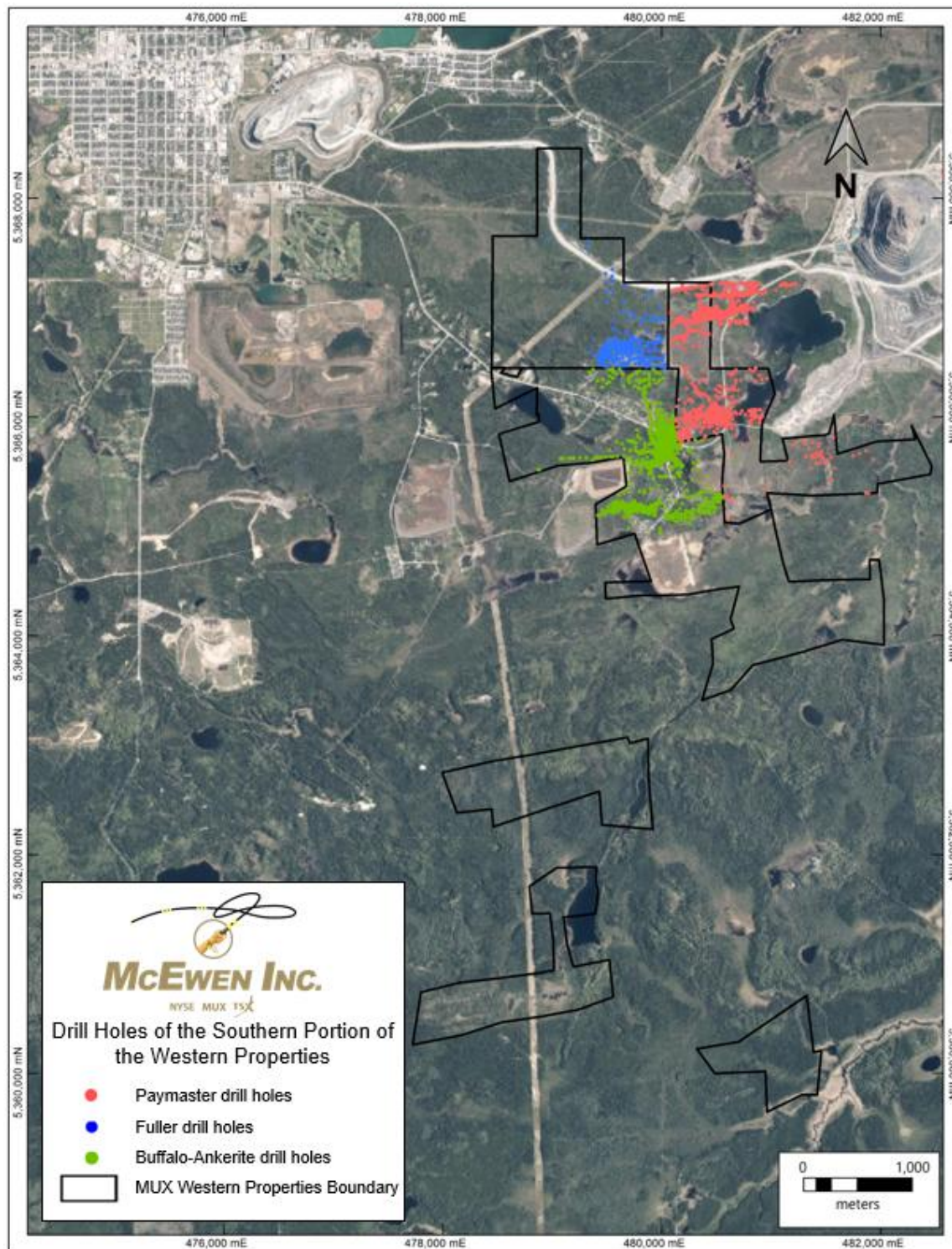


Figure 10-3: Southern Portion of the Western Properties Drill Hole Collar Location Plan





Figure 10-4: Northern Portion of the Western Properties Drill Hole Collar Location Plan



10.4 Drilling Methods

Early drilling operations conducted from both surface and underground were completed using primarily diamond drilling methods. Historical drill data have not been used to define Mineral Resources unless the data could be verified through documented QA/QC procedures, supported by confirmation from nearby drill holes or re-logged using McEwen's logging, sampling and QA/QC procedures.

10.4.1 Eastern Properties

Surface diamond drilling operations at the Eastern Properties have been conducted by several local drilling contractors since the 1970s. Standard drilling procedures and protocols have remained consistent as rigs shift from target to target within the property area. No underground exploration drilling has been completed at Grey Fox since the late 1980s, when Gold Post Resources conducted an underground drill program at the historical Gibson mine.

Fully hydraulic diamond drill rigs have been commonly utilized by the various drilling contractors since 2010. Advancements such as mechanized rod breaking, remote-control manipulation, and safety-perimeter cages have improved drilling efficiency, with coring performance increasing by approximately 30% to 50%. As a result, typical production rates today range from 80 m to 120 m per 24-hour period.

NQ-diameter coring (47.6 mm) has been the standard drill core size at the Grey Fox and Froome targets since 2010. Drill core is removed from the core tube carrier and immediately placed into wooden core boxes with a capacity of approximately 4.5 m of core. During drilling, contractors place a small wooden block identifying the downhole length of overburden and progressively place depth marker blocks at 3 m intervals of core. Once filled, the core boxes are transported to McEwen's nearby core facility for logging, photography, and sawing for assay. As drill rods are withdrawn in 3 m lengths, drillers carefully count the rods arriving to surface to verify that the lowest depth-block recorded in the core boxes is accurate.

10.4.1.1 Grey Fox

Sludge samples were collected at Grey Fox during drilling programs in 1979 and throughout the 1980s and were used as a guide for most of the core sampling. Core diameter was NQ from 1983 onwards with BQ used prior to that time. Heath & Sherwood served as the drilling contractor during this period. Reports from Mr. Peter Ginn (claim owner during that period) do not document the core sampling methods used. All information from this time period was acquired from the MLAS (Mining Lands Administration System – Ontario) and does not specify any details.

During Noranda-era drilling (1988-1994), NQ-diameter core was collected and transported to the Hemlo Gold Mines storage facility at the Aunor mine site in Timmins. Core was split for assay. Longyear was the drilling contractor used during this period.

From 2018 to the present, McEwen has employed several surface diamond drilling contractors at Grey Fox, including Norex (now Major Drilling), Black Diamond Drilling, NPLH Drilling, and Major Drilling. The primary core size has remained NQ. Drill core collected from 2018 to 2019 was shipped to temporary logging facilities located at the intersection of Pine Road and Tamarack Road, in proximity to the Black Fox mine.



From 2020 onward, Grey Fox drill core has been logged at either the Stock Mill facilities or the Black Fox mine core shack.

10.4.1.2 Stroud

Historical drilling (post 1970) on the property was completed by experienced Canadian diamond drill contractors using industry-standard methods of the time. The earliest programs used B- and A-diameter core, while later Chevron and other programs predominantly used wireline NQ core. Drill collars were surveyed from the property grid, and hole deviation was monitored at regular intervals, with gyroscopic surveys used where greater accuracy was required. Core handling, logging, sampling, and QA/QC procedures were conducted in accordance with accepted industry practice for the time, including standards, blanks, and check assays. Most drill core and supporting geological and assay records are still available, providing an adequate basis for interpretation and incorporation into the PFS. Since the McEwen acquisition of Stroud in 2024, the same practice that has been used for Grey Fox, which is described in Section 10.4.1.1, has been implemented.

10.4.1.3 Black Fox

Initial surface stage drilling operations at Black Fox were conducted by Noranda between 1989 and the mid-1990s and was performed using NQ-diameter core, with reduction to BQ diameter where ground conditions required. Drilling during this period was executed by Norex Drilling (since merged with Major Drilling). Drill core was sent to the Hemlo Gold Mines storage facility at the Aunor mine site in Timmins for storage and logging. The core was split and sent for assay.

Subsequent diamond drilling at Black Fox, completed up to 2026, was a combination of surface and underground drilling programs. Specific drilling details are summarized in Table 10-3. Drilling during this period was completed by several drilling contractors. Core diameters varied from HQ to BQ, depending on ground conditions and hole length. The exploration core was processed, split and stored at the Black Fox core facility prior to being sent for assay. Delineation and definition core were similarly processed and sampled at the Black Fox core facility and sent for assay. Any remaining portions of delineation and definition core were discarded.

10.4.1.4 Stock Property

Drilling operations completed at the Stock Property since 2018 have employed the same drilling contractors and operating procedures described for the Eastern Properties, using NQ-diameter core. Sealed core boxes are delivered daily by the foreman to McEwen's logging facilities at the Stock Mill facility.

Brigus Gold performed a drilling program in 2011 using Norex Drilling. Samples were taken from zones that indicated mineralization or areas that are commonly associated with mineralization.

According to the RPA report for the property (Roscoe et al., 2003), most of the surface diamond drilling was completed between 1961 and 1989, while most of the underground diamond drilling was carried out between 1981 and 1994. Limited documentation is available describing the specific sampling methods used during these time periods. Most underground drilling conducted between 1989 and 1994 utilized AQ-



diameter core, while most of the surface drilling completed before 1996 used BQ-diameter core. All of the underground and surface drilling from 1996 to 2000 employed NQ-diameter core.

Generally, the diamond drill core was split, with one half being assayed and the remaining half retained. While the mine was in production from 1989 to 1994, assays were carried out on some whole core from infill drilling.

A manual core splitter was used prior to 1994. Since 1996, a diamond core saw with a continuous supply of fresh water has been used for core cutting.

10.4.2 Western Properties

Lexam VG exclusively employed Norex Drilling for all surface drilling across the Western Properties. After 2010, the majority of Norex's drill rig fleet consisted of hydraulic rigs configured to recover NQ-diameter core.

The Western Properties have large footprints from historical underground development. The development is now flooded or infilled with sand. The QP considers it reasonable to assume that drilling contractors reduced the core diameter to BQ (36.5 mm) when a surface-staged exploration drill hole encountered underground voids and proceeded onwards into solid rock.

Under Lexam VG ownership, drill core from the rigs was picked up twice per day by core technicians and taken to core logging facilities in nearby South Porcupine. Towards the end of Lexam VG's activities, the exploration team was based at the Davidson-Tisdale property.

10.5 Logging Procedures

Standard logging procedures (described herein) have been followed since approximately 1996. Since January 2018 regionally accepted gold industry standard practices have been consistently implemented.

Core boxes are opened and immediately checked for missing or damaged core when they arrive to the core shack from the drill rigs. Core is logged immediately or placed into steel holding racks adjacent to the logging stations. Core is first fitted together and aligned to ensure continuity and proper alignment of any structural fabric. The geologist marks off 1 m standard intervals and calculates rock quality data (for every 3 m drilling run). Lithology, alteration, and mineralization styles are identified and logged, and structural features are carefully recorded, including foliations, offsets and displacements, and ductile deformations of linear flow features.

10.5.1 Eastern Properties

Gemcom logging software was commonly used on site between 2015 and 2018 and has since been replaced by Datamine DH Logger.

Drill core is photographed from an elevated perspective to capture images of three to five core boxes at a time. These images are downloaded, labelled using a standard numbering format, and stored with the corresponding drill logging records.



Paper logging of historical drilling includes records of mineralization, lithology, and geological structure.

10.5.2 Stock Property

Since 2018, McEwen has followed the same logging procedures at the Stock Property as those for the Eastern Properties.

Paper logging of historical drilling includes records of mineralization, lithology, and geological structure.

10.5.3 Western Properties

Lexam followed a consistent core logging protocol for all properties, recording the lithological, structural, alteration, and mineralogical features.

According to the technical report prepared by RPA (Altman et al., 2014), surface drilling at Buffalo Ankerite between 2009 and 2012 followed a system where the Lexam technicians collected rock quality measurements and indicated core recoveries in addition to the usual core logging. Geological features such as alteration, mineralization and rock unit descriptions were logged into the Geotic Log software system. The Geotic lithological legend was later incorporated into the Gemcom GEMS-3D geological modelling software. Drill core was photographed after samples were marked up.

Paper logging of historical drilling includes records of mineralization, lithology, and geological structure.

10.6 Core Recovery

10.6.1 Eastern Properties

Recovery of diamond drill core during advancement through bedrock is generally reliable across the Black Fox and Froome properties, with recoveries of at least 85% achieved unless localized fault zones or voids are intercepted. The country rock underlying the Eastern Property claims is generally weakly deformed, and faults in the deposit areas are relatively competent when drilled at angles greater than 45° to the structural zone. The majority of modern drilling within the Grey Fox cluster of targets has been oriented at high angles to the north-south foliation. Core recoveries have typically exceeded 85%.

After 2015, drill holes at Froome were designed to dip at high angles to the GKDZ, adjacent to the targeted mineralized zones, maximizing driller's core recovery. This pattern was intended to reduce the influence of the structural fabric of the Gibson-Kelore Fault, lowering the tendency for core to naturally part along foliation planes within soft chloritic ± talc schistose ultramafic volcanic rocks.

Drill logs from historical drilling that informed Mineral Resource estimates indicate similar recovery values.

10.6.2 Stock Property

Country rock underlying the Stock Property targets is relatively strongly deformed due to the proximity of the PDDZ and the Nighthawk Lake fault structures. Core recovery rates between 85% and 100% are achieved when drilling through competent mafic volcanic flows or hard, unfractured felsic intrusive rocks.



However, localized zones of soft, chloritic and talcose rubble, including chips and gravel, can occasionally impede the driller's coring to depth. Approximately 5% of drill holes completed since 2018 failed to penetrate such zones to target depth, often resulting in drill rod breakage and loss.

Drill logs from historical drilling used to inform resources indicate similar recovery performance.

10.6.3 Western Properties

Country rock in the south-central portion of the Porcupine Mining Camp is moderately to strongly deformed due to the proximity of the PDDZ and associated South Tisdale Anticline structures. According to the technical report prepared by P&E and RPA (Altman et al., 2014), core recoveries of approximately 99% were reported at the Paymaster, Buffalo Ankerite, and Fuller sites where drill holes passed through competent mafic volcanic flows or hard, unfractured felsic intrusives. The QP is unaware of any significant fault gouge or structural conditions that materially limited core recovery at the Davidson-Tisdale exploration site.

Historical drill logs used to inform mineral resource estimation indicate similar core recovery rates.

10.7 Collar Surveys

10.7.1 Eastern Properties

Historical drill sites were selected and laid out using a 50 m tape from established cuts and were picketed relative to a 1,000 m-spaced gridline system developed along the flanks of Tamarack Road.

Between 2008 and 2015, diamond drill hole collars at the Grey Fox, Froome, and Black Fox targets were spotted in the field using a handheld GPS unit with an accuracy of approximately 1 m. Since 2016, collar locations have been surveyed using a differential GPS accurate to within approximately 1 m and linked to the base station at the Black Fox mine.

Since 2019, drilling contractors have been using Reflex Instruments EZ-Gyro TN-14 gyroscopic tool to capture survey and collar alignment data. The TN-14 tool uses a north-seeking gyroscopic compass digitally linked to the local GPS grid.

Drill holes completed from 1988 onward were surveyed using handheld GPS units. Follow-up surveys using conventional surveying methods were required to accurately locate hole collars.

Drill holes completed prior to 1980 were surveyed using transit methods.

10.7.2 Stock Property

Between 2018 and 2019, diamond drill holes were spotted using a handheld GPS with an accuracy of approximately 1 m. Since 2019, McEwen has utilized differential GPS to establish accurate survey control for drill hole collars. All data collected are reported in NAD83 coordinates, UTM Zone 17U, and are referenced to the base station established at the Stock Mill.



Since October 2020, drillers have been using Reflex Instruments TN-14 collar gyroscopic survey tool.

Since spring 2021, McEwen Inc. began using an independent DeviSight APS differential GPS tool, a true-north azimuth gyro-alignment system, to improve the accuracy of planned collar locations and drill azimuth alignments. In summer 2023, McEwen returned to surveying drill hole collars using differential GPS with real-time kinematic correction.

Earlier drill holes from 1988 were surveyed using handheld GPS units. Follow-up surveys using conventional surveying methods have since been carried out to accurately locate those hole collars.

10.7.3 Western Properties

Drill logs coupled with the technical report prepared by RPA (Altman et al., 2014) indicate that drill hole collars were surveyed using a differential corrected GPS instrument commonly achieving positional accuracy of approximately 0.1 m under favourable conditions.

Drill holes dating back prior to 1988 were re-surveyed using handheld GPS units. Conventional surveying methods for the period were used to accurately locate hole collars.

10.8 Downhole Surveys

10.8.1 Eastern Properties

Between 2014 and 2017, drillers used an EX-Shot downhole survey tool that utilizes a magnetic-north-pointing single-shot gimballed globe compass, with readings collected at 50 m intervals.

Since 2017, drillers have been using Reflex Instruments EZ Gyro or multi-shot Sprint orientation tools, which are north-seeking gyroscopic compasses proven to reduce magnetic effect. This is particularly important given the presence of high-iron mafic and ultramafic rock units within the Black Fox and Froome mines and the Grey Fox stratigraphic sequences. Survey data were collected at 50 m intervals, with downhole data digitally relayed to a handheld receiver at surface. Paper copies were inserted into core boxes at the appropriate survey locations.

Noranda drill logs indicate that Tropari or acid Pajari methods were used for downhole surveys. No indication of downhole survey methods is available on drill logs prepared by Ginn or Parsons.

10.8.2 Stock Property

Paper drill logs generated in the 1990s indicate that downhole survey readings were collected on a hole-by-hole basis; however, the survey method and sensitivity are rarely noted. Drill logs from the 1990s indicate that Tropari or acid Pajari methods were used.

From 2018 onward, drillers have followed the same downhole survey procedures as those described for the Eastern Properties, using EZ Gyro tools with data collected at 50 m intervals.



10.8.3 Western Properties

According to the technical report prepared by RPA (Altman et al., 2014), Lexam supervised drillers to collect downhole survey data using a Reflex EZ-Shot instrument, with readings taken at 50 m intervals.

10.9 Geotechnical, Hydrological and Metallurgical Drilling

Geotechnical diamond drill holes have been completed on the Eastern and Stock properties.

At Grey Fox, six geotechnical holes totaling 1,218 m were drilled in 2013. Seven NQ-sized holes totaling 525 m were drilled at the Stock Main deposit in 2010.

Twenty-two (22) boreholes have been completed as monitoring wells at Grey Fox, primarily in overburden, with a total length of approximately 375 m. Overburden drilling was typically completed using 200 mm hollow-stem augers, with bedrock portions drilled using NQ-diameter coring. These installations were completed by various consultants between 2011 and 2015.

In 2006, four NQ-diameter holes totaling 630 m were drilled at Black Fox. Ten additional holes totaling 2,017 m were drilled by Primero between 2015 and 2016, with partial geotechnical logging (including weathering, rock strength, rock quality designation, and joint roughness), completed on selected exploration holes. In 2017, six NQ3-diameter holes totaling 1,646 m were drilled along the Black Fox ramp toward Froome.

Forty (40) monitoring wells were installed primarily in overburden at Black Fox in 2004 and 2005. The total length of boreholes with installations is approximately 700 m. Overburden drilling was typically completed using 150 mm to 200 mm hollow-stem augers, with bedrock portions drilled using NQ-diameter coring.

In 2025, an approximately 1,400 m HQ drill program was executed at Grey Fox to support metallurgical studies for the project. In addition, some samples for metallurgical testing were obtained from half-core intervals of NQ-diameter exploration drill holes.

10.10 Sample Length and True Thickness

10.10.1 Eastern Properties

Mineralization at Grey Fox is controlled by northeast-trending (C1) which dip to the Northwest and East-West trending, south-dipping (C2) structures and, locally, by the interaction of these structures with steeply east-dipping lithological contacts. Due to the variable orientations of these structures, the relationship between sample length and true thickness can be variable.

Most drilling at Grey Fox completed between 2008 and 2016 was oriented in an east-to-west azimuth. This drilling orientation resulted in low-angle intersections of mineralized veinlets and breccias relative to the core axis, in which cases true widths may be as low as 10% to 20% of the drilled core length. Since 2018, the majority of drilling in the Gibson, Grey Fox South, 147, and 147NE zones at Grey Fox has been oriented northwest to southeast, resulting in higher-angle intersections, typically greater than 60° to the core axis, where true widths may be as high as 80% to 90% of the core length. Low-angle south-dipping



mineralized structures corresponding to the C2 structural set are also intercepted in this northwest to southeast orientation.

10.10.2 Stock Property

Exploration drilling over the past 40 years at the Stock Property has been designed to intersect the PDDZ corridor of alteration and mineralization at near-perpendicular angles, with most drill holes oriented toward azimuths between 330° and 360°. The steeply southeast-dipping stratigraphic sequence allows these drill holes to cut the conformable mineralized structures at high angles, typically greater than 60° to the core axis, where the true width may be as high as 80% to 90% of the drilled core length.

10.10.3 Western Properties

According to the technical report prepared by RPA (Altman et al., 2014), Lexam attempted to intercept exploration targets at near-perpendicular attack angles. This approach included near-vertical drilling to intersect flat-lying mineralized zones at Davidson-Tisdale, as well as the swinging/reactive dip angles to bracket the folded stratigraphy at the northern and southern target zones at Buffalo Ankerite.

10.11 Drilling Completed Since Database Close-out Date

Drilling has continued at Grey Fox and Froome since the Fusion SQL database close-out date of March 31, 2026. Drilling completed since that date includes the following:

- **Grey Fox:** Drilling has continued, targeting potential mineralization expansion at the Gibson, Whiskey Jack and Stroud Zones.
- **Froome:** Infill and definition drilling is ongoing as of the date of this report.

10.12 QP Comment on Section 10

Mineral deposits within the Fox Complex are relatively narrow and exhibit variable orientations. As geological understanding improves, drill programs must be reoriented accordingly to more accurately capture the true volume of mineralization. Insufficient drilling in orientations that are perpendicular to the fabric of the deposits risk overestimation of the volumes of the Mineral Resource.

Many of the deposits remain open at depth and along strike.

The QP is of the opinion that the quantity and quality of lithological logging, collar and downhole survey data collected during exploration and infill drilling programs at the Fox Complex deposits are sufficient to support Mineral Resource estimation.



11 Sample Preparation, Analyses and Security

This section describes the sampling methods, sample preparation, analytical procedures, QA/QC programs, and sample security measures used to support the Grey Fox and Stroud Mineral Resource estimates which are part of the Eastern Properties and the main focus of this PFS.

For all other resource areas (remaining Eastern Properties, Stock and the Western Properties) which are not integral to this PFS, technical details of the sample preparation, analysis, QA/QC and security are as described in the 2025 Mineral Resource estimation technical report (McEwen, 2025).

11.1 Sampling Methods and Sample Preparation

11.1.1 Current Sampling Programs (McEwen)

McEwen's sampling protocols for drill core are designed to ensure that sample intervals are representative and appropriate for the geological features observed. Sample intervals are selected by McEwen geologists during the core logging process based on geological features such as lithological contacts, alteration, and mineralization. Sample lengths typically range from 0.3 m to 1.5 m, with key geological boundaries not crossed where possible.

Drill core samples are identified using uniquely barcoded sample tags that record the sample from-to interval and relevant notes. Sample tags are placed between the drill core and the core box at the sample interval. Sample booklet stubs are labelled, archived in boxes, and stored adjacent to the sampling facility.

For exploration drill core, the core is sawn into two equal halves using an industry-standard diamond saw. One consistent half of the core sample is returned to the core box in proper sequence, with the corresponding ticket stub placed in the core box. The other half of the sawn core is bagged, tagged with the corresponding sample number, sealed, and grouped into batches for shipment.

Sample bags are grouped into batches of 10 and placed into larger woven polypropylene bags for transport. All sample shipments are assigned specific work order numbers, with digital PDF documentation sent to the receiving laboratory prior to sample arrival.

11.1.1 Historical Sampling

Historical drilling data from previous operators that are relied upon for the Mineral Resource estimate includes programs completed by Primero and Brigus Gold, where drill core was logged and sampled by, or under the supervision of, Primero or Brigus Gold geologists. Drill core was cut in half using a diamond saw, with one half submitted for assay and the remaining half retained.

Historical sampling data from Exall, Noranda Exploration, and Apollo Gold were validated through previous Mineral Resource estimate studies performed by McEwen.



11.2 Analytical Methods and Laboratories

11.2.1 Analytical Methods

Until 2023, gold analysis was undertaken using fire assay (FA) with atomic absorption (AA) finish (FA-AA) for standard-grade samples and FA with gravimetric finish (FA-Grav) for over-limit samples. Since 2023, McEwen has adopted the PhotonAssay™ analysis method, which uses high-energy X-rays to determine gold content through the measurement of gamma ray emissions. PhotonAssay provides accurate results up to 350 ppm Au, with over-limit values undergoing secondary analysis at reduced X-ray beam intensity. This non-destructive method allows for the re-use of certified reference material (CRM) jars.

Multi-element analysis has been performed using inductively coupled plasma optical emission spectroscopy (ICP-OES) following aqua regia or four-acid digestion.

PhotonAssay differs from traditional fire assay in that it is non-destructive, uses larger sample volumes (300 g –500 g versus 30 g–50 g), and has a broader analytical range without requiring separate over-limit analysis in most cases.

11.2.2 Laboratories

A summary of laboratories used for sample preparation and analysis is provided in Table 11-1. All laboratories used since 2018 are commercial facilities operating under ISO/IEC 17025 accreditation. Affiliation with McEwen is noted where applicable.



Table 11-1: Analysis Laboratories Used in Resource Estimation

Laboratory	Period of Use	Location	Country	Analysis Method	Accreditation	Prep facility only	Analysis Lab	Affiliation to McEwen
PolyMet Laboratories	2009-2016	Cobalt, ON	Canada	FA	no information found		X	independent
Accurassay Laboratories	2012-2016	Thunder Bay, ON	Canada	FA	ISO 17025		X	independent
Swastika Laboratories	2012-2016	Swastika, ON	Canada	FA	ISO 17025		X	independent
Paragon Advanced Labs	2025-2026	Timmins, ON	Canada	PA	ISO 17025	X		affiliated (27.3%)
Paragon Advanced Labs	2025-2026	Surrey, BC	Canada	PA	ISO 17025		X	affiliated (27.3%)
Paragon Advanced Labs	2025-2026	Hamilton, ON	Canada	PA	ISO 17025		X	affiliated (27.3%)
Paragon Geochemical	2025-2026	Reno, NV	United States	FA	ISO 17025		X	affiliated (27.3%)
MSALABS	2023-2026	Timmins, ON	Canada	PA	ISO 17025		X	independent
MSALABS	2023-2026	Val-d'Or, QC	Canada	PA	ISO 17025		X	independent
MSALABS, Head Office	2023-2026	Langley, BC	Canada	FA	ISO 17025		X	independent
Pangea Laboratories	2021-2023	Guamuchil, Sinaloa	Mexico	FA	ISO 17025		X	affiliated (100%)
ALS Ltd.	2017-2020	Timmins, ON	Canada	FA	ISO 17025	X		independent
ALS Ltd.	2017-2020	Vancouver, BC	Canada	FA	ISO 17025		X	independent
Activation Laboratories Ltd.	2014-2022	Timmins, ON	Canada	FA	ISO 17025		X	independent
Activation Laboratories Ltd.	2014-2022	Ancaster, ON	Canada	FA	ISO 17025		X	independent



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11 Sample Preparation, Analyses and Security

Laboratory	Period of Use	Location	Country	Analysis Method	Accreditation	Prep facility only	Analysis Lab	Affiliation to McEwen
Activation Laboratories Ltd.	2020-2022	Geraldton, ON	Canada	FA	ISO 17025		X	independent
AGAT Laboratories	2009-2022	Timmins, ON	Canada	FA	ISO 17025	X		independent
AGAT Laboratories	2009-2022	Rouyn, QC	Canada	FA	ISO 17025	X		independent
AGAT Laboratories	2009-2022	Mississauga, ON	Canada	FA	ISO 17025		X	independent
SGS Canada	2011-2017	Sudbury, ON	Canada	FA	ISO 17025		X	independent
SGS Canada	2011-2017	Toronto, ON	Canada	FA	ISO 17025		X	independent
SGS Canada	2011-2017	Cochrane, ON	Canada	FA	ISO 17025		X	independent

Notes:

- Method abbreviations: FA = Fire Assay, PA = PhotonAssay™
- "Preparation Facility Only" marked with "X" indicates laboratories that only prepare samples.
- "Analysis Lab" marked with "X" indicates laboratories that perform analysis.
- Affiliation indicates relationship to McEwen (independent or affiliated with ownership% where applicable).



Exploration samples are shipped multiple times per week to Paragon Advanced Labs for preparation and photon assay analysis. Since September 2025, most photon assay work has been completed by Paragon.

Check assays (approximately 5%) for fire assay are submitted to ALS Vancouver, while PhotonAssay check assays (approximately 20%) are sent to MSALABS (Timmins, Ontario).

11.2.3 Historical Laboratories

Historical laboratories used for assays incorporated into the Mineral Resource estimate include those listed below and in Table 11-1.

- Swastika Laboratories (Swastika, Ontario)
- SGS Canada (Cochrane, Ontario; Rouyn-Noranda, Québec)
- AGAT Laboratories (Mississauga, Ontario)
- PolyMet Laboratories (Cobalt, Ontario)
- Techni-Lab (Timmins, Ontario)
- ALS Canada (various locations)
- XRAL Laboratories (Intertek Testing Services-Bondar Clegg)
- Bell White Laboratories
- Accurassay Laboratories

11.3 Quality Assurance and Quality Control

11.3.1 QA/QC Program

McEwen maintains a QA/QC program that includes the insertion of blank samples, CRMs, and duplicate samples into the drill core sampling sequence.

Blanks are inserted at a ratio of one per 20 samples (or two per 21 samples for photon assay). Prior to 2019, blanks consisted of core blanks from unmineralized core sections and certified blank pulp material. Since 2019, coarse blanks composed of commercial high-silica aggregate purchased in bulk from Technominex in Rouyn-Noranda, Québec, have been used.

CRMs are inserted at a ratio of one per 21 samples. Prior to 2019, CRMs purchased from Rocklabs Ltd. (SK78, SG66, SN75, SE68, Ox98) were used. Beginning in February 2019, OREAS standards replaced Rocklabs standards. For PhotonAssay, CRMs specifically certified for the PhotonAssay process are utilized. Insertion of standards with photon assay is performed at the external laboratory using code-matching CRM jars provided by McEwen.

Duplicate samples were inserted between 2017 to 2019, with pulp duplicates at a ratio of one per 20 samples and field duplicates inserted between 2017 and 2018. Following external review (Bloom and Jollette, 2019), which indicated very good duplicate repeatability, McEwen discontinued the insertion of pulp and field duplicates and now relies on each laboratory's internal duplicate protocols. For PhotonAssay, duplicate readings of the assay jar are used to calculate a weighted average result, which is reported on the certificate. The lab also performs preparation duplicates at a rate of one per 21 samples.



External umpire testing has been conducted since at least 2017, with pulps representing approximately 5% to 7% of the annual assay volume submitted by McEwen for check analysis. Following adoption of Paragon as the primary assay laboratory, the rate of comparison testing has increased to 20%. MSALABS has been designated as the referee sample laboratory.

11.3.2 QA/QC Results

The QA/QC results from McEwen's current sampling programs demonstrate good accuracy and precision. The insertion of CRMs has shown acceptable accuracy, with the vast majority of results falling within acceptable tolerance limits (typically ± 2 standard deviations of certified values). Blank results consistently return values below detection limits, indicating minimal contamination. Duplicate analyses show good precision, with coefficient of variation values typically below 20%, indicating acceptable repeatability.

QA/QC performance is monitored daily as assays come in and reviewed as a group on a monthly basis. Where CRM failures are identified, corrective actions are implemented immediately, including re-analysis of affected batches and communication with the laboratory. No systematic bias has been identified in the assay data.

The QP has reviewed the QA/QC data and considers the results sufficient to support the reliability of the assay data used in the Mineral Resource estimate.

11.3.3 Historical QA/QC

Historical QA/QC programs from previous operators varied in their approach and rigor. Review of historical QA/QC indicates that programs completed by Primero and Brigus Gold included in-field insertion of blanks, CRMs, and field duplicates, with in-laboratory QA/QC managed by the contracted analytical laboratories. Based on review, the QA/QC program is considered adequate for the data quality.

Historical QA/QC programs from operators whose data are not used in the Mineral Resource estimate (including Apollo Gold, Exall, and Noranda Exploration) have not been relied upon and are not considered material to this report.

Overall, the historical QA/QC data are considered adequate for use in the Mineral Resource estimate and do not introduce material uncertainty.

11.4 Sample Security

11.4.1 Chain of Custody and Handling

Sample chain of custody is maintained through standardized procedures including tagging, bagging, sealing, and documented transport.

Sample bags are sealed with heavy plastic tie-wraps after filling and labelled with the sample number. Sealed transport bags are stored within a secure area immediately adjacent to the logging and sampling facility. From 2014 to 2021, exploration drill core samples were shipped off site to commercial laboratory preparation facilities. Sample transport was handled by McEwen personnel or contracted carriers.



From 2021 to 2023, samples were sent to Pangea Laboratory in collapsible plastic storage containers, sealed using numbered plastic seals. Samples were shipped via Gardewine (intermediary only) to a broker in Naco, Arizona, who then reshipped the samples to the laboratory.

Since 2023, samples have been transported directly to MSALABS and Paragon by McEwen employees. Samples shipped to Paragon facilities in Hamilton, Ontario or Surrey, BC are transported in sealed wooden shipping crates to maintain sample integrity.

Drill core, rejects, and pulps are stored in secure locations at the Fox Complex facility. The core logging facility is locked, with access restricted to authorized personnel.

11.4.2 Security Practices by Period

Historical operators maintained sample security through chain-of-custody procedures appropriate for their periods of operation. Drill core was boxed, covered, and sealed at the drill rigs and transported to logging facilities, where company personnel assumed responsibility for core handling. Samples were cut by technicians, bagged, sealed, and grouped into batches for shipment.

11.4.3 Databases

Since 2017 when McEwen acquired the Fox Complex from Primero, a SQL database in Datamine Fusion has been the primary location for the collation, storing and tracking of all geological data including the sampling and assaying procedures.

Datamine's DH Logger software is used to record all logging and sampling data entry. Upon completion of logging, and subsequent satisfaction of a senior geologist and the database manager, the files are compiled, organized, and uploaded to the Datamine Fusion SQL database. Currently, the data from DH Logger is uploaded electronically, as is assay data from the external laboratories. Laboratory data is uploaded by the Database Manager into the Datamine Fusion SQL database.

The Fusion database uses a series of read-write permissions to ensure database integrity; access is restricted only to authorised personnel. Back-ups of the digital data archive are performed on a regular basis.

For the earlier 2014–2017 programmes, drill hole logging data (collar locations, down hole surveys, geology etc.) and related assays were collected and added to Gemcom project files. Separate workspaces were used for Grey Fox and Froome.

Regular QA/QC audits on collar locations, down-hole surveys, and assay certificates were performed over the period. Once verified, drill holes would be locked in the Gemcom project to ensure no changes to the data could occur. After McEwen's acquisition, four GEMCOM databases were merged into one SQL database in Datamine Fusion.

At the Stock East Zone, early exploration holes (2018 campaign) were logged and processed utilizing the Gemcom software and captured as PDF files using Primero's formatting. In 2019, the team transitioned to using the Datamine products and procedures described above.



Historical information before 1993 is not relied on for resource estimation at Grey Fox.

11.5 QP Opinion

The QP has reviewed the sampling methods, sample preparation and analytical procedures, QA/QC programs, and sample security measures described in this section. Based on this review, the QP offers the following opinions:

- The sampling methods used are appropriate and adequate to support representative sample collection for Mineral Resource estimation.
- Sample preparation and analytical methods, including fire assay, PhotonAssay™, and ICP techniques, are appropriate for the style of mineralization encountered and are consistent with industry-standard practices.
- QA/QC procedures implemented by McEwen, including the insertion of blanks, CRMs, and duplicates, along with external umpire testing, are sufficient to demonstrate the reliability of assay data.
- Sample security measures, including chain-of-custody documentation, sealed transport, and secure storage, are adequate to maintain data integrity.
- The data derived from the sampling, analytical, and QA/QC programs are suitable for use in Mineral Resource estimation.



12 Data Verification

12.1.1 Drill Core Verification

The QP reviewed all assay, collar, survey, and QA/QC data associated with drilling conducted from 2014 to the most recent results available in 2026, including McEwen drilling programs on the Eastern Properties and relevant historical data incorporated into the Mineral Resource model.

The QPs reviewed prior technical reports and verification studies and conducted independent audits, data checks, and physical inspections, where possible.

The QP applied the following data verification procedures:

- **Database Audits:** Assay results were cross-checked against original laboratory certificates to confirm data integrity. Assay intervals, sample IDs, and QA/QC sample insertions were validated within the digital assay database.
- **Collar and Survey Verification:** Collar locations and downhole surveys were verified against original survey records and field GPS data. Discrepancies were identified for the Stroud project and reconciled.
- **QA/QC Review:** The QP reviewed QA/QC data including blanks, CRMs, duplicates, and umpire laboratory results as described in Section 11. Statistical evaluations confirmed acceptable accuracy and precision.
- **Core Inspections:** The QP conducted site visits to assess drill core recovery, logging practices, and core storage procedures to confirm consistent geological and sampling protocols.
- **Independent Validation:** Randomly selected samples and pulps from the database were independently arranged and submitted for umpire laboratory analysis to confirm assay accuracy, with results reviewed by the QP.

12.1.2 Verification of Historical Data

Historical drilling data incorporated into the Mineral Resource model originates from multiple operators, including Primero, Brigus Gold, Apollo Gold, Chevron Minerals Ltd., Pancontinental Mining Ltd., and Gunnex Ltd. The QP audited a significant portion of these datasets by cross-checking assay certificates, collar records, and QA/QC information against the compiled database.

Verification efforts focused primarily on data generated from the 2010s onward, where documentation is more complete. Key assay results were confirmed where original certificates were available.

Although some early datasets lack complete QA/QC records or chain-of-custody information, most historical data incorporated into the model have been verified either directly or through corroborative evidence, including prior technical reports and independent reviews.



12.1.3 Limitations on Data Verification

The QP notes the following limitations, which are typical of historical exploration datasets:

- Incomplete QA/QC records and missing assay certificates for some early drilling programs.
- Downhole surveys for certain underground holes completed prior to 1995 rely on collar orientation only, without continuous deviation measurements.
- Gaps in chain-of-custody documentation for older historical drill programs.
- Physical core inspection was limited to available core; some legacy core storage conditions were suboptimal but adequately documented.

These limitations have been considered in the classification of Mineral Resources and are not considered to materially impact the overall reliability of the dataset.

12.1.4 Data Verification Results

The QP's verification work indicates that:

- QA/QC results show consistent accuracy and precision within accepted industry limits.
- Duplicate and umpire assay comparisons confirm repeatability and do not indicate systematic bias.
- CRM recoveries indicate acceptable analytical performance across all laboratories, including McEwen-affiliated facilities (Paragon Advanced Labs and Pangea Laboratories) and independent laboratories (MSALABS, ALS, AGAT, SGS).
- Survey, collar, and assay data are internally consistent and suitable for Mineral Resource estimation.

12.1.5 QP Opinions on Data Adequacy

The QPs responsible for the respective disciplines are satisfied that:

- Sampling, assay, survey, and QA/QC data used in the Mineral Resource estimate are reliable and of sufficient quality for the purposes of this technical report.
- Verification procedures support the accuracy and precision of the dataset.
- Identified limitations are typical of historical datasets and have been appropriately considered in resource classification.
- Overall, the QPs are satisfied that the data supporting the Mineral Resource estimate meet the requirements of NI 43-101 and CIM Best Practice Guidelines.

12.2 Property-Specific Verification

12.2.1 Eastern Properties Data Verification

Verification indicates that McEwen drilling data since 2014 are of sufficient quality and integrity, with strong QA/QC performance and consistent survey and collar accuracy.



12.2.2 Stroud Project Data Verification

Verification work indicates that the historical Stroud drill hole database is sufficiently reliable for use in the current Mineral Resource estimate. Collar locations were validated through RTK re-surveying and transformation from the local mine grid to UTM NAD83, resulting in acceptable spatial agreement.

Assay verification included recovery of historical certificates, re-logging and re-sampling of select holes, and a confirmatory drill program that twinned approximately 15% of historical holes. Independent checks by both the database manager and the QP identified only minor discrepancies (3 of 1,246 assay records), all conservative relative to original certificates.

Overall, drilling, survey, and assay data are considered adequate to support Mineral Resource estimation and geological interpretation.

12.3 Geochemical and Water Analysis Verification

12.3.1 QP Verification Procedures

For water quality data used in the water balance model, the QP reviewed available datasets and reports provided by McEwen to assess their suitability as geochemical source terms for different areas and rock types across the site. Calculation methods and model outputs, including spreadsheet and GoldSim models, were also reviewed.

The QP reviewed geochemical and water quality data used in the water and solute balance model described in Section 20. These data include site monitoring results (e.g., from the Black Fox East and West stockpiles), as well as laboratory-based static and kinetic testing, including shake flask extraction and humidity cell test results.

The QP evaluated the available data relative to their intended application in defining geochemical source terms for site features and considers the selected datasets appropriate for use in the model. Where available, data provided in project reports were compared with original laboratory analytical results to confirm consistency, including review of kinetic testing datasets.

12.3.2 Data Verification Results

The QP considers the available data to be of acceptable quality. In cases where both summarized project report data and laboratory analytical results were available, comparisons indicate consistency between reported values and underlying analytical data.

The QP has determined that a greater degree of uncertainty is associated with the representativeness of specific datasets for application to certain source terms, particularly for the Grey Fox underground mine and associated surface features. This uncertainty does not reflect deficiencies in analytical data quality, but rather limitations in the availability of appropriate analogues to characterize future site-specific conditions. For example, there is uncertainty associated with predicting the quality of water to be pumped from underground workings, due to the absence of directly comparable datasets.



Where uncertainty exists in the applicability of specific datasets, the QP considers that conservative assumptions have been applied in the selection of water quality inputs for modelling purposes.

The QP reviewed the water balance methods and key data inputs, including runoff coefficients, catchment areas, precipitation data, and geochemical source terms, and considers them appropriate for the intended level of study. GoldSim is recognized as an industry-standard tool for water and solute mass balance modelling in mining applications.

The QP recommends that, as additional geochemical characterization testing is completed in the future, these data be compared with historical datasets to further refine the selection of representative inputs for modelling of full-scale waste rock runoff and mine dewatering conditions.

12.3.3 QP Opinions on Data Adequacy

The QPs responsible for the respective disciplines are satisfied that:

- Water quality datasets used in development of the water balance model are of acceptable quality. Although there is uncertainty in the applicability of specific datasets to certain geochemical source terms (e.g., identifying appropriate analogues for underground dewatering), conservative assumptions have been applied in assigning water quality inputs.

12.4 Tailings Storage Data Verification

The QP reviewed data used in the development of the tailings storage design and associated cost estimates. The review included geospatial, geotechnical, process, and cost input data, with the objective of assessing their suitability and reliability for the PFS level of study.

12.4.1 Source Data

The following primary data sources were reviewed by the QP:

- Topographic and Geospatial Data:
 - 2025 aerial drone LiDAR data provided by Arcadis for the Stock Mill TMA and surrounding areas.
 - The 2016 Central Ontario Orthophotography Project (COOP) 2-m digital elevation model (DEM) for areas not covered by the LiDAR survey.
- Geotechnical Data:
 - Historical borehole and laboratory test data as referenced in prior design work (Golder 2015), including subsurface conditions relevant to dam foundations and embankment performance.
- Tailings Production Data:
 - Historical operating data and projected mill throughput and tailings production rates used to define storage requirements and facility life.

The QP reviewed these datasets in the context of their application to tailings storage design and considers them appropriate for use in support of the current level of study.



12.4.2 Data Verification Methods

The QP applied the following data verification procedures:

- **Topography and Volumes:**
Digital terrain models were cross-referenced against both recent LiDAR data and historical DEM datasets to assess the accuracy of derived dam surface elevations and tailings storage volumes. Comparisons between data sets indicate consistency in overall site geometry. Volumetric estimates were evaluated using industry-standard software tools (e.g., Autodesk Civil 3D) to confirm agreement between conceptual design volumes and actual site topography.
- **Geotechnical Conditions:**
Foundation conditions and embankment design parameters were reviewed against available borehole logs and laboratory test results. Where data gaps existed, the QP notes that conservative design values were applied, and that additional investigation is appropriately recommended for subsequent study phases.
- **Process and Tailings Data:**
Tailings production rates, physical tailings characteristics, and storage assumptions were reviewed relative to historical site operating data and typical industry benchmarks. The QP evaluated these inputs to confirm that projected tailings volumes and facility life are reasonable and consistent with the proposed mining and processing rates.
- **Supplemental Capacity and Site Constraints:**
Available lease boundary mapping was reviewed and compared with aerial imagery to assess land availability for potential future expansion. The QP evaluated the consistency between mapped boundaries and physical state conditions to support conclusions regarding expansion potential.
- **Tailings Storage Facility Costing:**
Tailings storage facility capital costs were developed as a conceptual estimate using published, auditable benchmarks, including RECLAIM 7.0, escalated via Statistics Canada CPI to 2026 CAD; publicly available government engineering and remedial-action cost estimates, including US Environmental Protection Agency and US Army Corps of Engineers; and selective vendor budgetary quotations. The estimate includes contingency appropriate to a Class 5/4 study.

12.5 Water Treatment

Water treatment inputs are based on water chemistry and flow rate data derived from the water balance. Water treatment costs are based on input from a water treatment specialist with experience in the design and specification of water treatment facilities and are supported by historical project pricing for the purchase and installation of vessels, pumps, and industrial metal buildings.

12.6 Metallurgical Testing and Analysis

The QP was unable to independently verify the representativeness of all samples used in historical metallurgical testwork programs. A significant portion of the metallurgical testwork was completed by previous property owners and operators, and the documentation available to the QP did not, in all cases,



include sufficient detail regarding sample selection methodologies, compositing procedures, sample locations, or the rationale used to define metallurgical domains.

Consequently, the QP cannot confirm that the samples tested are fully representative of the range of mineralization and metallurgical variability present within the deposit. The QP has relied upon the available historical records and the results of the metallurgical programs as documented. In the absence of complete sample selection records, it is inferred, based on industry practice, that the individuals responsible for the test programs selected and composited samples were selected and composited in a manner intended to reasonably represent the anticipated processing characteristics of the mineralized material. While the testwork results are considered suitable for the purposes of this study, the uncertainty associated with sample representativeness should be recognized when evaluating the metallurgical performance projections presented herein. Additional variability testwork utilizing samples selected under current QA and documentation protocols is recommended to further validate the metallurgical response of the deposit.

The QP is familiar with the laboratories that conducted the historical metallurgical testwork and understands them to be established facilities with recognized metallurgical testing capabilities. Based on the QP's industry experience and prior interactions with these laboratories, their testing procedures, analytical methods, and QC practices are considered to be consistent with accepted industry standards for metallurgical investigations.

12.6.1 QP Opinions on Data Adequacy

Although the QP was not directly involved in the execution of the historical test programs and has not independently audited the laboratories' quality management systems, there is no indication from the available documentation or test results that the metallurgical data are unreliable or unsuitable for the purposes of this study.



13 Mineral Processing and Metallurgical Testing

13.1 Summary

13.1.1 Comminution

Comminution testwork was conducted on Grey Fox drill cores across several test campaigns, and included Steve Morrell Comminution (SMC) testing, Bond Ball Mill Work index (BWi) testing, and grind calibration work. The testwork indicates a requirement of high specific grinding energy. SMC testing yielded drop-weight index (DWi) values of 8 to 10 kWh/m³ and Axb parameter values of 30 to 35, corresponding to the 76th to 89th percentile range of the JKTech SMC database. These results reflect the highly competent nature of the ores tested. The BWi results also confirmed the extremely high energy requirement to grind the ore, with values for the harder composites reaching up to 27 kWh/t (147 Zone) and the softer end (some parts of the underground zones) closer to 18 kWh/t.

With the harder samples being from the 147 Zone, which represents approximately 2/3 of the deposit, it is evident that the average and design hardness values should be skewed towards the harder end of the range of test results. BWi test results were weighted according to ore tonnage distribution by domain. The 85th percentile is 27.1 kWh/t. The highest BWi value (27.1 kWh/t) applies to Domain 31 (147 Zone iron-rich mafic volcanics), which comprises 28.3% of the total orebody (refer to Table 13-1 for domain-specific distributions). Additional variability testing is scheduled for the next phase of the project to gain more granularity at the top end of the hardness curve.

Table 13-1: Orebody Breakdown by Zone and BWi

Domain ID	Tonnage	Proportion%	BWi (kWh/t)
Domain 10 - Gibson South	3,506,192	20.9	18.7 ⁽¹⁾
Domain 11 - Gibson North	1,770,573	10.6	19.7 ⁽¹⁾
Domain 20 - Contact Zone	1,253,925	7.5	20.7 ⁽²⁾
Domain 30 - 147 Zone Mafic	4,763,982	28.4	25.3 ⁽²⁾
Domain 31 - 147 Zone Fe-rich mafics	4,741,755	28.3	27.1 ⁽³⁾
Domain 32 - Whiskey Jack	717,137	4.3	19.5 ⁽⁴⁾
Total Orebody	16,753,564	100	23.4

Notes:

- (1) Average of three samples tested in 2025.
- (2) Average of two samples tested in 2014 (XPS Phase 2).
- (3) Single result of one sample tested in 2014 (XPS Phase 2).
- (4) Average of two samples tested in 2025.



13.1.2 Gravity Testing

Gravity recovery testwork conducted at ALS Metallurgy and XPS (on open pit samples) confirms the presence of coarse free gold, with gravity recoverable gold (GRG) test values ranging between 4% and 32%. Extended gravity recoverable gold (EGRG) tests achieved mass pulls below 0.2% of new feed mass, with high gold grades recovered to a concentrate between 200 g/t and 1,200 g/t Au, confirming the presence of free coarse gold in the samples tested. The underground samples tested in 2025 contained significantly less GRG compared to the open pit samples tested during preceding test programs. For the underground ore, a gravity concentration step is not essential and can be omitted. Inclusion of a gravity recovery step is recommended for treatment of the open pit ores.

13.1.3 Flotation

Bulk sulphide flotation has been shown to be highly effective on Grey Fox ore types, especially for the pyritic and high-sulphur ore types. Testing by XPS and SGS on open pit samples yielded gold recoveries ranging from 91% to 97% into an 8% to 12% mass pull primary concentrate, using standard xanthate-type collectors at natural pH. The highest recoveries exceeding 95% were achieved at finer grind sizes ($P_{80} = 60 \mu\text{m}$). However, subsequent testwork confirmed that similar recoveries could be achieved at coarser grind sizes (up to $P_{80} = 120 \mu\text{m}$). The flotation concentrates produced were primarily pyrite with lower amounts of base metals, amenable to direct cyanidation. A carbon pre-float was ineffective due to the presence of carbonates, although this was not found to have a significant impact on overall flotation performance. Underground mine samples tested in 2025 yielded marginally lower overall recoveries using a flotation flowsheet, with an average gold recovery of 87.2%.

For all samples tested to date, a gravity-flotation flowsheet outperformed a gravity-leach flowsheet by 7% to 14% in terms of overall gold recovery. However, the rougher flotation concentrates produced did not achieve saleable concentrate grades as a standalone product. Further upgrading through additional cleaning stages would be required, but these have not been tested to date.

13.1.4 Cyanidation of Flotation Products

Cyanidation of the flotation concentrates resulted in variable gold extraction, dependent on pyrite content and grain size. In general, as-floated concentrates ($P_{80} \sim 60 \mu\text{m}$) yielded gold recoveries via cyanidation of 77% to 82% for the open pit samples. Regrinding of the concentrates to a $P_{80} \sim 25 \mu\text{m}$ improved gold extraction by approximately 2% to 4%. No additional benefit was observed by grinding to finer sizes approaching a $P_{80} \sim 12 \mu\text{m}$. This suggests that some of the contained gold in pyrite is either very fine and locked within the pyrite lattice or is in solid solution and therefore not accessible through cyanidation.

For the underground samples, the results were more variable, with cyanidation gold extractions after regrinding ranging between 46% and 95%. The 46% result is considered anomalous, as leaching was not complete by the end of the test. After discounting this result as anomalous, two of the other seven samples yielded extractions on the order of 70%. Therefore, the underground samples underperformed compared to the open pit samples.



Cyanidation of flotation tailings consistently yielded high extractions. The open pit samples again outperformed the underground samples, with extractions exceeding 74% and final tailings grades below 0.1 g/t Au achieved. By comparison, the underground samples yielded an average extraction of 72% and an average final tailings grade of 0.19 g/t Au.

A split-leach approach, comprising concentrate and tailings leaching, is supported by the testwork, with the potential to match or exceed whole-ore leach recoveries while decoupling primary grind size, to some extent, from overall recovery, thereby offering potential savings in grinding costs.

13.1.5 Whole-Ore Cyanidation

Whole-ore leach testwork was conducted on a range of composite materials and showed a wide range of recoveries tied to head grade and sulphur content. Free-milling ores from the 147 Zone returned gold recoveries of 88% to 94% at moderate grind sizes (P_{80} of 75 to 100 μm). In contrast, low-grade pyritic ores or those with high S/Au ratios from the Contact Zone (CZ) and Grey Fox South showed reduced recovery, with values on the order of 71% to 80%, even after extending leach durations beyond 24 h.

The open pit samples tested in 2025 exhibited similar variability, ranging from as low as 64% recovery to as high as 92% recovery, with an average of 80%.

Preg-robbing is minimal and appears to correlate with total organic carbon (TOC) levels above 0.5%. The whole-ore cyanidation route is viable for higher-grade, low-sulphur ores, but should be avoided for high-pyrite or low-grade domains due to lower recoveries and high reagent consumption.

13.1.6 Cyanide Detoxification

Cyanide (CN) destruction via the SO_2 /air process was demonstrated to be effective. In batch and continuous tests conducted on leached slurries from both the 147 Zone and CZ master composites (MCs), Weak Acid Dissociable (WAD) CN levels were reduced from approximately 165 mg/L to <0.1 mg/L using SO_2 doses of 4.0 to 4.8 g/g CN_{WAD} and a copper catalyst at 0.06 to 0.12 g/g CN_{WAD} . The resulting solutions met Ontario Provincial Water Quality Objectives (PWQO) guidelines for cyanide and most metals. Some elevated molybdenum levels were noted in decant solutions, but all other metals remained within acceptable limits. The process is considered robust and scalable.

The underground mine samples from Whiskey Jack and Gibson Zones have not been tested to date but their response to this method of detoxification is expected to be similar. Testing is recommended to confirm this assumption.

13.1.7 Sedimentation

Settling tests show that underflow densities of approximately 65% solids (w/w) can be achieved using non-ionic flocculants at dosages on the order of 25 g/t. Dynamic settling tests yielded optimum solids loadings of 0.41 to 0.47 t/m²/h for the 147 Zone and CZ, respectively. Rheology testing confirms manageable yield stresses (27 to 52 Pa at 68% solids), indicating that conventional thickening and deposition strategies are applicable. The sedimentation behavior of the Whiskey Jack and Gibson Zone underground samples was



not tested. Future sedimentation testwork should include flotation concentrate samples to confirm the sizing of a concentrate thickener for the proposed flowsheet.

13.1.8 Environmental

Static geochemical testing, including acid-base accounting, net acid generating, and shake flask extraction (SFE) testing of waste rock and tailings materials, suggests that most samples are non-potentially acid generating (NPAG), with 70% to 78% exceeding the neutralization potential ratio (NPR) >2 threshold. The remainder are considered uncertain and subject to ongoing kinetic testing. Paste pH values for all samples were alkaline (8.5 to 9.6), and sulphur was dominantly present as sulphide. Tailings residues from cyanidation and detoxification were classified as NPAG, with a Neutralizing Potential to Acid Potential (NP:AP) ratio greater than 40. Decant water chemistry met applicable discharge criteria, aside from minor molybdenum exceedances. No environmental testwork was conducted on underground samples as part of the 2025 program.

13.2 Introduction

This chapter summarizes all available metallurgical testwork results for the Grey Fox deposit, together with other pertinent information required to support preliminary feasibility and trade-off studies. For the purposes of this report, the testwork programs were grouped as historical programs, referring to programs conducted for previous owners, and recent programs conducted by McEwen. Process design criteria are defined based on these test results and are used as the basis flowsheet development and subsequent equipment sizing for the studies.

13.3 Test Work Completed to Date

This section summarizes all completed metallurgical and geochemical testwork available on the Grey Fox deposit. Please refer to Table 13-2, where each document is listed by its name, title, and general theme, excluding any reports that are strictly proposals or quotations for future work.

Table 13-2: Test work Programs

Authoring Entity and Date	Report Title	Scope of Work
JKTech, Oct. 2012	SMC Test Report on Three Samples from Grey Fox	Comminution parameters for 147, breccia, and tuff domains
SGS Canada, March 27, 2013	13327-002 Met Report: A continuation of testwork on samples from Grey Fox	Gravity, flotation, whole-ore leaching, cyanide detox, rheology
SGS Canada, March 27, 2013	13327-002 QEMSCAN Report: A continuation of testwork on samples from Grey Fox	Bulk mineralogy and gold deportment
XPS Consulting and Test work Services, Dec. 6, 2013	Geometallurgical Definition and Metallurgical Assessment of Grey Fox Zones, Phase 1	Initial scoping-level gravity and cyanidation tests



Authoring Entity and Date	Report Title	Scope of Work
XPS Consulting and Test work Services, Feb. 10, 2014	Geomet Definition and Metallurgical Assessment of Grey Fox Zones, Phase 2	Variability testing of multiple 147 Zone and CZ composites
XPS Consulting and Test work Services, Nov. 10, 2014	Geometallurgical Assessment of Grey Fox Pyritic (Phase 3) and Grey Fox South	Flotation and leaching of flotation products. Inclusion of Grey Fox South.
XPS Consulting and Test work Services, June 9, 2015	Brigus Grey Fox Phase IV: Leaching with Flotation	Flotation optimization and carbon prefloat analysis
Golder Associates, March 30, 2016	Grey Fox Deposit – Interim kinetic geochemistry results update #2	Leach columns and kinetic test results over short durations
ALS Metallurgy, Feb. 11, 2020	KM6034 –Metallurgical and Environmental Test Work for the Grey Fox Deposit	Cyanidation leach, grind sensitivity, environmental residue characterization. Grey Fox 147NE Composite
ALS Metallurgy, October 2025	KM7663 –Metallurgical and Environmental Test Work for Grey Fox Underground Zones	Comprehensive test program for the previously untested Whiskey Jack and Gibson Zones

13.4 Historical Metallurgical Testing Results

13.4.1 2013 SGS Master Composite & Variability Program

The report titled “An Investigation into Continuation of Testwork on Samples from Grey Fox” (SGS Project 13327-002, March 27, 2013) (SGS, 2013) was commissioned to support the ongoing metallurgical characterization of ore from both the 147 Zone and the CZ and included an expanded testwork scope focused on processing and environmental aspects.

13.4.1.1 Sample Selection and Compositing Rationale

Two (2) MCs were generated from coarse assay rejects (stage-crushed to <10 mesh) representing the 147 Zone and CZ ore types. Sub-sampling instructions were provided by EHA Engineering; preparation, splitting and head sampling were completed at SGS. Composite design targeted representative lithologies and gold grade population, while providing ≥10 kg for each downstream test per domain.

13.4.1.2 Head Assay

Screened-metallics fire assay indicates head grades of 3.35 g/t Au for the 147 Zone MC and 4.64 g/t Au for the CZ MC, with >97% of the gold reporting to the –105 µm fraction. There is good agreement between the grades determined through the screened metallics procedure and the direct fire assays. This suggests the absence of a significant nugget effect. This observation is supported by the high proportion of gold in the –150 mesh fraction.

Multi-element Inductively Coupled Plasma scans showed low deleterious element levels (Cu ≤0.015%, As <50 ppm), while total carbon occurred mainly as carbonate (~95% of C_T). QEMSCAN analysis identified



a predominantly quartz–carbonate gangue with 3% to 5% pyrite; visible gold was free-milling and largely liberated at <75 µm. Key results are summarized in Table 13-3.

Table 13-3: Head Assays for 147 Zone and CZ Master Composites

Master Composite	Screened Metallics Au (g/t)	Direct Fire Assay Au (g/t)	Cu (%)	Ag (g/t)	S (%)	C _t (%)	CO ₃ (%)
147 Zone	3.35	3.36	0.010	1.6	1.70	2.76	9.77
CZ	4.64	4.56	0.015	1.8	1.90	1.91	6.61

13.4.1.3 Bulk Modal Mineralogy

Modal mineralogy determined by QEMSCAN at a grind of P₈₀ ~150 µm revealed that both MCs were dominated by silicate gangue (>70 vol%), with carbonates constituting the next most abundant group, and sulphides making up only a few percent. Please refer to Figure 13-1. The graphite composite exhibited similar silicate proportions but contained ~4% pyrite and trace graphite.

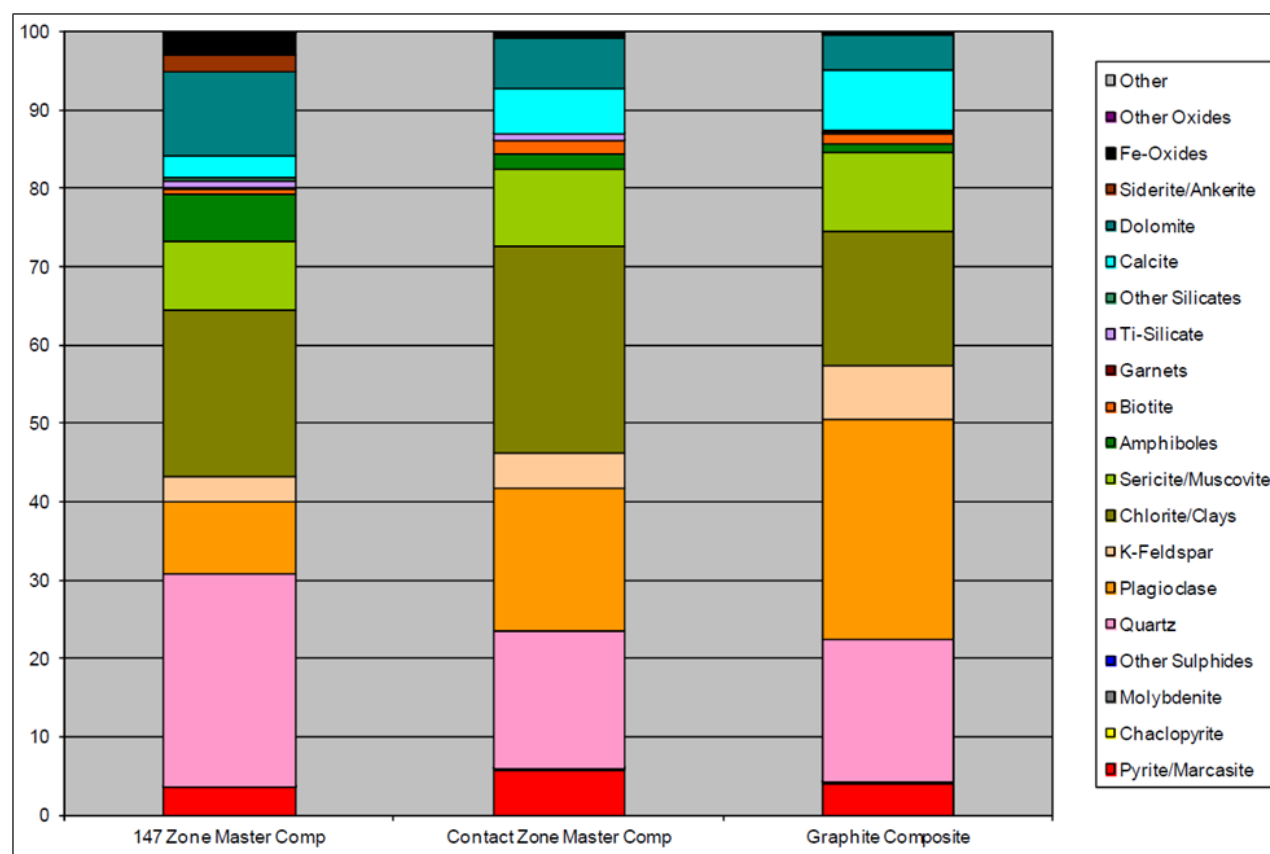


Figure 13-1: Modal Compositions of 147 Zone, CZ and Graphite Master Composites



13.4.1.4 Grain Size and Liberation Statistics

QEMSCAN particle sectioning across four sieve fractions (+150 µm, –150 µm/+75 µm, –75 µm/+25 µm and –25 µm) showed that:

- Pyrite liberation averaged 70% across the whole feed. Liberation improved markedly with size reduction, reaching ~80% in the –75 µm/+25 µm band and >90% in the –25 µm band for both MCs. Non-liberated pyrite was predominantly intergrown with silicates (hard and soft) with only minor complex locking.
- Silicate gangue (quartz + feldspars + chlorite/sericite) was inherently well liberated (>90% free or liberated) at all sizes.
- Carbonate liberation was limited in the coarse fractions, with only ~45% free/liberated above 150 µm. Liberation improved to ~60% in the –75 µm/+25 µm interval and to 85%–90% below 25 µm. Residual locked carbonate was mainly associated with silicates.

Table 13-4 below summarizes how much of the pyrite (the main sulphide of economic interest) was either free or liberated. Values were calculated from the liberation data tables in the SGS QEMSCAN report [3] for the three composites. These data confirm that a primary grind of $P_{80} = 75 \mu\text{m}$ is sufficient to expose the majority of gold-bearing pyrite surfaces, consistent with the >90% Au flotation recoveries obtained in SGS Tests F7–F8 [2].

Table 13-4: Pyrite Liberation Per Size Range for Three Composites

Size Fraction	147 Zone MC	CZ MC	Graphite Composite
Combined	74.4%	74.9%	72.1%
+150 µm	58.9%	51.0%	36.2%
-150 µm/+75 µm	69.6%	69.2%	69.4%
-75 µm/+20 µm	80.9%	77.7%	80.8%
-20 µm	91.3%	92.6%	95.3%

13.4.1.5 Gravity Recovery

EGRG tests on the 147 Zone MC achieved 28% gold recovery to a Mozley concentrate grading 629 g/t Au at $P_{80} \approx 91 \mu\text{m}$, while a finer grind ($P_{80} \approx 69 \mu\text{m}$) increased recovery to ~32% with concentrate grades exceeding 1,100 g/t Au. The CZ MC yielded ~22% GRG at a similar grind, reflecting a higher proportion of gold hosted in sulphide minerals. Refer to Table 13-5 for detailed results.

Table 13-5: Gravity Recovery for 147 Zone and CZ

Test ID	Ore Type	Grind P_{80} (µm)	Conc. Mass (%)	Conc. Au (g/t)	Au Recovery (%)
G 5	147 MC	91	0.13	629	28.0
G 8	147 MC	69	0.10	1 142	32.1
G 3	CZ MC	97	0.18	511	17.2
G 9	CZ MC	74	0.09	1 242	21.6



The gravity recoveries achieved are marginally below the rule-of-thumb minimum of 25% recovery required during laboratory testing to justify inclusion of a gravity recovery step in the flowsheet.

13.4.1.6 Flotation

Sulphide flotation with carbon pre-flotation of gravity tails was conducted using a reagent suite of kerosene, potassium amyl xanthate (PAX), methyl isobutyl carbinol (MIBC), and MaxGold 900 (modified dithiocarbamate). This testing showed gold recoveries exceeding 90% at mass pulls between 10% and 15%, producing concentrates grading 21 g/t to 33 g/t Au.

Combined gravity and flotation recoveries of +93% and +94% were observed for 147 Zone and CZ composites, respectively.

At the time of the flotation tests, only total carbon assays were available, and it was suspected that the carbon was only present in organic or graphitic forms. However, it was later determined that most of the carbon present was in the form of carbonate, which explains the poor performance of the carbon pre-float, which recovered only 1.8% of the contained carbon in tests F-7 and F-8.

Refer to Table 13-6 and Table 13-7 for additional information.

Table 13-6: Flotation Results for 147 Zone and CZ

ID Test	Product	Wt%	Au (g/t)	TOC (%)	S (%)	Au Dist (%)	C Dist (%)	S Dist (%)
F-8 (147)	Carbon Pre-Float Conc	1.6	35.8	3.25	2.72	25.3	1.8	2.8
F-8 (147)	Bulk Ro Conc	8.2	18.1	1.96	17.4	66.3	5.8	93.6
F-8 (147)	Carbon + Ro Conc	9.8	21.0	2.17	15.0	91.5	7.6	96.4
F-8 (147)	Ro Tail	90.2	0.21	2.85	0.06	8.5	92.4	3.6
F-8 (147)	Head (Calculated)	100	2.24	2.78	2.78	100	100	100
F-8 (147)	Head (Direct)		2.08	2.76	1.57			
F-7 (CZ)	Carbon Pre-Float Conc	1.7	30.0	2.13	4.13	9.9	1.8	2.8
F-7 (CZ)	Bulk Ro Conc	12.5	33.7	0.87	18.0	83.5	5.6	92.3
F-7 (CZ)	Carbon + Ro Conc	14.2	33.3	1.02	16.4	93.4	7.5	95.7
F-7 (CZ)	Ro Tail	85.8	0.39	2.08	0.14	6.6	92.5	4.9
F-7 (CZ)	Head (Calculated)	100	5.05	1.93	1.93	100	100	100
F-7 (CZ)	Head (Direct)		4.36	1.91	2.59			



Table 13-7: Stage Recoveries after Carbon Pre-flotation Results for 147 and CZ Samples

	147 Zone	CZ
Gravity	28.0	17.2
Flotation	91.5	93.4
Combined	93.9	94.5

13.4.1.7 Cyanidation of Gravity Tailings

Bottle-roll cyanidation tests were conducted on gravity tailings at 40% (w/w) solids, pH 10.5–11.0 and 0.5 g/L NaCN for 48 h, following a 1-h pre-aeration step. Results are shown in Table 13-8.

Table 13-8: Bottle Roll Cyanidation Results for 147 and CZ MC Samples

Composite	Head Feed Au (g/t)	Cyanidation Stage Au Extraction (%)	Overall Au Recov. (%)	Residue Au (g/t)	NaCN Cons. (kg/t)	CaO Cons. (kg/t)
147 MC	2.36	75	83	0.58	0.51	0.97
CZ MC	4.22	69	76	1.29	0.55	0.88

Combined gravity + cyanidation extractions yielded overall gold recoveries of 83% (147 MC) and 76% (CZ MC).

13.4.1.8 Detoxification

Leach tails from CN-16 (147 MC) and CN-17 (CZ MC) were subjected to an SO₂/Air–Cu-catalyzed detox process consisting of one (1) batch and three (3) continuous trials for each composite. The retention time for the continuous trials were approximately 60 minutes. Results are summarized as follows:

- **147 Zone MC:** Weak-acid-dissociable cyanide (CN_{WAD}) dropped from 169 mg L⁻¹ to ≤ 0.1 mg L⁻¹ in every continuous run. Sulphur dioxide consumption ranged from 4.75 g to 4.34 g SO₂ per gram of CN_{WAD}, with a constant Cu catalyst dose of 0.11 g g⁻¹ CN_{WAD}.
- **CZ MC:** CN_{WAD} was similarly reduced from 162 mg L⁻¹ to ≤ 0.1 mg L⁻¹ across all three continuous tests. Sulphur dioxide demand varied between 4.50 g and 3.98 g SO₂ per gram of CN_{WAD}. Copper addition started at 0.12 g per gram of CN_{WAD} (CND 2) and was lowered to 0.06 g per gram of CN_{WAD} for trials CND 2-1 and CND 2-2.

13.4.1.9 Sedimentation

Settling and rheology tests were performed on 10 kg lots of the 147 MC and CZ MC, each milled to P₈₀=83 µm. Flocculant optimization showed that a non-ionic polymer (Magnaflow 351) proved optimal at a dosage rate of 15 g/t. Static settling tests were assessed at varying feed rates, after which it was determined that a 10% pulp density produced the best clarity/underflow compromise for both composites. Refer to Table 13-9 for sedimentation test results.



Table 13-9: Dynamic Solid-Liquid Separation Tests Results

Test No.	Feed Solids (%)	Underflow Solids (%)	Unit Area (m ² /t/d)	Initial Rate (m/h)
5	5	62	0.084	0.94
6	10	68	0.078	0.56
7	15	67	0.119	0.20
8	20	66	0.232	0.06

Dynamic thickening tests showed that, for the 147 MC, 26 g/t flocculant resulted in 64.5% solids underflow density at 88 m³/m²/d net rise rate, 0.41 t/m³/h solids loading, and 3.6 m³/m²/h net hydraulic loading. For the CZ MC, at a 20 g/t flocculant dosage, a 65% underflow density was obtained, with net rise rate of 103 m³/m²/d, solids loading of 0.47 t/m³/h, and net hydraulic loading of 4.3 m³/m²/h.

Both thickener underflows exhibited a critical solids density of ~68%. At this density, the 147 Zone MC displayed yield stresses of 41 Pa (unsheared) and 27 Pa (post-shear), while the Contact-Zone MC measured 52 Pa and 30 Pa, respectively. JKTech Comminution Testing – “Final SMC Report for Grey Fox Project” (JKTech Job 12007/P53, 12 October 2012).

13.4.1.10 Sample Handling and Density Checks

SMC test data for three (3) Grey Fox composites (147 Zone MC, Breccia, and Tuff) were received by JKTech from SGS Minerals Services on October 12, 2012 and analyzed to generate JKSimMet comminution parameters. The SMC protocol, a drill-core variant of the JK Drop-Weight test, provides ore breakage characteristics (A, b, t_a, DWi) and derives specific-energy indices (M_{ia}, M_{ih}, M_{ic}) essential for preliminary comminution circuit design.

The core for all three (3) Grey Fox composites (147 Zone MC, Breccia, and Tuff) was quarter-cut, weighed in air and water, and its density established prior to breakage testing. Average specific gravities determined from the SMC Test were 2.91 for both the Breccia and 147 Zone MCs and 2.71 for the Tuff sample.

13.4.1.11 Drop-Weight Index Test Results

The Drop-Weight Index (Dwi) indicates a consistently hard ore. Axb values corroborate this, with the Breccia sample being the most resistant to impact breakage. Note that lower Axb values indicate a more competent sample and vice versa. The resultant test parameters and obtained indices are shown in Table 13-10.

Table 13-10: JKTech Results

Sample	DWi (kWh/m ³)	Axb	Category	t _a	M _{ia} (kWh/t)	M _{ih} (kWh/t)	M _{ic} (kWh/t)
147 Zone	9.01	32.6	Hard	0.29	22.9	18.0	9.3
Breccia	9.58	30.1	Hard	0.27	24.0	19.1	9.9



Sample	DWi (kWh/m ³)	Axb	Category	ta	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)
Tuff	7.82	35.3	Moderately Hard	0.33	21.7	16.6	8.6

DWi values plot between the 76th and 89th percentiles of the >11,000-sample SMC database, underscoring the ore's high competency, while Axb percentiles are in the 13% to 26% range, meaning 74% to 87% of the ores in the database are softer. Breccia is, as expected, the hardest sample tested. The t_{10} @1 kWh/t averages 26.9%, which is also in the harder quartile of JKTech data.

High DWi and low Axb dictate a power-intensive comminution circuit. JKTech correlations indicate that ore with these hardness characteristics typically requires:

- Elevated SAG mill specific energy or the inclusion of a pebble crusher is required to maintain throughput.
- Potential benefit from high-pressure grinding rolls (HPGR) ball-mill flowsheets where M_{ih} (~17 to 19 kWh/t) falls below M_{ia} , offering energy savings of ~10% to 15% relative to SAG mill–Pebble Crusher–Ball mill Circuit (SABC) designs. This is offset by a relatively modest throughput and low cost of electricity in Ontario which makes the HPGR option less attractive

13.4.2 Phase 1 Variability Assessment

The Phase 1 metallurgical testwork program was conducted by XPS in 2013 and documented in “Geomet Definition & Metallurgical Assessment of Grey Fox Zones, Phase 1” (Report 5013826.00, dated December 6, 2013) (XPS, 2013) represents the first systematic geometallurgical assessment of the Grey Fox deposit and was conducted to better understand ore variability and its implications for gold recovery across different geological and lithological domains.

The primary aim of the program was to assess how gold recovery performance varies with geological characteristics—such as lithology, sulphide content, alteration, and depth—by conducting metallurgical and mineralogical testing on a carefully selected suite of composite samples.

The Phase 1 metallurgical testwork program provides critical early-stage insights into ore variability and its implications for gold recovery across the deposit. The 2013 study forms the foundation for defining geometallurgical units and supports the development of a predictive framework for processing behavior, laying the groundwork for optimization in both mine design and processing strategy.

13.4.2.1 Variability Sample Matrix

Eight (8) variability composites were selected to benchmark the principal geometallurgical controls at Grey Fox. Selection criteria targeted lithology (147 Zone vs CZ), host unit, and depth (Upper vs Lower horizons) while maintaining a similar head-grade window (~3 g/t) to isolate textural and sulphide effects. The resulting matrix provides a representative first-pass dataset on which subsequent phases will be built (Table 13-11).



Table 13-11: Sample Compositing Matrix

Comp #	Zone	Host Lithology	Horizon	Target Au (g/t)	Au (g/t)
1	147 FW	Variolitic Volcanic (VIV)	Upper	2.97	4.31
2	147 FW	VIV	Lower	2.92	6.37
3	147 HW	Mafic Intrusive (MI)	Upper	3.08	2.20
4	147 HW	MI	Lower	2.90	3.15
5	CZ FW	Sedimentary (SED)	Upper	2.96	3.37
6	CZ FW	SED	Lower	2.99	5.70
7	CZ HW	Mafic Volcanic (MV)	Upper	2.96	2.36
8	CZ HW	MV	Lower	2.97	3.03

Note: HW = Hangingwall, FW = Foot Wall

13.4.2.2 Head Assay and Mineralogical Correlation

Head assays showed gold grades from 2.20 g/t to 6.37 g/t and sulphur grades from 0.93% to 2.92%. Elevated sulphur (proxy for pyrite) was most pronounced in CZ units and correlates negatively with leach recovery. Composite-scale QEMSCAN and backscatter electron imaging confirmed that CZ MV samples host finer, pyrite-locked gold grains, whereas 147 Zone samples contain coarser, partially liberated gold. Typical major-element chemistry for the eight (8) composites is summarized in Table 13-12.

Table 13-12: Head Assays of Composites

Comp #	Zone	Au (g/t)	S (%)	Fe (%)	Pyrite vol. (%) [†]
1	147 FW	4.31	1.18	10.49	2.8
2	147 FW	6.37	1.47	10.28	4.1
3	147 HW	2.20	0.93	9.79	1.6
4	147 HW	3.15	1.14	9.79	2.0
5	CZ FW	3.37	2.12	4.33	4.6
6	CZ FW	5.70	1.33	3.57	6.6
7	CZ HW	2.36	1.41	7.27	4.0
8	CZ HW	3.03	2.92	8.46	6.6

Note: [†] Pyrite volume derived from modal mineralogy tables (XPS Phase 1 Report Appendices 2).



13.4.2.3 Whole-Ore Cyanidations

Whole-ore cyanidation testing was conducted using a batch (bottle-roll) methodology on all eight variability composites. Test conditions were standardized across all samples to enable direct performance comparison:

- Charge size: 2 kg.
- Initial pH: 11.0 (adjusted with lime addition).
- NaCN dosage: 2,500 mg/L (maintained at equilibrium throughout test).
- Pulp density: 40% solids (w/w).
- Dissolved oxygen: 8–10 mg/L (maintained by air sparging).
- Sampling intervals: 2, 4, 6, 24, and 48 h.

Final gold recovery varied considerably across the composites, ranging from 70% to 93%, and correlated primarily with ore head grade and sulphur content.

Dissolved oxygen levels were maintained within the 8-10 ppm range through air sparging. Sodium cyanide consumption spanned 1.26 kg/t (MV Upper) to 2.45 kg/t (147 FW Lower), as shown in Table 13-13.

Table 13-13: Whole-Ore Cyanidation Results

Comp #	Zone	Au Recovery (%)	NaCN Consumed (g/t)	Average Au grain size (µm)
1	147 FW	84.8	1,859	5.6
2	147 FW	93.4	2,452	4.4
3	147 HW	86.2	1,383	3.2
4	147 HW	89.3	2,081	7.3
5	CZ FW	87.5	2,109	9.2
6	CZ FW	92.8	1,725	31.9
7	CZ HW	76.6	1,258	5.4
8	CZ HW	70.0	1,667	2.7

Kinetic analysis of leach solutions sampled at 2, 4, 6, 24, and 48 h revealed different dissolution kinetic profiles by Zone: 147 Zone composites leached fast, reaching 65% of their ultimate extraction within the first 2 h, whereas CZ ores lagged significantly, with averaged 2-h recoveries of ~41%. These kinetic differences have implications for leach tank sizing: CZ ore required longer residence times to achieve comparable recovery to 147 Zone material at the same grind size.

With the average recovery of 85% after 2 days of leaching, it is evident that the ore body contains significant portions that are mildly refractory.

Gold extraction is chiefly grade-driven with a secondary penalty from pyrite. 147 Zone composites averaged 88% recovery, whereas CZ composites averaged 81%, primarily due to the poor-performing MV subgroup.



Within zones, upper vs lower horizons did not materially affect recovery at this grind size. The data also confirms that higher sulphur grades (>2%) combined with lower gold grades (<4 g/t) generally yields a sub 80% recovery. The two worst performing variability samples (7 and 8) also exhibited below average gold grain sizes, again hinting at some encapsulation of gold within the pyrite matrix. The results suggest that higher grades occur when coarse free gold is present, while lower grades are associated with much finer gold, often locked in sulphides and hence not amenable to cyanidation.

13.4.3 Phase 2 Composite Program

The Phase 2 metallurgical testwork program was conducted by XPS and documented in “Geomet Definition & Metallurgical Assessment of Grey Fox Zones, Phase 2” (Report 5013826.00, dated February 10, 2014) (XPS 2014a). This Phase 2 campaign continued the systematic geometallurgical evaluation of the Grey Fox deposit initiated in Phase 1, with a specific focus on validating and refining metallurgical domains and assessing their predictive value for processing performance.

The results of XPS's Phase 1 testwork program demonstrated that the four laboratory composites comprising high- and low-grade material from the 147 Zone, and the two principal lithologies from the CZ (MV and SEDS units), are accurate, homogeneous, and representative of the current resource. The testwork further confirmed that these composites provide a suitable and reliable basis for predicting full-scale plant performance.

The main objectives of the Phase 2 program were:

- To confirm and validate the metallurgical behavior of key geometallurgical units through comparative leaching and flotation performance of the four selected composites.
- To assess the reproducibility and internal variability of leach recoveries and flotation responses, ensuring these units are suitable for metallurgical modelling and process planning.
- To refine and update Phase 1 conclusions using material from recent drilling, with particular emphasis on head-grade characterization, sulphur and carbon content, and ore hardness.
- To reaffirm the geometallurgical drivers for processing behavior, namely that Grey Fox remains a coarse-gold, pyrite-hosted deposit where metallurgical response is closely tied to sulphur and carbon variability. This included observations on the influence of carbonaceous content, pyrite deportment, and gold grain size on recovery.
- To explore the potential for flotation as a recovery route for high-sulphide, low-recovery leach material by quantifying gold recovery into sulphide-rich flotation concentrates.

In summary, the Phase 2 metallurgical testwork program provides enhanced confidence in the use of the four geometallurgical composites as representative processing units for Grey Fox. By validating composite consistency and updating metallurgical profiles using current drilling, the program confirms that recovery variability is chiefly controlled by sulphide and carbon content. These findings support the development of predictive recovery models and provide a basis for informing plant design decisions.

13.4.3.1 Composite Definition and Representativeness

Four (4) geometallurgical composites were prepared from drill core intervals considered representative of the Grey Fox orebody: 147 Zone High Grade (HG), 147 Zone Low Grade (LG), CZ–Massive Volcanic (MV)



and CZ–Siltstone/Epidote/Diopside Skarn (SEDS). Core was stage-crushed to –10 mesh, blended and rotary-split. Triplicate external reference distribution assays confirmed homogeneity with relative standard deviations $\leq 7\%$ for gold. The resulting head grades bracket the expected resource cut-off ranges and provide a robust basis for flowsheet selection (Table 13-14).

Table 13-14: Composites Definitions

Composite	Lithology / Zone	Au (g/t)
147 HG	Quartz feldspar porphyry	9.49
147 LG	Mafic volcanic	1.83
Contact MV	Massive volcanic	3.91
Contact SEDS	Bedded tuffaceous siltstone	2.96

13.4.3.2 Head Grade and Mineralogy

Head assay results (as shown in Table 13-15) confirm gold association with pyrite. Arsenic and total carbon remain modest (As $< 0.02\%$, TOC $< 3\%$) across all composites. Sulphur in Contact SEDS (3.1 %) is ~60% higher than Contact MV (1.9%), reversing Phase 1 trends and explaining the lower leach response for SEDS. QEMSCAN imagery showed ~90% of gold as $\leq 10\ \mu\text{m}$ inclusions in pyrite or at pyrite–silicate boundaries, supporting a sulphide flotation flowsheet.

Table 13-15: Composite Head Assays

Composite	Au (g/t)	As (%)	Fe (%)	S (%)	TOC (%)	Remarks
147 HG	9.49	0.013	10.12	1.86	2.12	Head grade higher than model; blended later
147 LG	1.83	0.006	10.06	1.03	1.62	Provides dilution for 6 g/t blend
Contact MV	3.91	0.008	7.81	1.95	3.09	Highest carbon; potential preg robbing
Contact SEDS	2.96	0.005	4.69	3.05	1.49	Most pyritic sample

13.4.3.3 Comminution

BBWi values placed Grey Fox ores among the very hardest gold deposits globally (19–26 kWh/t). These results supported a high-power conventional SAG/ball-mill circuit or, alternatively, a secondary crush / closed-circuit ball-mill configuration should SAG throughput prove limiting. Please refer to Table 13-16 for a summary of results.

Table 13-16: Composite Head Assays

Composite	BWi (kWh/t)	Percentile*	Comment
147 HG	25.9	99	Very hard – upper variability limit
147 LG	24.6	99	Comparable hardness



Composite	BWi (kWh/t)	Percentile*	Comment
Contact MV	22.1	97	Slightly softer, very hard
Contact SEDS	19.2	89	Hardness advantage

13.4.3.4 Gravity Recovery

Dedicated GRG testing was not repeated in Phase 2; however, rougher flotation concentrates showed gold call factors up to 120% (147 HG), indicating a persistent coarse-gold/nugget component. Phase 1 testwork suggested 20% to 30% of gold is amenable to gravity; similar performance is inferred here, justifying a gravity circuit ahead of leach or flotation.

13.4.3.5 Flotation

Single-stage rougher flotation was carried out on all four composites at a grind $P_{80} \approx 60 \mu\text{m}$ (consistent with the anticipated primary-mill product), 40 min residence in a 5 L Denver cell, and a reagent suite comprising 110 g/t PAX and 25 g/t MaxGold 900, supported by 40 g/t MIBC frother. Flotation was done using tap water at natural pH of 8. Under these conditions gold recoveries ranged from 92.9% to 96.8% for the 147 Zone composites and 94.3% to 95.7% for the two CZ composites; pyrite recovery exceeded 95% in most cases while mass pull to concentrate ranged between 8% and 12%. Please refer to Table 13-17 for summary of the flotation results.

Table 13-17: Initial Flotation Results

Composite	Au Rec. (%)	FeS ₂ Rec. (%)	SiO ₂ Rec. (%)	Au Call Factor (%)
147 Zone HG	96.8	96.0	7.8	120
147 Zone LG	92.9	95.0	7.6	125
Contact MV	95.7	95.0	7.8	104
Contact SEDS	94.3	95.5	11.7	85

Concentrate grades varied between 18 g/t and 105 g/t Au, with the higher values coinciding with the 147 Zone samples that contain a coarse gold component. The elevated "call factors" (>120%) calculated for 147 HG and LG indicate the presence of unassayed liberated gold remaining in the feed, whereas the lower call factor of 85% for Contact SEDS points to a finer gold-in-pyrite grain size. Overall, the data confirm that a single reagent recipe and 60 μm grind size can accommodate Grey Fox variability with minimal loss in performance, providing a sound baseline for the forthcoming locked-cycle and concentrate treatment tests.

13.4.3.6 Whole-Ore Cyanidation

Extended bottle-roll cyanidation tests (sampling at 2, 4, 6, 24 and 48 h) conducted at $P_{80} \approx 60 \mu\text{m}$, 2,500 mg/l NaCN dosage, dissolved oxygen at $>8 \text{ mg L}^{-1}$ and a target pH 11.0. Pulp Eh and residual CN^- were logged at 2, 8, 24 and 48 h intervals, but only the 24 h and 48 h gold extractions are reported by XPS and shown in Table 13-18.



Table 13-18: Extended Cyanidation Results

Sample	Head Au (g/t)	NaCN @24h (kg/t)	NaCN @48h (kg/t)	Au Recovered at 24h (%)	Au Recovered at 48h (%)	Comments
147 HG	10.3	3.1	4.6	94.2	93.6	Fast, diffusion limited after 8 h; no preg-robbing or thiocyanate build up
147 LG	2.42	1.7	2.4	82.2	83.2	Kinetics governed by finer Au; plateau reached only after ~36 h
147 Blend A	7.09	1.0	1.4	90.5	93.0	Confirms near linear grade vs recovery trend; CN efficiency improves at intermediate grade
147 Blend B	5.86	1.3	1.4	92.5	92.0	Replicate validates projection at 6 g/t design grade
Contact MV	3.87	2.1	3.2	88.0	85.7	Slight Au drop between 24 h & 48 h due to preg robbing by organic carbon (3% TOC)
Contact SEDS	3.07	1.6	2.2	70.8	70.5	High sulphide slows dissolution; pre oxidation or finer grind recommended

Based on the cyanidation results in Table 13-18, the 147 Zone behaved as a classic free-milling system, with extraction almost linearly proportional to head grade. Blending high- and low-grade material to a ~6 g/t Au feed grade yielded ~90% recovery after 24 h and exceeded 92% at 48 h. Cyanide consumption scaled with sulphide content but stabilised below 1 kg/t at the design grade, indicating that reagent efficiency improved when the excess sulphide surface area of low-grade ore was diluted with high-grade feed.

The CZ composited diverged markedly. Contact MV responds well initially (88% in 24 h) but showed a modest regression by 48 h, diagnosed by XPS as preg-robbing from graphitic carbon that became active once dissolved gold concentrations fell below ~2 mg/L. In contrast, Contact SEDS extraction was inhibited from the outset, with only 71% of gold dissolved after 48 h. This behavior was attributed to the higher pyrite (~3% S) competing for cyanide and passivation of gold surfaces. Pre-oxidation or finer grinding was therefore required for this domain.

Overall, these results confirmed that a whole-ore leach flowsheet will be feasible for 147 Zone ore, provided that head grades stay above ~5 g/t (refer to Figure 13-2). For CZ ores, a flotation-first route appears to be optimal. The blended tests clarified recovery at the reserve grade and will provide input into the recovery algorithms in the Grey Fox economic model.



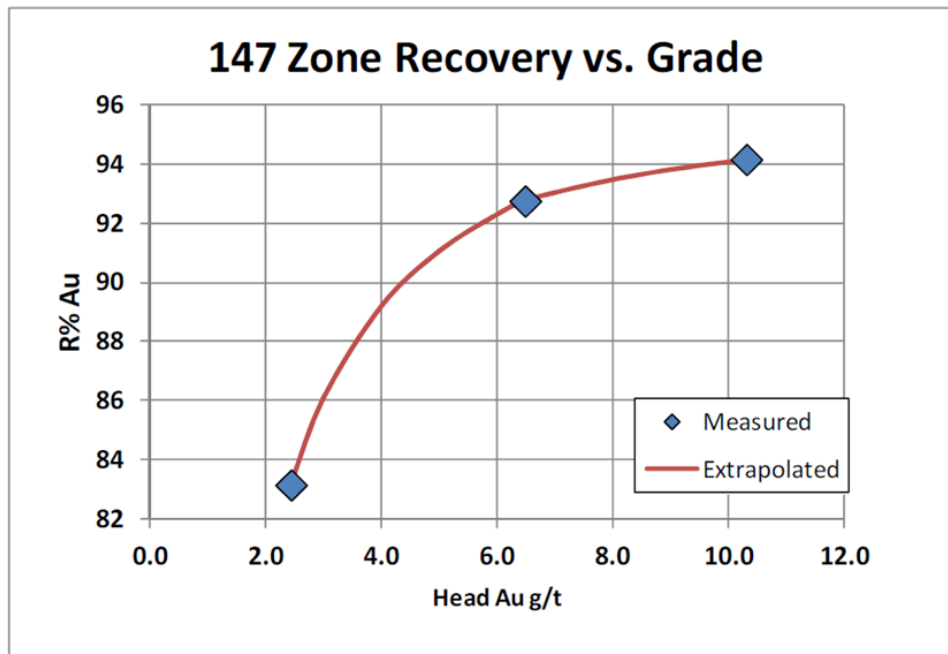


Figure 13-2: Head Grade vs Recovery for 147 Zone

13.4.3.7 Flotation Tailings Cyanidation

Although flotation tailings were not separately leached in Phase 2, the high (>95 %) pyrite recoveries imply low residual sulphide and a remaining unrecovered gold fraction of only 2%–3%, placing doubt on the efficacy and value of leaching flotation tails.

13.4.4 2014 Phase 3 Pyritic & South Study

The test program conducted in 2014 represents Phase 3 of metallurgical and geometallurgical investigation into the Grey Fox deposit by XPS for then-owner Brigus Gold. The findings of this testwork were documented in “Geometallurgical Assessment of Grey Fox Pyritic & Grey Fox South” (Report 5014812.00, dated November 10, 2014) (XPS 2014b). This phase was aimed at advancing the process design by exploring the recovery behavior of both high-sulphur and variable-grade ore types through alternative flowsheet strategies.

The 2014 report documents Phase 3 metallurgical and geometallurgical testwork carried out by XPS on two ore sources from the Grey Fox deposit: a high-sulphur “Contact Zone Pyritic” composite and a suite of variability samples from Grey Fox South. The program combined bulk sulphide flotation, bottle-roll cyanide leaching, QEMSCAN mineralogy, and hardness testing to define geometallurgical units, measure gold deportment in pyrite, and establish how head grade and sulphur content control recovery. Its purpose was to generate the data needed to compare whole-ore leaching against flowsheets that float, regrind, and separately leach sulphide concentrates and tailings, thereby guiding process design for both the pyritic and Grey Fox South ores.



This Phase 3 report extends the geometallurgical understanding of the Grey Fox deposit by targeting two challenging ore domains—high-sulphur pyritic and variable Grey Fox South ores. The comprehensive approach combining flotation, leaching, mineralogy, and hardness testing delivers the critical data needed to evaluate and compare processing strategies, optimize recovery, and support design decisions for future development phases.

13.4.4.1 Sample Classification

Two distinct groups of samples were selected from the Grey Fox deposit to represent the expected variability in ore properties and associated metallurgical performance. One composite specifically represented the high-sulphur, low-grade CZ Pyritic ore, identified as potentially refractory due to its elevated sulphur content. In addition, six variability samples were sourced from the Grey Fox South zone. These samples spanned a range of lithologies including Variolitic Volcanics (VIV), Mafic Volcanics (MV), Mafic Intrusive (MI), and Sedimentary (SEDS) rocks, each reflecting a spectrum of gold and sulphur contents.

These Grey Fox South variability samples were later consolidated into four clearly defined geometallurgical units, facilitating easier interpretation and prediction of metallurgical performance:

- High-grade (HG) ore with Au >4 g/t (Grey Fox South-HG).
- Low-grade (LG) ore with Au <4 g/t (Grey Fox South-LG).
- Low-grade, low-pyrite (LGC) ore defined by a S/Au ratio below 0.75 (Grey Fox South-LGC).
- Low-grade, high-pyrite (LGP) ore defined by a S/Au ratio above 0.75 (Grey Fox South-LGP).

In parallel with the metallurgical classification, a detailed mineralogical analysis using QEMSCAN was undertaken on all feed samples and specifically on the CZ Pyritic concentrate after flotation, to thoroughly characterize pyrite liberation, gold deportment, and mineral associations impacting cyanide leach recoveries.

13.4.4.2 Head Assays

Detailed head assays were conducted to characterize the ore samples prior to metallurgical testing. These assays highlighted the critical influence of sulphur and gold contents on downstream processing and recoveries. The CZ Pyritic composite specifically showed a relatively low gold grade of 1.92 g/t Au coupled with an elevated sulphur content of about 4.0%, resulting in a very high sulphur-to-gold ratio (S/Au $\approx$ 2.1), indicative of challenging recovery conditions.

The Grey Fox South geometallurgical units demonstrated a diverse set of head assay values, clearly delineating their metallurgical response, as shown in Table 13-19.

Table 13-19: Geomet Units Head Assay

Composite	Gold Grade (Au, g/t)	Sulphur (S,%)	S/Au Ratio
CZ Pyritic	1.92	4.0	2.1
Grey Fox South-HG	5.3	1.5	0.28
Grey Fox South-LG	0.95	1.4	1.5



Composite	Gold Grade (Au, g/t)	Sulphur (S,%)	S/Au Ratio
Grey Fox South-LGC	3.40	1.1	0.31
Grey Fox South-LGP	2.99	3.9	1.3

13.4.4.3 Comminution

Ore hardness is critical for process design and throughput forecasting. Historical BWi data indicated that ores from the Grey Fox deposit vary from moderately hard to extremely hard. The CZ ores exhibited moderate hardness, with a BWi range from 19 kWh/t to 22 kWh/t. In contrast, the 147 Zone and particularly the VIV-rich ores from Grey Fox South displayed significantly higher hardness values, typically ranging from 24 kWh/t to 27 kWh/t, placing them in the category of very hard ores (Table 13-20).

Table 13-20: BWi Results by Zone

Zone / Lithology	BWi (kWh/t)
Contact – SEDS-Tuff	19.2
CZ – MV	22.1
147 Zone – VIV + MI (LG)	24.6
147 Zone – VIV + MI (HG)	25.9
147 Zone – VIV + MV Blend	27.1

In the case of Grey Fox South, this zone is largely comprised of VIV hosted ore with marginal amounts of MV and MI hosted ore making up the balance. This indicates that the expected BWi hardness of the Grey Fox South-hosted ore may fall between 24 kWh/t and 27 kWh/t and be considered in the 98th to 99th percentile for hardness.

To evaluate liberation and recovery trade-offs, all metallurgical testwork in this phase used a consistent primary grind size of approximately 60 µm (P_{80}). In addition, pyrite concentrates from flotation were further reground to finer sizes (25 µm and 12 µm) to investigate the impact on gold liberation and the subsequent extraction efficiencies via cyanide leaching.

13.4.4.4 Whole-Ore Cyanidations

Whole-ore cyanidation tests involved duplicate 48-h bottle-roll leaches at a controlled pH of about 11 and cyanide concentration of approximately 2,500 ppm NaCN, with 50% solids loading. Both solutions and solids residues were assayed in duplicate, enhancing the reliability of the recovery estimates.

The CZ Pyritic composite achieved only moderate recovery (approximately 79%), indicating a significant proportion of gold being refractory and encapsulated in pyrite minerals. Grey Fox South geometallurgical composites yielded higher recovery rates reflecting the influence of gold grade and sulphur content, with HG achieving about 89%, LG around 73%, LGC around 75%, and LGP approximately 81% recovery, as shown in Table 13-21.



Table 13-21: Geomet Units Cyanidation Results

Composite	Gold Grade (g/t)	Sulphur Grade (%)	S/Au Ratio	Averaged Au Recovery (%)
CZ Pyritic	1.92	4.0	2.1	79.4
Grey Fox South-HG	5.3	1.5	0.28	89.3
Grey Fox South-LG	0.95	1.4	1.5	73.2
Grey Fox South-LGC	3.40	1.1	0.31	75.5
Grey Fox South-LGP	2.99	3.9	1.3	81.3

13.4.4.5 Rougher Flotation

Rougher flotation tests were performed on 2-kg samples, at a grind size of P_{80} of about 60 μm to evaluate the recovery of gold-bearing sulphides into a concentrate. The focus of the flotation tests was to determine the effect of pyrite-bearing ore; hence, only the CZ Pyritic composite and Grey Fox South-LGP composites were tested. The CZ Pyritic composite exhibited excellent flotation response, recovering approximately 93% of the gold and 97% of the sulphur into only 14% of the total mass. Similarly, the Grey Fox South-LGP composite showed comparable flotation efficiency, achieving about 95% gold recovery into a concentrate comprising approximately 15% of the total feed mass (Table 13-22).

Table 13-22: Geomet Units Flotation Results

Composite	Mass Pull (%)	Au Recovery (%)	Sulphur Recovery (%)
CZ Pyritic	14.0	93%	97%
Grey Fox South-LGP	14.6	95%	Not reported

13.4.4.6 Flotation Tailings Cyanidation

Flotation tails were cyanide-leached without further regrinding. The tailings from the CZ Pyritic composite were successfully leached to a residual gold content of approximately 0.05 g/t Au. From the initial flotation tail feed (0.18 g/t Au), this represented around 74% recovery. Similarly, for the LGP composite, a gold recovery of 74.3% was achieved (Table 13-23), indicating minimal refractory gold in non-sulphide gangue minerals. Cyanide consumption was approximately 1.2 kg/t but maybe have been inflated due to the use of a very high cyanide concentration during the test. It is expected that cyanide consumption for tailings leaching would be <1 kg/t due to the removal of cyanide-consuming components from the tails.

Table 13-23: Flotation Tailings Cyanidation Results

Sample Type	Flotation Tail Grade Au (g/t)	Leach Tails Residual Au (g/t)	Au Recovery from Flotation Tails (%)
CZ Pyritic composite	0.18	0.05	73.6%
Grey Fox South-LGP	0.16	0.04	74.3%



13.4.4.7 Flotation Concentrate Cyanidation

Leaching tests on pyrite concentrates indicated limited recovery gains from intensive regrinding. At the as-floated grind size (~60 µm), the CZ Pyritic concentrate yielded about 80% gold recovery. Regrinding the concentrate to a P₈₀ of 25 µm improved recovery only slightly to 83%, while finer grinding to 12 µm provided no further recovery benefit, indicating diminishing returns at finer sizes due to pyrite-locked gold inclusions. The Grey Fox South-LGP composite reflected similar recovery compared to the CZ Pyritic composite (Table 13-24). Cyanide consumption was approximately 5 kg/t of leach feed but again from a very high cyanide concentration in solution.

Table 13-24: Flotation Concentrate Cyanidation Results

Sample Type	Conc Grade Au (g/t)	Residual Au (g/t)	Au Recovery from Conc (%)
CZ Pyritic Composite Concentrate @ 60 µm	12.6	2.59	80.2%
CZ Pyritic Composite Concentrate @ 22 µm	12.6	2.11	83.5%
CZ Pyritic Composite Concentrate @ 12 µm	12.6	2.09	82.9%
Grey Fox South-LGP	17.06	3.26	80.9%

13.4.5 2015 Phase 4 Leaching-with-Flotation Investigation

The Phase 4 testwork program was completed by XPS and documented in the report titled “Brigus Grey Fox Phase IV: Leaching with Flotation” (Report 4015803.00, dated June 9, 2015) (XPS 2015). This program was designed to evaluate processing optimization opportunities for Grey Fox South ore: the southernmost HG zone of the deposit. Specifically, Phase 4 targeted optimization opportunities around grind size, flotation, and leaching strategies.

The primary goals of this Phase 4 testwork program were:

- To assess the response of Grey Fox South ore to flotation and leaching at grind sizes coarser than the baseline P₈₀ of 60 µm, with test grind sizes up to 120 µm. This was driven by previous observations of high energy consumption due to ore hardness.
- To evaluate whether flotation could effectively concentrate gold into a low-mass sulphide product, and if subsequent leaching of this concentrate and its tailings could maintain gold recovery while allowing for coarser primary grinding.
- To investigate whether a split leach approach (leaching of both flotation concentrate and tailings separately) offered any metallurgical advantages over conventional whole-ore leaching.

To explore the operational flexibility provided by decoupling recovery from grind size, and assess if energy savings or increased throughput could offset minor losses in gold recovery, especially given the ore's high hardness characteristics.



13.4.5.1 Sample Selection

A composite sample was prepared from mineralized drill core collected from both the 147 Zone and the CZ at Grey Fox South. The final composite was made from 37 core intervals comprising various lithologies (SED, VIV, MI/MV, FZ) and weighed approximately 40 kg. The expected head grade was 3.0 g/t Au. The assayed composite showed a slightly higher gold grade due to a known nugget effect. Sulphur levels across lithologies ranged from 0.4% to 2.9%. Refer to Table 13-25 for details pertaining to the samples used in this phase of testing.

Table 13-25: XPS Phase 4 Sample Selection

Lithology (Zone)	Core Intervals (m)	Avg. S (%)	Mass Weight (%)	Au Grade (g/t)
SED (CZ)	8	1.2	8.8	3.12
VIV (CZ)	6	0.4	6.6	3.27
MI/MV (CZ)	5	2.9	5.6	3.03
FZ (CZ)	4	2.8	4.4	3.40
Subtotal Contact	23	1.8	25.4	3.19
VIV (147)	6	1.0	6.6	3.09
MV (147)	6	1.7	6.6	2.54
MI (147)	1	1.1	0.5	2.07
SED (147)	1	0.5	1.1	2.62
Subtotal (147)	14	1.1	14.8	2.78
Full Composite	37	1.43	40.2	3.00

13.4.5.2 Flotation

Flotation tests were conducted on the composite sample using 2 x 2.0 kg charges. Reagents used included 110 g/t PAX and 25 g/t Aero Maxgold 900 promoter, with MIBC frother. Flotation was performed at natural pH over 40 minutes. The aim was to recover gold into a low-mass sulphide concentrate for further treatment. Rougher flotation was performed on multiple grind sizes (P_{80} of 60 μm , 90 μm , and 120 μm), and consistently achieved a gold recovery of ~94% with a mass pull of ~14%, indicating flotation performance is insensitive to grind over the range tested. Sulphur recovery followed the same trend, as shown in Table 13-26.

Table 13-26: XPS Phase 4 Flotation Results

Primary Grind P_{80} (μm)	Mass Pull to Con. (%)	Au Recovery to Con. (%)	S Recovery to Con. (%)
60	13.8	93.6	96.7
90	12.7	94.0	95.7
120	14.5	93.9	95.7



13.4.5.3 Whole-Ore Cyanidation

Cyanidation of whole-ore samples was performed at 2,500 ppm CN dosage. This level was maintained throughout the leaching, which included solution collection and analysis after 6, 24 and 48 h of leaching. Leaching performance was evaluated at different grind sizes. The recalculated head grades differed considerably between the tests due to variations in the amount of gold found in solution; however, the duplicate tailings assays showed a clear trend toward higher gold losses as a function of increasing size consistent with plant experience.

As shown in Table 13-27, the tailings gold grade increased from 0.85 g/t Au at 60 µm to 0.97 g/t Au at 120 µm, or 0.12 g/t increase over a 60 µm increase in the P₈₀. This is equivalent to 0.02 g/t Au, or ~0.6% recovery, per 10 µm change in the P₈₀.

Table 13-27: Grind-Cyanidation Results

Primary Grind P ₈₀ (µm)	Head Grade (g/t)	Solution Grade (g/t)	Tails Grade (g/t)	Call Factor	Calculated Au Recovery (%)
60	3.56	2.70	0.85	100	76.0
90	3.56	5.89	0.91	191	74.2
120	3.56	3.60	0.97	128	72.4

It is important to note that the nugget effect became more pronounced with coarser grind sizes, making it difficult to draw definitive conclusions regarding the grind-recovery relationship.

13.4.5.4 Flotation Tailings Cyanidation

The flotation tailings, defined as non-sulphide gangue (NSG), were subjected to cyanidation at varying primary grind sizes. Leaching of NSG tails showed that gold recovery was less sensitive to grind size compared to whole-ore leaching. The indicated leach recovery was 73.9% at 60 µm, declining to 63% at 90 µm and 57% at 120 µm (Table 13-28). Expressed as a tailings grade, the final leach tailings rose from 0.10 g/t Au at 60 µm to 0.15 g/t Au at 120 µm, or an 0.05 g/t Au increase over a 60 µm increase in the P₈₀. This is equivalent to about 0.08 g/t Au, or approximately 0.25% recovery, per 10 µm change in the P₈₀. This indicates the presence of fine free gold for which recovery is independent of liberation size.

Table 13-28: Grind vs Flotation Recovery Results

Primary Grind P ₈₀ (µm)	Flotation Tails Grade (g/t)	Solution Grade (g/t)	Tails Grade (g/t)	Call Factor	Assayed Au Recovery (%)
60	0.30	0.28	0.10	128	73.9
90	0.36	0.22	0.13	97	62.9
120	0.31	0.19	0.15	108	56.7



13.4.5.5 Flotation Concentrate Cyanidation

The flotation concentrate (~14% of feed mass) was reground to a P_{80} of ~35 μm in all cases before cyanidation. In Phase 3 testwork, further regrinding to finer sizes (P_{80} 25 μm and 15 μm) showed only minor improvements in recovery for finer grinds, and therefore was not repeated. In the Phase 4 program, the focus was on confirming these results under different primary grind sizes. No significant change in flotation concentrate leach recovery was observed with changes in the original grind size of the ore, as indicated in Table 13-29.

Table 13-29: Grind vs Flotation Concentrate Cyanidation Results

Primary Grind P_{80} P_{80} (μm)	Flotation Conc Grade (g/t)	Leach Solution Grade (mg/L)	Tails Grade (g/t)	Assayed Au Recovery (%)
60	31.3	25.6	5.78	81.9
90	38.7	32.1	5.91	84.5
120	25.3	20.1	5.14	79.7

13.5 Recent Metallurgical Testing Results

13.5.1 2020 ALS Metallurgy Program KM6034

This program comprised metallurgical and environmental testwork conducted on mineralized material from the 147NE Zone of the Grey Fox deposit as part of McEwen's ongoing evaluation and process development. The program was carried out by ALS Metallurgy in 2019 and 2020 and was designed to support process design through assessment of gold recovery potential via cyanidation, while also evaluating environmental considerations associated with leach residues. Findings from this testing program were documented in the report titled "Metallurgical & Environmental Testwork for the Grey Fox Deposit," dated February 11, 2020 (ALS, 2020).

This program used six individual drill core composites and a MC from the 147NE Zone. The testing included bottle-roll cyanidation leach tests, grind size sensitivity analysis, head assay determination, environmental characterization of residues, and deportment analysis of refractory gold. The goal was to determine the cyanide leach amenability of the samples, understand the influence of grind size and reagent consumption on recovery, and gather supporting environmental data for residue management.

This testwork program provided critical information on the cyanidation leach performance of Grey Fox 147NE material, showing how grind size and composite variability influence recovery, reagent consumption, and leach kinetics. It also delivered environmental characterization data to support future permitting and waste management planning, and improved understanding of refractory gold deportment within the deposit. The results will guide process design decisions for gold recovery and support metallurgical modelling efforts for the Grey Fox project.



13.5.1.1 Master and Variability Samples Compositing Strategy

ALS Metallurgy prepared a robust and methodically constructed composite sampling strategy to characterize the metallurgical behavior of ore samples from the 147NE Zone of the Grey Fox deposit. Samples were received by ALS on two separate dates (November 7 and 22, 2019), comprising 28 individual drill-core intervals of approximately 4 kg each.

From these individual samples, six variability composites (Composite 1 through Composite 6A) were carefully assembled, ensuring representative blending from specified drill-core intervals. Each variability composite aimed to capture unique metallurgical characteristics expected within the 147NE Zone.

This assembly approach enabled an accurate representation of spatial and geological variability, ensuring comprehensive coverage of anticipated ore variability. Individual variability composites ranged from approximately 4.7 kg (Composite 5) to 30.0 kg (Composite 2), allowing for sufficient mass to perform standardized cyanidation tests. MC 1 (totaling 40 kg), blended from equal proportions (8 kg) of Composites 1, 2, 3, 4, and 6A, provided a larger representative bulk sample for detailed grind sensitivity, comminution characterization, environmental testing, and gold deportment studies.

13.5.1.2 Head Assays

Head assays conducted on each of the composites (Table 13-30) confirmed gold grades ranging from approximately 2.60 g/t to 8.09 g/t Au, with notable variations observed within duplicate assays, suggesting nuggety gold occurrences. Composite 2 yielded the highest average gold grade (8.09 g/t Au), whereas Composite 1 was the lowest grade sample (2.60 g/t Au). Significant assay variance in Composites 3, 4, and 6A suggested the presence of coarse gold particles, emphasizing the recommendation by ALS to employ screened metallic assays in future to enhance assay reliability and mitigate variability risks in head-grade determination.

Table 13-30: ALS Composite Sample Head Assays

Composite	Duplicate 1 Au (g/t)	Duplicate 2 Au (g/t)	Average Au (g/t)
1	2.72	2.48	2.60
2	7.99	8.20	8.09
3	8.74	5.01	6.87
4	2.34	3.59	2.96
5	3.86	3.74	3.80
6A	5.99	4.18	5.09
Master 1	4.49	5.15	4.82



13.5.1.3 Comminution

ALS performed rod-mill grind-calibration tests on every variability composite and on MC 1. Each test used a 2 kg charge, 1 L water and 20 kg mild-steel rods (Mill M2). The mill was stopped at timed intervals and the product screened through ASTM screens to generate full particle-size distributions. While these results are not used for sizing full-scale grinding mills, they do provide indicative information regarding differences in hardness.

The grind-calibration tests reveal a clear range in relative hardness: Composites 2 and MC 1 are the most resistant, needing more than 30 minutes of rod-milling to fall below a 100 µm P₈₀, whereas Composite 5 is the softest, achieving that size in only 22 minutes. This 20%–25% spread in grind time implies proportionally higher specific energy requirements for the harder ore samples—consistent with the very hard BWi of ~25 kWh/t recorded for MC 1. Nonetheless, every composite reached the design grind of ~100 µm within a practical 22–43 minute window. MC 1's grind-sensitivity profile (157 µm→98 µm→66 µm) supplied the data underpinning subsequent cyanidation optimization. Together, these calibrated grind curves and work-index measurements provide a robust basis for sizing the primary grinding circuit and forecasting power demand across the deposit's varying ore domains.

13.5.1.4 Whole-Ore Cyanidation

Cyanidation leach tests performed at various grind sizes (66 µm, 98 µm, and 157 µm P₈₀) on MC 1 at a pulp density of 33% solids, pH 11 (maintained with lime), and a cyanide concentration of 600 ppm showed robust gold extraction performance between 66 µm (90.2% Au recovery) and 98 µm (91.0% Au recovery), with only a minor decrease observed at the coarser grind size of 157 µm (87.0% Au recovery).

Refer to Table 13-31 for results. Gold leach kinetics were rapid, achieving near-complete extraction within the first 6 h.

Table 13-31: ALS Grind-Whole Ore Cyanidation Results

Test	P ₈₀ (µm)	Au Extraction (%)	Tail Au (g/t)	NaCN (kg/t)	Lime (kg/t)	Calc. Head Au (g/t)
1	66	90.2	0.56	0.56	0.9	5.70
2	98	91.0	0.57	0.57	0.8	6.33
4	157	87.0	0.84	0.84	0.5	6.48

These findings indicated limited sensitivity to grinding finer than ~100 µm, providing a metallurgical basis for a design decision at or near 100 µm as a target grind size.

As shown in Table 13-32, leach testing of the variability composites at a standardized grind size (P₈₀ = 100 µm) showed consistently high gold extractions ranging from approximately 85.1% (Composite 3) to 95.8% (Composite 1). Composites with higher gold grades and higher sulphide content (e.g., Composite 2 and Composite 3) tended toward somewhat lower extraction efficiencies and higher cyanide and lime consumptions. This underscores the importance of monitoring sulphide-gold associations to optimize recovery and manage reagent use.



Table 13-32: ALS Variability Cyanidation Results

Composite	Au Extraction (%)	Tail Au (g/t)	NaCN (kg/t)	Lime (kg/t)	Head Au (g/t)
1	95.8	0.21	0.2	0.5	4.94
2	90.8	0.96	1.1	1.1	10.41
3	85.1	1.05	1.0	1.2	7.05
4	93.0	0.22	0.4	0.5	3.15
5	91.1	0.42	0.4	0.9	4.74
6A	91.3	0.65	1.1	1.1	7.49

13.5.2 2025 ALS Metallurgy Test Work Program KM7663

13.5.2.1 Sample Description and Test Work Objectives

This testwork program, completed in October 2025, focused on samples from the underground zones Whiskey Jack and Gibson. These two zones constitute the bulk of the underground mine resource, whereas most of the previous testing was conducted using samples from the open pit mining zones. The primary objective of the program was to assess the amenability of these underground zones to the flowsheet as proposed for the open pit zones. A secondary objective was to adjust Process design criteria, if necessary, to include this part of the overall orebody.

Eight (8) drill core samples were provided to ALS Metallurgy for testing:

- Samples 1, 2 and 3 from Gibson South.
- Samples 4, 5 and 6 from Gibson North.
- Samples 7 and 8 from Whiskey Jack.

The testwork program consisted of head assays, mineralogy, comminution testing, flotation testing and leaching of flotation products.

13.5.2.2 Head Assays

Head assays conducted on each of the underground samples (Table 13-33) show gold grades ranging from approximately 2.05 g/t to 6.28 g/t with an average of 3.8 g/t. The average for this set of samples is well above the life-of-mine (LOM) average which is expected to be closer to 2.5 g/t. This implies that average recovery results may be somewhat elevated.

Table 13-33: Underground Samples Head Assays

Sample ID	Fe (%)	S(t) (%)	TOC (%)	Au (g/t)
Sample 1	2.35	0.50	0.06	2.98
Sample 2	3.81	0.20	0.07	4.25
Sample 3	6.75	0.84	0.05	2.05



Sample ID	Fe (%)	S(t) (%)	TOC (%)	Au (g/t)
Sample 4	2.71	1.04	0.07	2.12
Sample 5	2.49	0.41	0.05	3.88
Sample 6	2.74	0.63	0.04	6.01
Sample 7	6.25	0.79	0.06	6.28
Sample 8	6.55	0.45	0.06	2.72

Gold assays were performed using the screen metallics method, whereby a representative sub-sample was screened at 75 µm and the oversize fraction, which may contain coarse particulate gold, was digested to extinction and the solution subsequently analyzed. The more voluminous -75 µm fraction was split into smaller fractions, two of which were analyzed for gold using conventional fire assay. The results of the screen metallics procedure are shown in Table 13-34.

For the three higher-grade samples (2, 6 and 7), the increase in average grade appears to be due to an increase in coarse gold content in the +75 µm size fraction. Comparing the +75 µm and -75 µm split grades indicates that grades are similar when the average grade is below 4 g/t Au; however, for the three higher-grade samples, the grade of the +75 µm fraction is noticeably higher than that of the -75 µm fraction. This assessment is supported by the mass and gold distribution splits, which are similar for the lower-grade samples, but for the three higher-grade samples show gold distributions that are much higher than the corresponding mass split to the +75 µm oversize.

Table 13-34: Screen Metallics Gold Assay

Sample ID	Average Au (g/t)	Size	Au grade (g/t)	Mass%	Gold Distribution (%)
Sample 1	2.98	>75 µm	2.14	8.0	5.7
		<75 µm	3.05	92.0	94.3
Sample 2	4.24	>75 µm	5.36	7.0	8.9
		<75 µm	4.16	93.0	91.1
Sample 3	2.05	>75 µm	2.17	6.8	7.2
		<75 µm	2.04	93.2	92.8
Sample 4	2.12	>75 µm	2.07	4.9	4.8
		<75 µm	2.12	95.1	95.2
Sample 5	3.88	>75 µm	2.78	5.7	4.1
		<75 µm	3.95	94.3	95.9
Sample 6	6.01	>75 µm	8.36	5.8	8.1
		<75 µm	5.87	94.2	91.9
Sample 7	6.28	>75 µm	9.04	6.0	8.7
		<75 µm	6.10	94.0	91.3
Sample 8	2.72	>75 µm	1.65	7.0	4.3
		<75 µm	2.80	93.0	95.7



The implication of this difference is that it supports the inclusion of a gravity concentration step in the flowsheet to remove coarse gold as early as possible. This would result in less variable gold grade feed to the downstream processes and would also increase the accuracy of metallurgical accounting.

13.5.2.3 Mineralogy

Table 13-35 presents the mineral composition of the samples as determined by QEMSCAN.

Table 13-35: Minerals Composition of the Underground Samples

Minerals	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Sample 7	Sample 8
Chalcopyrite	0.1	0.1	<0.1	1.3	<0.1	<0.1	<0.1	<0.1
Bornite	0.0	<0.1	0.0	<0.1	<0.1	0.0	0.0	0.0
Chalcocite	0.0	0.0	0.0	0.0	0.0	<0.1	0.0	0.0
Covellite	0.0	0.0	0.0	0.0	0.0	<0.1	0.0	0.0
Tetrahedrite	<0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cuprite/Native Cu	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Other Sulphides	<0.1	<0.1	0.0	0.0	<0.1	<0.1	0.0	0.0
Pyrite	0.8	0.3	1.7	1.1	0.7	1.0	1.5	0.8
Iron Oxides	0.5	2.1	0.3	0.3	0.2	0.1	0.1	0.3
Quartz	8.1	7.6	1.0	4.8	35.8	34.1	33.4	37.9
Feldspars	56.4	53.2	57.9	66.3	29.9	23.5	2.3	7.7
Chlorite	2.9	7.9	30.1	7.9	6.5	3.7	17.0	9.6
Muscovite	10.3	6.5	4.6	9.7	14.7	15.2	4.5	8.0
Biotite/Phlogopite	1.4	3.0	1.7	2.1	1.5	1.7	3.2	2.2
Titanium Minerals	0.6	1.0	0.6	0.4	0.6	0.5	1.2	2.2
Apatite	0.8	1.3	0.7	0.5	0.3	0.3	0.1	0.2
Calcium Carbonates	17.5	16.2	0.8	5.0	9.0	19.2	34.6	29.9
Kaolinite	0.2	0.2	0.1	0.2	0.2	0.3	0.2	0.2
Others	0.5	0.7	0.5	0.4	0.6	0.4	1.8	1.0

Feldspars, quartz and calcium carbonates are the most abundant minerals and constitute about 75% of the total mass. Chlorite and muscovite are also present in significant quantities. The low composition of kaolinite (0.2%) and absence of other clay minerals indicates that the ore should not pose significant handling challenges. Sulphide minerals contribute only 1.5% to the total mass.

Table 13-36 presents the breakdown of sulphide minerals in these samples.



Table 13-36: Sulphide Minerals Proportions in the Underground Samples

Sulphide Minerals	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Sample 7	Sample 8
Copper Sulphides	6.6	11.5	0.9	42.4	3.1	1.0	1.4	1.5
Pyrite	92.2	87.6	99.1	57.5	96.7	98.8	98.5	98.3
Molybdenite	0.1	0.2	0.0	0.0	<0.1	0.0	0.0	0.0
Galena	<0.1	0.0	0.0	0.0	0.0	0.1	<0.1	0.0
Sphalerite	0.0	0.0	0.0	0.0	0.0	<0.1	0.0	0.0
Other Sulphides	1.1	0.6	<0.1	0.1	0.2	<0.1	0.1	0.1

Pyrite is by far the most abundant sulphide, contributing more than 90% to the total sulphide mass in most of the samples. Minor quantities of copper sulphides are also present. Sample 4 is distinct in that it contains a significant quantity of chalcopyrite (42.4% of the total sulphides). For this sample, elevated cyanide consumption could be expected, and it may also cause adsorption problems because dissolved copper can compete with gold for adsorption sites on activated carbon. When treating this part of the orebody, it may be prudent to blend this chalcopyrite-rich feed with other run-of-mine (ROM) feed. Carbon stripping frequency would also need to be increased to accommodate the negative effects of this mineral.

13.5.2.4 Comminution

Eight (8) samples were subjected to BWi testing and results are presented in Table 13-37.

Table 13-37: BWi Results

Sample ID	F₈₀ (µm)	P₈₀ (µm)	Grams/Revolution	BWi (kWh/t)
Sample 1	2308	80	1.05	17.7
Sample 2	2418	77	0.89	19.7
Sample 3	2328	78	0.97	18.6
Sample 4	2345	81	1.01	18.3
Sample 5	2483	76	0.82	20.8
Sample 6	2427	75	0.85	20.1
Sample 7	2349	76	0.97	18.2
Sample 8	2343	75	0.83	20.7
Average				19.3

The closing screen size for all of the BWi tests was 106 µm.

The underground mine samples, with an average BWi of 19.3 kWh/t, are not as hard as the open pit ore samples tested in previous campaigns, which yielded BWi results around 24 kWh/t. These results display a moderate degree of variability which implies that the grinding circuits will not be subjected to excessive



peaks and troughs with respect to ore feed hardness and should therefore be relatively easy to control and operate.

13.5.2.5 Whole-Ore Cyanidation

The eight samples were subjected to whole-ore cyanidations after grinding to a P_{80} of approximately 80 μm . Results are presented in Table 13-38. Test conditions were as follows:

- Pulp density = 45%w/w solids.
- Total test duration = 48 h.
- NaCN concentration = 471 g/m³ (maintained by manual addition of NaCN).
- pH = 11.0 (maintained manually).
- Oxygen sparged in headspace of bottle roll.

Table 13-38: Whole-Ore Cyanidation Results

Sample ID	Au Extraction (%)	NaCN consumed (kg/t)	Ca(OH) ₂ consumed (kg/t)	Residence time required (h)	Calc Head (g/t)	Measured Head (g/t)	Tails Residue (g/t)
Sample 1	90.9	0.3	2.4	48	3.64	2.98	0.33
Sample 2	93.3	0.2	3.4	40	5.34	4.25	0.36
Sample 3	78.6	0.2	2.9	30	2.82	2.05	0.61
Sample 4	71.1	1.2	3.4	48	2.04	2.12	0.59
Sample 5	70.8	0.5	5.9	30	4.05	3.88	1.19
Sample 6	64.6	0.4	2.7	40	6.58	6.01	2.33
Sample 7	93.1	0.4	2.4	24	6.69	6.28	0.47
Sample 8	91.8	0.5	1.7	28	3.41	2.72	0.28
Average	81.8	0.46	3.10	36	4.32	3.79	0.77

Whole-ore cyanidation results were similar to that of the open pit samples tested previously with a wide range of gold recoveries ranging from as low as 64.4% to 93.3%. Cyanide consumption rates were reduced compared to the open pit testing with the notable exception of Sample 4, where the abundance of cyanide-soluble copper resulted in a 1.2 kg/t cyanide consumption rate. The samples leached relatively slowly with many still leaching after 40 h of cyanidation.

13.5.2.6 Gravity Concentration

The eight samples were subjected to gravity separation tests after grinding to a P_{80} of approximately 80 μm . Two sets of gravity separation tests were conducted. In the first set, 2-kg sub-samples were used to measure gravity recoveries and grades. In the second set, 10-kg bulk gravity separation tests were conducted primarily to generate gravity tailings for downstream flotation and cyanidation testing. In both cases, a centrifugal concentrator was used as the first stage, followed by panning of the primary



concentrate to produce an upgraded final concentrate. The two-stage approach is generally considered to produce results that best mimic full-scale operations.

Results are summarized in Table 13-39 and Table 13-40.

Table 13-39: Gravity Concentration Results (2 kg tests)

Sample ID	Mass Pull (%)	Grade (g/t)	Au Recovery (%)	Calc Head (g/t)	Measured Head (g/t)
Sample 1	0.6	122	19.2	3.54	2.98
Sample 2	0.9	141	26.5	4.71	4.25
Sample 3	0.5	58.2	12.6	2.45	2.05
Sample 4	0.6	53	15.9	2.06	2.12
Sample 5	0.5	89	11.6	3.88	3.88
Sample 6	0.6	76	7.3	6.25	6.01
Sample 7	0.8	115	15.3	6.27	6.28
Sample 8	0.7	83.1	21.3	2.72	2.72
Average	0.65	92.2	16.2	3.99	3.79

Table 13-40: Gravity Concentration Results (10 kg bulk tests)

Sample ID	Mass Pull (%)	Grade (g/t)	Au Recovery (%)	Calc Head (g/t)	Measured Head (g/t)
Sample 1	0.16	163	11.0	2.32	2.98
Sample 2	0.19	208	14.8	2.64	4.25
Sample 3	0.19	614	30.0	3.85	2.05
Sample 4	0.18	86.2	8.7	1.76	2.12
Sample 5	0.18	256	12.2	3.74	3.88
Sample 6	0.18	220	6.9	5.78	6.01
Sample 7	0.22	115	4.2	5.90	6.28
Sample 8	0.20	83.1	6.8	2.43	2.72
Average	0.19	218.2	11.8	3.55	3.79

Comparing the results for the 2-kg and 10-kg charges, the mass pull to concentrate was lower for the bulk test, which is attributed to differences in equipment operation. The lower mass pull is more representative of what would be expected in a full-scale plant. As a result of the lower mass pull, the resulting concentrate exhibited, on average, higher grades and a decrease in gold recovery to the gravity concentrate.

No meaningful correlations were observed between head grade and either recovery or gravity concentrate grade. The occurrence of GRG appears to be random and unrelated to grade variations. For these underground zones, gravity recoveries are noticeably lower than those recorded during earlier test programs for open pit samples. The average gravity recovery for the underground zones was approximately



12%, compared to 25% recorded for the open pit zones. At these lower levels, a gravity recovery unit operation would likely not be included in the flowsheet for the full-scale plant.

Given that the mine plan has the underground zones being mined first, followed by the open pit later, there is an opportunity to defer capital expenditure by delaying the installation of the gravity circuit until the start of open pit mining.

13.5.2.7 Cyanidation of Gravity Tails

A 2 kg sub-sample of the tailings produced from the bulk gravity separation tests (Table 13-39) was subjected to standard bottle-roll cyanidation. Cyanidation conditions were consistent with those used for the whole-ore cyanidation tests. Results are summarized in Table 13-41.

Stage gold recovery during cyanidation is reported as the percentage of gold extracted from the gravity tailings sample into solution. Overall gold recoveries, inclusive of both the preceding gravity step and subsequent cyanidation of the gravity tailings, are also presented. Overall recovery was calculated using both solution gold analysis by AA spectrometry and solids gold analysis by fire assay.

Good agreement was observed between recovery values calculated from solution and solids analyses, providing confidence in the accuracy of the test and analytical procedures used by the laboratory.

Table 13-41: Cyanidation of Gravity Tails Results

Sample ID	Au Stage Recovery (%)	NaCN (kg/t)	Lime (kg/t)	Residence time (h)	Overall Recovery% (Calculated from Solution Analyses)	Overall Recovery% (Calculated from Solids Analyses)
Sample 1	76.4	0.23	0.64	24	79.0	79.5
Sample 2	89.6	0.28	0.85	24	91.1	92.0
Sample 3	89.2	0.19	0.82	24	92.4	85.9
Sample 4	69.3	1.04	0.54	48	72.0	75.0
Sample 5	65.4	0.19	0.82	48	69.6	69.8
Sample 6	63.2	0.24	0.98	30	65.7	65.2
Sample 7	90.7	0.28	0.91	24	91.1	90.6
Sample 8	77.9	0.44	1.20	32	79.4	76.5
Average	77.7	0.36	0.85	31.8	80.0	79.3

Comparing these results with the whole-ore cyanidation tests shows that overall recoveries were similar, with average recoveries of 80.7% for the whole-ore cyanidation tests compared to 79.7% for the gravity plus cyanidation route. This indicates that the addition of a gravity step does not necessarily improve overall gold extraction when sufficient time was allowed for coarse gold dissolution in whole-ore leach testing.

Comparison of minimum required residence times indicates that the gravity tailings leached faster than the whole-ore samples. This implies that a residence time of approximately 32 h will be sufficient for leaching of gravity tailings, whereas the same residence time may result in incomplete dissolution of coarse gold if the



gravity step is omitted. Alternatively, a whole-ore cyanidation flowsheet would require more leach volume to achieve the same gold extraction as a gravity-leach flowsheet.

Measured reagent consumptions also differed between two routes, with gravity-leach cyanidation requiring less cyanide (0.36 kg/t vs 0.46 kg/t) and substantially less lime (0.85 kg/t vs 3.1 kg/t). These results indicate that a significant portion of the reagent consumers reported to the gravity concentrate. It is possible that the reagent consumption may be the same after intensive cyanidation of the gravity concentrate; however, there is potential to reduce reagent costs if intensive cyanidation incorporates a pre-oxygenation step, whereby cyanide-soluble sulphides may be oxidized to sulphate. The oxidation of reagent consumers was not tested in this test program but should be considered in future test programs if a gravity-plus-leach flowsheet is pursued.

13.5.2.8 Rougher Flotation of Gravity Tails

A 2 kg sub-sample of the same bulk gravity tailings used for the cyanidation tests was subjected to rougher flotation testwork using the following conditions:

- Denver flotation cell (4.4 L).
- Natural pH ranging from 8.3 to 8.6.
- Total duration of 8 minutes with sampling at 2-minute intervals.
- PAX additions ranging from 40 to 70 g/t (depending on visual observation of the froth).
- MIBC additions ranging from 44 to 60 g/t (visual observation of froth).

Table 13-42 summarizes the results. All values in the table refer to the head grade (i.e., rougher flotation recovery represents the percentage of gold reporting to the rougher concentrate relative to the head sample, rather than the gravity tailings sample. The mass pull, similarly, is based expressed relative to the original ROM sample mass and does not represent a stage recovery from the previous step.

Table 13-42: Rougher Flotation of Gravity Tails Results

Sample ID	Gravity Mass Pull%	Gravity Au Recovery %	Rougher Float Mass Pull (%)	Rougher Float Au Recovery (%)	Overall Au Recovery% (Gravity + Float)	Overall Sulphur Recovery% (Grav + Float)
Sample 1	0.16	11.0	6.8	77.4	88.4	94.6
Sample 2	0.19	14.9	4.4	74.8	89.7	92.3
Sample 3	0.19	30.0	4.7	59.3	89.3	91.0
Sample 4	0.18	8.7	10.3	78.9	87.6	95.9
Sample 5	0.16	12.3	5.9	73.9	86.2	88.3
Sample 6	0.18	6.9	7.1	72.0	78.9	89.0
Sample 7	1.10	1.3	5.2	84.8	86.1	92.8
Sample 8	1.10	3.3	4.0	82.1	85.4	89.6
Average	0.41	11.1	6.1	75.4	86.5	91.7



Comparison with results previously reported indicates that the rougher concentrate mass pull of 6.1% for these underground samples is lower than the average of 12% recorded for the open pit samples tested previously. This difference may be partly due to the shorter 8 minute total duration compared to 40 minutes for the open pit samples. However, evaluation of the incremental mass recoveries it is evident that very little additional mass or sulphide recovery is expected after 8 minutes.

Overall gold and sulphur recoveries for the underground samples, at 86.5% and 91.7%, respectively, were also lower than the corresponding recoveries of 93% and 95% reported for the open pit samples. The implication is that both cyanidation and flotation achieved lower overall gold recoveries for the underground samples relative to the open pit material.

13.5.2.9 Whole-Ore Rougher Flotation

A 2 kg sub-sample of the original as-received samples was subjected to rougher flotation testwork after grinding to 80% passing 90 µm. The purpose of this set of tests was to evaluate whether the omission of an upstream gravity concentration step had any effect on flotation performance. The same conditions were employed as for the flotation test on gravity tailings.

Table 13-43 summarizes the results for the set of whole-ore rougher flotation tests.

Table 13-43: Whole-ore Rougher Flotation

Sample ID	Mass pull (%)	Gold Recovery (%)	Sulphur Recovery (%)
Sample 1	4.8	89.7	93.0
Sample 2	4.7	89.6	87.6
Sample 3	6.7	91.0	96.7
Sample 4	11.6	88.7	84.9
Sample 5	6.3	88.4	88.9
Sample 6	8.2	79.4	89.8
Sample 7	5.6	88.8	89.8
Sample 8	3.9	88.1	86.3
Average	6.5	88.0	89.6

Comparison of these results with the previous set of tests that included a gravity step indicates that the overall gold recoveries were marginally higher (88.0% vs 86.5%). However, given that recoveries were calculated from gold analyses of the gravity concentrate, which represents a small mass at a much higher grade, the observed difference is likely within the margin of error. The absence of a significant decrease in performance suggests that the gravity step is not an essential component of the overall flowsheet.

Samples were analyzed at 2 minute floatation intervals to assess kinetics. Figure 13-3 shows the averaged curve for all eight samples combined.



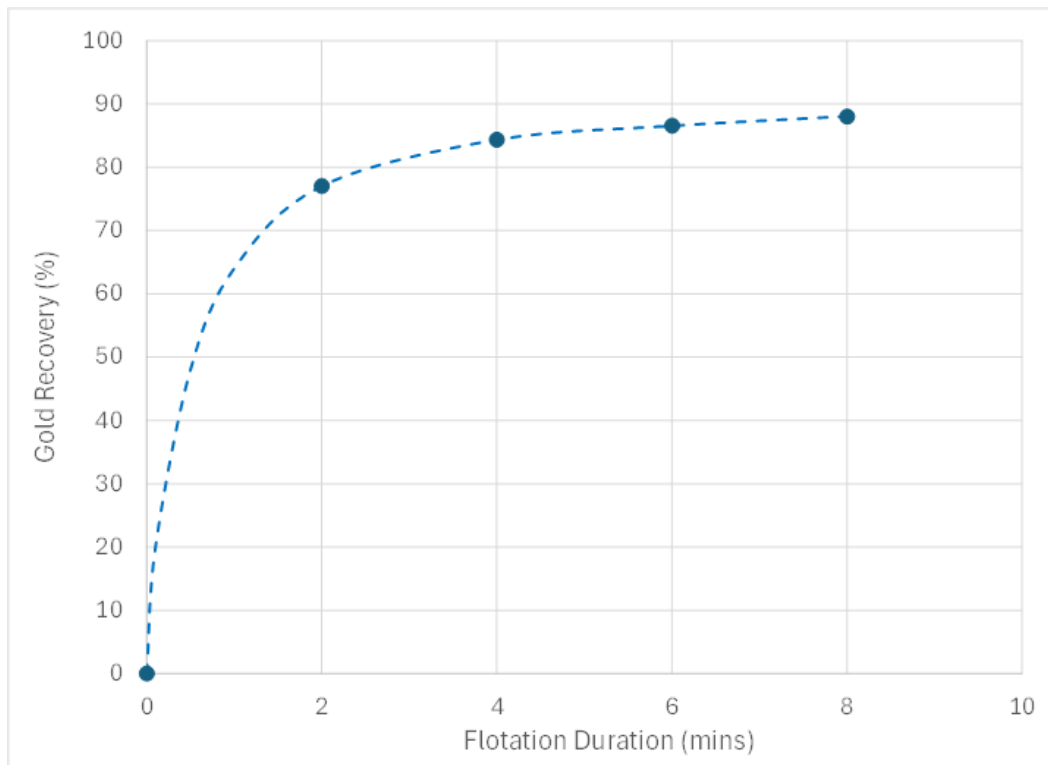


Figure 13-3: Rougher Flotation Kinetics for Whole-Ore Samples

Flotation kinetics were fast, with most of the recoverable gold reporting into the concentrate within the first 2 minutes. Recovery then slowed significantly but continued at a steady rate over the remainder of the 8-minute test duration. It appears that extending the test time beyond 8 minutes would likely have resulted in gold recoveries exceeding 90%. Given that previous test campaigns on open pit samples achieved approximately 93% but with 40 minutes of flotation time, it is likely that the shorter duration of the underground sample tests was a primary factor contributing to the lower recoveries achieved.

Figure 13-4 shows a comparison of the overall gold recoveries achieved through cyanidation versus flotation for all eight samples. For both cyanidation and flotation routes, the average results of the tests with and without gravity separation were used. The graph shows that the flotation route yielded less variable and, on average, superior results compared to cyanidation. The average recovery for the eight samples combined was 87.2% through flotation, compared to 80.2% through cyanidation. While the recoveries are marginally lower than for open pit samples, mostly due to shorter float times, the underground samples nevertheless exhibited good amenability to flotation.



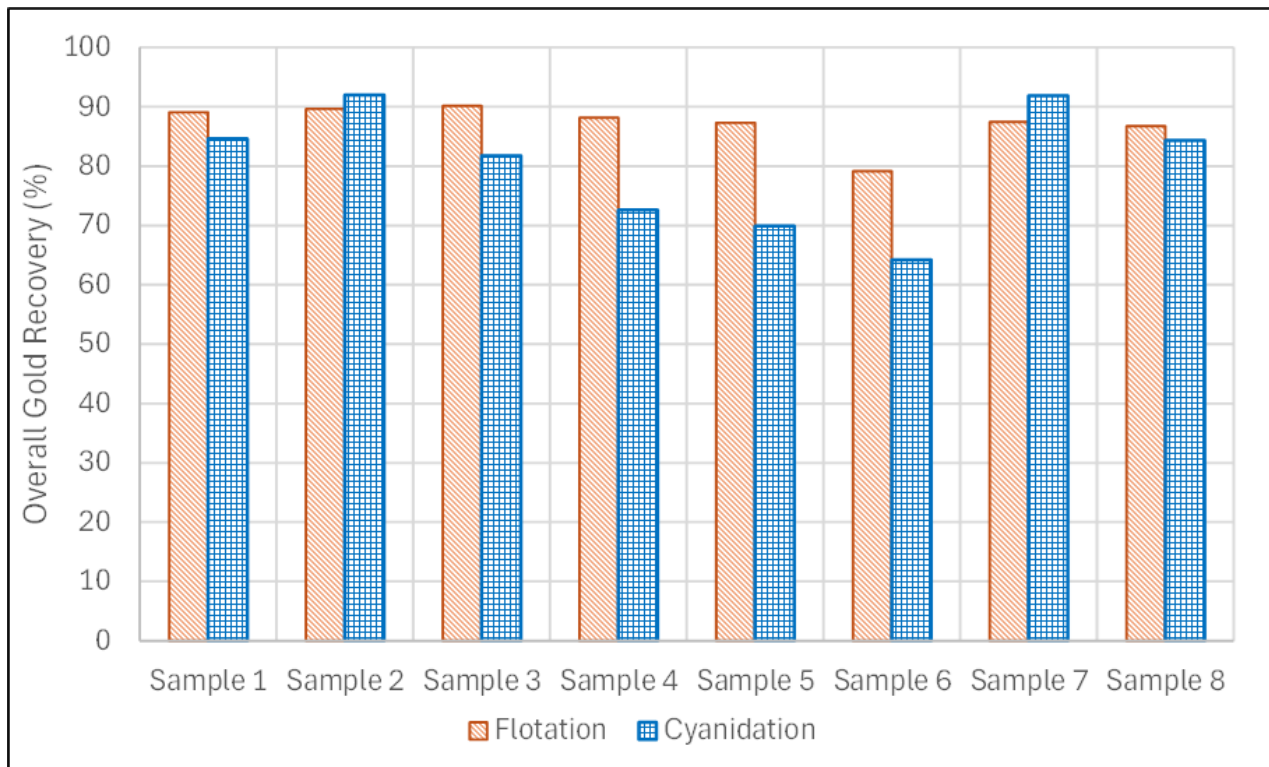


Figure 13-4: Overall Recoveries through Cyanidation versus Flotation for Underground Samples

13.5.2.10 Cyanidation of Rougher Tails

The rougher flotation tailings produced during the previous set of tests were subjected to bottle-roll cyanidations using the standard conditions as listed in Section 13.5.2.5. Results are tabulated in Table 13-44.

Note that the gold stage recoveries represent the average of recoveries calculated from the solution analysis by AA spectroscopy and from the fire assays of the tailings residues. The results shows that approximately 72% of the gold contained in the rougher tails was extracted through cyanidation. Given that the rougher tails contain approximately 12% of the ROM feed gold, the downstream cyanidation of the rougher tails contributes approximately 8.6% to the overall gold recovery.

Table 13-44: Cyanidation of Rougher Flotation Tails

Sample ID	Au Stage Recovery (%)	Calc Feed Grade (g/t)	Tails grade (g/t)	NaCN (kg/t)	Lime (kg/t)	Residence time (h)
Sample 1	77.7	0.39	0.09	0.50	0.7	24
Sample 2	76.5	0.54	0.13	0.42	0.69	48
Sample 3	80.1	0.34	0.07	0.16	0.67	48
Sample 4	68.3	0.25	0.08	0.39	0.63	48
Sample 5	52.9	0.46	0.22	0.40	0.81	24



Sample ID	Au Stage Recovery (%)	Calc Feed Grade (g/t)	Tails grade (g/t)	NaCN (kg/t)	Lime (kg/t)	Residence time (h)
Sample 6	55.7	1.53	0.68	0.83	1.05	24
Sample 7	78.4	0.87	0.19	0.39	1.13	24
Sample 8	85.7	0.49	0.07	0.46	0.75	48
Average	71.9	0.61	0.19	0.44	0.80	36

Cyanide and lime consumptions were similar to those recorded during the gravity tailings series of tests. Four of the samples exhibited slower leach kinetics, with a significant amount of leaching continuing into the second day. This behavior is unexpected because leaching from silicates is usually fast and complete within 24 h.

Another unexpected result was the tailings grade of 0.68 g/t Au recorded for Sample 6. It is possible that the gold in this sample was not liberated, which is consistent with the low flotation recovery observed for this sample. No follow-up testing was conducted on this material. It is therefore recommended that a similar sample be collected for future testing and that its products be subjected to a diagnostic procedure to determine the cause of its lower recovery.

13.5.2.11 Cyanidation of Rougher Concentrate

Rougher flotation concentrates produced during the previous set of rougher flotation tests were ground to 80% passing 14 µm and then subjected to cyanidation. Regrinding of flotation concentrates allowed additional liberation and gold recovery during previous test campaigns for open pit samples. The purpose of this set of rougher concentrate cyanidations was to assess if the same holds true for the underground samples. The same set of standardized cyanidation conditions were used for this set of tests.

Results are tabulated in Table 13-45.

Table 13-45: Cyanidation of Rougher Flotation Concentrate

Sample ID	Au Stage Recovery (%)	Calc Feed Grade (g/t)	Tails grade (g/t)	NaCN (kg/t)	Lime (kg/t)	Required Residence time (h)
Sample 1	93.0	47.4	3.33	2.90	2.40	24
Sample 2	94.0	69.9	4.19	2.40	2.50	24
Sample 3	82.1	33.6	6.03	1.30	1.70	30
Sample 4	45.8	14.1	7.63	5.70	0.70	48
Sample 5	69.7	45.4	13.75	0.80	2.30	40
Sample 6	70.2	54.7	16.3	0.50	0.20	16
Sample 7	95.1	80.3	3.94	0.50	2.20	24
Sample 8	94.2	46.6	2.72	0.80	1.40	18
Average	80.5	49.0	7.24	1.86	1.68	28



Samples 1, 2, 7 and 8 yielded stage gold recoveries exceeding 90%. However, the other four samples all yielded lower than expected recoveries, with some as low as 45.8% (Sample 4). This sample contained a large amount of cyanide soluble copper so it is possible that cyanide deficiency may have been partly responsible for the result (even though the cyanide was measured and replenished at timed intervals). The results suggest that it would be worthwhile to explore the possible application of the Albion process (which could destroy the sulphide matrix in an atmospheric oxidation step ahead of cyanidation).

Cyanide consumption was skewed by the 5.7 kg/t outlier recorded for Sample 4. Both cyanide and lime consumptions were highly variable as can be expected from concentrates.

Leach kinetics were, in most cases, faster than expected. The only sample requiring more than 40 h of leaching was Sample 4 which may, again, be due to insufficient cyanide.

13.5.2.12 Combining the Results for the Float-Regrind-Leach Flowsheet

In this section, results from the entire underground test campaign were combined to estimate overall gold recovery for the float-regrind-leach flowsheet. Table 13-46 summarizes the results and illustrates the distribution of gold among the various products generated, including gravity concentrate, flotation concentrate, and leach solutions following leaching of the flotation tails and concentrate, respectively. The final two columns in Table 13-46 compare the overall recovery achieved using the flotation-based flowsheet with recovery obtained without the flotation step, using whole-ore cyanidation only.

The comparison shows that the addition of the flotation-regrind steps adds significant value to the flowsheet in terms of increased gold recoveries. Sample 4 is the only exception with the very low recovery of 54.4% being significantly lower than the 72.6% achieved for the leaching only flowsheet. This result for sample 4 is believed to be anomalous and due to cyanide deficiency and insufficient leach time during the cyanidation of the rougher concentrate. Flotation followed by regrinding will liberate more gold hence the expectation is that the result must be either the same or better when adding these additional steps. A significant decrease in recovery is not possible hence the observed result can be ignored as an anomaly. In the last two rows the averages are calculated first with sample 4 included then when excluding Sample 4 data. With sample 4 excluded the overall recovery for the flotation-regrind flowsheet is 86.1% which is significantly better than the 81.2% achieved without flotation. This outcome supports the decision to add the flotation step into the full-scale plant flowsheet.

It should be noted that the average recovery achieved for the underground samples with the float-regrind flowsheet is about 2% lower than what was previously achieved for the open pit samples. This difference is even more pronounced (about 5%) for the simpler flowsheet which excludes the flotation-regrind steps.



Table 13-46: Cyanidation of Rougher Flotation Concentrate

Sample ID	Gold Distribution (% of head feed)				Overall Au Recovery (%)	
	Gravity Conc.	Flotation Conc.	Tails Leach	Conc. Leach	With Float + regrind	Without Float + regrind
Sample 1	15.1	77.4	5.8	72.0	92.9	84.6
Sample 2	20.7	74.8	3.5	70.3	94.4	92.0
Sample 3	21.3	59.3	15.5	48.7	85.5	81.8
Sample 4	12.3	78.9	6.0	36.1	54.4	72.6
Sample 5	11.9	73.9	7.5	51.5	70.9	69.9
Sample 6	7.1	72.0	11.6	50.5	69.3	64.2
Sample 7	9.8	84.8	4.3	80.6	94.7	91.8
Sample 8	14.1	82.1	3.3	77.3	94.7	84.3
Average	14.0	75.4	7.2	60.9	82.1	80.1
Avg excl. Sample 4	14.3	74.9	7.4	64.4	86.1	81.2

13.6 Process Design Criteria Derived from Test Results

The following Process design criteria (Table 13-47) consolidates all operating and design parameters that can be directly referenced to the Grey Fox metallurgical test programs (2013 to 2025). Values were extracted from MC, variability and bulk tests undertaken to date. Parameters are grouped by unit operation—mineralogy/head grade, comminution, gravity, flotation, cyanidation (whole ore, tails and concentrate), detoxification, and solid-liquid separation.

All reagent dosages are stated as consumption on a dry-solids basis (kg/t); residence times and grind sizes are those that achieved target metallurgical performance in the lab.

A flowsheet incorporating flotation followed by regrinding of the flotation concentrate yielded higher gold recoveries for some of the samples tested to date and should be the preferred route for any greenfield development. However, the existing Stock Mill flowsheet does not include flotation. Retrofitting flotation and regrinding into the existing mill flowsheet and layout would be costly and operationally disruptive. Additionally, the remaining Stock Main and West ore is not expected to benefit from flotation and represents approximately half of the total tonnage projected to be treated through the Stock Mill with Grey Fox material. Consequently, the existing whole-ore cyanidation flowsheet at the Stock Mill will be adopted, unchanged, for Grey Fox ore treatment.

These criteria are intended as a baseline for mechanical and mass-balance design for the trade-off and PFS studies.



Table 13-47: Process Design Criteria

Area	Parameter	Design Criterion / Typical Range	Notes / Basis
Mineralogy and Head Grade	Gold head grade (Au)	Nominal = 3.8 g/t	Test work Samples Typical Range = 3 – 5 g/t
	Total sulphur (S)	1.5%–2.0%	Dominated by pyrite; rises to $\approx$ 3% S in high-sulphide SEDS domains. Needs to be assessed with further variability testing.
	Gangue minerals	Quartz-carbonate host; CO ₂ 7%–10%; SG $\approx$ 2.9 t/m ³	Carbonate buffering limits acid generation; density used for all volumetric throughputs.
Comminution	BBWi	Nominal = 23 kWh/t; Design = 27 kWh/t; Range 18–27 kWh/t	Suggest a high-power SABC or HPGR-ball-mill circuit.
	SMC parameters	DWi = 9.0–9.6 kWh/m ³ ; Axb = 30–33	Confirms competent, impact-resistant ore.
	Rock Density	2.91 t/m ³ (ore)	Used for equipment sizing and mass/volume conversions.
Whole-Ore Leach	Grind P ₈₀	P ₈₀ = 75 μ m	Diminishing returns < 100 μ m. Unliberated gold appears to be fine.
	Au Recovery	64%–95% (Avg 84% for open pit zones and average 88% for the underground samples)	Average recovery for underground zones excludes anomalous results from the Gibson North zone.
	Pre-aeration conditions	1 h at 40% w/w solids, pH 10.5–11 (for open pit, not used for underground tests)	
	Leach time	36h	Tests range was 26 to 48 h
	NaCN dosage	1.1 kg/t for open pit samples; 0.46 kg/t for underground	MC & variability tests – KM6034 report. Important to note underground tests were done at lower cyanide levels.
	Lime (CaO)	1.2 kg/t (with pre-aeration) but >2 without (underground tests were without)	To hold pH $\sim$ 11
Detoxification	Method	SO ₂ /Air + Cu catalyst	
	CN-WAD start and end conc	165 mg/t to 0.1 mg/l	
	Residence	180 min	Batch-tested
Sedimentation	Thickener underflow density	65% solids	



Area	Parameter	Design Criterion / Typical Range	Notes / Basis
	Unit area	88 and 103 m ³ /m ² /day net rise rate	147 Zone and CZ
		0.41 to 0.47 t/m ² /h solids loading	147 Zone and CZ
		3.6 m ³ /m ² / h net hydraulic loading	147 Zone
	Flocculant	15–25 g/t Magnafloc 351 (non-ionic)	

13.7 Recovery Model

The Grey Fox deposit comprises a combination of free milling gold and gold intricately associated with pyrite (semi-refractory). Recovery is generally positively correlated to liberation and negatively correlated to sulphur grade. With a limited volume of distinct variability samples tested to date it is not appropriate to attempt to define a predictive model at this juncture. Consequently, only averaged recoveries will be used for the PFS.

Whole-ore cyanidation results for the Gibson North Zone yielded unexpectedly low recoveries into the leach solutions. Upon investigation, it was observed that the selected intercepts had unusually high concentrations of graphite, phyllosilicate, and galena. It is possible that the poor recoveries could be attributed to re-adsorption of leached gold, surface coatings, or physical encapsulation of liberated gold. For the purposes of the PFS, these results were excluded from the estimation of average recovery for the combined underground zones.

Dissolved gold losses and losses to carbon fines are expected to be constant and independent of grade. A well-run carbon in pulp (CIP) adsorption circuit should maintain dissolved gold losses below 0.01 mg/L. At a solid-to-liquid ratio of roughly 1:1, this equates to a constant efficiency loss of 0.01 g/t. Gold losses to carbon fines are negligible by comparison.

The resultant recovery model for the Grey Fox underground deposit can therefore be formulated as follows:

$$\text{Gold Recovery (\%)} = 87.8 - (1 / \text{Head Grade})$$

Where 87.8 represents the average extraction achieved through cyanidation, and the second term represents dissolved gold losses as a function of head grade (g/t).

Opportunities to improve the overall recovery will be explored during future testwork programs. Adding a flotation step followed by fine grinding and leaching of flotation products has shown some promise. Adding an oxidation step after fine grinding to destroy the sulphide host matrix will also be explored.



13.8 Deleterious Elements

The current flowsheet will produce doré bars; therefore, no penalties are anticipated. Mineralogical testing showed that the ore does not contain significant quantities of arsenic, while head-grade analysis showed that mercury levels are not significant.

13.9 Interpretation and Conclusions

Cyanidation, with or without gravity concentration, yielded mixed results, with some samples registering recoveries exceeding 94%, while a significant proportion of samples tested registered recoveries below 90%. Based on the testwork results, the average recovery with this type of flowsheet is estimated at approximately 87% for the underground zones. Additional testing is required to improve the accuracy of the recovery estimate.

Given the hardness of the samples tested, a SABC comminution circuit is recommended.

13.10 Recommendation for Future Work

Historical testwork provides sufficient information on the Grey Fox deposit to support a PFS. However, beyond the PFS stage, and contingent upon the consideration of an optimized flowsheet that is not processing Grey Fox ore at the Stock Mill, several information gaps and opportunities remain, including the following:

- The effects of pre-oxygenation, oxygen sparging, and the addition of chemical oxidants (e.g., PbNO_3) on cyanide consumption have not been investigated.
- For fine grinding, vendor-specified fine-grinding tests are required, as Bond's Fourth Law is not valid below a grind size of approximately 50 μm . This recommendation applies only to a flowsheet that contains a fine-grinding or regrinding step.
- Carbon adsorption testing, including assessment of the effects of carbon fouling by flotation reagents.
- The addition of a cleaning stage after the initial bulk flotation step should be tested, as this could reduce the costs for downstream processing of the concentrate. Of particular interest would be to assess the leach efficiency of the cleaner tails if they are rejected to tailings rather than recycled. Cleaner flotation should be tested both with and without regrinding. Producing a saleable flotation concentrate could result in an overall recovery above 90% if the cleaner tails are also amenable to cyanidation.
- Specific testing to assess the presence of preg-robbars.
- Sedimentation and rheology of the flotation concentrate have not been tested.
- Intensive cyanidation of gravity concentrate should be tested.
- Filtration of tailings could be considered, as this would provide information should paste backfill ever be considered. While this is not currently the case, it may be prudent to generate this information.



- Water chemistry and the effects of recirculating process water need to be determined. Test work to date was conducted using neutral-pH tap water. The effects of flotation reagents, lime, and NaCN recirculating through the plant process water system must therefore be investigated.
- Testing of the Albion Process to evaluate the potential for overall recoveries exceeding 95%. Test work performed to date indicates that remaining unrecovered gold is locked within sulphides, which may be liberated if subjected to the full Albion Process.
- Locked-cycle testing should be undertaken if cleaner flotation is considered.
- Dilution testing to assess amenability to the use of Jameson flotation cells.

While the current metallurgical and geochemical testwork programs have provided useful baseline data, it is important to note that the testing has predominantly been conducted on master or composite samples rather than on discrete variability samples. As such, the spatial and lithological variability across the deposit has not been thoroughly evaluated. Further investigation is warranted to assess the geometallurgical variability across the resource, particularly in relation to lithological domains, alteration styles, and sulphide content, and how these factors align with the mine plan and processing schedule. A structured variability testing program—incorporating both metallurgical and geochemical analyses on domain-specific intervals from representative boreholes—is recommended to ensure process performance assumptions remain valid across the full LOM.

Specific areas that should be confirmed through variability testing include:

- Whole-ore cyanidation leach performance, especially in relation to sulphur content and Au/S ratios.
- GRG response across different lithologies and head grades.
- Flotation response and variability in sulphide recovery rates across domains.
- Cyanidation of flotation concentrates and tailings to quantify recovery contributions and assess leach kinetics.
- Variability in reagent consumption (cyanide, lime, flotation reagents) as a function of mineralogy.
- Confirmation of preg-robbing potential associated with variable carbonaceous content and its deportment.



14 Mineral Resource Estimates

14.1 Approach

The 2026 Grey Fox Pre-Feasibility Study, and thus the more in depth engineering and economic aspects of this Technical Report, are based on the mineral resources defined at Grey Fox. In addition, the report also includes Resource Model updates for the following deposits which also occur on the broader Fox Complex Project areas:

- Black Fox
- Froome
- Stroud
- Stock West and Main
- Stock East
- Fuller
- Davidson-Tisdale

Two employees of McEwen are the QPs for all resource estimates in this report and have calculated the Mineral Resource estimates for Grey Fox, Black Fox, Froome, Stroud, Stock West and Main, Stock East, Fuller and Davidson-Tisdale. The Resource Models were prepared and/or updated using Datamine Studio RM estimation software, with lithological modelling prepared using Leapfrog Geo software. Geostatistics were analyzed using Datamine's Snowden Supervisor software module. To demonstrate the Reasonable Prospects for Eventual Economic Extraction (RPEEE), all deposits were evaluated using Datamine's Mineable Shape Optimizer (MSO) extension.

In general the methodology is as follows:

- Compilation of the Resource Database and Data Validation
- Exploratory Data Analysis
- Geological Modelling
- Compositing of Sample Data
- Geostatistics and Block Estimation
- Model Validation
- Resource Classification
- Reasonable Prospects for Eventual Economic Extraction
- Mineral Resource Statement
- Factors That Could Affect Mineral Resource Estimates

14.2 Resource Estimation

The accuracy of resource (and later reserve) estimates is, in part, a function of the quality and quantity of available data and of geological interpretation (and later engineering) and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable.



However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material.

14.3 Grey Fox

14.3.1 Mineral Resource Database

The close-out date for the Mineral Resource database was 11 July 2025. A total of 23 drill holes were excluded from the drill database because they are geotechnical or metallurgical holes and were never assayed for gold. The database now consists of 1,659 core drill holes totalling 619,504 m of drill core and 322,665 m of sampled core. Unsourced intervals were considered barren and assigned a background gold grade of 0.0005 g/t based on one quarter of the assay laboratory's lower detection limit.

Drilling campaigns at Grey Fox have taken place since 2021 with 29 new holes in 2021, 14 new holes in 2022, 21 new holes in 2023, ramping up to 126 holes in 2024 and 65 holes in 2025. These holes were primarily drilled in the Gibson, 147, South and Whiskey Jack zones. Logging and assay results continue to indicate strong continuity in Whiskey Jack and an increase in confidence in the model. Drilling in the Gibson zone resulted in the modelling of additional C-veins, expanding resources to the North, as well as the validation of previously modeled C-veins. Drilling in the South and 147 Zone resulted in validation of previously modeled C-veins, intersecting them at optimal angles.

On importing the drill hole database from the central SQL database to Datamine Studio software in preparation for resource estimation, sample data were checked for missing and overlapping intervals in the assay and survey tables, and collar locations checked against the LiDAR topography surface.

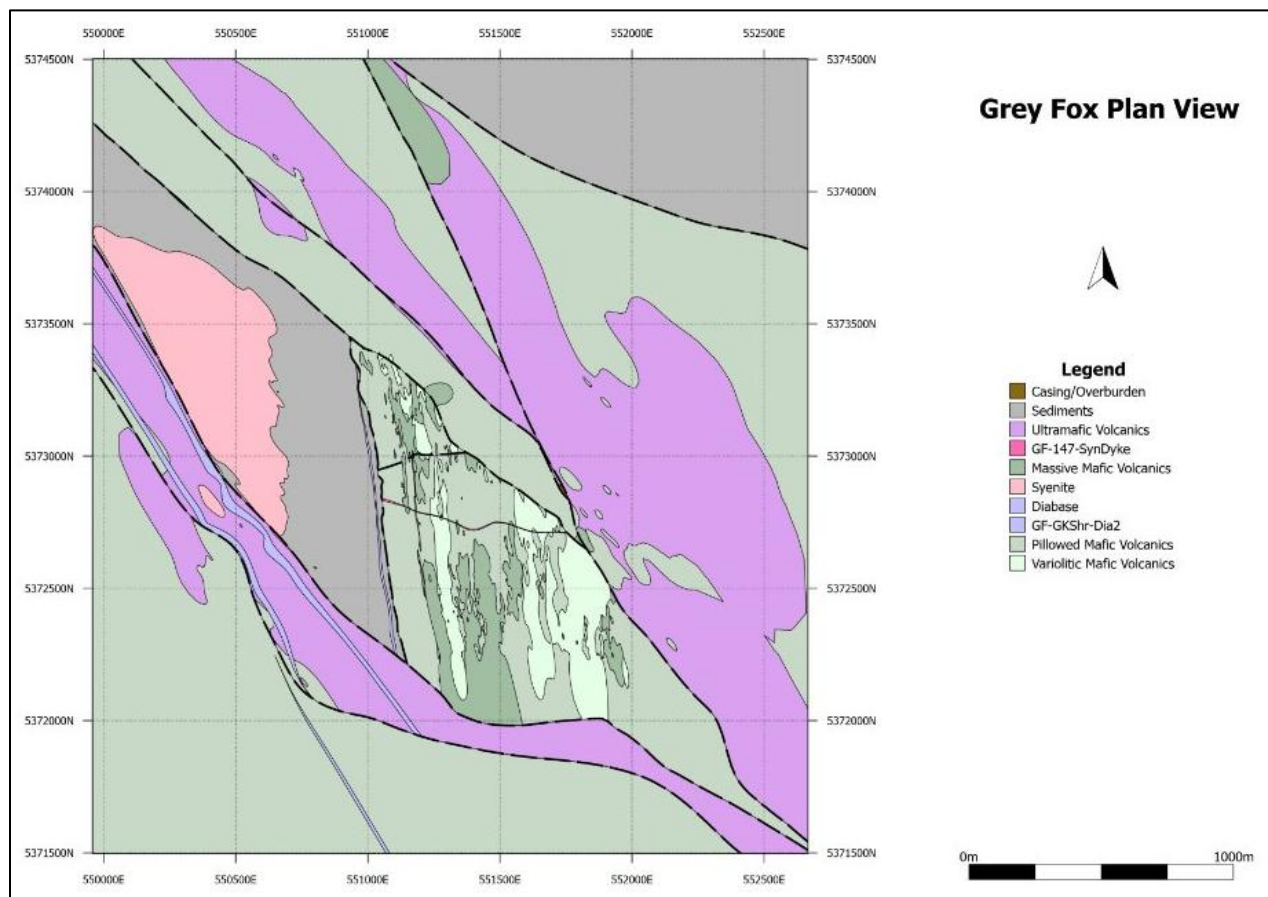
14.3.2 Geological Modelling

14.3.2.1 Lithology

A lithology model was constructed to capture the seven key deposit lithologies as shown in Figure 14-1. The primary deposit area is bounded in the east by ultramafic metavolcanics and in the west by chlorite-talc schist ultramafic metavolcanics. Each are separated from the main deposit area by regional faults and deformation zones. The western portion of the deposit is hosted in a metasedimentary unit intruded by a brecciated syenitic- to dioritic-feldspar porphyry intrusion. The metasedimentary unit has an angular unconformity contact with an assemblage of metavolcanic mafic rocks in the east. There is a prominent diabase dyke (believed to be post mineralization) running through the assemblage sub-parallel to the unconformable sediment-mafic contact. The mafic metavolcanic units can be described as massive/pillowed, or variolitic textured and have been separated in the model based on those primary textures.

A 3D surface of the bedrock/overburden contact was generated from the drill holes and used to constrain the upper extent of mineralization.





Source: (McEwen, 2025)

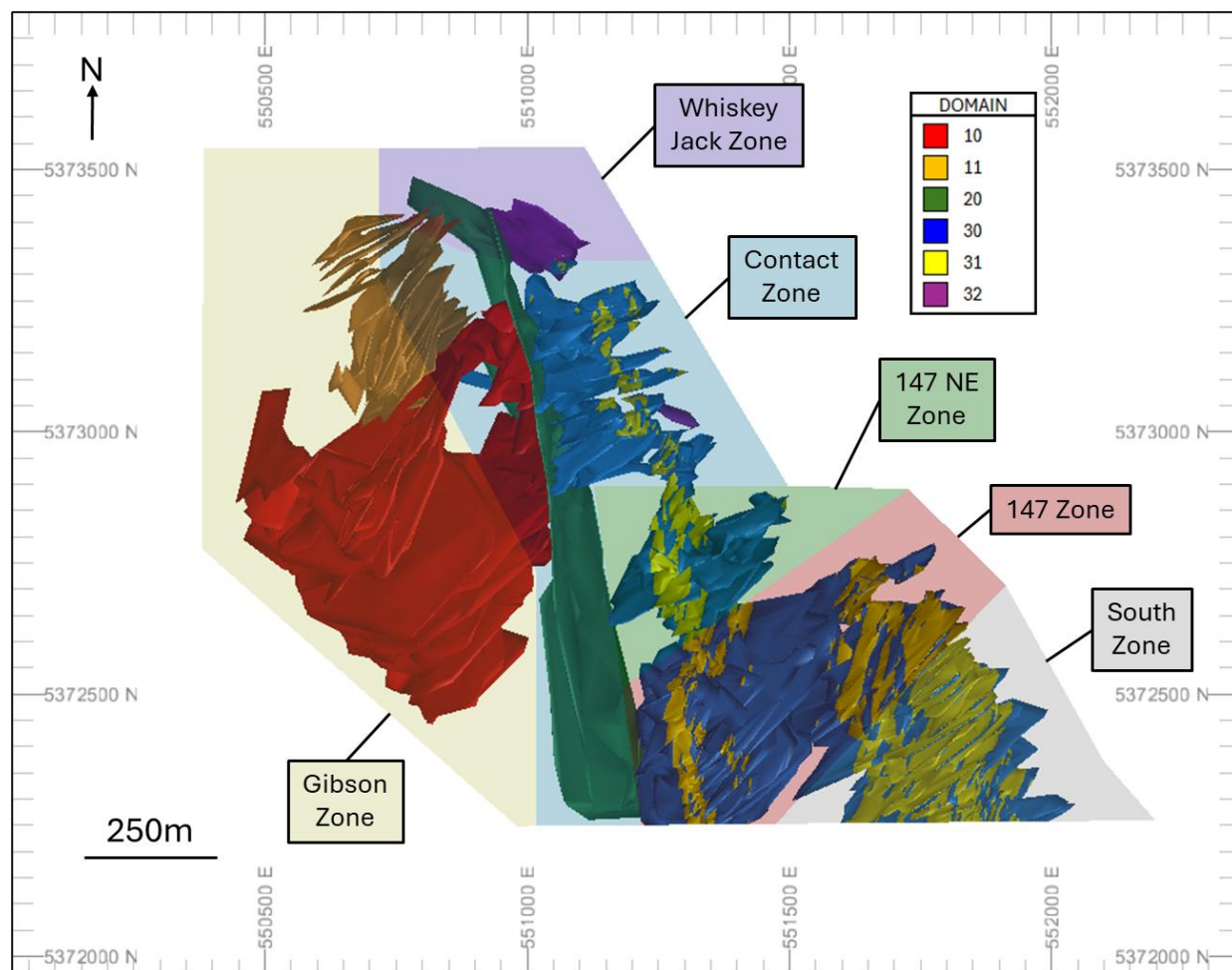
Figure 14-1: Plan View of the Grey Fox Project Lithology Model

14.3.2.2 Mineralized Zones

The Grey Fox deposit area is subdivided into six mineralized zones differing in host lithology. The zones include the Contact, 147, 147NE, Gibson and South Zone with the northern most mineralized lens of the Contact Zone area named the Whiskey Jack Zone (Figure 14-2). The mineralization is further categorized into six domains based on mineralization style, orientation, grade population, and host lithologies.

The mineralization wireframes were created by geologically interpreting the structures controlling mineralization, guided by structural data and gold grades. These were then explicitly modelled creating closed volumes around the defining drill hole intercepts at an approximate cut-off grade of 0.5 g/t Au.





Source: (McEwen, 2025)

Figure 14-2: Plan View of the Grey Fox Project Mineralized Zones and Domains

A description of each mineralized zone is summarized in Table 14-1.

Veins hosted in the syenite and sediments form the local Domains 10 and 11; Domain 10 has a lower grade population, and an orientation that strikes Southwest and forms the Gibson Main; Domain 11 has a slightly higher grade population, strikes Southwest and forms the Gibson North. Veins hosted in the mafic intrusive and volcanic rocks form Domain 30. The veins hosted in the variolitic units appear to be a continuation or subset of the veins in the surrounding mafic rocks and form Domain 31. Due to their physical continuity with Domain 30, but having a slight grade population difference, Domain 31 veins were estimated using a soft boundary contact with veins of Domain 30.



Table 14-1: Grey Fox Mineralized Domains

Domain	Description
10	Southwest trending, steeply northwest dipping quartz veins and breccias hosted in sediments and the Gibson syenite intrusive (70° dip @ 280° dip direction)
11	Southwest trending, steep northwest dipping quartz veins and breccias hosted in sediments and the syenite Gibson intrusive (80° dip @ 310° dip direction). This is a new domain, separated from Domain 10, due to a change in lens orientation and indicated by geostatistical data.
20	North-south trending unconformity contact between mafic rocks and sedimentary rocks. Outcrop is steeply east dipping (75° dip @ 80° dip direction), at the northern extent and at depth the zone rolls over to a steep westerly dip (75° dip @ 250° dip direction)
30	Southwest trending, steeply northwest dipping quartz veins and breccias hosted in mafic volcanic flows and mafic intrusive rocks (65° dip @ 310° dip direction)
31	Southwest trending, steeply northwest dipping quartz veins and breccias hosted in dominantly iron-rich variolitic textured mafic rocks (65° dip @ 310° dip direction)
32	East trending sub-vertical to steep south dipping quartz breccia veins hosted in mafic volcanic and intrusive rocks (75° dip @ 175° dip direction)

14.3.2.3 Background Envelope

A background envelope (Domain 100) of approximately 25 m was constructed around all mineralized domains to capture mineralized samples outside the mineralized domains and to estimate adjacent dilution.

14.3.3 Composites

More than 70% of the assays have a sample length of 1 m. The modal composite length of 1 m was applied to samples within the mineralized zones which generated composites close to 1 m with a minimum composite length of 0.3 m. A modal composite length of 4 m was applied to samples within the background envelope (dilution) based on the parent cell size of the block model.

14.3.1 Capping

The impact of high-grade outlier populations was analyzed using histograms, log probability plots, mean and variance, and cumulative metal plots. Results are presented in Table 14-2 for each mineralized domain with metal removed ranging from 4.6% in Domain 11 and 27.4% in Domain 20.

Table 14-2: Grey Fox – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
10	1.60	3.67	27	1.47	1.98	29	8.5
11	1.99	2.37	27	1.90	2.06	17	4.6
20	2.23	10.02	34	1.62	2.39	19	27.4
30	1.65	23.53	87	1.23	3.94	20	25.4



Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
31	2.02	8.58	70	1.76	2.95	27	13.0
32	4.22	2.41	48	3.93	2.06	16	6.8
100	0.10	19.07	10	0.09	4.65	115	13.8

14.3.2 Density

There are 14,459 density samples contained in the Mineral Resource database. Currently, one in twenty assay samples are selected for density analysis by the assay laboratory. A study of all density data was conducted to determine an average density value for each of the host lithology units. Blocks were assigned the average density per unit as listed in Table 14-3.

Table 14-3: Grey Fox – Density Values by Lithology

Lithology	No. of Samples	Avg. Density
Ankerite Altered Ultramafic Volcanics (AUV)	17	2.82
Fuchsite Altered Ultramafic Volcanics (CGR)	98	2.89
Diabase (DIA)	49	2.94
Mafic Volcanics (AMV, PMV, MV, MI, MR)	3580	2.90
Quartz Veins (QV, QBX)	55	2.86
Sediments (SED, GR)	719	2.76
Felsic Intrusives (SYN, FI, FP)	1660	2.76
Ultramafic Volcanics (UMV, UV, CUV, TUV)	488	2.83
Variolitic Mafic Volcanics (VIV)	4902	2.86
All Lithologies Combined	11568	2.85
* 14,459 Total samples. Lowest 5 and highest 5 values removed from evaluation as outliers		
* Density measurements belonging to minor, unmodeled lithologies removed from density calculation.		
*Overburden given a specific gravity of 2.0		

14.3.3 Variography

The spatial distribution (continuity) of gold within each mineralized zone was evaluated using variograms.

Primary directions of the variograms were modeled based on observed mineralization orientations. These orientations were then examined statistically to ensure they represented the best possible fit with the variography. The stability of the variograms was evaluated by varying the direction specification and comparing the resulting experimental variograms. Variograms were modelled with two spherical structures, three rotations to match the strike, dip and plunge of the mineralized zone. Nugget effects range from 10% in Domain 30 and 32 to 20% in Domain 10, 11, 20, and 31.

14.3.4 Grade Estimation

A block size of 4 m x 4 m x 4 m was selected based on drill hole spacing, composite length, the geometry of the modelled mineralized zones, and anticipated mining methods. Sub-cells down to 0.5 m x 0.5 m x 0.5 m



were used to better reflect the shape and volume of the mineralized domains. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized domains using a four-pass approach, although estimated blocks within only the fourth search neighborhood are not classified. Search neighborhoods are based on variography, with the range doubling from search pass 1 to search pass 2, multiplying by 2.5 from search pass 2 to search pass 3 and doubling from 3 to 4. Datamine's dynamic anisotropy function was implemented due to the anisotropic nature and widths of the mineralized domains.

Data selection for mineralized domains in passes one and two used a minimum of nine and a maximum of 16 composites from at least three drill holes to estimate a given block. Data selection for mineralized domains in pass three and four used a minimum of five and a maximum of 12 composites from at least two drill holes to estimate a given block. Blocks within the background envelope were estimated using inverse distance to the power of three with its ellipsoid guided by the general lithology contact orientation determined through variography. Any blocks that were not estimated in one of the three passes were assigned a background gold value of one quarter of detection limit, or 0.0005 g/t.

The search parameters are presented in Table 14-4 below.

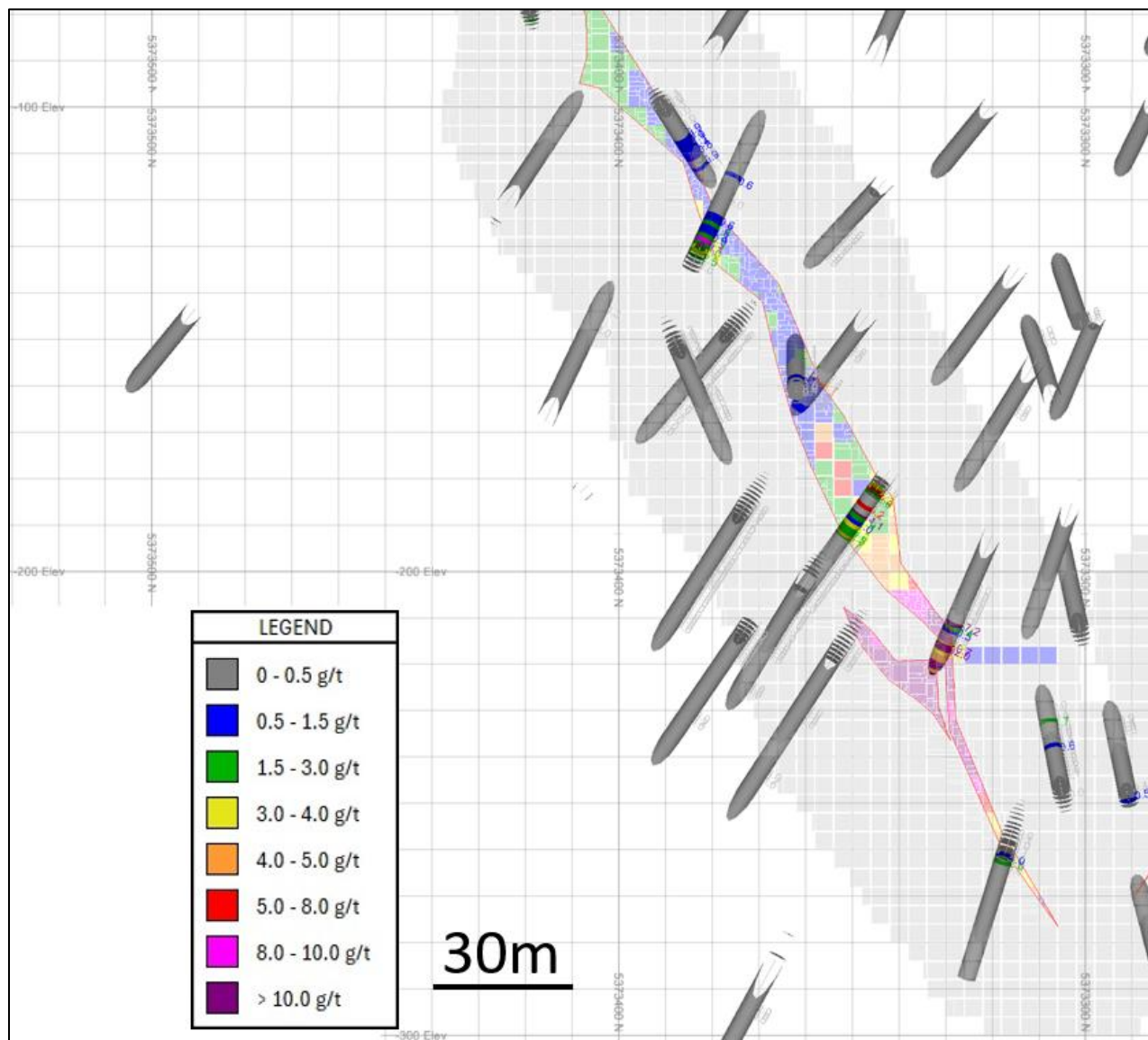
Table 14-4: Grey Fox Estimation Search Parameters

Domain	Orientation Rotation			Search Volume 1			Search Volume 2			Search Volume 3			Search Volume 4 Distances					
	Z	X	Z	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	Min Composites	Max Composites	Max Composites per Drillhole
Dynamic Anisotropic Search Volume																		
10			120	17	13	9	34	26	18	85	65	45	170	130	90	9	Search Vol 1&2	
11			150	11	15	12	22	30	24	55	75	60	110	150	120		16	
20			90	17	28	6	34	56	12	85	140	30	170	280	60		4	
30	Dip Direction	Dip	90	19	11	14	38	22	28	95	55	70	190	110	140	5	Search Vol 3&4	
31			80	11	10	6	22	20	12	55	50	30	110	100	60		12	
32			100	13	7	2	26	14	4	65	35	10	130	70	20		4	
100	-170	80	150	39	54	32	78	108	64							5	12	4

14.3.1 Model Validation

The estimated block model was validated by visually comparing drill hole data with block grades and by reviewing swath plots to determine whether any local bias exists. Figure 14-3 shows a section view looking east for the Whiskey Jack Zone, and Figure 14-4 shows a section view looking northeast for Gibson Zone North, both comparing block grades with drill hole data using ± 25 m clipping. These analyses indicate that the block model reasonably reflects the assay sample data, with only minor local bias in areas of sparse data.

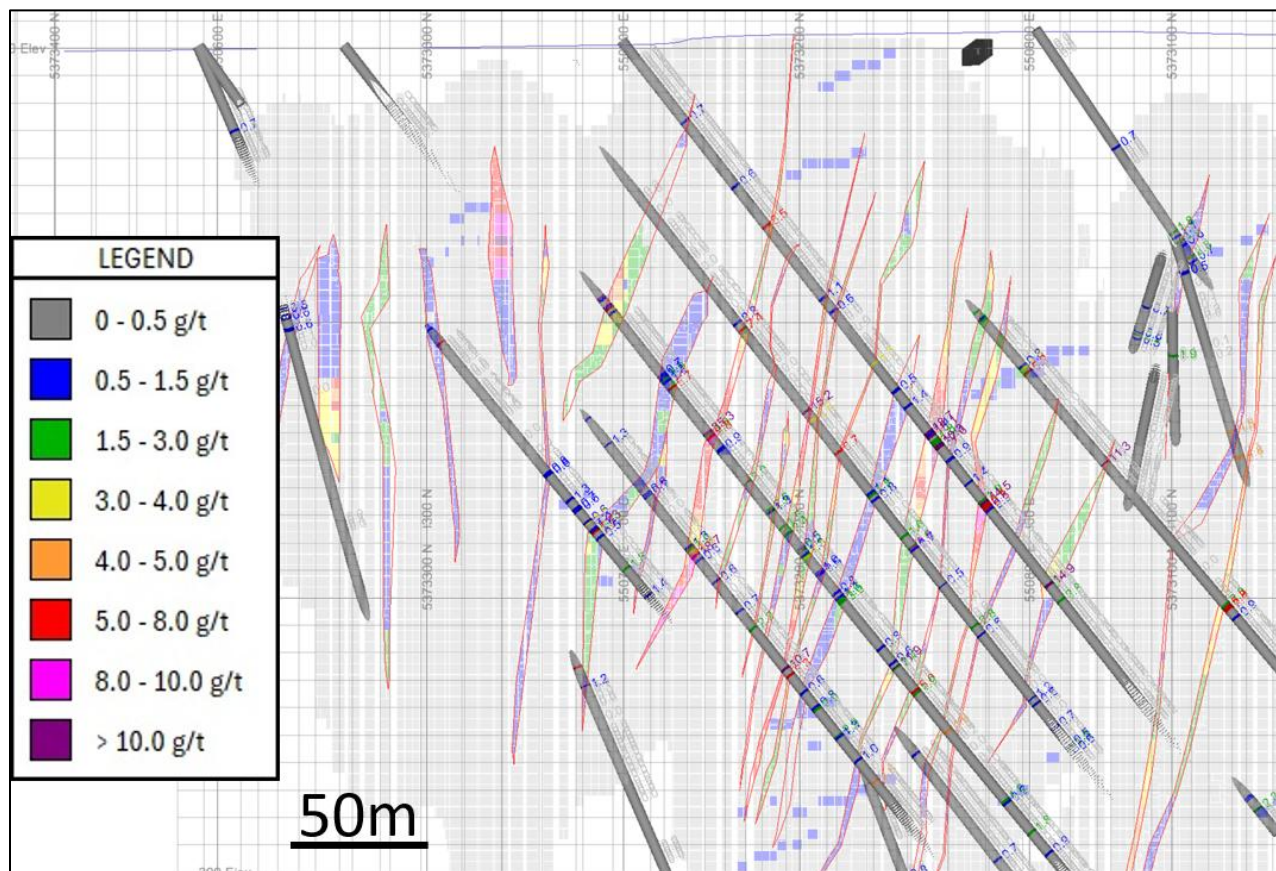




Source: (McEwen, 2025)

Note: Looking east, ± 25 m clipping

Figure 14-3: Section View Comparing Block Grades and Drill Hole Data, Whiskey Jack Zone, Grey Fox Model



Source: (McEwen, 2025)

Note: Looking northeast, ± 25 m clipping

Figure 14-4: Section View Comparing Block Grades and Drill Hole Data, Gibson Zone N, Grey Fox Model

14.3.2 Confidence Classification

The QP is confident that the Grey Fox mineralization model honours the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The Mineral Resource model is constrained by mineralized domains based on lithological, structural and grade criteria and is modelled from core holes drilled on an irregular grid, with a spacing between 5 and 50 m. The controls on the distribution of the gold mineralization are understood, and the confidence in geological continuity is reasonable.

Classification criteria considers the confidence in the continuity of mineralization, average distance to informing samples (weighted on distance), and continuity of drill hole spacing. The following classification criteria were defined:

- **Indicated:** Assigned to those continuous blocks within the mineralized domains and within a weighted average informing sample distance of 40 m. The estimates had to be informed by, and



the sample spacing had to be continuous between, three or more drill holes and estimated within the first three search passes.

- **Inferred:** Assigned to those continuous blocks within the mineralized domains informed by at least two drill holes and within a weighted average informing sample distance of 60 m and estimated within the first three search passes.

Blocks within the mineralized zones not meeting the above criteria were not classified. Blocks within the background envelope were not classified.

14.3.3 Reasonable Prospects for Eventual Economic Extraction

To report a Mineral Resource in accordance with the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards, 2014), the Mineral Resource statement must demonstrate RPEEE.

Due to the narrow vein nature, geometry, and location of the Grey Fox mineralization, the QP considers the deposit to be amenable to underground mining methods. A cut-off grade of 1.35 g/t gold was determined considering the costs and input parameters (Table 14-5) based on recent quotations and benchmarked with similar projects.

Table 14-5: Grey Fox Mineral Resource Cut-off Grade Parameters

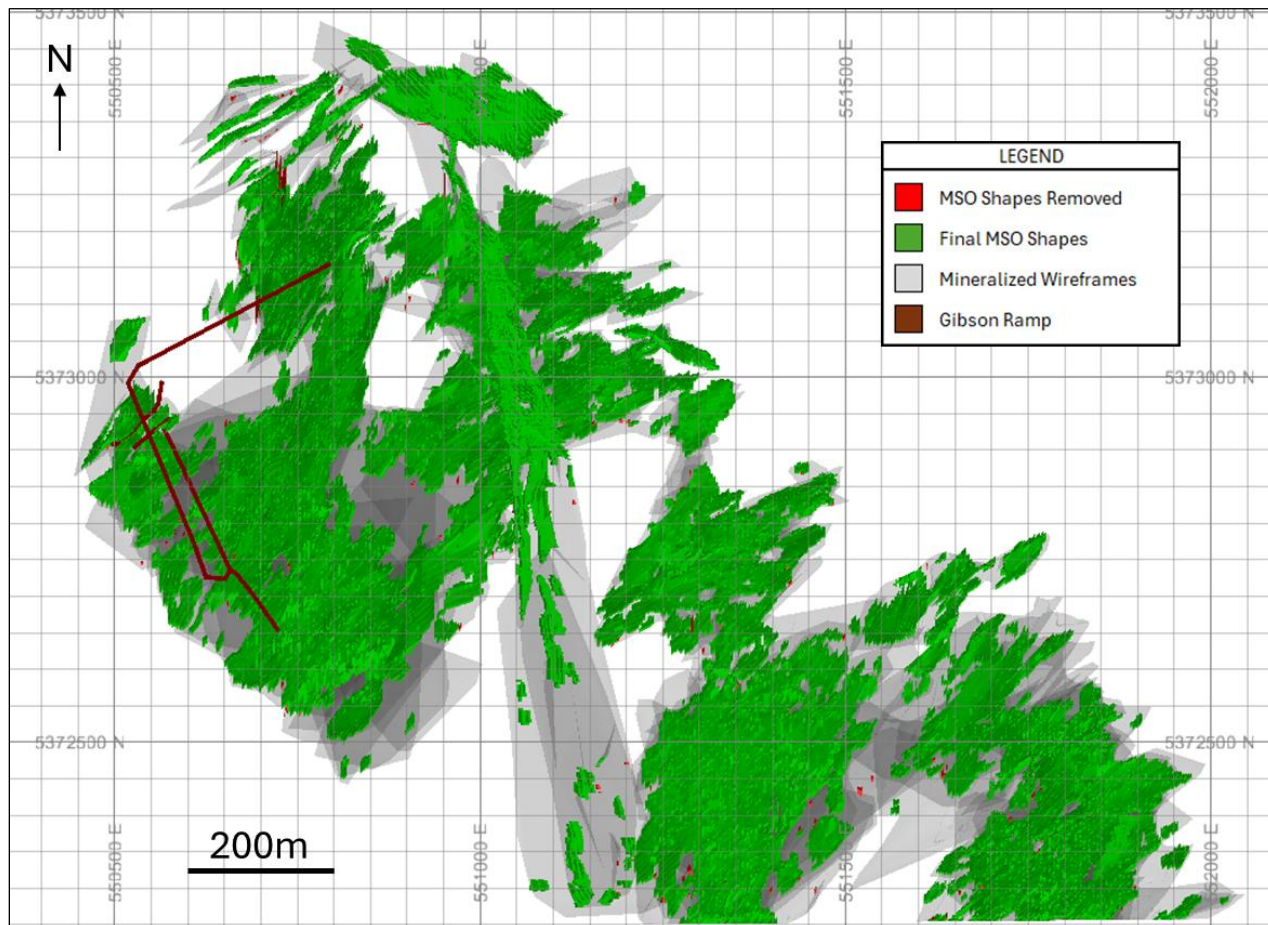
Parameter	Unit	Value
Mining Cost	\$US/t	67.72
G&A Cost	\$US/t	16.60
Milling Cost	\$US/t	30.77
Transport Cost	\$US/oz	1.52
Payable Gold	%	99.95
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Royalty (NSR)	%	*
Mill Recovery	%	90

Note: *Royalty is variable from 0.15% NSR to 3% NSR, with a calculated cut-off grade varying from 1.33g/t to 1.37g/t, ultimately a rounded 1.35 g/t cut-off grade was used for all of Grey Fox.

Reasonable prospects for eventual economic extraction were derived from potentially mineable shapes developed using Datamine's Mineable Shape Optimizer function. A stope size of 12 m vertical height by 6 m wide along strike by a minimum 3 m wide across strike was adopted. This size was benchmarked against mined stopes at the nearby Black Fox underground mine. Sub stopes were developed at half height and half width along strike (6 m x 3 m) to account for local anisotropy of the grade continuity. Stopes with a width less than 3 m across strike were not considered potentially economical and excluded. Stope shapes were developed using the economic cut-off-grade of 1.35 g/t Au. All classified blocks were analyzed by the mineable shape optimizer with resulting stopes shown in Figure 14-5.



All classified blocks were flagged by the potentially mineable shapes including those below the 1.35 g/t Au cut-off grade, this is known as “must take material” and as a combined mineable shape they meet the necessary parameters, including the cut-off grade. Isolated, discontinuous stope shapes were not considered economic and were therefore removed and not included in the Mineral Resource Statement.

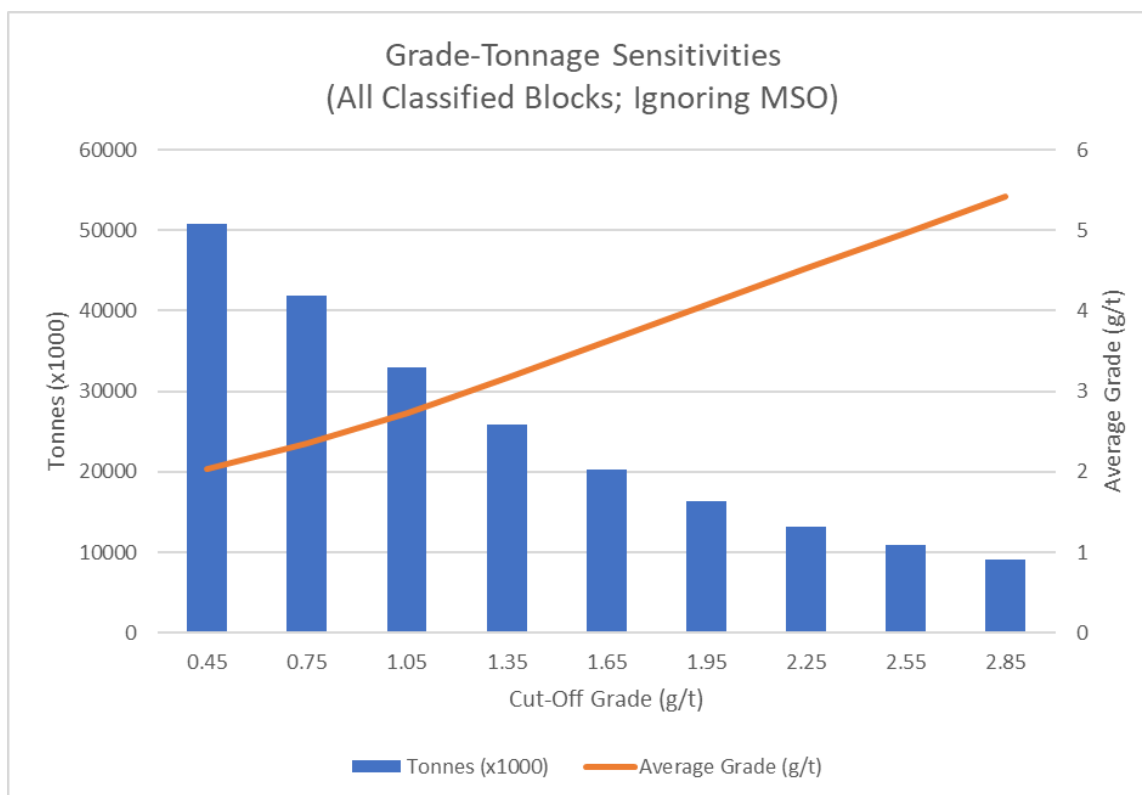


Source: (McEwen, 2025)

Figure 14-5: Plan View of the Potentially Mineable Stope Shapes at Grey Fox

Figure 14-6 presents the experimental grade–tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include both Indicated and Inferred classified blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.





Source: (McEwen, 2026)

Note: Sensitivities use raw block grades and include Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-6: Experimental Grade-Tonnage Sensitivities at Various Cut-Off Grades

14.3.4 Mineral Resource Statement

Table 14-6 summarizes the Mineral Resource estimates for Grey Fox, assuming underground stoping methods, reported in accordance with the CIM Definition Standards (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are reasonable prospects for eventual economic extraction via underground stope extraction. This Mineral Resource Statement presents the mineral resource as exclusive of Reserves reported in Section 15, which was generated by processing the RPEEE after Reserve shapes were removed.

Table 14-6: Grey Fox Mineral Resource Statement, 11 July 2025, **exclusive of Reserves**

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Indicated	9,681	2.25	701
Inferred	4,696	2.57	388

Notes:

- Effective date of the Mineral Resource estimate is 11 July 2025. The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources are reported using the 2014 CIM Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.



- Resources are reported exclusive of the reserves reported in Section 15
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t gold assuming underground extraction methods and based on a mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, metallurgical recovery of 90%, variable royalty NSR from 0.15% to 3.0%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.

Table 14-7 summarizes the Grey Fox resource as inclusive of Reserves reported in Section 15.

*Table 14-7: Grey Fox Mineral Resource Statement, 11 July 2025, **inclusive** of Reserves*

Classification	Tonnes, kt	Au Grade, g/t	Contained Au, koz
Indicated	18,819	3.28	1,986
Inferred	5,056	2.69	438

Notes:

- Effective date of the Mineral Resource estimate is 11 July 2025. The QP for the estimate is Mr Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources are reported using the 2014 CIM Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Indicated resources are reported inclusive of the reserves reported in Section 15.
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t gold assuming underground extraction methods and based on a mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, metallurgical recovery of 90%, variable royalty NSR from 0.15% to 3.0%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.

14.3.5 Factors That Could Affect the Mineral Resource Estimate

Factors that may affect the Grey Fox Mineral Resource estimate include:

- Changes in the local geological interpretations and assumptions used to generate the estimation domains.
- Changes in mineralization and geological geometry and continuity of mineralized zones.
- Changes in assumptions of mineralization and grade continuity.
- Changes in the treatment of high-grade gold values.
- Changes in the grade interpolation methods and estimation parameter assumptions.
- Changes in the confidence assumptions and methods used in the mineral resource classification.
- Changes in the density and the methods used in the density assignments.
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination.
- Changes in input and design parameter assumptions that pertain to the underground mining constraints.
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this report.



14.4 Stroud

14.4.1 Mineral Resource Database

In March 2026, McEwen Inc. developed the initial Stroud mineral resource model. This model incorporates historic drilling results with follow up drilling completed by McEwen from the late 2025 and Q1 2026 confirmation drill campaign. The resource database includes 149 historic holes drilled from 1974 to 2007 as well as 31 holes drilled in Q4 2025 and Q1 2026 to confirm the mineralization described and captured in the historic database. The current drill hole database was closed on March 26, 2026 and consists of 180 holes, comprised of 36,486 m of drill core and 27,201 intervals (25,932 m) assayed for gold.

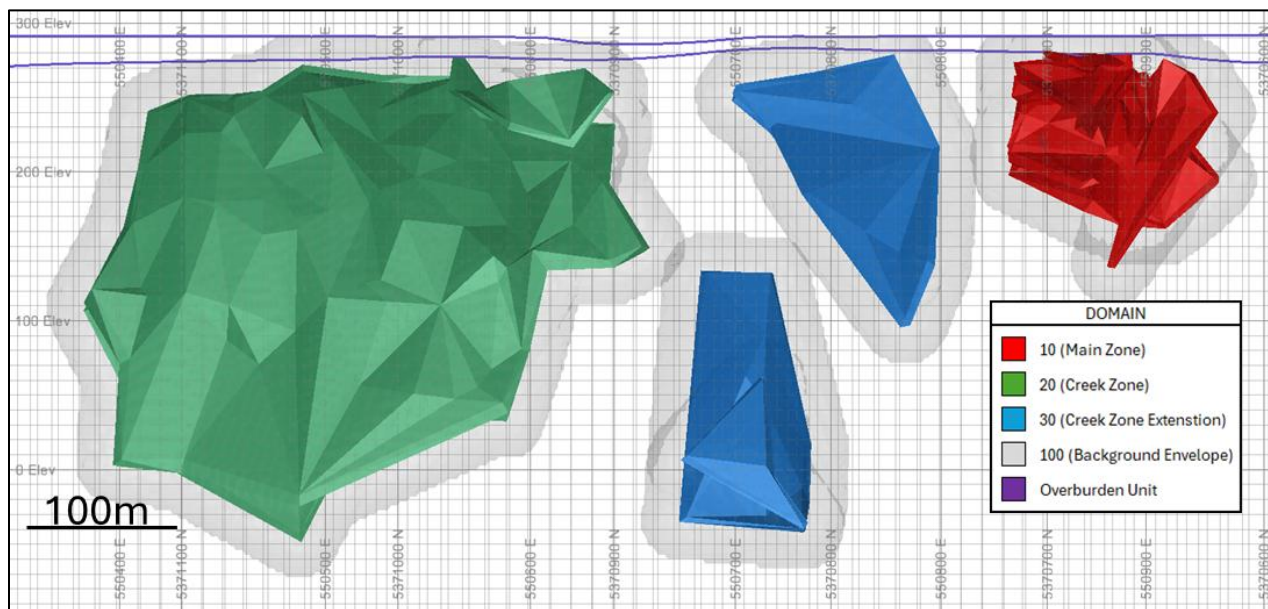
14.4.2 Geological Modelling

The Stroud deposit occurs proximal to the Hislop fault, mostly within sheared, altered mafic metavolcanic rocks, but also within felsic intrusives, and to a lesser extent metasediments, volcanic tuffs and ultramafic intrusives. These units, along with the mineralization, are crosscut by younger diabase dykes. The mineralization was explicitly modelled by members of the McEwen Exploration team and the QP using Datamine Studio. Wireframes were snapped to drill intercepts, using approximately 0.5 g/t as a general cut-off for mineralization. Core logs, photos, assays, and previously modeled wireframes were used as a guide to orient the wireframes. The mineralization was divided into three domains based on geostatistics, geology, and geographic areas. All three domains show similar geology in that they are predominantly composed of altered (sericite, carbonate, +/-hematite) metavolcanic mafics within a shear zone:

- Domain 10, historically the “Main Zone,” is composed of sheared, stacked veins within altered mafic metavolcanics, and also contains felsic porphyries and metasediments. This domain strikes at ~135° with a variable dip of ~35°-60°.
- Domain 20, historically the “Creek Zone,” is composed of sheared, stacked veins within altered mafic metavolcanic, and also contains felsic porphyries. This domain strikes at ~130° with a ~65° dip.
- Domain 30, historically the “Creek Zone Extension,” is composed of sheared, stacked veins within altered mafic metavolcanics, and also contains felsic porphyries. This domain has a more variable orientation with the strike at ~135°-150° with a ~55° dip.

A longitudinal view of the different domains present at Stroud are shown in Figure 14-7.





Source: (McEwen, 2026)

Figure 14-7: Longitudinal View of the Mineralized Domains at Stroud (looking NE)

14.4.3 Composites

Over 97% of all samples have core lengths between 0.5 m and 1.5 m. The modal composite length of 1 m was applied to samples within the mineralized zones which generated composites close to 1 m with a minimum composite length of 0.3 m and a maximum of 1.5 m. A composite length of 3 m was applied to samples within the background envelope (dilution) based on the parent cell size of the block model.

14.4.4 Capping

The impact of high-grade outlier populations was analyzed using histograms, log probability plots, mean and variance, and cumulative metal plots. Results are presented in Table 14-8 for each mineralized domain, with metal removed ranging from 4.2% in Domain 30, to 18.4% in Domain 20. The capping values derived from this analysis were applied to drill hole sample composites.

Table 14-8: Stroud – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
10	2.56	2.21	27.0	2.40	1.85	16	6.4
20	2.63	4.54	25.0	2.14	1.32	3	18.4
30	2.45	1.12	8.5	2.35	1.04	6	4.2

Note: CV = coefficient of variation



14.4.5 Density

As no certified density measurements are in the historical data base, a value of 2.65 g/cm³, reported in previous estimation reports is considered as representative for the mineralized rock types and used in all tonnage calculations. As density measurements are taken from further additional drilling, this will be reevaluated and combined with the lithological model. This default value of 2.65 g/cm³ is considered adequate for Inferred classification but represents a potential source of uncertainty. Further testwork is underway and a density study will be conducted to create and refine the density model.

14.4.6 Variography

The spatial distribution (continuity) of gold within each mineralized domain was evaluated using variograms.

Primary directions of the variograms were modelled based on observed mineralization orientations. The stability of the variograms were evaluated by varying the direction and comparing the resulting experimental variograms. The developed variograms displayed two spherical structures and three rotations to match the strike, dip and plunge of the modelled mineralization. Nugget effects are 20% for all Domains. Variography and Kriging Neighbourhood Analysis were conducted on Datamine's Supervisor software and were used to calculate search ellipse dimensions.

14.4.7 Grade Estimation

A block size of 3 m x 3 m x 3 m was selected based on drill hole spacing, composite length, the geometry of the modelled mineralized zones, and anticipated mining methods. Sub-cells down to 0.5 m x 0.5 m x 0.5 m were used to better reflect the shape and volume of the mineralized domains. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized domains using a three-pass approach. Search neighbourhoods are based on variography with ranges doubling in length with each pass. Datamine's dynamic anisotropy function was implemented due to the anisotropic nature and widths of the mineralized domains. This function aligns the search ellipse with the local dip and dip direction with the plunge defined as per the variography direction.

Data selection for all mineralized domains remained the same for all passes using a minimum of nine and a maximum of 16 composites from at least three drill holes to estimate a given block in the first three search passes. Data selection for the fourth search pass used a minimum of five and a maximum of 12 composites from at least two drill holes to estimate a given block, ultimately no blocks within the fourth search pass were classified.

Blocks within the background dilution envelope were estimated using ordinary kriging with its ellipsoid guided by dimensions determined through variography without the use of dynamic anisotropy. Any blocks that were not estimated in one of the four passes were assigned a background gold value of ¼ of detection limit, or 0.0005 g/t.



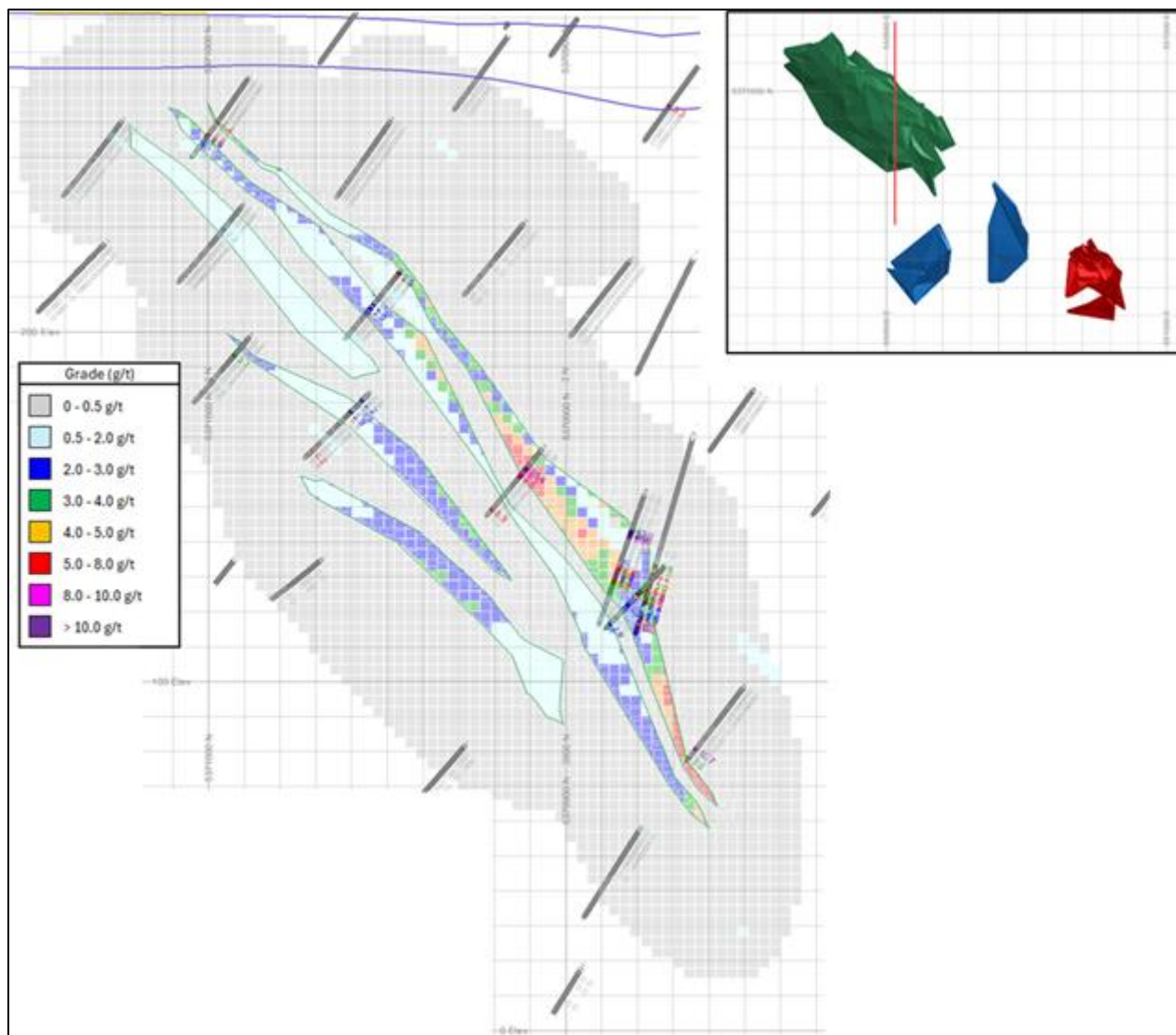
14.4.8 Model Validation

The estimated block model was validated by visually comparing the drill holes with the block grades (Figure 14-8 and Figure 14-9), with swath plots to determine whether any statistical local bias exists, and a nearest neighbour estimate versus the kriged estimate to determine if any global bias exists. A histogram comparison of the estimated grades versus the composite grades was also used to investigate grade smoothing. These analyses indicate the block model acceptably reflects the assay sample data with only minor local bias in areas of sparse data. There is a minor amount of local smoothing of the higher grades; however, the grade distribution of the samples is reasonably reproduced by the estimate.



Source: (McEwen, 2026)

Figure 14-8: Plan Section of Estimated Blocks and Informing Drill Holes at Stroud, Domains 20 and 10



Source: (McEwen, 2026)

Note: Looking east (550515E); ± 15 m clipping

Figure 14-9: Vertical Section of Estimated Blocks and Informing Drill Holes in Domain 20 at Stroud

14.4.9 Classification

The QP is confident that the Stroud mineralization model honours the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The Mineral Resource model is constrained by mineralized domains based on lithological, structural and grade criteria and is modelled from core holes drilled on an irregular grid. The controls on the distribution of the gold mineralization are understood, and the confidence in geological continuity is reasonable.

Classification criteria considers the confidence in the continuity of mineralization, average distance to informing samples, distance to the closest informing sample and continuity of drill hole spacing. The following classification criteria were defined:

- **Measured:** No blocks were assigned the measured category. The QP requires the geological model and grade interpolation to be validated with either 10 m spaced definition drilling or physical mapping from surface or underground to assign a measured classification.
- **Indicated:** Until more robust data validation on historical data can be performed and density data can be confirmed, no blocks were assigned to the indicated category.
- **Inferred:** Assigned to those continuous blocks within the mineralized domains, informed by three or more drill holes, less than 30 m to the nearest sample, and within an average drill hole spacing of 50 m.

14.4.10 Reasonable Prospects for Eventual Economic Extraction

To report a Mineral Resource in accordance with the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards, 2014), the Mineral Resource statement must demonstrate RPEEE.

Due to the narrow vein nature, geometry, and location of the Stroud mineralization, the QP considers the deposit to be amenable to underground mining methods. A cut-off grade of 1.35 g/t Au was determined considering the costs and input parameters (Table 14-9) benchmarked with similar projects, in particular the nearby Grey Fox deposit.

Table 14-9: Stroud Mineral Resource Cut-off Grade Parameters

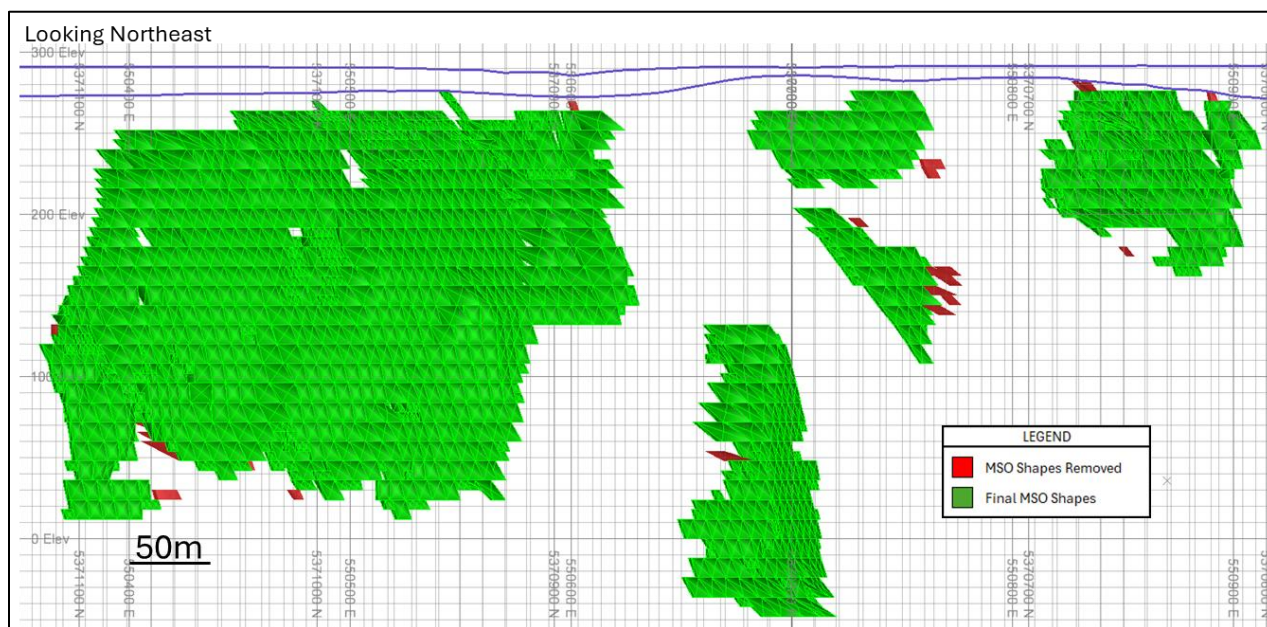
Parameter	Unit	Value
Mining Cost	\$US/t	67.72
G&A Cost	\$US/t	16.60
Milling Cost	\$US/t	30.77
Transport Cost	\$US/oz	1.52
Payable Gold	%	99.95
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Royalty (NSR)	%	0
Mill Recovery	%	90

RPEEEs were derived from potentially mineable shapes developed using Datamine's Mineable Shape Optimizer function. A stope size of 12 m vertical height by 6 m wide along strike by a minimum 3 m wide across strike was adopted. This size was benchmarked against mined stopes at the nearby Black Fox underground mine. Sub stopes were developed at half height and half width along strike (6 m x 3 m) to account for local anisotropy of the grade continuity. Stopes with a width less than 3 m across strike were not considered potentially economical and excluded. Stope shapes were developed using the economic



cut-off-grade of 1.35 g/t Au. All classified blocks were analyzed by the mineable shape optimizer with resulting stopes shown in Figure 14-10. All classified blocks were flagged by the potentially mineable shapes including those below the 1.35 g/t Au cut-off grade, this is known as “must take material” and as a combined mineable shape they meet the necessary parameters, including the cut-off grade.

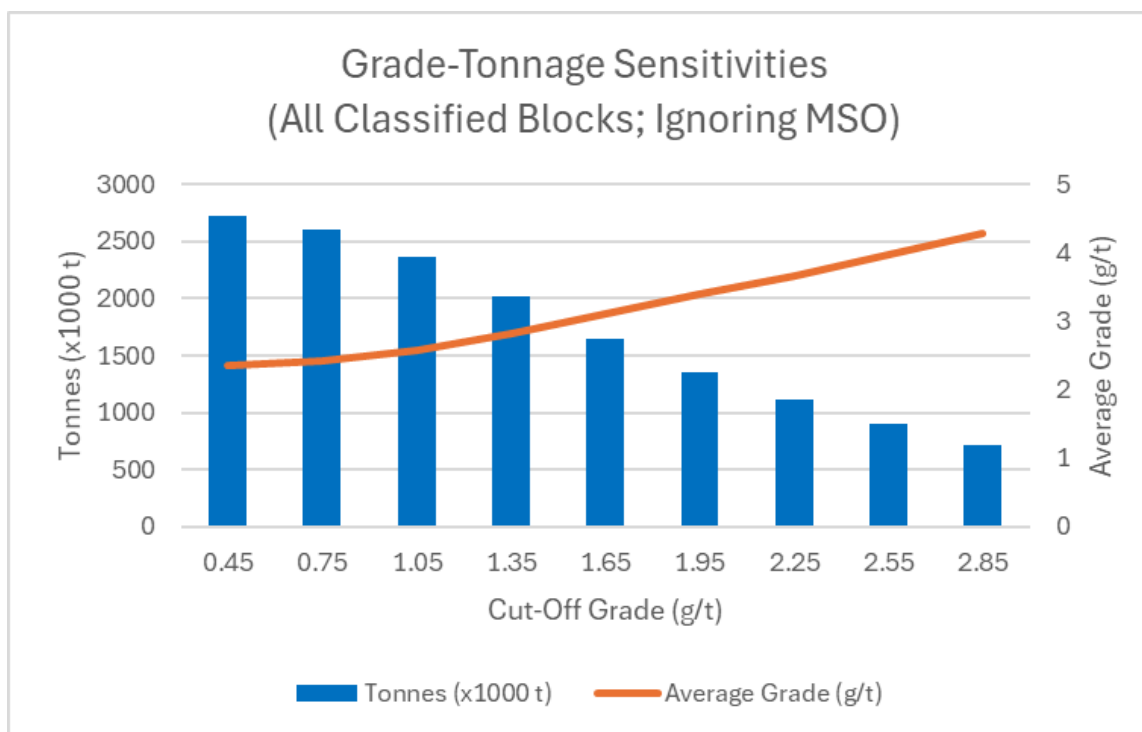
Isolated, discontinuous stope shapes were not considered economic, and were therefore removed and not included in the Mineral Resource Statement.



Source: (McEwen, 2026)

Figure 14-10: Long Section NE View of the Generated Stroud MSO Shapes

Figure 14-11 presents the experimental grade-tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.



Source: (McEwen, 2026)

Note: Sensitivities use raw cut-off grades and include all Inferred blocks. This is for comparison to show what potential effect a change in cut-off grade would have and does not represent Mineral Resources.

Figure 14-11: Experimental Sensitivities at Various Cut-Off Grades

14.4.11 Mineral Resource Statement

Table 14-10 summarizes the Mineral Resource estimates for Stroud, assuming underground stoping methods, reported in accordance with the 2014 CIM Definition Standards (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are reasonable prospects for eventual economic extraction via underground stope extraction.

Table 14-10: Stroud Mineral Resource Statement, March 26, 2026

Classification	Tonnes, kt	Au Grade, g/t	Contained Au, koz
Indicated	0	0.00	0
Inferred	1,866	2.80	168

Note:

- Effective date of the Mineral Resource estimate is March 26, 2026. The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources are reported using the 2014 CIM Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t gold assuming underground extraction methods and based on a mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, metallurgical recovery of 90%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.



14.4.12 Factors That Could Affect the Mineral Resource Estimate

Factors that may affect the Stroud Mineral Resource estimate include:

- Changes in the local geological interpretations and assumptions used to generate the estimation domains.
- Changes in mineralization and geological geometry and continuity of mineralized zones.
- Changes in assumptions of mineralization and grade continuity.
- Changes in the treatment of high-grade gold values.
- Changes in the grade interpolation methods and estimation parameter assumptions.
- Changes in the confidence assumptions and methods used in the mineral resource classification
- Changes in the density and the methods used in the density assignments.
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination.
- Changes in input and design parameter assumptions that pertain to the underground mining constraints.
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this report.

14.5 Black Fox

14.5.1 Mineral Resource Database

The close-out date for the Mineral Resource database is 31 December 2025 and consists of the following:

- 7,731 core drill holes totalling 1,097,798 m of drill core and 438,200.1 m of sampled core (144 drill holes were excluded from the drill hole database, primarily due to low confidence in their locations).
- 13,332 strings of chip samples totalling 52,295.3 m from development headings were sampled for a total of 56,884 assays.

Unsampled intervals were considered barren and were assigned a background gold grade of 0.0005 g/t, based on one-quarter of the assay laboratory's lower detection limit.

On importing the drill hole database from the central SQL database to Datamine Studio software in preparation for resource estimation, sample data were checked for the following:

- Missing, duplicated and overlapping intervals in the assay and lithology tables.
- Missing and overlapping intervals in the survey table.
- Visually checking collar locations in 3D against mine excavations and surface topography.

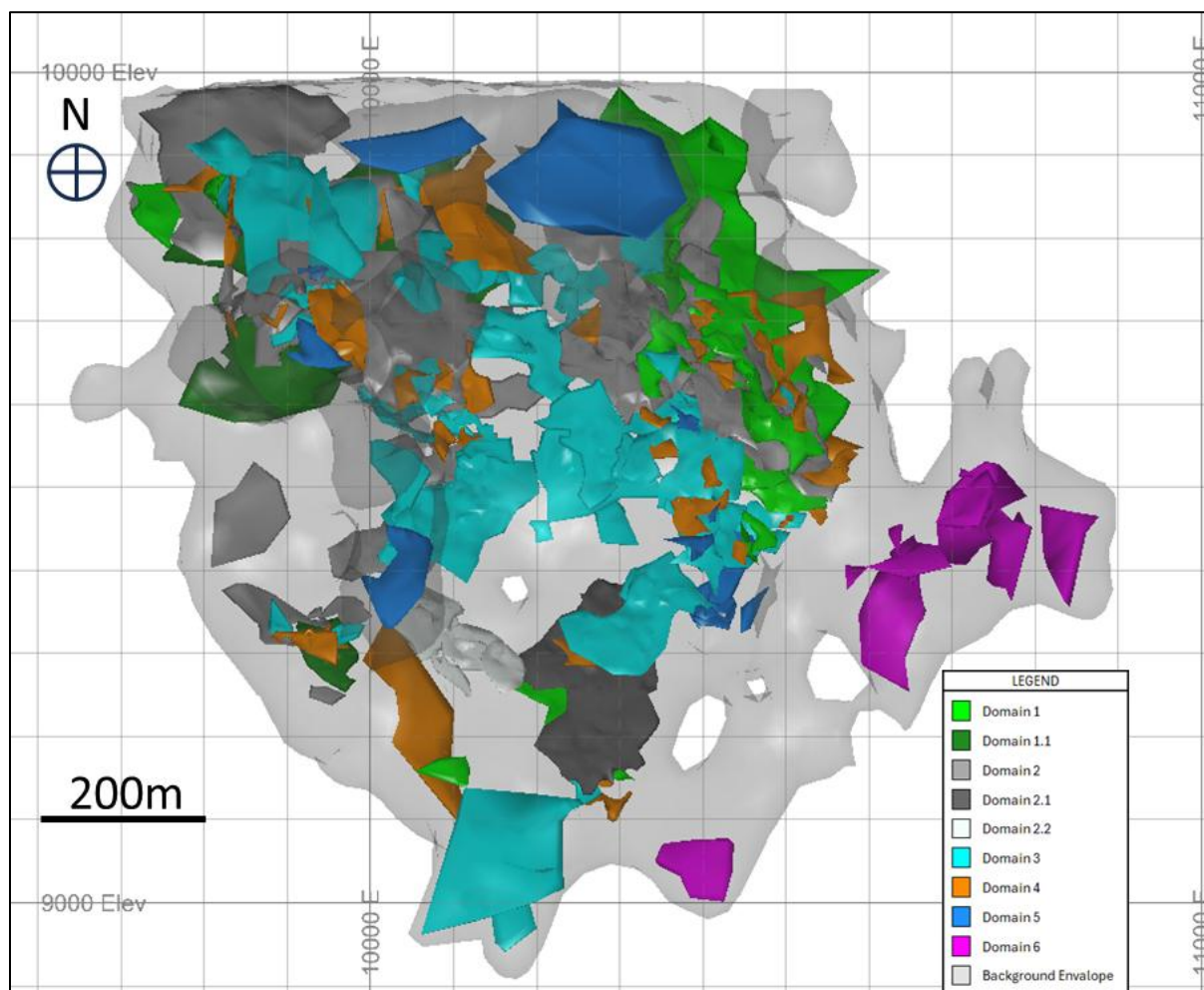


14.5.2 Geological Modelling

14.5.2.1 Mineralized Domains

Gold mineralization at Black Fox occurs in several different geological settings within the main Ankerite Alteration envelope. The mineralization occurs within both ultramafic and mafic volcanic rock types within the outer boundaries of the PDDZ. The mineralization and alteration envelopes are primarily parallel/subparallel and adjacent to the regional A1 fault. The lenses of the mineralization have been developed within the main lithological and alteration units and primarily occur in vein arrays, shear veins, and replacement zones within the interleaved mafic and ultramafic host rocks.

The mineralized lenses were interpreted and explicitly modelled considering structural and lithological controls, and a grade threshold of approximately 0.5 g/t Au to demonstrate mineralized continuity. The mineralized lenses were divided into nine mineralized domains (Figure 14-12) based primarily on mineralization style, grade population statistics and primary host lithologies.



Source: (McEwen, 2025)

Figure 14-12: Long Section Looking North of Black Fox Mineralized Domains



14.5.2.2 Background Envelope

A background envelope (Domain 100) representing a low-grade alteration package surrounding the mineralization, was constructed around all mineralized domains to capture mineralized samples outside the mineralized domains and to estimate adjacent dilution.

14.5.3 Composites

A large percentage of the assays within the mineralized domains have a sample length of 1 m. The modal composite length of 1 m was applied to all samples (drill hole and chip) in the database. This generated composites close to 1 m with a minimum composite length of 0.3 m.

14.5.4 Capping

The impact of high-grade outlier populations was analyzed using histograms, log probability plots, mean and variance analysis, and cumulative metal plots. Results are presented in Table 14-11 for each mineralized domain, with metal removed ranging from 0.6% in Domain 6, to 45.9% in Domain 2. Capping was warranted on all mineralized domains except for Domain 5. The capping values derived from this analysis were applied to both drill hole and chip sample composites.

Table 14-11: Black Fox – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
1	3.86	4.00	52.0	3.53	2.05	70	8.4
1.1	2.14	4.45	20.0	1.64	1.99	40	23.3
2	6.43	6.45	28.0	3.48	2.00	522	45.9
2.1	8.83	3.46	140.0	7.86	2.73	38	11.0
2.2	2.58	4.57	16.5	1.63	2.00	19	36.9
3	4.37	5.53	30.0	2.93	2.11	259	32.9
4	1.99	3.63	29.0	1.73	2.03	40	13.3
5	1.46	1.42	-	1.46	1.42	-	-
6	0.90	2.20	33.0	0.90	2.09	3	0.6
100 (background)	0.26	28.5	15.0	0.16	5.44	681	36.1
100 (background)	0.26	28.5	30.0	0.18	6.95	313	28.5

Note: CV = coefficient of variation

14.5.5 Density

A constant density value of 2.84 g/cm³ was applied to all material. This value has been used in previous estimates and for production tracking. This value has been validated by analyzing the average value of 150



specific gravity measurements, which are restricted to samples within mineralized domains, and taken from drill hole samples since 2018.

14.5.6 Variography

The spatial distribution (continuity) of gold within each mineralized domain was evaluated using variograms. Primary directions of the variograms were modelled based on field observations and interpreted mineralization orientations. The stability of the variograms were evaluated by varying the direction and comparing the resulting experimental variograms. The developed variograms displayed two spherical structures and three rotations to match the strike, dip and plunge of the modelled mineralization. Nugget effects range from 21% in Domain 6 to 33% in Domains 2 and 3.

14.5.7 Grade Estimation

A block size of 3 m x 3 m x 3 m was selected based on drill hole spacing, composite length, the geometry of the modelled mineralized zones, and anticipated mining methods. Sub-cells down to 0.5 m x 0.5 m x 0.5 m were used to better reflect the shape and volume of the mineralized domains. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized domains using a three-pass approach with increasing search neighbourhoods with each pass. Datamine's dynamic anisotropy function was implemented due to the anisotropic nature and widths of the mineralized domains. This function aligns the search ellipse with the local dip and dip direction of the mineralization, with the plunge component defined as per the variography direction.

Data selection for mineralized domains remained the same for the first and second passes using a minimum of 7 and a maximum of 15 composites from at least three drill holes to estimate a given block. The third pass reduced the minimum requirement to four composites from at least two drill holes. The search parameters are presented in Table 14-12.

Table 14-12: Black Fox Estimation Search Parameters

Domain	Orientation Rotation			Search Volume 1 Distances			Search Volume 2 Distances			Search Volume 3 Distances			Number of samples used to estimate			
	Z	X	Z	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	Search Pass	Min Composites	Max Composites	Max Composites per Drillhole
Dynamic Anisotropic Search Volume																
1			160	13	19	4	39	57	12	52	76	16				
1.1			110	31	15	4	93	45	12	124	60	16	1	7	15	3
2			180	18	7	8	54	21	24	72	28	32				
2.1			90	7	4	2	21	12	6	28	16	8	2	7	15	3
2.2	Dip Direction	Dip	-150	9	5	2	27	15	6	36	20	8				
3			150	4	3	2	12	9	6	16	12	8	3	4	15	3
4			-130	12	12	5	36	36	15	48	48	20				
5			-150	16	13	3	48	39	9	64	52	12				
6			-150	10	15	14	30	45	42	40	60	56				
100 0 run	170	60	130	5	5	5							1	1	12	3
100 Main Run	170	60	130	50	50	10	100	100	20	250	250	50	1, 2	4	18	3
													3	3	20	3

Blocks within the background envelope (Domain 100) were estimated using inverse distance to the power of three with its ellipsoid guided by the deposit's general mineralized orientation. An initial estimation pass focused on estimating areas where high-grade composites were not captured in the mineralized domains. This estimation pass used composites capped at 15 g/t gold with a tight isometric search ellipse to limit the



number of blocks estimated to one or two in each direction. Three subsequent passes using composites capped at 30 g/t were made with increasing search ranges and any blocks not estimated were assigned a background gold value of one-quarter of the detection limit, or 0.0005 g/t.

14.5.8 Model Validation

In validating the estimated block model, nearest neighbour and inverse distance models were constructed. The nearest neighbour model produces a theoretically globally unbiased estimate of the average value when no cut-off grade is applied and is a good basis for checking the performance of different estimation methods.

The estimated block model was validated by visual inspection, with swath plots to determine whether any local bias exists, and with local and global reconciliations using randomly selected stopes in mined out areas.

Results of the validation show the estimated model acceptably reflects the sample data and past production results.

14.5.9 Confidence Classification

The QP is confident that the Black Fox mineralization model honours the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The Mineral Resource model is constrained by mineralized domains based on lithological, structural and grade criteria and is modelled from core holes drilled on an irregular grid. The controls on the distribution of the gold mineralization are understood, and the confidence in geological continuity is reasonable.

Block classification scenarios were analyzed using a combination of criteria, including confidence in the mineralization's continuity, drill spacing, sample spacing (in the form of average distance to informing samples - weighted on kriging weights), and the number of drill holes used to estimate. The QP considers the average distance at approximately one-third the modelled variogram (10 m) to adequately represent the local mineralization grades considering the amount of informing data used in the estimate. The QP also considers the estimated grades, using a search distance equal to the average full sill ranges of the modelled variograms (30 m), to adequately represent the global mineralization grades.

The following classification criteria were defined:

- **Measured:** Assigned to those continuous blocks within the mineralized domains, estimated within 10 m of informing composites from at least three drill holes.
- **Indicated:** Assigned to those continuous blocks within the mineralized domains, estimated within 10 m of informing composites from at least two drill holes.
- **Inferred:** Assigned to those continuous blocks within the mineralized domains estimated within 30 m of informing composites from at least two drill holes.

Blocks within the mineralized zones not meeting the above criteria were not classified. Blocks within the background envelope were not classified.



14.5.10 Reasonable Prospects for Eventual Economic Extraction

To report a Mineral Resource in accordance with the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards, 2014), the Mineral Resource statement must demonstrate RPEEE.

Due to the narrow vein nature, geometry, and location of the Black Fox mineralization, the QP considers the deposit to be amenable to underground mining methods. A cut-off grade of 1.35 g/t Au was determined considering the costs and input parameters (Table 14-13) based on recent quotations and benchmarked with similar projects.

Table 14-13: Black Fox Mineral Resource Cut-off Grade Parameters

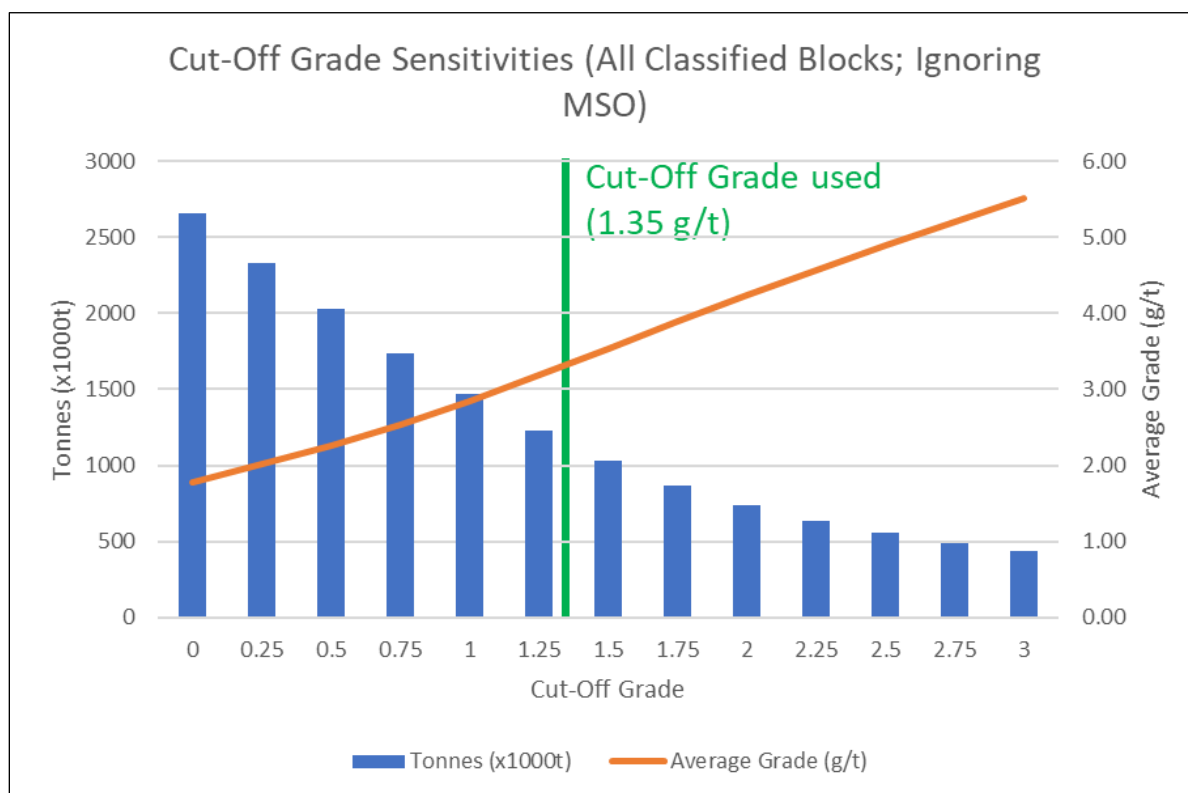
Parameter	Unit	Value
Mining Cost	\$US/t	62.66
G&A Cost	\$US/t	20.49
Milling Cost	\$US/t	32.21
Transport Cost	\$US/oz	1.52
Payable Gold	%	99.95
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Royalty (NSR)	\$US/t	7.20
Mill Recovery	%	95

RPEEEs were derived from calculating potentially mineable shapes developed using Datamine's MSO function. A stope size of 12 m vertical height by 6 m wide, along strike by a minimum 3 m wide across strike was adopted. Sub stopes were developed at half height and half width along strike (6 m x 3 m) to account for local anisotropy of the grade continuity. Stope shapes were developed using the economic cut-off-grade of 1.35g/t Au. All classified blocks were analyzed by the mineable shape optimizer.

Isolated, discontinuous stope shapes were not considered economic and were therefore removed and not included in the Mineral Resource Statement. Mineralization below the 300 m level is not considered economic at this time due to the lack of accessibility and the current condition of workings at the deeper levels. All blocks beneath the 300 m level are coded as such and are not included in the Mineral Resource Statement.

Figure 14-13 presents the experimental grade–tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include Measured, Indicated and Inferred blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.





Source: (McEwen, 2026)

Note: Sensitivities use raw block grades and include Measured, Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-13: Experimental Grade-Tonnage Sensitivities at Various Cut-Off Grades

14.5.11 Mineral Resource Statement

Table 14-14 summarizes the Mineral Resource estimates for Black Fox, assuming underground stoping methods, reported in accordance with the 2014 CIM Definition Standards (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are reasonable prospects for eventual economic extraction via underground stope extraction.

Table 14-14: Black Fox Mineral Resource Statement, December 31, 2025

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Measured	244	3.59	28
Indicated	141	3.54	16
Total Measured + Indicated	385	3.57	44
Inferred	385	2.90	36

Note:

- Effective date of the Mineral Resource estimate is 31 December 2025. The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.



- Mineral Resources are reported using the 2014 CIM Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported above an economic cut-off grade 1.35 g/t gold assuming underground extraction methods and based on a mining cost of \$62.66/t, process cost of \$32.21/t, G&A cost of \$20.49/t, metallurgical recovery of 95%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.
- Informing sample database cut-off date is 2 October 2024. Mining depletion date up to and including 31 December 2025.

14.5.12 Factors That Could Affect the Mineral Resource Estimate

Factors that may affect the Black Fox Mineral Resource estimate include:

- Changes in the local geological interpretations and assumptions used to generate the estimation domains
- Changes in mineralization and geological geometry and continuity of mineralized zones
- Changes in assumptions of mineralization and grade continuity
- Changes in the treatment of high-grade gold values
- Changes in the grade interpolation methods and estimation parameter assumptions
- Changes in the confidence assumptions used in the resource classification
- Density assignment
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination
- Changes in input and design parameter assumptions that pertain to the underground mining constraints
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report.

14.6 Froome

14.6.1 Mineral Resource Database

The close-out date for the Mineral Resource database was 31 December 2025 and consisted of 1,112 core drill holes totalling 221,479 m of drill core and 128,304 m of sampled core. The database also includes 1,599 strings of chip samples totalling 9,196 gold assays values from 8,617 m of underground development. Unsourced intervals were considered barren and assigned a background gold grade of 0.0005 g/t, based on one-quarter of the assay laboratory's lower detection limit.

On importing the drill hole and chip databases from the central SQL databases to Datamine Studio software in preparation for resource estimation, sample data were checked for missing and overlapping intervals in the assay, lithology and survey tables, and density values inspected to ensure no erroneous entries were present.



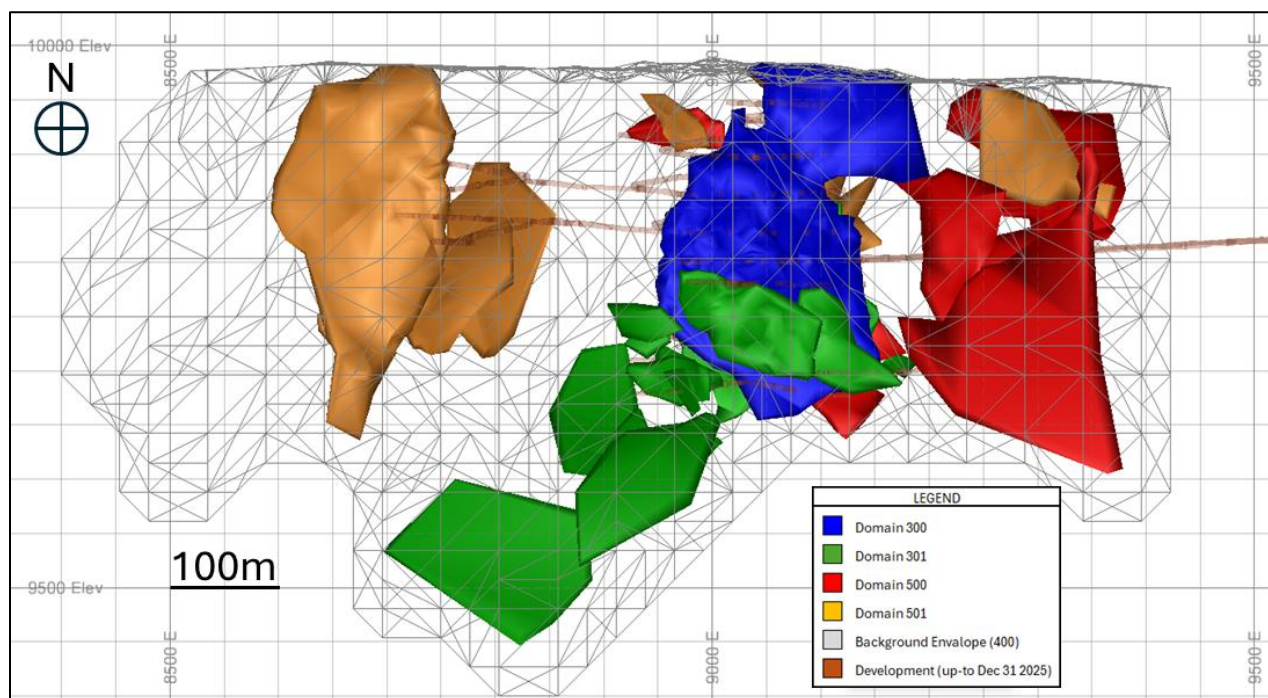
Underground grade control drilling has continued since the close-out date for the Mineral Resource database in areas of proposed mining. Small local geological interpretations have been improved in these areas; however, no material difference is indicated by the results.

14.6.2 Geological Modelling

14.6.2.1 Mineralized Domains

The Froome project area is subdivided into four mineralized domains. There are two grade populations hosted primarily in metasediments and two grade populations hosted in FW mafic and ultramafic metavolcanics. All mineralized domains were created using explicit wireframing techniques around static drill holes. Figure 14-14 shows the location of the mineralized domains.

Domain 300 is comprised of the relatively higher gold grade population hosted in metasediments. A low grade mineralized mafic intrusive dyke (domain 300-MI) running through the core of Domain 300 was modelled separately to reduce the smearing of low-grade samples into the surrounding sediments and also to restrict high-grade samples in the surrounding sediments from the mafic dyke. No major structures were noted as gold mineralization is hosted within disseminated sulphide mineralization. However, grade trends were analyzed and planes representing those trends were created.



Source: (McEwen, 2025)

Figure 14-14: East-West Long Sectional View of Froome Project Mineralized Domains

Domain 301 is disseminated through the metasediments, primarily along the north (FW) contact and consists of a lower gold grade population.



Both domains 500 and 501 are hosted primarily in the mafic and ultramafic metavolcanics in the FW of the metasediment unit. The mineralization of these domains is hosted in fault-filled quartz carbonate veins and breccias typical of Archean lode gold vein structures along the PDDZ. Statistical analysis of the gold grades within the mineralized lenses that make up these domains suggests there are two populations of gold grade. Domain 501 has a higher probability of containing higher-grade mineralization.

Average structural planes through the FW domain lenses were created to determine each lens anisotropy. The bedrock/overburden contact was generated from the drill holes and used to constrain the extent of mineralization.

14.6.2.1.1 Background Envelope

A background envelope (Domain 400) was constructed around all mineralized domains to capture mineralized samples outside the mineralized domains and to estimate adjacent dilution.

14.6.3 Composites

More than 50% of the assays within the mineralized domains have a sample length of 1 m. The modal composite length of 1 m was applied to all samples in the database. This generated composites close to 1 m with a minimum composite length of 0.3 m and a maximum of 1.5 m.

14.6.4 Capping

The impact of high-grade outlier populations was analyzed using histograms, log probability plots, mean and variance, and cumulative metal plots. Results are presented in Table 14-15 with metal removed ranging from 0.4% in Domain 300 to 27.7% in Domain 500.

Table 14-15: Froome – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
300	3.19	1.07	19.0	3.17	1.05	21	0.4
300-MI	2.7	1.11	8.5	2.55	0.99	55	5.4
301	1.23	1.64	10.0	1.19	1.44	43	3.5
500	2.1	7.3	19.0	1.52	1.58	7	27.7
501	2.03	5.72	45.0	1.74	2.73	17	14.3
400	0.07	3.56	2.3	0.06	2.61	92	4.5

Note: CV = coefficient of variation



14.6.5 Density

A bulk density analysis was completed for the previous resource update (dated 31 December 2024) and is still considered valid. In that resource update there were 2,973 density samples used in the Froome Mineral Resource bulk density analysis. Density samples were measured on NQ sized core with lengths between 10 and 15 cm, using the Archimedes' Principle method. A study of all the density data in the resource database was conducted and used to determine an average density value for each of the modelled domains. The average densities for each domain are presented in Table 14-16 and applied to the blocks contained within.

Table 14-16: Froome – Density Values by Mineralized Domain

Domain	Density (g/cm³)
300	2.76
301	2.76
500	2.85
501	2.85
400	2.81

14.6.6 Variography

The spatial distribution (continuity) of gold within each mineralized domain was evaluated using variograms.

Primary directions of the variograms were modelled based on observed mineralization orientations. The stability of the variograms were evaluated by varying the direction and comparing the resulting experimental variograms. The developed variograms displayed two spherical structures and three rotations to match the strike, dip and plunge of the modelled mineralization. Nugget effects range from 0% in Domain 500, and 300-MI to 23% in Domain 301.

14.6.7 Grade Estimation

A block size of 3 m x 3 m x 3 m was selected based on drill hole spacing, composite length, the geometry of the modelled mineralized zones, and anticipated mining methods. Sub-cells down to 0.5 m x 0.5 m x 0.5 m were used to better reflect the shape and volume of the mineralized domains. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized domains using a three-pass approach. Search neighbourhoods are based on variography with ranges doubling in length with each pass. Datamine's dynamic anisotropy function was implemented due to the anisotropic nature and widths of the mineralized domains. This function aligns the search ellipse with the local dip and dip direction with the plunge defined as per the variography direction.

Data selection for mineralized domains 300, 300-MI & 301 remained the same for all passes using a minimum of seven and a maximum of 15 composites from at least three drill holes to estimate a given



block. Data selection for mineralized domains 500 & 501 used a minimum of 5 and a maximum of 15 composites from at least two drill holes to estimate a given block. Blocks within Domains 300-MI and 400 were estimated using ordinary kriging with its ellipsoid guided by the general lithology contact orientation determined through variography without the use of dynamic anisotropy. Any blocks that were not estimated in one of the three passes were assigned a background gold value of one quarter of the detection limit, or 0.0005 g/t.

Estimation search parameters are presented in Table 14-17 below.

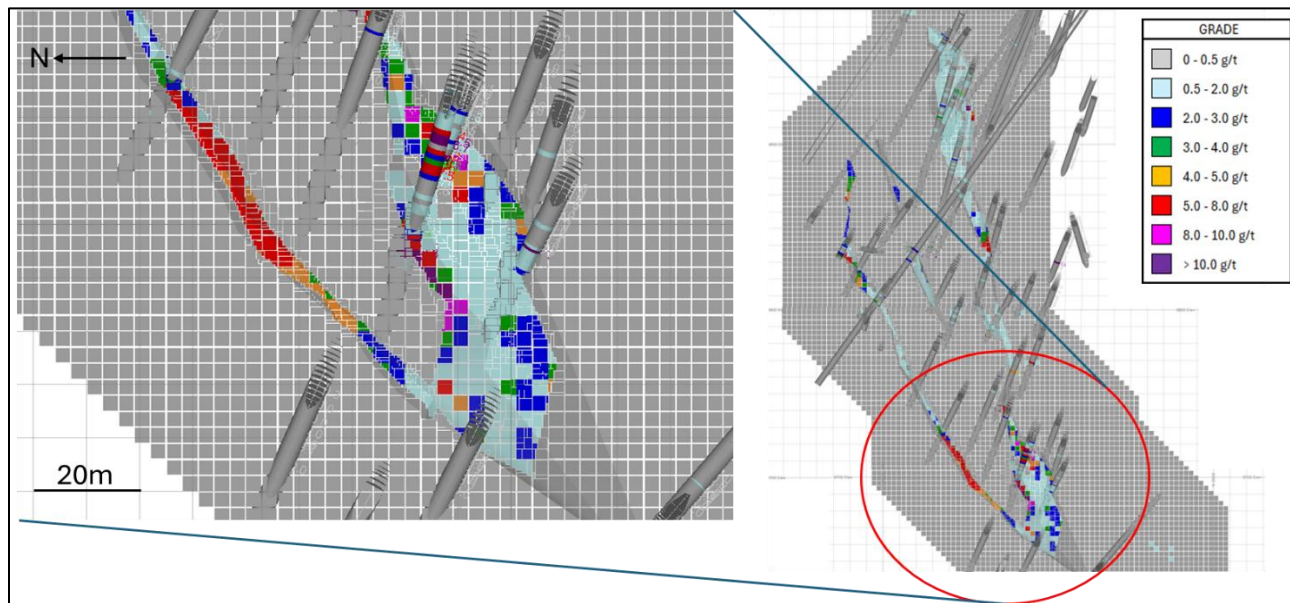
Table 14-17: Froome Estimation Search Parameters

Domain	Orientation Rotation			Search Volume 1			Search Volume 2			Search Volume 3			Search Volume 4 Distances			Number of samples used to Estimate		
	Z	X	Z	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	X(Strike)	Y(Dip)	Z(Perp)	Min Composites	Max Composites	Max Composites per Drillhole
Dynamic Anisotropy																		
300			90	15	10	2.5	30	20	5	60	40	10	300	300	50	7	15	3
301			90	7	10.5	4	14	21	8	28	42	16	300	300	50			
500	TRDIPDIR	TIRDIP	130	7.5	4	2.5	15	8	5	30	16	10	300	300	50			
501			80	6.5	3.5	3	13	7	6	26	14	12	300	300	50			
300-mi	Z	X	Z													7	15	3
	-170	75	110	2.5	9	1	5	18	2	10	36	4				1	12	3
400 O Run	0	0	70	5	5	3										7	15	3
400 Background	0	0	70	20	18	18	40	36	36	60	54	54						

14.6.8 Model Validation

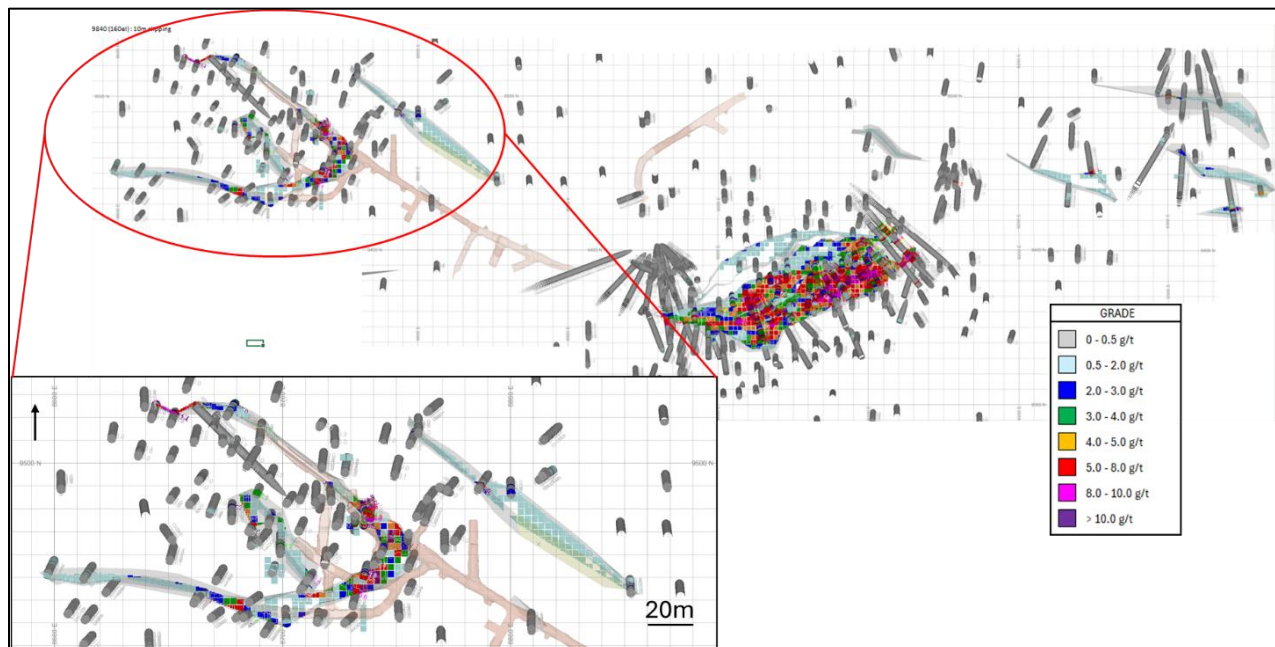
The estimated block model was validated by visually comparing the drill holes with the block grades (Figure 14-15 and Figure 14-16), with swath plots to determine whether any statistical local bias exists, a nearest neighbour estimate versus the kriged estimate to determine if any global bias exists, and a local stope reconciliation. A histogram comparison of the estimated grades versus the composite grades was also used to investigate grade smoothing. These analyses indicate the block model acceptably reflects the assay sample data with only minor local bias in areas of sparse data. There is a minor amount of local smoothing of the higher grades; however, the grade distribution of the samples is reasonably reproduced by the estimate.





Source: (McEwen, 2025)
Note: 10 m clipping, looking east

Figure 14-15: Section View Comparing Block Grades with Drill Hole Data in Froome Domain 501



Source: (McEwen, 2025)
Note: Zoomed in on Froome West Domain 501, 10 m clipping.

Figure 14-16: Plan View at Elevation 9840 Comparing Block Grades with Drill Hole Data



14.6.9 Confidence Classification

The QP is confident that the Froome mineralization model honours the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The Mineral Resource model is constrained by mineralized domains based on lithological, structural and grade criteria and is modelled from underground mapping and core holes drilled on an irregular grid, with an average spacing between drill holes of 10 and 30 m. The controls on the distribution of the gold mineralization are understood, and the confidence in geological continuity is reasonable.

For Domains 300 and 301, classification criteria considers the confidence in the continuity of mineralization, distance to the closest informing sample, and average drill hole spacing based on the three closest drill holes. The following classification criteria were defined:

- **Measured:** Assigned to those continuous blocks within the mineralized Domains 300 and 301, informed by three or more drill holes, less than 7 m to the nearest drill hole, and within an average drill hole spacing of 10 m
- **Indicated:** Assigned to those continuous blocks within the mineralized Domains 300 and 301, informed by three or more drill holes, less than 15 m to the nearest drill hole, and within an average drill hole spacing of 25 m
- **Inferred:** Assigned to those continuous blocks within the mineralized Domains 300 and 301, informed by three or more drill holes, less than 20 m to the nearest drill hole, and within an average drill hole spacing of 40 m.

The geological nature of the gold mineralization in Domains 500 and 501 exhibits higher variability in grade than other domains and therefore was assigned a stricter classification criteria. The following classification criteria were defined:

- **Measured:** Due to lower confidence, no blocks in these domains were assigned to the measured classification (the QP requires an area with 10m spaced drilling or visual mapping to confirm the geological model before considering the measured category).
- **Indicated:** Assigned to those continuous blocks within the mineralized Domains 500 and 501, informed by three or more drill holes, less than 10 m to the nearest drill hole, and within an average drill hole spacing of 25 m.
- **Inferred:** Assigned to those continuous blocks within the mineralized Domains 500 and 501, informed by two or more drill holes, less than 20 m to the nearest drill hole, and within an average drill hole spacing of 40 m.

Blocks within the mineralized domains not meeting the above criteria were not classified. Blocks within the background envelope were not classified.

14.6.10 Reasonable Prospects for Eventual Economic Extraction

To report a Mineral Resource in accordance with the CIM Definition Standards on Mineral Resources and Reserves (2014), the Mineral Resource statement must demonstrate RPEEE.



Due to the geometry and location of the Froome mineralization, the QP considers the deposit to be amenable to underground mining methods. A cut-off grade of 1.35 g/t Au was determined considering the costs and input parameters (Table 14-18) based on current rates and costs and benchmarked with similar projects.

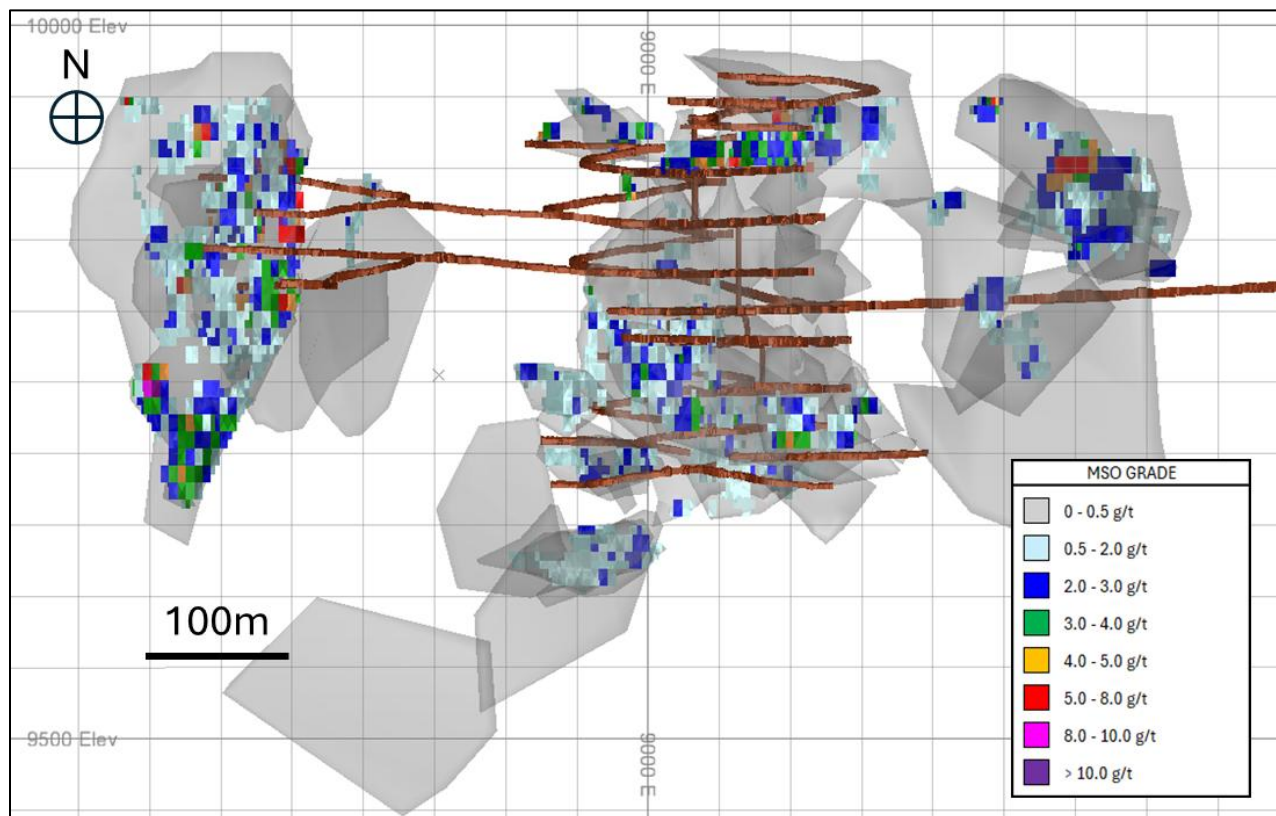
Table 14-18: Froome Mineral Resource Cut-off Grade Parameters

Parameter	Unit	Value
Mining Cost	\$US/t	62.66
G&A Cost	\$US/t	16.07
Milling Cost	\$US/t	32.21
Transport Cost	\$US/oz	1.52
Payable Gold	%	99.95
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Royalty (NSR)	\$US/t	7.20
Mill Recovery	%	89.5

Reasonable prospects for eventual economic extraction were derived from potentially mineable shapes developed using Datamine's MSO function. A stope size of 12 m vertical height by 6 m wide along strike by a minimum 3 m wide across strike was adopted. This size is consistent with current mining methods and was benchmarked against mined stopes at the nearby Black Fox underground mine. Sub stopes were developed at half height and half width along strike (6 m x 3 m) to account for local anisotropy of the grade continuity. Stopes with a width less than 3 m across strike were not considered RPEEE. Stope shapes were developed using the economic cut-off-grade of 1.35 g/t Au. All classified blocks were analyzed by the MSO with resulting stopes shown in Figure 14-17.

All classified blocks were flagged by the potentially mineable shapes including those below the 1.35 g/t Au cut-off grade, this is known as "must take material" and as a combined mineable shape they meet the necessary parameters, including the cut-off grade. Isolated, discontinuous stope shapes were not considered economic, and were therefore removed and not included in the Mineral Resource Statement.

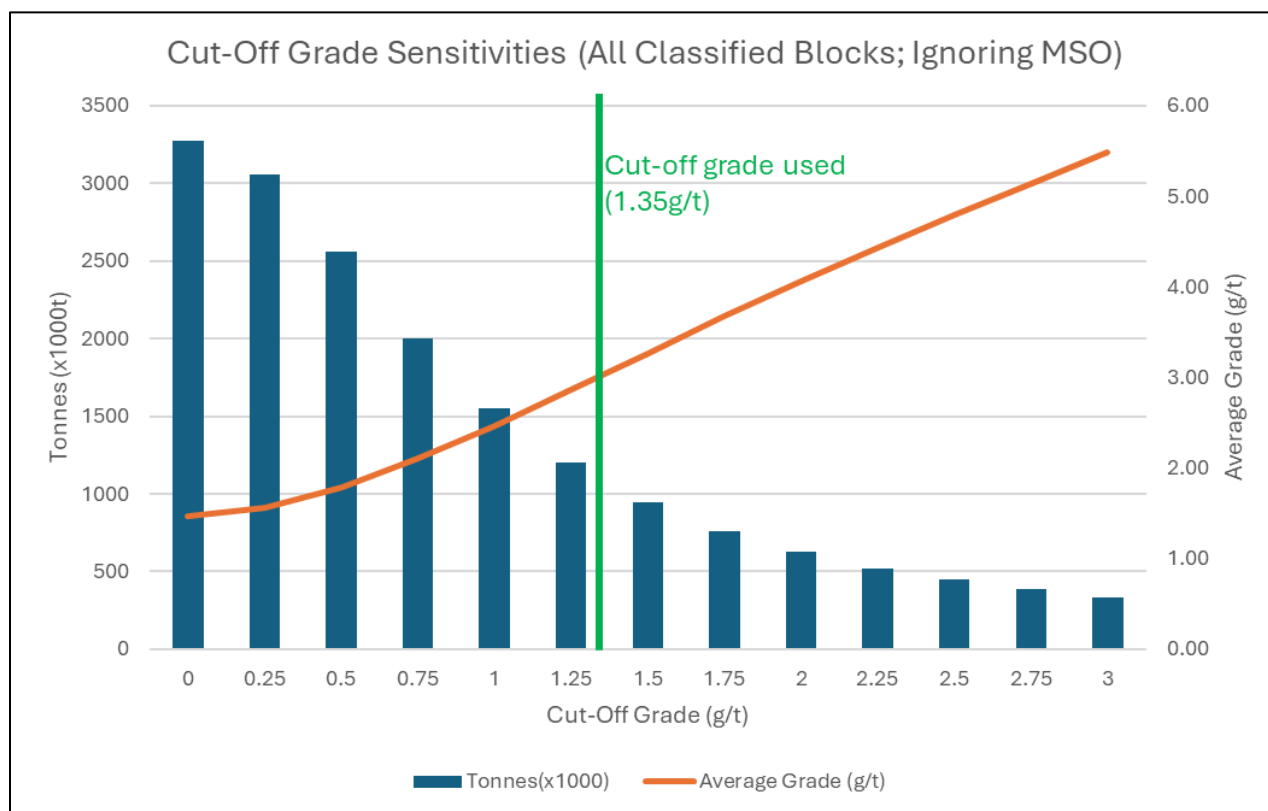




Source: (McEwen, 2025)

Figure 14-17: Long Section Looking North of Potentially Mineable Stope Shapes at Froome

Figure 14-18 presents the experimental grade–tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include Measured, Indicated and Inferred blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.



Source: (McEwen, 2026)

Note: Sensitivities use raw block grades and include Measured, Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-18: Experimental Grade-Tonnage Sensitivities at Various Cut-Off Grades

14.6.11 Mineral Resource Statement

Table 14-19 summarizes the Mineral Resource estimates for Froome, assuming underground stoping methods, reported in accordance with the 2014 CIM Definition Standards (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are reasonable prospects for eventual economic extraction via underground stope extraction.

Table 14-19: Froome Mineral Resource Statement, December 31, 2025

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Measured	212	2.45	17
Indicated	440	2.94	42
Total Measured + Indicated	651	2.78	58
Inferred	302	2.93	28

Notes:

- Effective date of the Mineral Resource estimate is December 31, 2025. The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.



- Mineral Resources are reported using the 2014 CIM Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t Au assuming underground extraction methods and based on a mining cost of \$62.66/t, process cost of \$32.21/t, G&A cost of \$16.07/t, metallurgical recovery of 89.5%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.
- Informing sample database cut-off and mining depletion date is December 31, 2025.

14.6.12 Factors That Could Affect the Mineral Resource Estimate

Factors that may affect the Froome Mineral Resource estimate include:

- Changes in the local geological interpretations and assumptions used to generate the estimation domains
- Changes in mineralization and geological geometry and continuity of mineralized zones
- Changes in assumptions of mineralization and grade continuity
- Changes in the grade interpolation methods and estimation parameter assumptions
- Changes in the confidence assumptions used in the resource classification
- Density assignments
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination
- Changes in input and design parameter assumptions that pertain to the underground mining constraints
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

14.7 Stock Property – West and Main Zone

14.7.1 Mineral Resource Database

The close-out date for the Mineral Resource database is October 23, 2023 and consists of 567 core drill holes totalling 241,163.70 m of drill core and 73,148 intervals assayed for gold. Unsampled intervals were considered barren and assigned a background gold grade based on ¼ of the assay laboratory's lower detection limit.

There has been only limited drilling activity at the property since the database close-out. Logging and assay results of the new data indicate the geological and mineralization model reported here continues to be valid. As new data arrives in future drilling campaigns, the estimation methods and parameters will be refined, which will affect the Mineral Resources dynamically.



On importing the drill hole database from the central SQL database to Datamine Studio software in preparation for resource estimation, sample data were checked for missing and overlapping intervals in the assay and survey tables, and collar locations checked against the LIDAR topography surface.

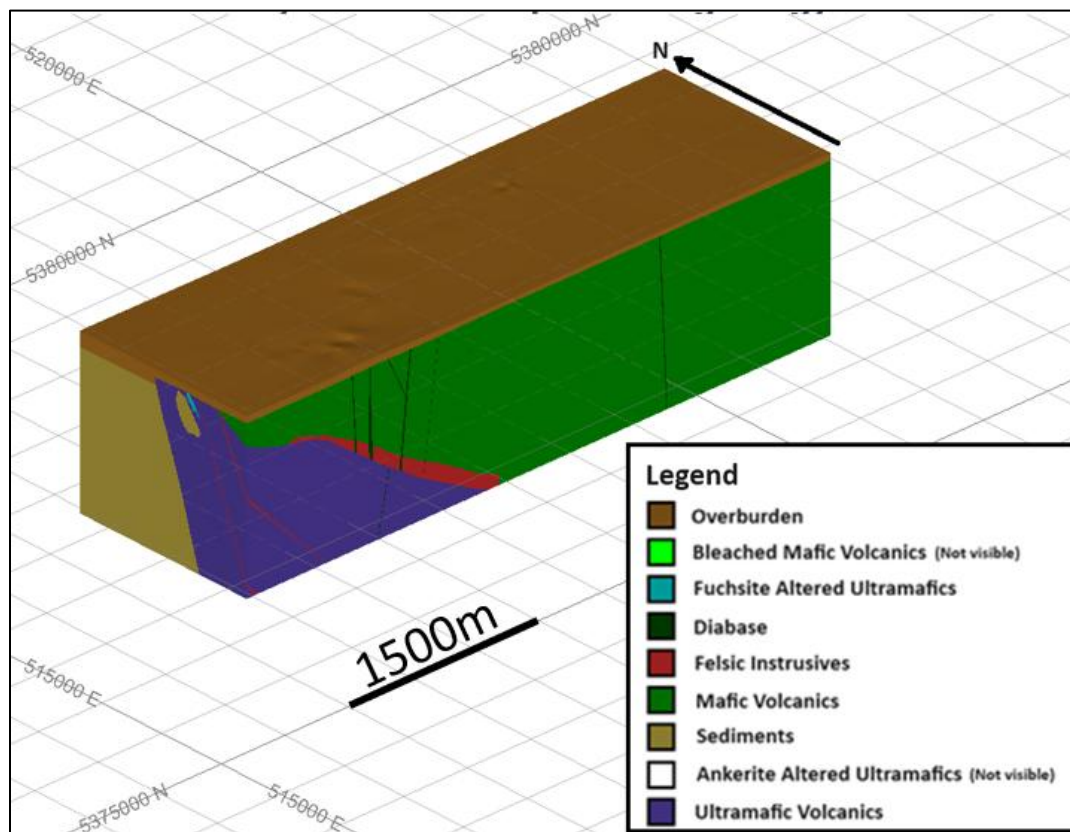
Mining as-builds and remnant shapes were up-to-date as of December 31, 2025 and used to deplete the model.

14.7.2 Geological Modelling

14.7.2.1 Lithology

A lithology model was developed implicitly to capture the eight key deposit lithologies and the overburden as shown in Figure 14-19.

Using the logged lithology, an overburden volume was modeled to the top of the bedrock and was used to cap the lithology wireframes.



Source: (McEwen, 2023)

Figure 14-19: Isometric View of the Stock Project Lithology Model

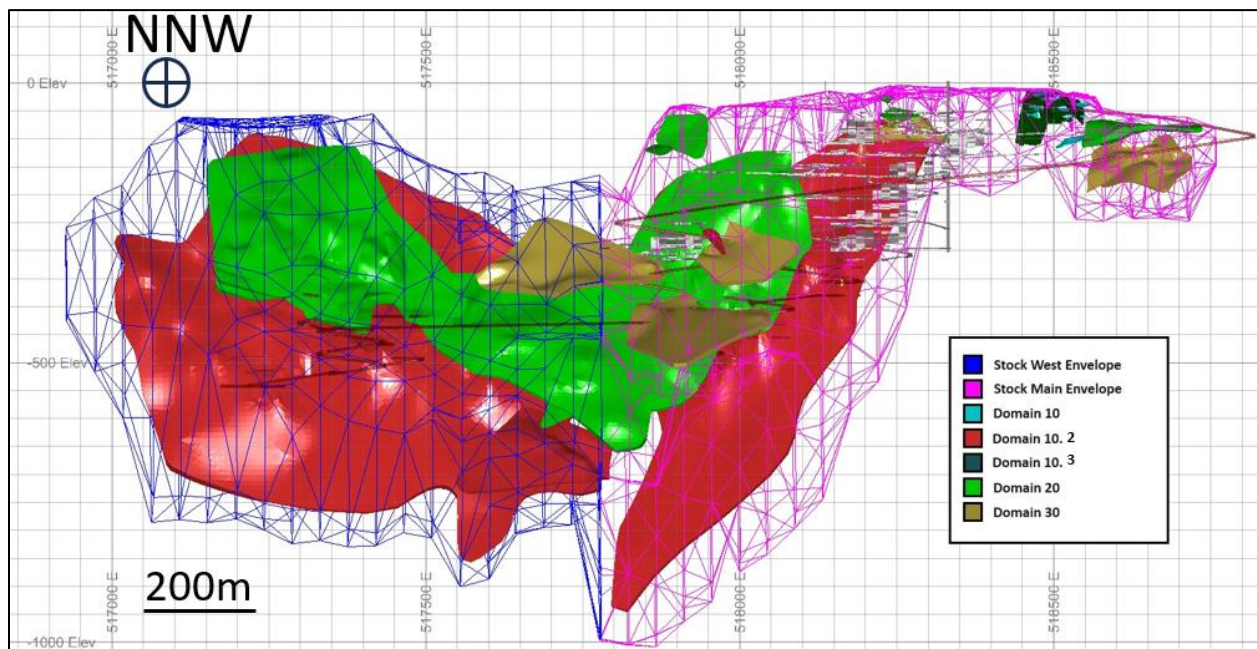


14.7.2.2 Mineralized Domains

The Stock West and Main deposit contain five main zones of mineralization shown in Figure 14-20 and described below:

- Domain 10 is a shear zone, hosted in highly altered ultramafic rocks. Coincident with the most intense mineralization within Domain 10 is the primary alteration carbonate mineral, fuchsite. Disseminated gold mineralization occurs in en-echelon quartz veins, quartz stockwork and matrix material. Domain 10 trends east-northeast and dips moderately to steeply southeast.
- Domains 10.2 and 10.3 form a mineralized zone near the Stock Portal, composed of conjugate sets of crosscutting quartz veins in fuchsite altered ultramafics; 10.2 veins are sub-horizontal; 10.3 veins form sub-vertical, north-south striking lenses.
- Domain 20 is a lower grade zone located in the HW of Domain 10 and is hosted primarily in a bleached altered mafic volcanic unit trending east-northeast and dipping moderately to steeply southeast.
- Domain 30 is generally HW to both Domain 10 and Domain 20, and hosted primarily in felsic porphyry and intrusive units, as well as mineralized mafic metavolcanic units.

Mineralized lenses were interpreted (usually following a threshold grade of 0.5 g/t), flagged in the drill hole file and modeled using Datamine's implicit Vein Modelling module with a minimum thickness of 0.5 m.



Source: (McEwen, 2023)

Note: Looking NNW

Figure 14-20: Long-section view of Stock West and Main Mineralized Zones and Background Envelope



14.7.2.2.1 Background Envelope

A background envelope (Domain 200) was constructed around the mineralized domains to capture mineralized samples outside the mineralized domains and to estimate adjacent dilution (Figure 14-20). It is approximately a 50m buffer around mineralized lenses, constructed from strings using sections spaced every 50m.

14.7.3 Composites

More than 50% of the assays have a sample length of 1 m. The modal composite length of 1 m was applied to samples within the modeled domains which generated composites close to 1 m with a minimum composite length of 0.3 m.

14.7.4 Capping

The impact of high-grade outlier populations was analyzed using histograms, log probability plots, mean and variance, and cumulative metal plots. Results are presented in Table 14-20 for each domain with metal removed ranging from 6.7% in Domain 10 to 51.3% in Domain 10.3.

Table 14-20: Stock West & Main – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
10	1.38	4.07	20.0	1.29	2.01	30	6.7
10.2	2.96	5.64	15.0	1.62	1.94	27	45.1
10.3	4.27	5.65	15.0	2.08	1.69	22	51.3
20	0.80	3.31	13.0	0.71	1.99	24	10.9
30	0.86	2.12	7.0	0.78	1.57	9	9.3
200	0.07	20.77	2.4	0.05	4.02	262	28.4

Note: CV = coefficient of variation

14.7.5 Density

There are 2,124 density samples contained in the Mineral Resource database. Currently, one sample every 20 is collected by the assay laboratory for density analysis in the sample stream submitted for gold assay analysis. A study of all density data was conducted to determine an average density value for each of the host lithology units. Outliers were identified and removed resulting in the average density values listed in Table 14-21.

The block model was flagged by the major lithology wireframes and average densities were applied to the blocks accordingly. Any blocks outside the modelled lithologies were assigned overall average density value of 2.83 g/cm³. Blocks above the bedrock and below the topography were flagged as overburden and assigned a density of 2.0 g/cm³.



Table 14-21: Stock West & Main – Density Values by Lithology

Lithology	No. of Samples	Avg. Density
Bleached Mafic Volcanics (AMV, BMV)	298	2.83
Fuchsite Altered Ultramafic Volcanics (CGR)	287	2.89
Diabase (DIA)	11	2.89
Felsic Intrusive (AFP, FP, FI, II, QFP)	251	2.70
Mafic Volcanics (MV, PMV)	129	2.84
Sediments (SED)	63	2.76
Ankerite Altered Ultramafic Volcanics (AUV, CGY)	532	2.86
Ultramafic Volcanics (TUV, CUV, UMV, UV)	244	2.84
All Lithologies Combined	1815	2.83
* 2124 Total samples. Lowest 5 and highest 5 values removed from evaluation as outliers		
* Density measurements belonging to minor, unmodeled lithologies removed from density calculation.		
* overburden given a specific gravity of 2.00		

14.7.6 Variography

The spatial distribution (continuity) of gold within each mineralized domain was evaluated using variograms.

Primary directions of the variograms were modelled based on observed mineralization and structural orientations. These orientations were then examined statistically to ensure they represented the best possible fit with the variography. The stability of the variograms was evaluated by varying the direction specification and comparing the resulting experimental variograms. Variograms were modelled with spherical structures and two or three rotations to match the strike, dip and plunge of the mineralized zone. Nugget effects are 20% in all Domains.

14.7.7 Grade Estimation

A block size of 3 m x 3 m x 3 m was selected based on drill hole spacing, composite length, the geometry of the modelled mineralized zones, and anticipated mining methods. Sub-cells down to 0.5 m x 0.5 m x 0.5 m were used to better reflect the shape and volume of the mineralized domains. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized domains using a four-pass approach. For Domain 10, search neighbourhoods are based on variography with ranges doubling in length for passes two and three with the maximum search length in pass four equal to one and one half the range of the maximum variogram structure. For Domains 10.2, 10.3, 20 and 30, search neighbourhoods are based on variography with ranges doubling in length for all search ranges.

Data selection for mineralized domains remained the same for the first three passes using a minimum of 7 and a maximum of 15 composites from at least 3 drill holes to estimate a given block. These restrictions were reduced to a minimum of 4 composites from at least 2 drill holes to estimate a block in the fourth pass.

Blocks within the background envelope were estimated using inverse distance to the power of three with its ellipsoid guided by the general lithology contact orientation determined through variography. An initial



estimation pass focused on estimating areas where high-grade composites were not captured in the mineralized domains. This estimation pass used uncapped composites with a tight search ellipse to limit the number of blocks estimated to one or two in each direction. Three subsequent passes were made with increasing search ranges and any blocks not estimated were assigned a background gold value of a one-quarter of detection limit, or 0.0005 g/t.

14.7.8 Model Validation

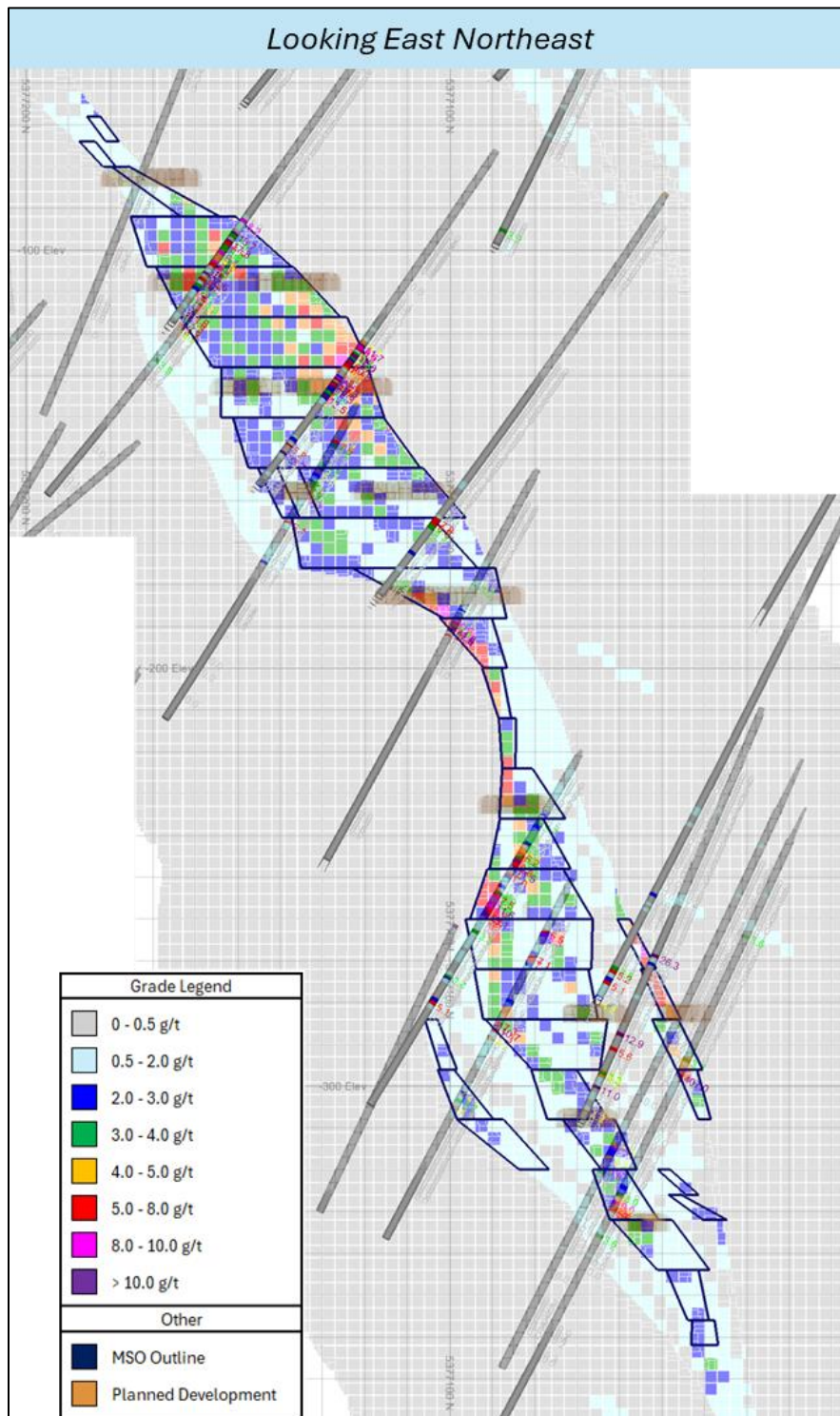
In validating the estimated block model, a nearest neighbour model was constructed. This model mimics a declustered data set and was generated using 3 m capped composites (block dimension) following the same search ellipse as the ordinary kriged estimate.

The estimated block model was validated by visual inspection, comparing the global statistics of the estimated model with the nearest neighbour model, and with swath plots to determine whether any local bias exists.

Inspecting drill holes with block estimates in section (Figure 14-21) and plan (Figure 14-22) confirms gold grades of the model follow the grades of the informing samples.

Swath plots indicate the mean grades of the estimates follow the mean grades of the declustered informing samples relatively well with small local biases in areas with less informing sample data.

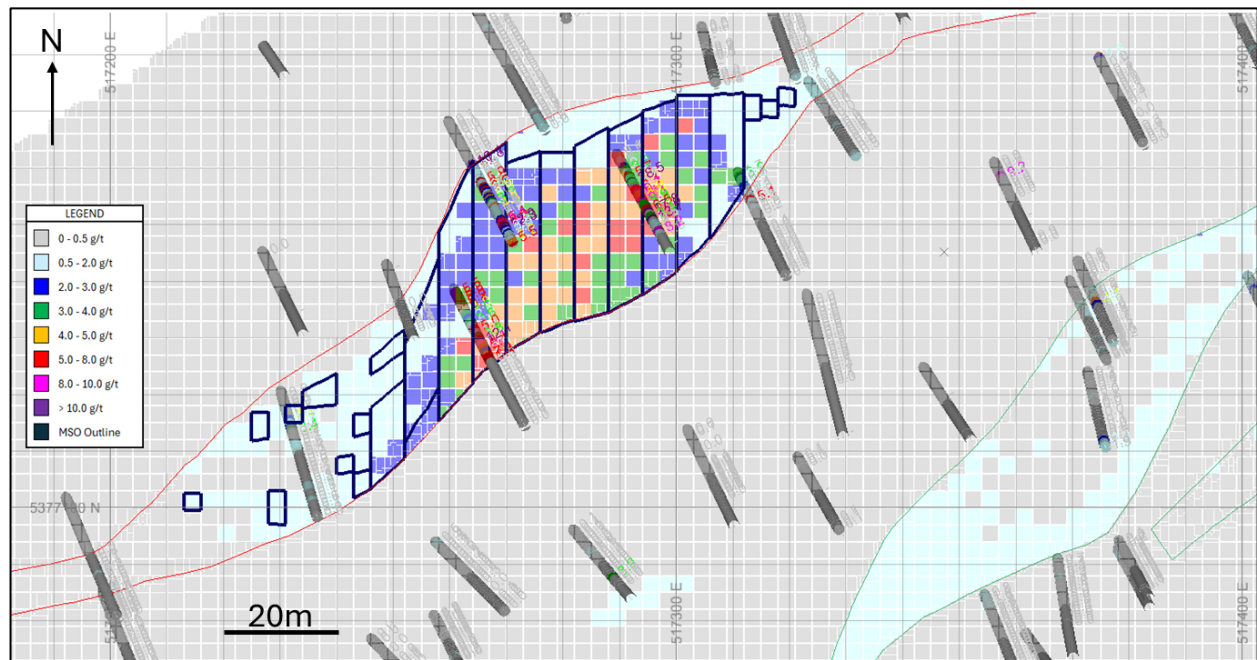




Source: (McEwen, 2025)

Note: Includes MSO outlines and planned development, ± 25 m clipping

Figure 14-21: Section View Comparing Block Grades in Domains 10 and 20 with Drill Hole Data



Source: (McEwen, 2025)

Note: ± 20 m clipping, -100 m elevation, includes MSO outlines

Figure 14-22: Plan View Comparing Block Grades with Drill Hole Data

Block estimates were checked for global bias by comparing the average grade (at a zero g/t cut-off) of the estimated model with that obtained from nearest neighbour estimates. The QP considers the estimate to be globally unbiased if the comparison is within a $\pm 5\%$ tolerance. The global bias check was made on the classified Mineral Resource resulting in less than a 5% difference showing no global bias exists.

14.7.9 Confidence Classification

The QP is confident that the Stock West & Main mineralization models honour the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support Mineral Resource evaluation. The Mineral Resource model is constrained by mineralized domains based on lithological, structural and grade criteria and is modelled from core holes drilled on an irregular grid, with a spacing between 15 m and 35 m. The controls on the distribution of the gold mineralization are understood, and the confidence in geological continuity is reasonable.

Classification criteria considers the confidence in the continuity of mineralization, average distance to the closest three drill holes, distance to the nearest drill hole, and search neighbourhood. The following classification criteria were defined:

- **Indicated:** assigned to those continuous blocks within mineralized domains 10, 10.2, 10.3, and 20, informed by three or more drill holes, within the first three search neighbourhoods, less than 15 m to the nearest drill hole, and an average drill spacing less than 25 m.
- **Inferred:** assigned to those continuous blocks within the mineralized domains and within the first three search neighbourhoods, informed by at least three drill holes, and an average drill spacing

less than 40 m. Also assigned to those continuous blocks within the mineralized domains and within the fourth search neighbourhood, informed by at least three drill holes, less than 12.5 m to the nearest drill hole, and within an average drill spacing less than 40 m.

For Domain 30, lenses are small and discontinuous, it was concluded that more data is needed to increase the confidence therefore the classification for this domain was forced as inferred; any material that might have been classified as indicated using the above criteria was classified as inferred.

Blocks within the mineralized domains not meeting the above criteria were not classified. Blocks within the background envelope were not classified.

14.7.10 Reasonable Prospects for Eventual Economic Extraction

To report a Mineral Resource in accordance with the CIM Definition Standards on Mineral Resources and Reserves (2014), the Mineral Resource statement must demonstrate RPEEE. Due to the geological nature, geometry, and location of the Stock West mineralization, the QP considers the deposit to be amenable to underground mining methods. A cut-off grade of 1.30 g/t Au was determined considering the costs and input parameters (Table 14-22) based on recent quotations and benchmarked with similar projects.

Table 14-22: Stock West and Main Mineral Resource Cut-off Grade Parameters

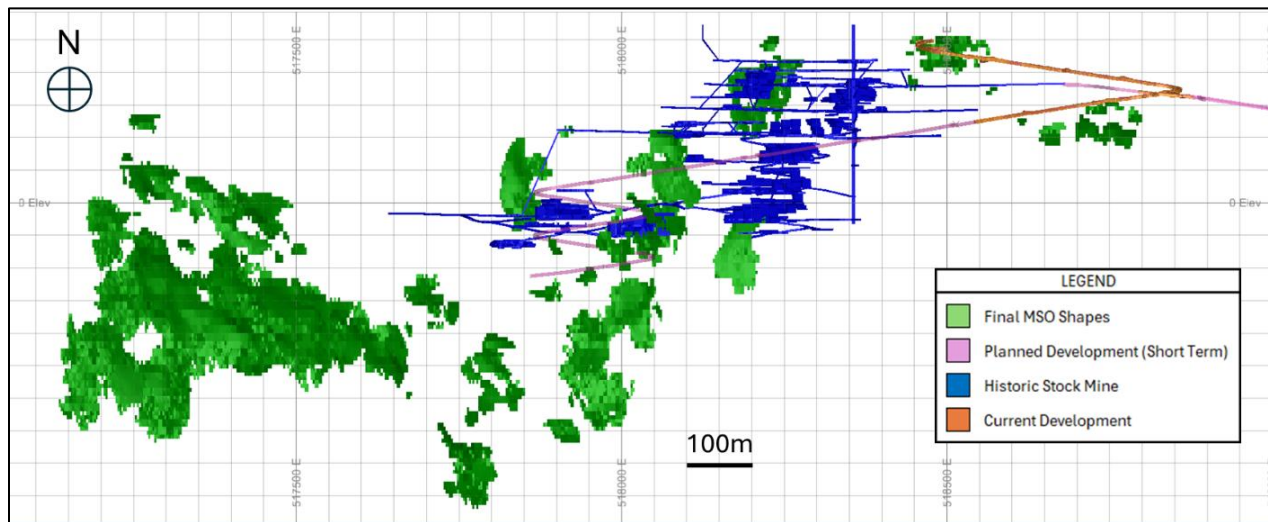
Parameter	Unit	Value
Mining Cost	\$US/t	62.66
G&A Cost	\$US/t	20.49
Milling Cost	\$US/t	32.21
Transport Cost	\$US/oz	1.52
Payable Gold	%	99.95
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Royalty (NSR)	%	*
Mill Recovery	%	93

Note: *Franco Nevada retains a 1.0% NSR Royalty on a portion of Stock Main, calculated cut-off grade for that zone is 1.30g/t vs 1.29g/t for the remainder of the deposit, ultimately a rounded 1.30 g/t cut-off grade was used for all of Stock.

Reasonable prospects for eventual economic extraction were derived from potentially mineable shapes developed using Datamine's Mineable Shape Optimizer function. A stope size of 12 m vertical height by 6 m wide along strike by a minimum 3 m wide across strike was adopted. This size was benchmarked against mined stopes at the nearby Black Fox underground mine. Sub stopes were developed at half height and half width along strike (6 m x 3 m) to account for local anisotropy of the grade continuity. Stopes with a width less than 3 m across strike were not considered potentially economical and excluded. Stope shapes were developed using the economic cut-off-grade of 1.30 g/t Au. All classified blocks were analyzed by the mineable shape optimizer with resulting stopes shown in Figure 14-23.



All classified blocks were flagged by the potentially mineable shapes including those below the 1.30 g/t Au cut-off grade that were included in the MSO shapes calculated at the economic cut-off grade. This can be considered as internal dilution or “must-take” material. Isolated, discontinuous stope shapes were not considered economic, and were therefore removed and not included in the Mineral Resource Statement.

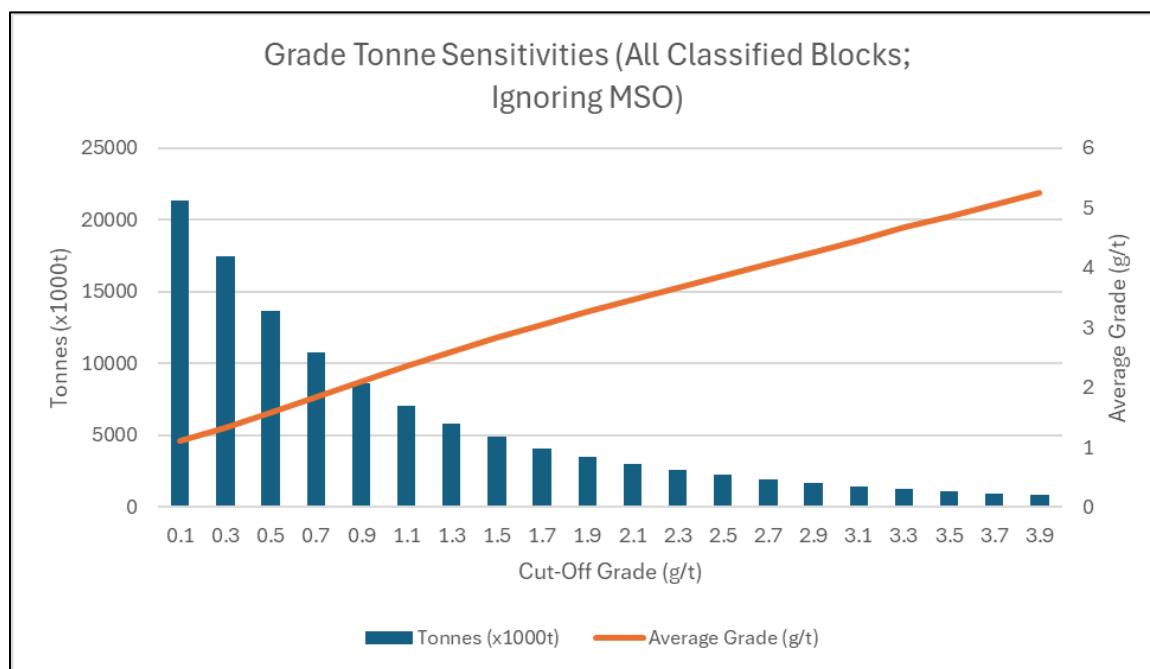


Source: (McEwen, 2026)

Note: Looking north

Figure 14-23: Long Section View of the Potentially Mineable Stope Shapes at Stock West & Main

Figure 14-24 presents the experimental grade–tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include both Indicated and Inferred classified blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.



Source: (McEwen, 2026)

Note: Sensitivities use raw block grades and include Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-24: Experimental grade-tonnage sensitivities at Various Cut-Off Grades

14.7.11 Mineral Resource Statement

Table 14-23 summarizes the Mineral Resource estimates for Stock West and Main, assuming underground stoping methods, reported in accordance with the 2014 CIM Definition Standards (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are RPEEEs via underground stope extraction.

Table 14-23: Stock Project - West & Main Mineral Resource Statement, December 31, 2025

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Indicated	3,179	2.63	268
Inferred	2,468	2.31	184

Notes:

- Effective date of the Mineral Resource estimate is December 31, 2025. Drill hole database cut-off date is October 23, 2023 and the effective depletion date is December 31, 2025. The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources are reported using the 2014 CIM Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported above an economic cut-off grade of 1.30 g/t gold assuming underground extraction methods and based on a mining cost of \$62.66/t, process cost of \$32.21/t, G&A cost of \$20.49/t, metallurgical recovery of 93%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.



14.7.12 Factors That Could Affect the Mineral Resource Estimate

Factors that may affect the Stock West and Main Mineral Resource estimate include:

- Changes in the local geological interpretations and assumptions used to generate the estimation domains
- Changes in mineralization and geological geometry and continuity of mineralized zones
- Changes in assumptions of mineralization and grade continuity
- Changes in the treatment of high-grade gold values
- Changes in the grade interpolation methods and estimation parameter assumptions
- Changes in the confidence assumptions and methods used in the mineral resource classification
- Changes in the density and the methods used in the density assignments
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination
- Changes in input and design parameter assumptions that pertain to the underground mining constraints
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report.

14.8 Stock Project – East Zone

14.8.1 Mineral Resource Database

The Stock Project East Zone has been tested by diamond drilling over a cumulative strike length of approximately 1,100 m and down to a vertical depth of about 500 m. The close-out date for the Mineral Resource database is May 20, 2024 and consists of 257 surface diamond drill holes. The average drill hole spacing in the best drilled areas of the deposit is about 20 m; the spacing in the more poorly drilled areas is between 100 m and 150 m.

Mining as-builds and remnant shapes were up-to-date as of December 31, 2025 and used to deplete the model.

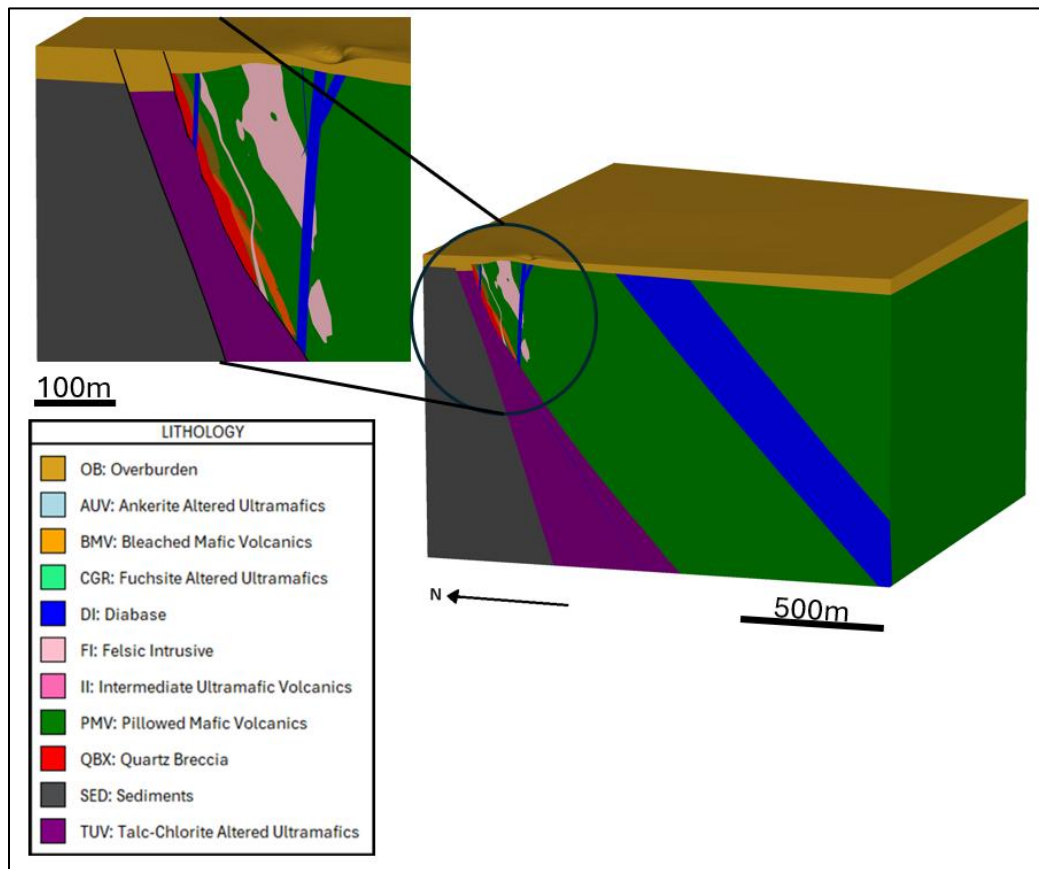
14.8.2 Geological Modelling

14.8.2.1 Lithology

A lithology model was developed implicitly to capture the 10 key deposit lithologies and the overburden as shown in Figure 14-25.

Using the logged lithology, an overburden volume was modeled to the top of the bedrock and was used to cap the lithology wireframes.





Source: (McEwen, 2024)

Figure 14-25: Isometric View of the Stock Project East Zone Lithology Model

14.8.2.2 Mineralized Domains

Gold mineralization in the Stock East deposit occurs primarily in a quartz breccia (QBX) and generally follows the bleached/altered mafic volcanic lithological unit (BMV). Mineralization is bounded on the footwall by the interpreted Night Hawk Break regional fault structure. The Night Hawk Break is a splay off the regional Porcupine Destor fault zone.

Mineralized veins were interpreted (usually following a threshold grade of 0.5 g/t), flagged in the drill hole file and modeled using Datamine's implicit Vein Modelling module with a minimum thickness of 0.5 m.

14.8.2.2.1 Background Envelope

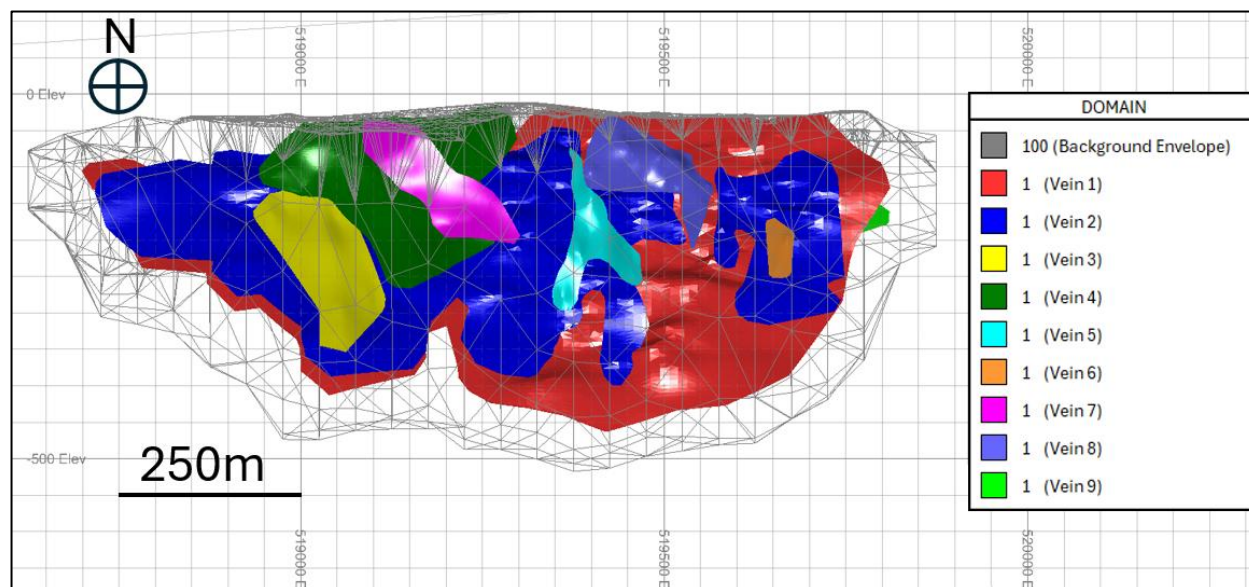
A background envelope (Domain 100) was constructed around the BMV Domain (Domain 1) to capture mineralized samples outside the mineralized domains and to estimate adjacent dilution. It is approximately a 50 m buffer around mineralized lenses, constructed from strings using sections spaced every 50 m.

The modelled domains for Stock's East zone are shown in Figure 14-26.



14.8.3 Composites

A composite length of 1 m was selected based on the mode of the sample lengths. Any residual length was distributed equally to the 1 m composites to ensure that the entire length of each drill intersection within a domain was utilized. The rationale behind adopting a standard sample length of 1 m within zones perceived to be geologically similar was to maintain resolution in terms of the gold grade distribution. That resolution is maintained by adopting 1 m as the composite length.



Source: (McEwen, 2024)

Figure 14-26: Long Section View of Stock East Zone Showing Modelled Domains 1 & 100 (Background)

14.8.4 Capping

Grade capping was conducted using population histograms and probability/log-probability plots to limit the influence of outlier values within each domain. Results are presented in Table 14-24 for each domain with 13.2% metal removed from the BMV Domain 1.

Table 14-24: Stock East – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
1	1.45	3.97	13.5	1.26	1.41	28	13.2
100	0.05	5.44	2.4	0.05	3.22	28	6.1

Note: CV = coefficient of variation



14.8.5 Density

There are 1,445 density samples contained in the Mineral Resource database for Stock's East Zone. A study of all density data was conducted to determine an average density value for each of the host lithology units. Outliers were identified and removed resulting in the average density values listed in Table 14-25.

The block model was flagged by the major lithology wireframes and average densities were applied to the blocks accordingly. Any blocks outside the modelled lithologies were assigned the overall average density value of 2.80 g/cm³. Blocks above the bedrock and below the topography were flagged as overburden and assigned a density of 2.0 g/cm³.

Table 14-25: Stock Project – East Zone Density Averages for Host Lithologies

Lithology	No. of Samples	Avg. Density
Altered Ultramafic Volcanics AUV	55	2.81
Bleached Mafic Volcanics BMV, AMV	557	2.86
Fuchsite Altered Ultramafic Volcanics CGR	35	2.88
Diabase DIA	15	2.87
Felsic Porphyry FP, FI	365	2.68
Mafic Volcanics MV, PMV	329	2.82
Quartz Breccia QBX, QV	11	2.77
Sediments SED	14	2.74
Talc-Chlorite Altered Ultramafics TUV	64	2.79
All Lithologies Combined	1445	2.80

* 1445 Total samples. Lowest 3 and highest 3 values removed from evaluation as outliers (2 BMV, 1 MV, 2 AUV, and 1 TUV).

* overburden given a specific gravity of 2.0

* Intermediate Intrusives (II) given a specific gravity of 2.80 (avg density)

* Gabbro (GAB) given a specific gravity of 2.80 (avg density)

14.8.6 Variography

The spatial distribution (continuity) of gold within each modeled domain was evaluated using variograms.

Precision in spatial analysis/variography is directly proportional to the quality of the sampling pattern. Due to the moderate- to steeply-dipping nature of the Stock East deposit, some of the drill holes from surface intersect the mineralization at oblique angles while some are at the desired right angles, culminating in a mixture of orientations. Thus, variography results are not perfectly representative of the spatial continuity/distribution patterns of the mineralization. Experimental variograms were modeled with two spherical structures and with a dip and strike. Nugget effects are 20% in the BMV domain and 0% in the background envelope.



14.8.7 Grade Estimation

A block size of 3 m x 3 m x 3 m was selected after comparing the kriging efficiency and slope of regression for various block sizes. The selected block size is supported by the average drill hole spacing of 20 to 30 m in well drilled areas of the deposit. Sub-cells down to 1 m x 1 m x 1 m were used to better reflect the shape and volume of the mineralized domains. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the modelled domains using a four-pass approach. Search neighbourhoods are based on variography with ranges doubling in length for the first three search ranges, with the fourth search range having a distance of 1.5x that of the third search range. The range of the third search range is based on the range of the second variogram structure.

Data selection for modeled domains require 11 composites from three drill holes to be estimated in passes 1 through 3. The fourth pass requires a minimum of 6 composites and a maximum of 18 from at least two drill holes.

14.8.8 Model Validation

In validating the estimated block model, nearest neighbour and inverse distance cubed models were constructed. The nearest neighbour model mimics a declustered data set and was generated using 3 m composites.

The estimated block model was validated by visually comparing the drill holes with the block grades, and with swath plots to determine whether any local bias exists.

Inspecting drill holes with block estimates in section and plan confirms gold grades of the model follow the grades of the informing samples.

Swath plots indicate the mean grades of the estimates follow the mean grades of the declustered informing samples relatively well with small local biases in areas with less informing sample data.

14.8.9 Confidence Classification

Classification criteria consider the confidence in the continuity of mineralization, average distance to informing samples (weighted on distance), and continuity of sample spacing. No material has been classified as a Measured Mineral Resource. The following classification criteria were defined:

- **Indicated:** assigned to those contiguous blocks within the mineralized domain, within the first three search neighborhoods, informed by between 11 and 18 composites from three or more drill holes, less than 15 m to the nearest drill hole, and an average drill spacing less than 25 m.
- **Inferred:** assigned to those contiguous blocks within the mineralized domain, and within the first three search neighborhoods, informed by between 11 and 18 composites from at least three drill holes, and an average drill spacing less than 40 m. Also assigned to those continuous blocks within the mineralized domains and within the fourth search neighborhood, informed by between 6 and 18 composites from at least three drill holes, less than 12.5 m to the nearest drill hole, and within an average drill spacing less than 40 m.



14.8.10 Reasonable Prospects for Eventual Economic Extraction

To report a Mineral Resource in accordance with the CIM Definition Standards on Mineral Resources and Reserves (2014), the Mineral Resource statement must demonstrate RPEEE.

The Stock East deposit area is proximal to the existing plant/mill infrastructure, and in the QP's opinion, the small size of the deposit does not justify relocating the processing facilities to allow for extraction via an open pit. Thus, it seems more feasible to mine the deposit using underground mining methods. A cut-off grade of 1.30 g/t Au was determined considering the costs and input parameters summarized in Table 14-26.

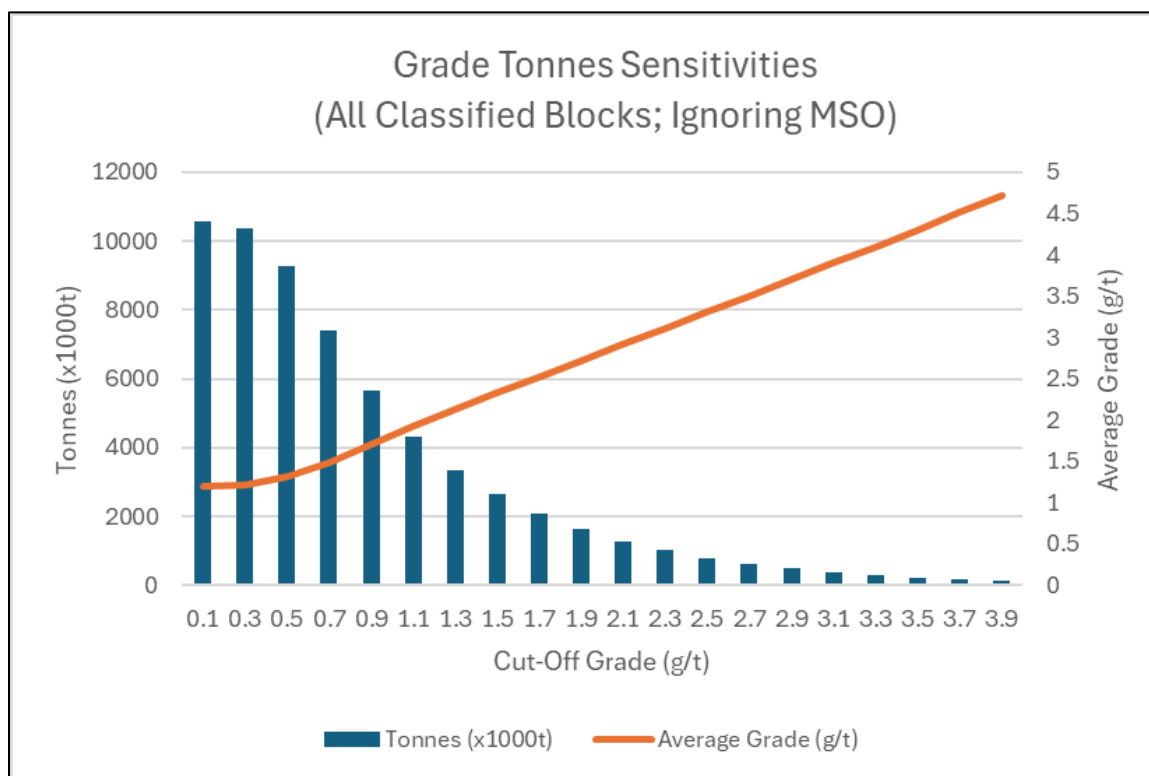
Table 14-26: Stock East Mineral Resource Cut-off Grade Parameters

Parameter	Unit	Value
Mining Cost	\$US/t	62.66
G&A Cost	\$US/t	20.49
Milling Cost	\$US/t	32.21
Transport Cost	\$US/oz	1.52
Payable Gold	%	99.95
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Royalty (NSR)	%	0
Mill Recovery	%	93

RPEEEs were derived from potentially mineable shapes developed using Datamine's MSO function. A stope size of 12 m vertical height by 6 m wide along strike by a minimum 3 m wide across strike was adopted. This size was benchmarked against mined stopes at the nearby Black Fox and Froome underground mines. Sub stopes were developed at half height and half width along strike (6 m x 3 m) to account for local anisotropy of the grade continuity. It should be noted that different stope dimensions will yield different resources at the same cut-off grade. Isolated, discontinuous stope shapes were not considered economic, and were therefore removed and not included in the Mineral Resource Statement.

Figure 14-27 presents the experimental grade-tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include both Indicated and Inferred classified blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.





Source: (McEwen, 2024)

Note: Sensitivities use raw block grades and include Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-27: Experimental Grade-Tonnage Sensitivities at Various Cut-Off Grades

14.8.11 Mineral Resource Statement

Table 14-27 summarizes the Mineral Resource estimates for Stock East, assuming underground stoping methods, reported in accordance with the 2014 CIM Definition Standards (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are RPEEEs via underground stope extraction.

Table 14-27: Stock Project – East Zone Mineral Resource Statement, December 31, 2025

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Indicated	1,790	2.12	122
Inferred	1,350	2.02	87

Notes:

- Effective date of the Mineral Resource estimate is December 31, 2025. Drill hole database cut-off date is May 20, 2024 and the effective depletion date is December 31, 2025. The QP for the estimate is Mr. Carson Cybolsky, P.Geo, an employee of McEwen.
- Mineral Resources are reported using the 2014 CIM Definition Standards and in accordance with the CIM Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.



- Mineral Resources are reported above an economic cut-off grade of 1.30 g/t gold assuming underground extraction methods and based on a mining cost of \$62.66/t, process cost of \$32.21/t, G&A cost of \$20.49/t, metallurgical recovery of 93%, and gold price of US\$3,000/oz.
- Mineral Resources include the 'must take' minor material below cut-off grade which is interlocked with masses of blocks above the cut-off grade within the mineable shape optimizer stopes.
- Figures may not sum due to rounding.

14.8.12 Factors That Could Affect the Mineral Resource Estimate

Factors that may affect the Stock Project – East Zone Mineral Resource estimate include:

- Changes in the local geological interpretations and assumptions used to generate the estimation domains
- Changes in mineralization and geological geometry and continuity of mineralized zones
- Changes in assumptions of mineralization and grade continuity
- Changes in the treatment of high-grade gold values
- Changes in the grade interpolation methods and estimation parameter assumptions
- Changes in the confidence assumptions and methods used in the mineral resource classification
- Changes in the density and the methods used in the density assignments
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination
- Changes in input and design parameter assumptions that pertain to the underground mining constraints
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

14.9 Fuller

In May 2017, SRK prepared and completed an internal Mineral Resource estimate for the Fuller gold deposit under a commission by Lexam. After the acquisition of Lexam by McEwen, SRK was again commissioned to prepare the Mineral Resource statement for the 2021 PEA for Fuller using updated economic parameters. In April 2026, McEwen (as QP) updated the Fuller Mineral Resource statement for this PFS using updated economic parameters. No additional resource drilling has been completed at Fuller recently and the existing 2017 resource model was used as a basis for reporting purposes.

This section summarizes the data, methodology, and parameters used to prepare the Mineral Resource model for Fuller.



14.9.1 Mineral Resource Database

A database containing information from Fuller, Paymaster and Buffalo Ankerite comprises 6,622 drill holes and underground workings with a total length of 501,523 m. The total number of samples with gold assays is 153,113 (200,730 m). The database also includes 496,064 m of lithological, structural and alteration logging.

The Fuller subset of data used for estimation comprises 1,514 drill holes and underground workings (chip samples) for a total length 118,070 m. The subset of drill holes and workings intersecting the estimation domains is 1,230 totalling 77,485.2 m. The total number of samples with gold assays is 35,562 (42,379 m). The total number of samples with gold assays is 35,562 (42,379 m). The database also includes 113,927 m of lithological, structural and alteration logging. It is understood that Buffalo Ankerite, Fuller and Paymaster properties are adjacent, and some drill holes can occur across several properties.

Five historic level plans for Fuller were digitised from the graphic data, adding 4,137 chips and bazooka samples to the database. On importing the drill hole database to Datamine Studio and Leapfrog software several validation steps were performed and any errors identified were corrected.

14.9.2 Geological Modelling

14.9.2.1 Lithology

The deposit comprises several lithologies that have contrasting density characteristics. To use this information for accurate tonnage calculations a simplified lithological model was developed using four key lithologies.

14.9.2.2 Mineralized Zones

Gold mineralization belongs to the class of structurally controlled Archean lode gold deposits. The mineralization is associated with foliated shear zones with strong sericite and fuchsite alteration and occurs stratigraphically above what appears to be the contact between the older ultramafic lower formation and the basaltic middle formation of the Tisdale group. The porphyry intrusive body also contains the lower grade mineralization. A combination of lithological, alteration, structural and grade criteria (in that order) were used to define resource domains.

14.9.2.2.1 Shear Zone

The Shear (or historically called “contact”) Zone is the main mineralization host occurring stratigraphically above the contact between the older ultramafic rocks and the basaltic middle formation.

The following combined criteria were used to define resource domains:

- Grade criteria: Au > 0.3 g/t.
- Lithological criteria: shear/mineralized zone, foliation, presence of quartz veins.
- Alteration criteria: presence of sericite or fuchsite.



An interval containing any data defined above was treated as mineralized with several non-mineralized intervals included to keep the continuity of the mineralized zone.

Considering the good general continuity but high variability on a smaller scale the Shear Zone wireframe was modelled in two steps:

- Building the **external** general wireframe manually containing some internal barren intervals within it (Figure 14-28).
- Building the **internal** wireframe within the external contour with semi-automatic wireframing algorithms which exclude the barren intervals and follow the general trend of the Shear Zone. This wireframe was used as the estimation domain for the Shear Zone (Figure 14-29).

14.9.2.2.2 Porphyry

The Porphyry Zone was developed based on the lithological logging data. Some adjustments were made to make the wireframe correspond better with the level plans interpretation. The internal mineralized zone was developed within the porphyry body based on the same criteria used for the Shear Zone.

14.9.2.2.3 Hangingwall Zone

The Hangingwall Zone has a similar morphology as the Shear Zone and follows the boundary between the Porphyry and Shear zones within the East area of the project. The modelling method for the Hangingwall Zone replicates that of the Shear Zone.

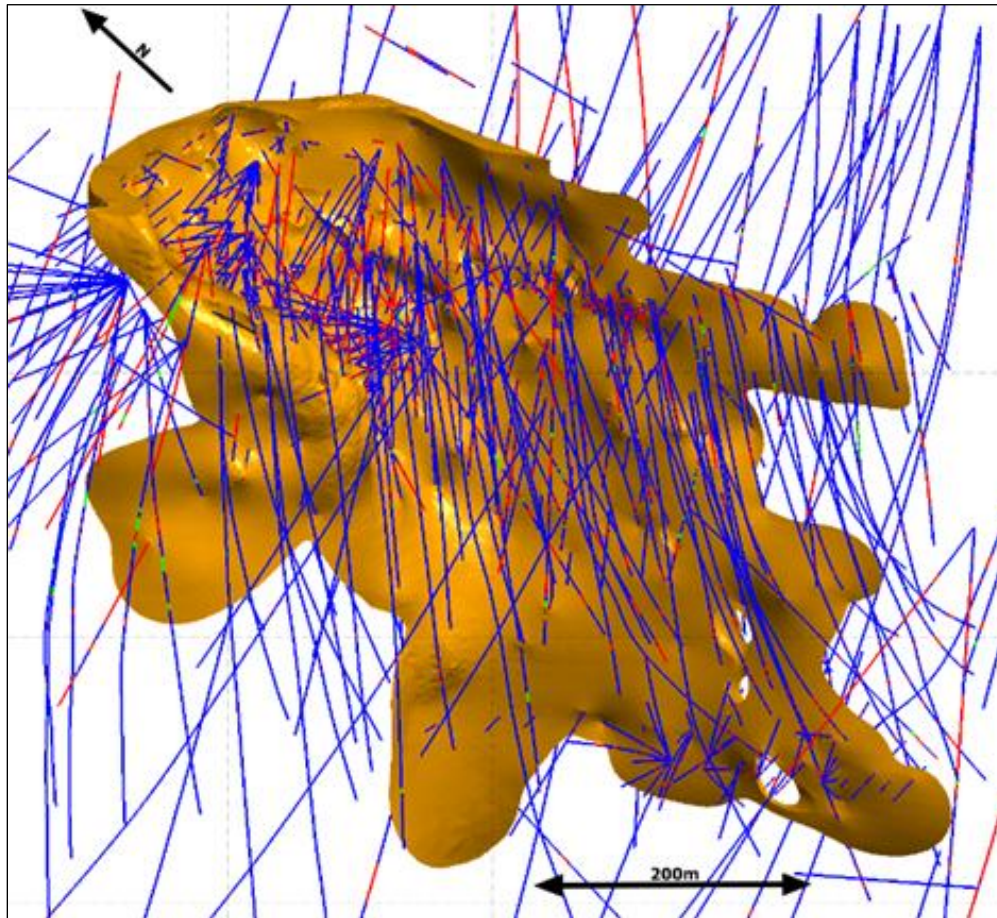
14.9.2.2.4 Low Grade Zone

The residual mineralized intervals with grade more than 0.3 g/t Au were contoured with an automatic contouring algorithm, but only wireframes with a total volume greater than 500 m³ were used to define a Low Grade (LG) Zone.

14.9.3 Composites

Most of the analytical samples within the mineralized zones were collected at 1.5 m intervals for drill holes and 1 m intervals for underground channels. A modal composite length of approximately 1.5 m was applied to all data generating composites as close to the 1.5 m as possible, while creating residual intervals of up to 0.75 m in length (drill hole assays and channel samples).

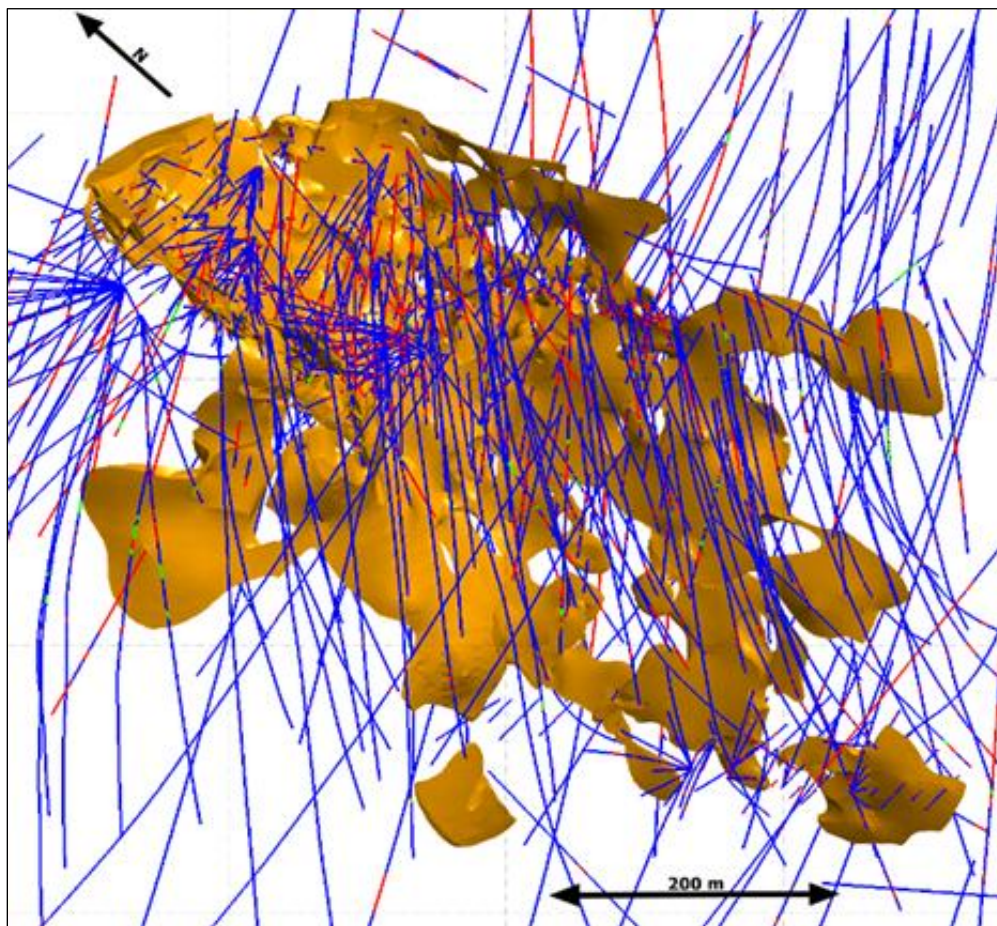




Source: (prepared by SRK 2021; reissued McEwen 2026)

Note: Red drill hole traces represent the mineralized intervals, blue – waste, green – potentially mineralized (logging anomaly with no grade confirmation).

*Figure 14-28: Isometric View of the Shear Zone, **External** Contours*



Source: (prepared by SRK 2021; reissued McEwen 2026)

Note: Red drill hole traces represent the mineralized intervals, blue – waste, green – potentially mineralized (logging anomaly with no grade confirmation).

Figure 14-29: Isometric View of the Shear Zone, **Internal** Contours

14.9.4 Capping

The impact of high-grade outlier populations was examined on composite data using log probability plots and cumulative statistics separately for mineralized zones. The capping value was determined separately for underground development sampling and drill hole data types and the changing of statistical parameters after compositing and capping. Capping was not warranted in the Porphyry Zone or for the underground samples of the LG Zone.

Results are presented in Table 14-28 for each zone with metal removed ranging from 6% in the drill hole data of the LG Zone to 16% in the underground samples of the Hangingwall Zone.

Table 14-28: Fuller – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
Drill Hole Data							
Shear Zone	2.85	5.19	40	2.47	1.87	10	13
Porphyry	0.98	1.59	-	0.98	1.59	-	-
HW Zone	2.93	3.05	30	2.54	1.73	5	13
LG Zone	2.03	2.30	30	1.91	1.72	3	6
Underground Samples							
Shear Zone	3.72	3.08	30	3.35	1.36	13	10
Porphyry	0.87	0.79	-	0.87	0.79	-	-
HW Zone	6.07	2.05	30	5.11	1.29	7	16
LG Zone	1.12	0.87	-	1.12	0.87	-	-

Note: CV = coefficient of variation

14.9.5 Density

Average densities for each modelled lithology are presented in Table 14-29 and applied to the blocks contained within. The density data is based on the results of 181 historical measurements; however, no original data was provided.

Table 14-29: Fuller – Density Values by Lithology

Lithology	Density (g/cm ³)
Dyke	2.92
Porphyry	2.69
Mafic Volcanics	2.82
Ultramafic Volcanics	2.85

14.9.6 Variography

The spatial distribution of gold in the mineralized zones was evaluated using variograms. Both channel and boreholes sample data were considered for variogram analysis. A variogram analysis was done within the east and the most flattened zone of the fold.

Variograms for the Shear Zone and Porphyry were modelled from the corresponding datasets. Variograms for the HW and LG Zones were modelled using composites from all mineralized zones due to the limited number of composites they each contain.



Variograms were modelled using two spherical structures and three rotations to match the strike, dip, and plunge of the modelled mineralization. Nugget effects are 40% for Shear Zone and 20% for the Porphyry.

14.9.7 Grade Estimation

The criteria used in the selection of block size included the drill hole spacing, composite length, the geometry of the modelled zone, and anticipated mining method. A block size of 5 m x 5 m x 5 m was selected. Sub-cells were used allowing a resolution of 1 m x 1 m x 1 m to better reflect the shape of the mineralized zone. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized domains using a four-pass approach. The first pass used a smaller search ellipsoid to constrain the influence of the channel samples.

The three following passes used increasing search neighbourhoods sized from variography results. Datamine's dynamic anisotropy function was implemented due to the anisotropic nature and widths of the mineralized domains.

Data selection for mineralized zones varied for each pass requiring a minimum number of composites between six and four, and a maximum number of composites between eight and 16 for passes 1 to 2, respectively. Estimated blocks required at least two holes in pass 1 and 3, and three holes in pass 2. Blocks estimated in pass 4 required at least one composite.

The waste material located outside the constrained zones was estimated with composites capped at 10 g/t Au, and an isotropic search requiring a minimum of seven composites and a maximum of 12.

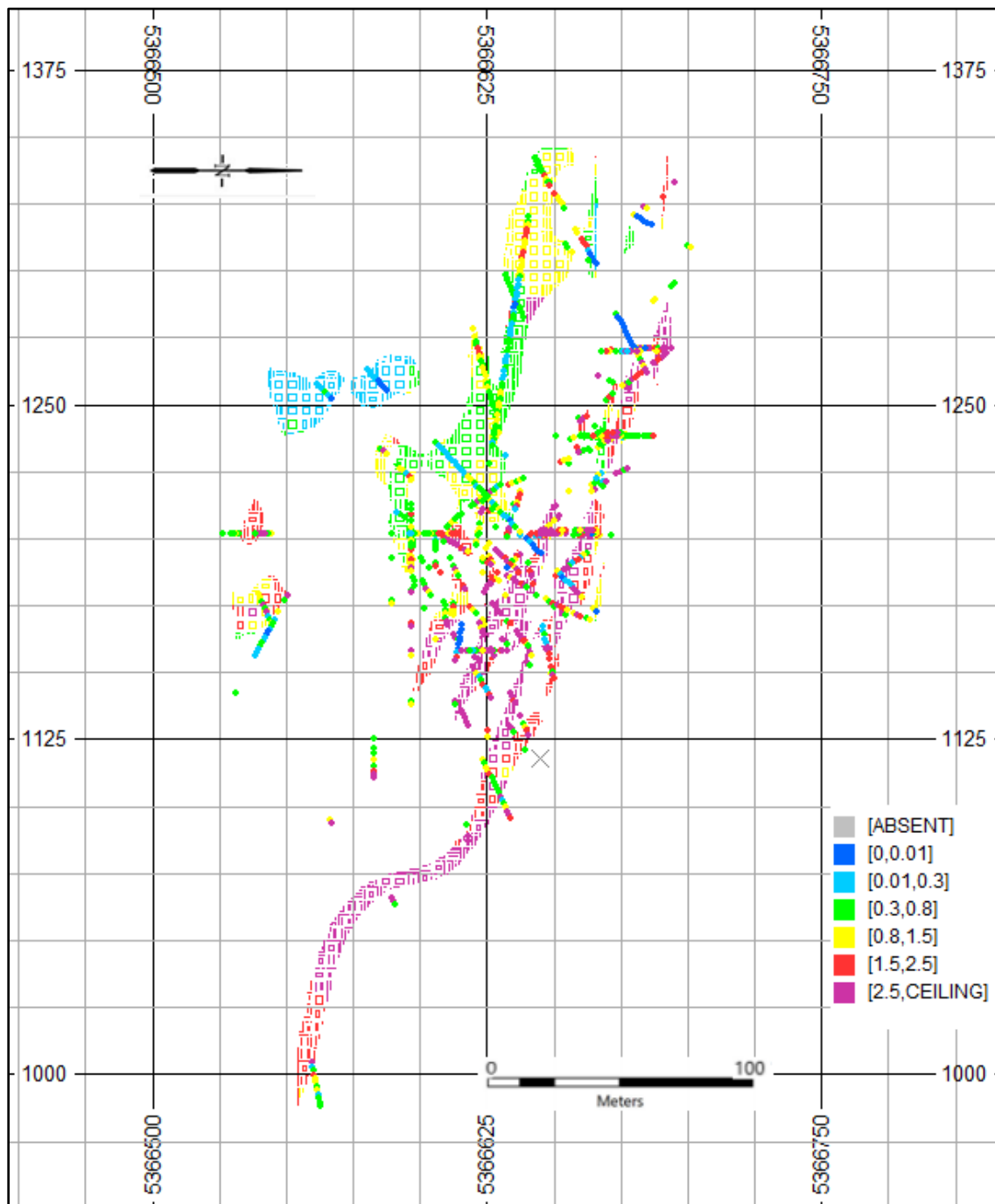
This estimation allowed the incorporation of several unconstrained mineralized intervals into the model.

14.9.8 Model Validation

The block model was validated by visual comparison of informing sample data with resource blocks data (on plan and section, as shown in Figure 14-30 and Figure 14-31) and with swath plots (section by section) comparing the ordinary kriged blocks with the informing composite data.

The results of the validation show that the block model adequately reflects the assay sample data. The deviations between the averages on swath plots appear only within the areas of the low data density.

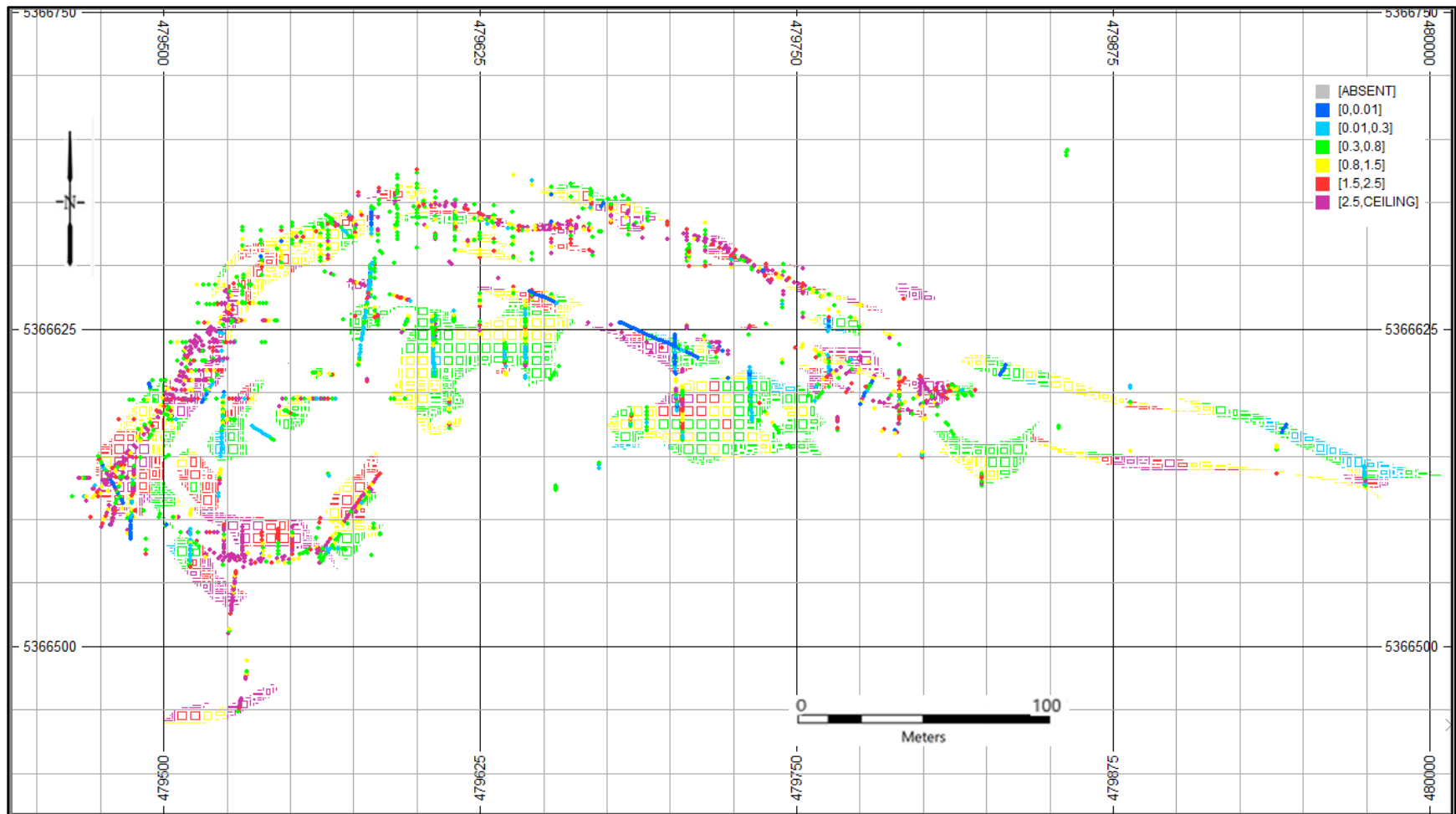




Source: (prepared by SRK 2021; reissued McEwen 2026)

Note: South-North view, drill hole data denoted by red dots

Figure 14-30: Section View at Easting 479,600 Comparing Block Grades with Drill Hole Data



Source: (prepared by SRK 2021; reissued McEwen 2026)

Note: Drill hole data denoted by red dots

Figure 14-31: Plan View at the 1230 El. Comparing Block Grades with Drill Hole Data



14.9.9 Confidence Classification

Industry best practices suggest that classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification as well as the continuity of the deposit at the reporting cut-off grade.

The QP is satisfied that the mineralization model honours the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support resource evaluation and do not present a risk that should be considered for block classification. The Mineral Resource model is constrained by mineralized zones based on lithological, structural and grade criteria and is modelled from drill holes and underground sampling on a somewhat irregular grid, with a spacing between 5 and 40 m.

The QP considers the drill spacing to be sufficient to assume reasonable continuity of the gold mineralization. Accordingly, block estimates were classified using a combination of criteria, including confidence in the mineralization's continuity, drilling spacing and estimation results. The following classification criteria were defined:

- **Indicated:** Assigned to those continuous blocks located within one of the mineralization zones (Shear Zone, Porphyry Zone, Hangingwall Zone or Low Grade Zone) and informed by the data from at least three drill holes within a 30 m buffer. A wireframe surface, which limits and smooths the Indicated category, was also developed.
- **Inferred:** Assigned to those blocks located within one of the mineralization domains (Shear Zone, Porphyry Zone, Hangingwall Zone or Low Grade Zone) and outside the Indicated contour. An Inferred classification was also assigned to the unconstrained blocks located within the Indicated contour.

All remaining blocks were not classified.

14.9.10 Reasonable Prospects for Eventual Economic Extraction

In accordance with the CIM Definition Standards on Mineral Resources and Reserves (2014), the Mineral Resource statement must demonstrate RPEEE.

The requirement for RPEEE generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade considering extraction scenarios and processing recoveries. The QP considers the mineralization to be amenable to underground mining methods.

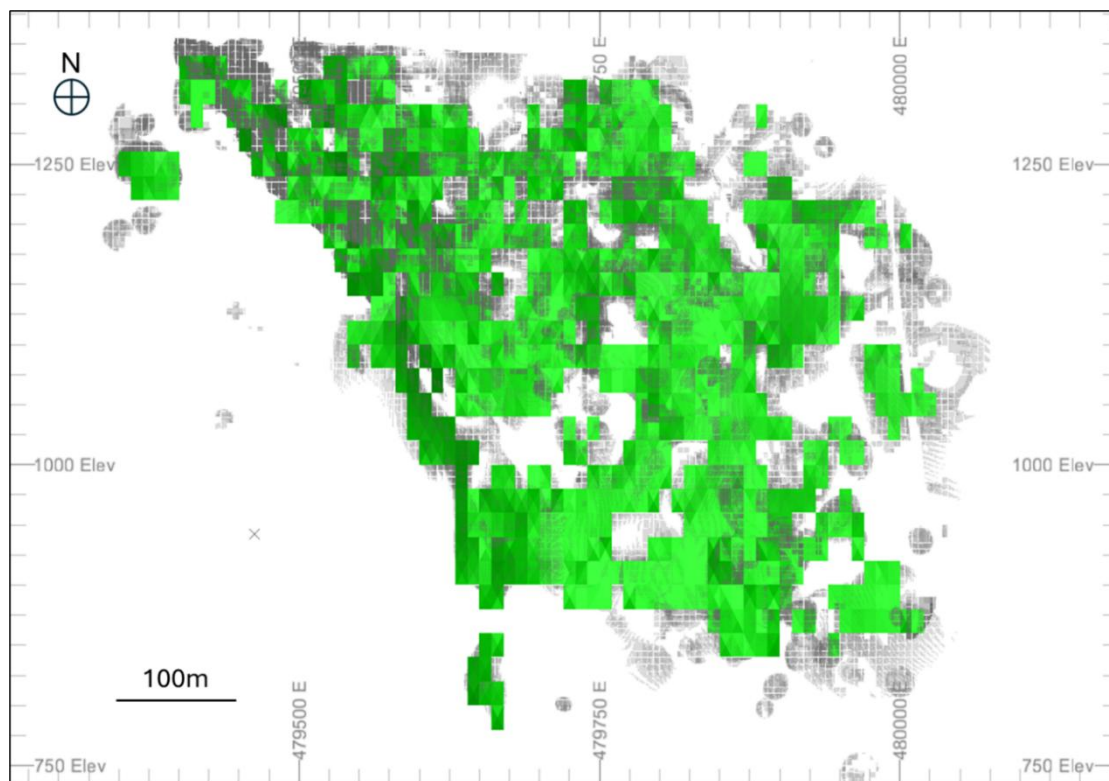
A cut-off grade of 1.35 g/t Au was determined considering the costs and input parameters summarized in Table 14-30.



Table 14-30: Fuller Optimization Parameters

Parameter	Unit	Value
Mining Cost	\$US/t	67.72
G&A Cost	\$US/t	16.60
Milling Cost	\$US/t	30.77
Transport Cost	\$US/oz	1.52
Payable Gold	%	99
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Mill Recovery	%	88

The MSO algorithm within the Datamine Studio RM software package was used to evaluate the profitability of each resource block based on its value (Figure 14-32). All classified blocks were flagged by the potentially mineable shapes including those below the 1.35 g/t Au cut-off grade. Isolated, discontinuous stope shapes were not considered economic and were therefore removed and not included in the Mineral Resource Statement.



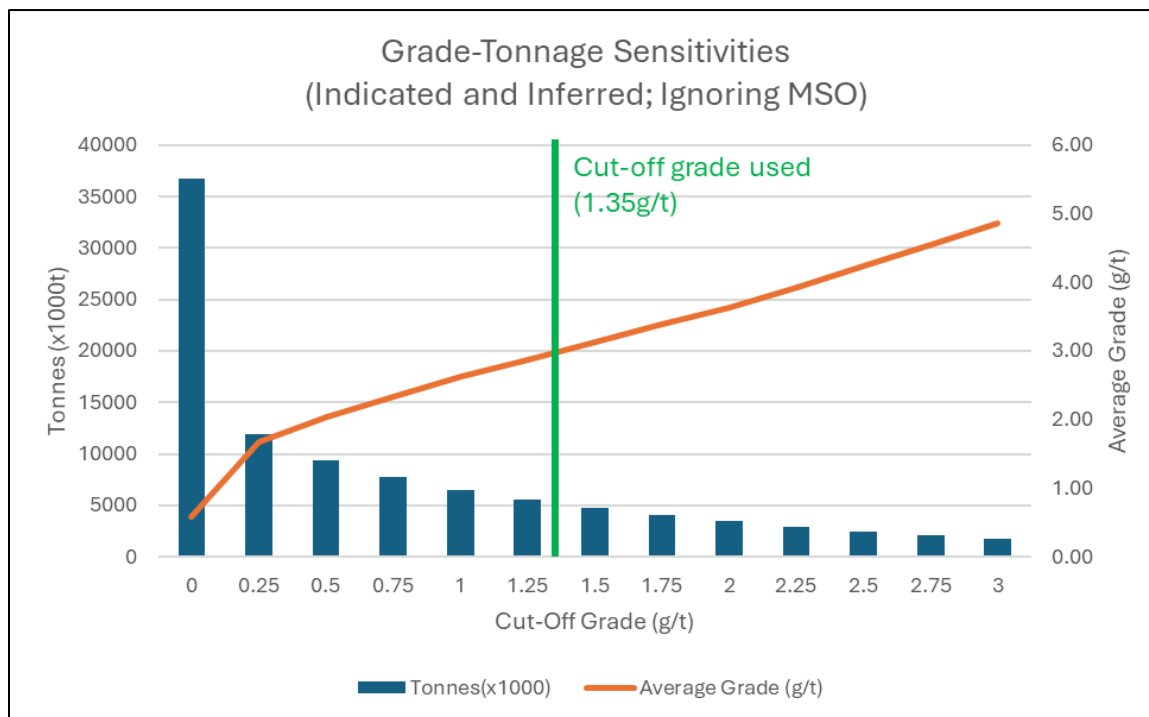
Source: (McEwen, 2026)

Note: Looking North, MSO shapes = green, Block Model above 1.35 g/t Au cut-off grade = grey

Figure 14-32: E-W Section of MSO Shapes and Block Model Above 1.35 g/t Au Cut-off Grade



Figure 14-33 presents the experimental grade–tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include both Indicated and Inferred classified blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.



Source: (McEwen, 2026)

Note: Sensitivities use raw block grades and include Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-33: Experimental Grade-Tonnage Sensitivities at Various Cut-off Grades

14.9.11 Mineral Resource Statement

Table 14-31 summarizes the Mineral Resource estimates for Fuller underground stoping methods, reported in accordance with the CIM Definition Standards on Mineral Resources and Reserves (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are RPEEEs via underground mining methods.

Table 14-31: Fuller Mineral Resource Statement, April 9, 2026

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Indicated	2,184	3.14	221
Inferred	1,625	2.19	114



Notes:

- Effective date of the Mineral Resource estimate is 9 April 2026. The QP for the estimate is Mr. Luke Willis, P.Geo, an employee of McEwen.
- Mineral Resources are reported according to the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards, 2014). Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Mineral Resources are reported above an economic cut-off grade of 1.35 g/t gold assuming underground extraction methods and based on a mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, metallurgical recovery of 88%, 10% NPI royalty, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.

14.9.12 Factors That Could Affect the Mineral Resource Estimate

Factors that could materially affect the Fuller Mineral Resource Estimate results include:

- Any new drilling and sampling information that may potentially extend the mineralization zones down dip and along strike.
- Metallurgical testwork results that might affect the recovery assumptions of the gold.
- Changes in the local geological interpretations and assumptions used to generate the estimation domains
- Changes in mineralization and geological geometry and continuity of mineralized zones
- Changes in assumptions of mineralization and grade continuity
- Changes in the treatment of high-grade gold values
- Changes in the grade interpolation methods and estimation parameter assumptions
- Changes in the confidence assumptions and methods used in the mineral resource classification
- Changes in the density and the methods used in the density assignments
- Changes in gold price and exchange rates and other economic assumptions used in the cut-off grade determination and mine optimization results
- Changes in input and design parameter assumptions that pertain to the underground mining constraints.
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.
- Changes due to social implications due to the proximity of neighbouring houses.

No other environmental, permitting, legal, title, taxation, socioeconomic, marketing, political or other relevant factors are known to the QP that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

14.10 Davidson-Tisdale

In May 2017, SRK prepared and completed an internal Mineral Resource Estimate for the Davidson-Tisdale gold deposit under a commission by Lexam. After the acquisition of Lexam by McEwen, SRK was again commissioned to prepare a Mineral Resource statement for the 2021 PEA for Davidson-Tisdale using updated economic parameters. In April 2026, McEwen (as QP) updated the Mineral Resource statement for Davidson-Tisdale using updated economic parameters. No additional resource drilling has been completed at Davidson-Tisdale recently, and the existing 2017 resource model was used as a basis for reporting purposes.



14.10.1 Mineral Resource Database

The resource database comprises primarily samples from surface and underground core drilling. The database comprises 691 drill holes with a total length of 80,026 m. The total number of samples with gold assays is 24,162 (26,665 m). The database also includes 80,847 m of lithological, structural and alteration logging.

Six level plans were digitised and used as well as the existing geological and structural maps to guide mineral resource modelling.

On importing the drill hole database to Datamine Studio and Leapfrog software several validation steps were performed and any errors revealed were corrected.

14.10.2 Geological Modelling

14.10.2.1 Mineralized Areas

Gold mineralization at Davidson-Tisdale belongs to the class of structurally controlled Archean lode gold deposits. The mineralization is associated with the quartz vein zones whose distribution and orientation generally corresponds with the structure of the faults.

Two main areas of mineralization were modelled. The north area comprises the most abundant and high-grade mineralization with a general strike of 060° and a dip of 30 to 50° northwest. Mineralization in the north area also has strike lengths up to 300 m and widths up to 20 m. The south area is lower grade, but with good continuity, an eastern strike of 060° and a dip of 10 to 20° north. Mineralization in the south area has strike lengths up to 700 m and widths up to 25 m.

14.10.2.1.1 North Area

Three zones of mineralization were modelled within the north area: Zones A, B and C. The B Zone was split into Zones B1 and B2 by the Main fault.

A combination of lithological, alteration, structural and grade criteria were used to define the resource domains. The lithological criteria included the presence of foliation, shearing and quartz veins. The grade threshold was chosen as 0.4 g/t Au. The intervals were prioritized based on the satisfaction of one or more criteria as follows:

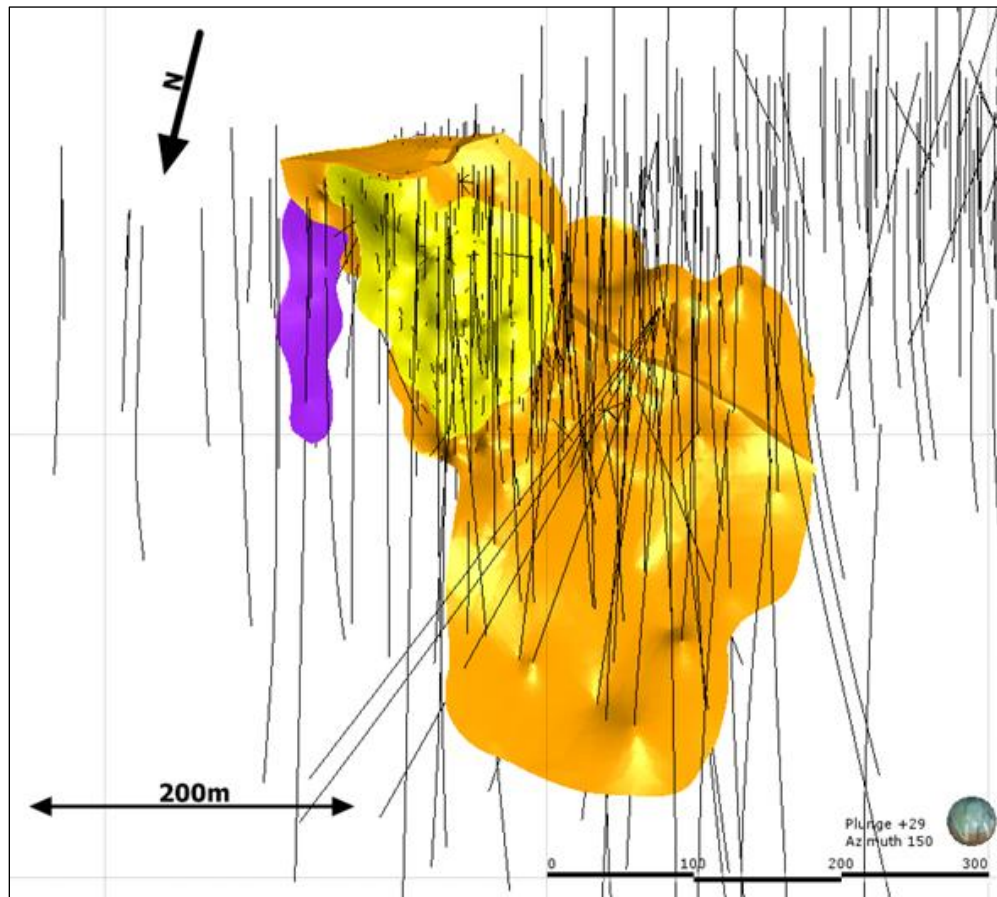
- High priority intervals – lithological + grade criteria
- Medium priority intervals – only grade criteria
- Low priority intervals – only lithological criteria.

The low priority intervals were used only to maintain the continuity of the mineralized zone and used with adjacent high and medium priority intervals.

Considering the good general continuity, but high variability on a smaller scale, the mineralized zone wireframes were modelled in two steps:



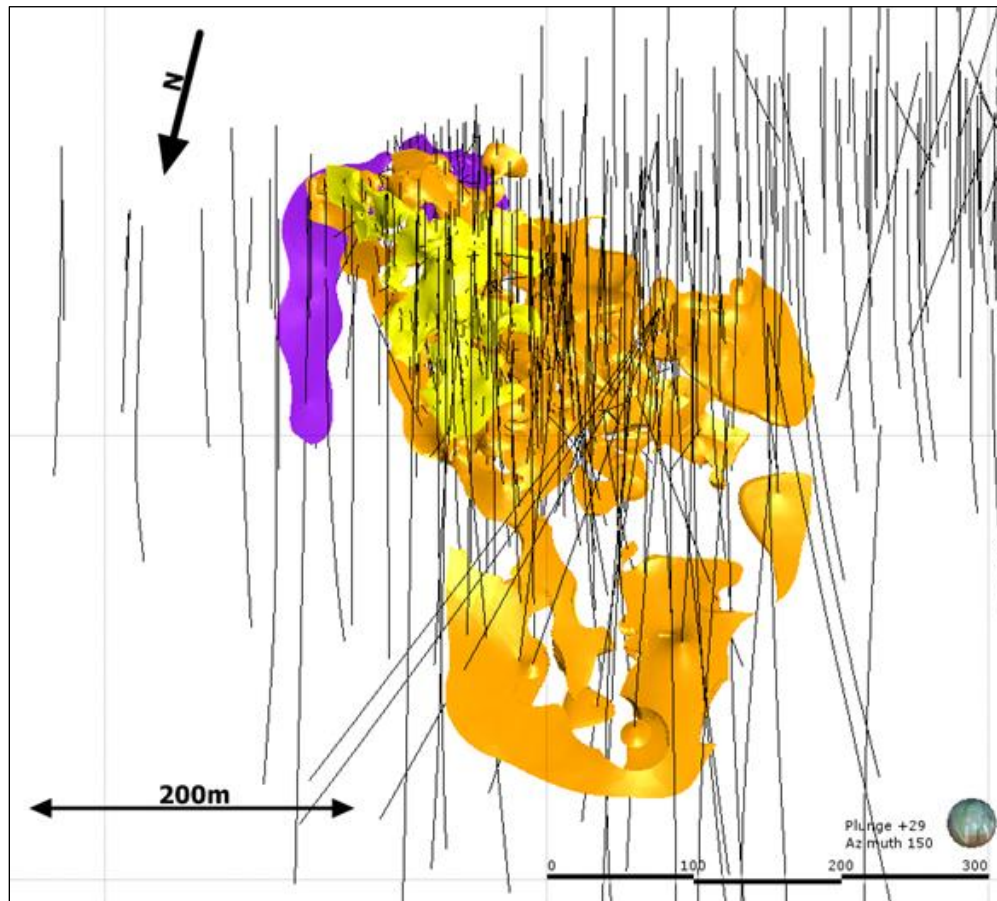
- Generation of the external general wireframe manually containing some internal barren intervals within it (Figure 14-34).
- Generation of the internal wireframe within the external contour with semi-automatic wireframing (Figure 14-35) algorithms which excludes the barren intervals and follows the general trend of the mineralized zone. This wireframe was used to constrain grade estimation.



Source: (prepared by SRK, 2021; reissued by McEwen, 2026)

Note: A Zone – purple, B Zone – orange, C Zone – yellow

Figure 14-34: Isometric View of the Davidson-Tisdale North Area, **External** Contours



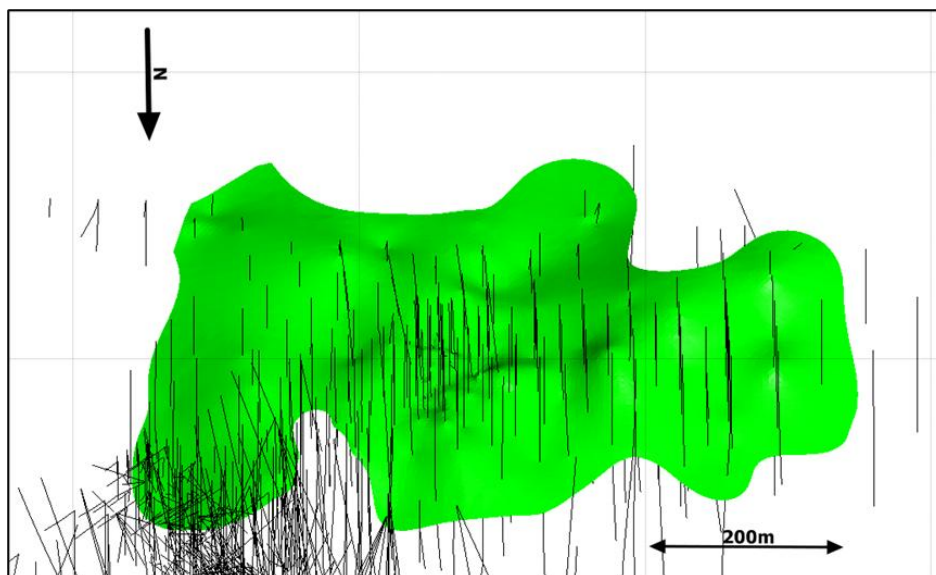
Source: (prepared by SRK, 2021; reissued by McEwen, 2026)

Note: A Zone – purple, B Zone – orange, C Zone – yellow

Figure 14-35: Isometric View of the Davidson-Tisdale North Area, **Internal Contours**

14.10.2.1.1 South Area

South Area wireframe (S Zone) was modelled based on quartz veins lithological criteria only. The S Zone is described as having very good continuity, but relatively low grades so no internal wireframe was constructed (Figure 14-36).



Source: (prepared by SRK, 2021; reissued by McEwen, 2026)

Figure 14-36: Isometric View Looking South of the S Zone Wireframe

14.10.3 Composites

Most of the analytical samples within the mineralized areas were collected at 0.5 m and 1 m intervals. A modal composite length of approximately 1 m was applied to all the data, generating composites as close to 1 m as possible, while creating residual intervals of up to 0.5 m in length.

14.10.4 Capping

The impact of high-grade outlier populations was examined on composite data using log probability plots and cumulative statistics separately for all mineralized areas. A capping value of 110 g/t was applied for the B Zone and 100 g/t for the C and S Zones. No capping was used for the A Zone. Results are presented in Table 14-32 for each zone with metal removed, ranging from 6% in C Zone to 29% in B Zone.

Table 14-32: Davidson-Tisdale – Metal Removed by Capping Composites

Domain	Raw Composites		Capped Grade (g/t)	Capped Composites		No. Capped	Metal Removed (%)
	Mean (g/t)	CV		Mean (g/t)	CV		
North Zone							
A Zone	5.54	1.61	-	5.54	1.61	-	-
B1+B2 Zone	5.59	6.40	110	3.99	3.16	16	29
C Zone	4.56	3.44	100	4.29	3.12	5	6
South Zone							
S Zone	1.71	7.33	100	1.45	6.01	5	15

Note: CV = coefficient of variation



14.10.5 Density

In 2012, a total of 55 core samples were taken and analyzed at AGAT (Altman et al., 2014). The average bulk density of 2.87 g/cm³ of the 55 samples was applied to this resource estimate for Zones A, B and C.

For Zone S, a more conservative density value of 2.70 g/cm³ was used on Lexam's recommendation.

14.10.6 Variography

The spatial distribution of gold in the North and South areas was evaluated using variograms and correlograms.

Variograms/correlograms were modelled using two spherical structures and three rotations to match the strike, dip and plunge of the modelled mineralization. Nugget effects are 40% for both areas.

14.10.7 Grade Estimation

The criteria used in the selection of block size included the drill hole spacing, composite length, the geometry of the modelled zone, and the anticipated mining method. A block size of 5 m x 5 m x 5 m was selected. Sub-cells were used allowing a resolution of 0.5 m x 0.5 m x 0.5 m to better reflect the shape of the mineralized area. No rotation was applied to the block model.

Ordinary kriging was used to estimate the blocks within the mineralized zones using a three -pass approach in the North Area and four passes in the South Area. An initial pass in the S Zone focused on estimating areas where there were high-grade composite values. Capped composites with a tight isotropic search ellipse were used to limit the number of blocks estimated to one or two in each direction. Subsequent passes were made with composites capped to 10 g/t gold to minimize smearing of high-grade outliers in lower grade areas and increasing search ranges based on the variogram ranges. Datamine's dynamic anisotropy function was implemented due to the anisotropic nature and widths of the mineralized zones.

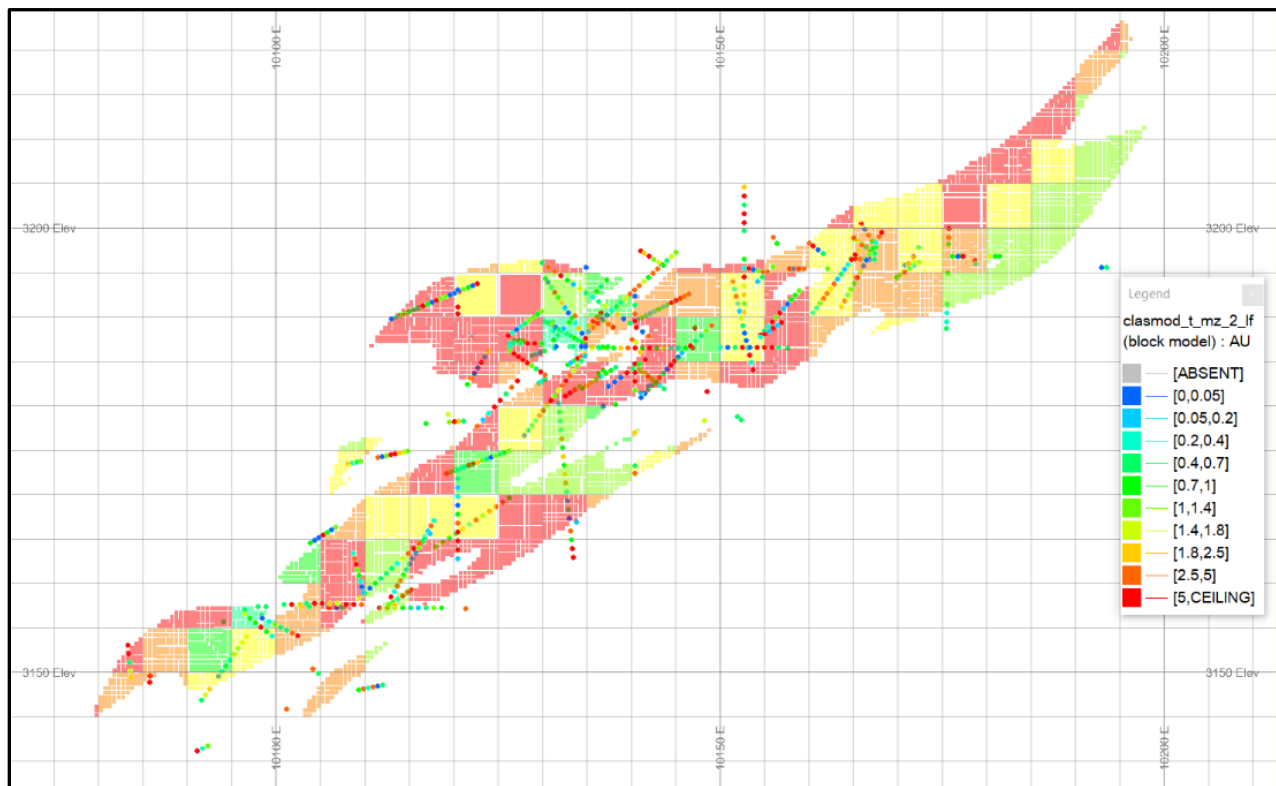
Data selection for mineralized zones varied for each pass requiring a minimum number of composites between seven and four and a maximum number of composites between 10 and 16 for passes 1 to 3. Estimated blocks required at least three holes in pass 1, and two holes in pass 2 and 3.

14.10.8 Model Validation

The block model was validated by visual comparison of informing sample data with resource blocks data (on plan and section, see Figure 14-37) and with swath plots (section by section) comparing the ordinary kriged blocks with the informing composite data.

The results of the validation show that the block model acceptably reflects the assay sample data. The deviations between the averages on swath plots appear within the areas of the low data density and within the S Zone where more restrictive capping was applied.





Source: (prepared by SRK 2021; reissued by McEwen 2026)

Note: East-West at Y=9,920

Figure 14-37: Cross-Section Comparing B Zone Estimated Blocks and Composite Data

14.10.9 Confidence Classification

Industry best practices suggest that classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification as well as the continuity of the deposit at the reporting cut-off grade.

The QP is satisfied that the mineralization model honours the current informing data from the geological database. The location of the samples and the assay data are sufficiently reliable to support resource evaluation and do not present a risk that should be considered for block classification. The Mineral Resource model is constrained by mineralized zones based on lithological, structural and grade criteria and is modelled from drill holes on a somewhat irregular grid, with a spacing between 5 m and 40 m.

The QP considers the drill spacing to be sufficient to assume reasonable continuity of the gold mineralization. Accordingly, block estimates were classified using a combination of criteria, including confidence in the mineralization's continuity, drilling spacing and estimation results. The following classification criteria were defined:



- **Measured:** Assigned to those continuous blocks located within Zones B and C of the North Area and informed by the data from at least three drill holes within a 15 m buffer. The wireframe surface, which limits and smooths the Measured category, was also developed.
- **Indicated:** Assigned to those continuous blocks informed by the data from at least three drill holes within one search radius (one variogram range). The wireframe surface, which limits and smooths the Indicated category, was also developed.
- **Inferred:** Assigned to all remaining blocks.

14.10.10 Reasonable Prospects for Eventual Economic Extraction

In accordance with the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards, 2014), the Mineral Resource statement must demonstrate RPEEE.

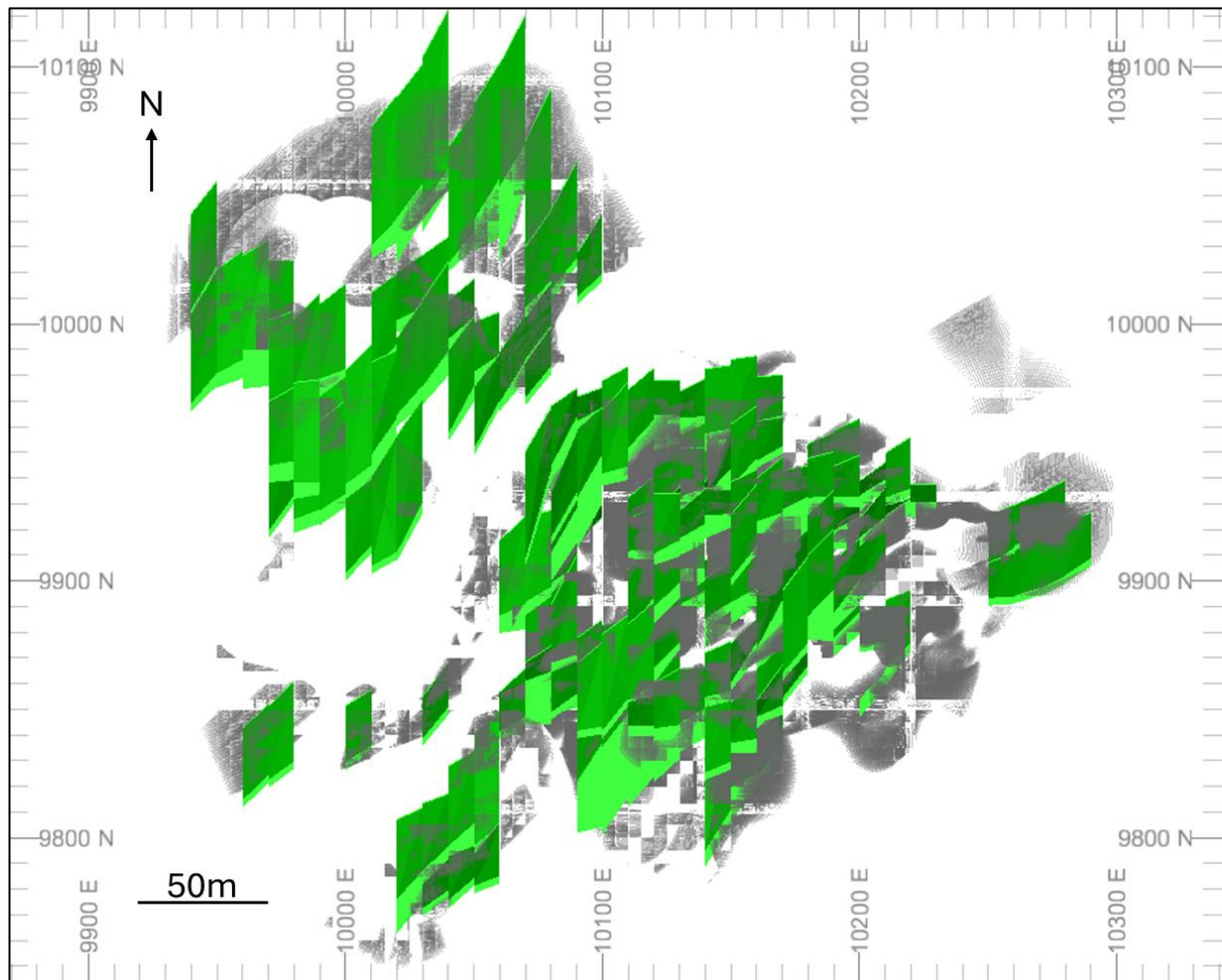
The requirement for RPEEE generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade considering extraction scenarios and processing recoveries. The QP considers the mineralization is amenable to underground extraction. A cutoff grade of 1.30 g/t Au was determined considering the cost and input parameters summarized in Table 14-33.

The MSO algorithm within Datamine Studio RM software package was used to evaluate the profitability of each resource block based on its value (Figure 14-38). All classified blocks were flagged by the potentially mineable shapes including those below the 1.30 g/t Au cut-off grade. Isolated, discontinuous stope shapes were not considered economic and were therefore removed and not included in the Mineral Resource Statement.

Table 14-33: Davidson-Tisdale Optimization Parameters

Parameter	Unit	Value
Mining Cost	\$US/t	67.72
G&A Cost	\$US/t	16.60
Milling Cost	\$US/t	30.77
Transport Cost	\$US/oz	1.52
Payable Gold	%	99
Dilution	%	15
Gold Price	\$US/oz	3,000
Exchange Rate (US\$/C\$)	-	1.35
Mill Recovery	%	92



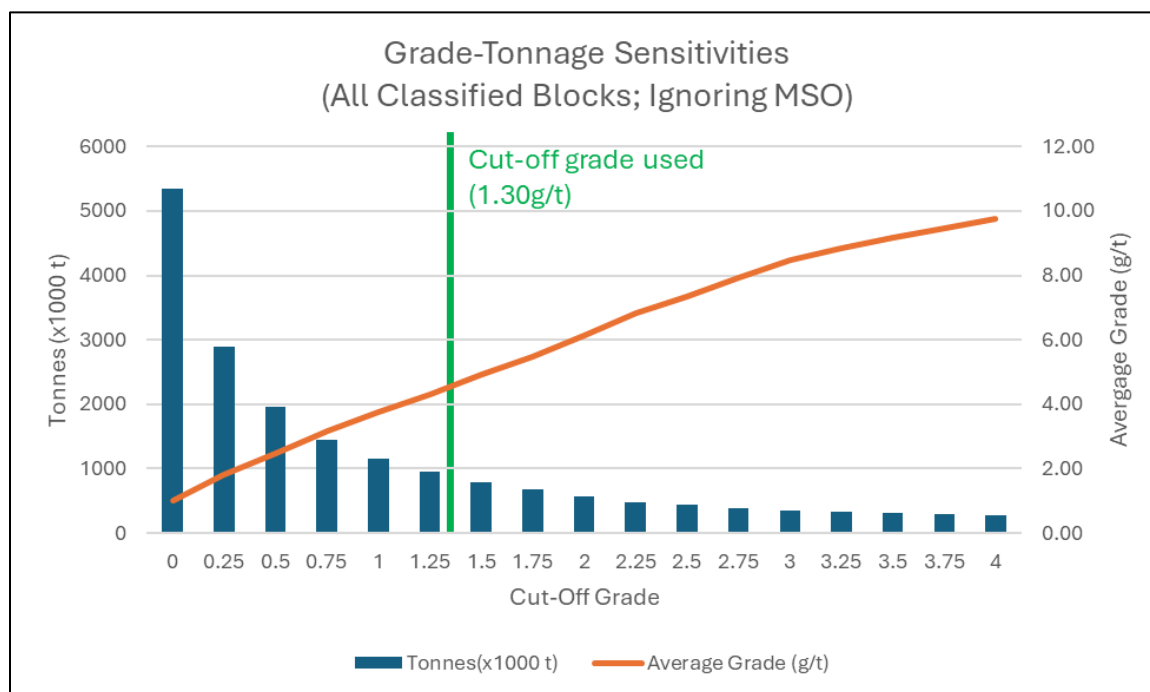


Source: (McEwen, 2026)

Note: MSO shapes = green, Block model above 1.30 g/t Au cut-off grade = grey

Figure 14-38: Plan View, Davidson-Tisdale MSO Shapes, Block Model Above 1.30 g/t Au Cut-off Grade

Figure 14-39 presents the experimental grade–tonnage sensitivities at various cut-off grades. The sensitivities were generated using raw block grades and include Measured, Indicated and Inferred classified blocks. Sensitivities include all classified blocks, not only those meeting RPEEE. These results are provided for comparative purposes only, to illustrate the potential effect of applying different cut-off grades on tonnage and grade. Sensitivity cases do not represent Mineral Resources and should not be interpreted as such.



Source: (McEwen, 2026)

Note: Sensitivities use raw block grades and include Indicated and Inferred blocks. These results are for comparison only and do not represent Mineral Resources.

Figure 14-39: Experimental Grade-Tonnage Sensitivities at Various Cut-off Grades

14.10.11 Mineral Resource Statement

Table 14-34 summarizes the Mineral Resource estimates for Davidson-Tisdale assuming underground stoping mining methods, reported in accordance with the CIM Definition Standards on Mineral Resources and Reserves (2014). The Mineral Resource was limited to those parts of the gold mineralization for which there are RPEEEs via underground mining methods.

Table 14-34: Davidson-Tisdale Mineral Resource Statement, April 9, 2026

Classification	Tonnes (kt)	Au Grade (g/t)	Contained Au (koz)
Measured	198	6.44	41
Indicated	95	5.80	18
Total Measured + Indicated	294	6.23	59
Inferred	200	3.44	22

Notes:

- Effective date of the Mineral Resource estimate is 9 April 2026. The QP for the estimate is Mr. Luke Willis, P.Geo, an employee of McEwen.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Mineral Resources are reported above an economic cut-off grade of 1.30 g/t gold assuming underground extraction methods and based on a mining cost of \$67.72/t, process cost of \$30.77/t, G&A cost of \$16.60/t, metallurgical recovery of 92%, dilution of 15% and gold price of US\$3,000/oz.
- Figures may not sum due to rounding.



14.10.12 Factors That Could Affect the Mineral Resource Estimate

Factors that could materially affect the Mineral Resource Estimate results include:

- Any new drilling and sampling information that potentially extend the mineralization zones down dip and along the strike.
- Metallurgical testwork results might affect the recovery assumptions of the gold and other metals.
- Changing gold prices may alter the mine optimization results.
- Changes in the local geological interpretations and assumptions used to generate the estimation domains
- Changes in mineralization and geological geometry and continuity of mineralized zones
- Changes in assumptions of mineralization and grade continuity
- Changes in the treatment of high-grade gold values
- Changes in the grade interpolation methods and estimation parameter assumptions
- Changes in the confidence assumptions and methods used in the mineral resource classification
- Changes in the density and the methods used in the density assignments
- Changes in metal price and exchange rates and other economic assumptions used in the cut-off grade determination
- Changes in input and design parameter assumptions that pertain to the underground mining constraints
- Changes to assumptions as to the continued ability to access the mine site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.



15 Mineral Reserve Estimates

15.1 Forward Looking Statement

The accuracy of the Mineral Reserve estimate is dependent, in part, on the quality and quantity of available data, as well as on geological interpretation and professional judgments. Based on the data available at the time this report was prepared, the estimate presented herein is considered reasonable. However, these should be accepted with the understanding that additional data or analyses obtained subsequent to the date of the estimate may necessitate revision, and such revisions may be material. There is no guarantee that all or any part of the estimated mineral reserve will be recoverable. Estimates of mineralization and other technical information presented herein have been prepared in accordance with the requirements of NI 43-101.

15.2 Risks

The Mineral Reserve is reported herein, in accordance with the 2014 CIM Definition Standards.

Factors that may affect the Mineral Reserve estimate include, but are not limited to, the following.

- Commodity price assumptions.
- Changes in interpretations of mineralization geometry and continuity of mineralization zones.
- The presence of unrecognized fault structures that could affect mine design, ground control costs, and/or ore dilution.
- Changes to geotechnical/geomechanical, hydrogeological, and metallurgical recovery assumptions.
- Input parameters used to assess dilution and recoveries.
- Assumptions that all required permits to operate can be obtained.
- Assumptions regarding social, permitting, and environmental conditions.
- Infill and/or definition diamond drilling.

The QP has reviewed the risks, opportunities, conclusions, and recommendations and is not aware of any conditions that would put the Mineral Reserve estimate at a high-risk level.

15.3 Mineral Reserve Estimate

15.3.1 Underground Mineral Reserve Estimation

The Grey Fox Mineral Reserve estimate, prepared by Stantec on March 27, 2026, is based on Indicated Mineral Resources identified in the block model provided by McEwen (rm_gf_1p5_0725_streamlined.dm, issued on September 10, 2025). All Mineral Resource material in the block model that was not classified as Measured or Indicated was removed prior to use. Table 15-1 outlines the estimated Mineral Reserve.



Table 15-1: Mineral Reserve Estimate, March 27, 2026

Classification	Ore Tonnes (t)	Au (g/t)	Gold Ounces (Oz)
Proven	0	0.00	0
Probable	9,405,829	3.24	980,303
Total Reserve	9,405,829	3.24	980,303

Notes:

- CIM definitions were followed for the Mineral Reserve
- Mineral reserve includes longitudinal stopes, overhand cut-and-fill stoping, ore development, as well as marginal longitudinal stopes, and marginal development all factored with external dilution (including host waste rock and estimated backfill tonnes) and mining recovery.
- Longitudinal stopes were estimated at an undiluted cut-off value of 2.40 g/t.
- Marginal longitudinal stopes were estimated at an undiluted cut-off value of 2.00 g/t.
- Cut-and-fill stopes were estimated at an undiluted cut-off value of 2.70 g/t.
- A minimum mining width of 3.0 m was applied for all stoping.
- Total dilution is 46.5%, consisting of 11.6% external dilution and 34.9% internal dilution.
- Total mineable recovery is 91.2%.
- Some numbers may not sum due to rounding.

Table 15-2 presents the Mineral Reserve estimate, categorized by mining method.

Table 15-2: Mineral Reserve Estimate by Mining Method

Mining Method	Ore Tonnes (t)	Au (g/t)	Gold Ounces (Oz)
Longitudinal Longhole Stoping	6,812,141	3.55	777,042
Marginal Longitudinal Longhole Stoping	910,806	1.85	54,090
Cut-and-Fill Stoping	117,984	3.94	14,959
Development Ore	1,564,899	2.67	134,212
Total Reserve	9,405,829	3.24	980,303

Table 15-3 presents the Mineral Reserve estimate, categorized by mining zones.

Table 15-3: Mineral Reserve Estimate by Zone

Zone	Ore Tonnes (t)	Au (g/t)	Gold Ounces (Oz)
Whiskey Jack	692,256	4.37	97,165
Remaining Grey Fox North (Gibson, Contact, and 147NE)	4,997,799	2.95	473,751
Grey Fox South (147 and South)	3,715,774	3.43	409,387
Total Reserve	9,405,829	3.24	980,303



15.3.2 Cut-off Grade

The Grey Fox mineral resource is gold. The cut-off grade will be based on the gold grade field (AU_C) within the block model, expressed in grams per tonne (g/t). To determine a mineral reserve for Grey Fox, a preliminary cut-off grade was estimated prior to preparing the mine design. The cost parameters used to estimate the cut-off grade were provided by McEwen in September 2025. The mining cost for cut-and-fill was estimated using data from McEwen's Black Fox and Froome deposits, as well as other benchmarks. The surface trucking cost was estimated from the provided mining cost.

The marginal cut-off grades for longhole stoping exclude sustaining capital costs. A processing-and-trucking-only cut-off grade was used to evaluate ore development, with other costs considered sunk, as the development would be required as waste regardless. Final cut-off grade values were rounded up to the nearest tenth of a gram.

Table 15-4 presents the gold revenue assumptions, modifiers, and unit costs used in the cut-off grade calculation. Table 15-5 presents the resulting cut-off grade calculations, adjusted for mill recovery and estimated mining dilution.

For longhole stoping, the mill feed cut-off grade is 2.1 g/t, with a marginal mill feed cut-off grade of 1.7 g/t. The undiluted target stope shape cut-off grade is 2.4 g/t, with a marginal undiluted target stope shape cut-off grade of 2.0 g/t.

For cut-and-fill mining, the mill feed cut-off grade is 2.4 g/t. The undiluted target stope shape cut-off grade is 2.7 g/t.

For development material, the mill feed cut-off grade is 0.5 g/t. The undiluted target development cut-off grade is also 0.5 g/t.



Table 15-4: Gold Revenue Assumptions, Modifiers, and Unit Costs Used in cut-off grade Calculations

Item	Value	Unit
Gold Revenue & Modifiers		
Gold Price	2,200	\$USD/oz
Refining Cost	0.55	\$USD/oz
Transport and Insurance	2.05	\$USD/oz
Excess Liability	0.92	\$USD/oz
Payable Gold	99.95	%
Penalties	0.00	%
Royalties	2.50-3.00	%
Revenue per Ounce after fees and royalties	2,140.54	\$USD/oz
Revenue per Gram after fees and royalties	68.82	\$USD/g
Unit Costs		
Mine Operating Cost (Longhole)	\$62.72	\$USD/t
Mine Operating Cost (Cut-and-Fill)	\$83.03	\$USD/t
Surface Trucking Cost	\$5.00	\$USD/t
Mill Operating Cost (Processing)	\$23.43	\$USD/t
General and Administration Operating Costs	\$16.60	\$USD/t
Sustaining Capital Cost (including reclamation and exploration)	\$20.91	\$USD/t

Table 15-5: Cut-off Grade Calculations

Cut-off Grade	Longhole	Marginal Longhole	Cut-and-Fill	Ore Development	
Item	Value	Value	Value	Value	Unit
Total Cost per Tonne	128.65	107.75	148.97	28.43	\$USD/t
Grams Gold per Total Cost per Tonne	1.87	1.57	2.16	0.41	g/t
Mill Recovery	90	90	90	90	%
Mill Feed Cut-off Grade	2.1	1.7	2.4	0.5	g/t
Estimated Mining Dilution	15	15	10	10	%
Undiluted Cut-off Grade	2.4	2.0	2.7	0.5	g/t



15.3.2.1 Consideration of Marginal Cut-off Grade and Low-Grade Material

Marginal material is below the planned break-even cut-off value but can still be processed economically. Marginal stoping occurs where gaps between stopes, in planned stoping sequences can be filled by stopes mined at a lower cut-off grade. The cut-off grade calculation for marginal stoping excludes sustaining capital and infrastructure costs, as this stoping occurs in areas where development is already planned for break-even stoping and the associated development costs are therefore considered sunk.

Ore development material is generated typically in operating crosscuts and sill development. It can be stockpiled separately on a per-round basis as low-grade material and used to fill gaps in the mill schedule without displacing higher-grade material.

15.3.3 Modifying Factors

Modifying factors include dilution and mining recovery, both of which were applied in the estimation of Mineral Reserve. The following sub-sections describe these modifying factors and the application to the mine design.

15.3.3.1 Internal Dilution

Internal dilution comprises of low-grade material and/or waste rock contained within the stope shape or development shape that will be mined along with the mineralized resource. This internal dilution is included within the mining shapes and reported as part of the reserve. Internal dilution will be defined as material below the mining method's cut-off grade as determined by the block model. Where no grade is assigned in the block model, the grade will be assumed to be 0.0 g/t.

Stoping-only internal dilution is estimated to be 35.3%, while overall internal dilution, including development, is estimated at 34.9%.

15.3.3.2 External Dilution and Backfill Dilution

External dilution includes low-grade material, waste rock, and/or backfill from the HW, FW, and/or end walls that will overbreak into the stope and be mined with the stope resource. External dilution will be calculated during the mine scheduling process, and the methodology is presented in Table 15-6.

Stope-only external dilution is estimated at 14.2%, of which stope backfill dilution accounts for 0.7%. Overall external dilution, including development, is estimated at 11.6%, with stope backfill dilution accounting for 0.5% of the total.

Table 15-6: Stope Dilution Methodology

Dilution Source	Methodology
External Dilution	Equivalent Linear Overbreak and Sloughage
	HW – 0.3 m of overbreak
	FW – 0.3 m of overbreak
	Stope End Walls (Ribs) – 0.3 m of overbreak where no stope adjacent



Dilution Source	Methodology
	Dilution grade 0.0 g/t
Backfill Dilution	0.3 m of overbreak for each exposed adjacent surface at Stope End Walls (Ribs). Dilution grade 0.0 g/t

15.3.3.3 Mining Recovery

A mining recovery factor has been applied to all stopes to account for losses incurred during the stoping process. These losses reflect material that is not recovered during extraction of blasted ore. Mining losses from longhole stopes were estimated based on the mining method and average stope size. Factors influencing mining losses include mucking line of sight, depth of sight, possible hang-ups on the footwall, and blast-related complications, especially with up-hole stopes. In some instances, rib pillars may also be required for up-hole stopes, as these stopes cannot be backfilled, and these losses are accounted for in the recovery factors. The mining recovery factors are summarized in Table 15-7.

Stope-only mining recovery is estimated at 89.5%, while overall mining recovery, including development, is estimated at 91.2%.

Table 15-7: Mining Recovery Factors

Ore Source	Mining Recovery Factor
Longitudinal Longhole Stope	95%
Up-hole only Stope (<20 m high)	75%
Up-hole only Stope (20 to 30 m high)	65%
Cut-and-Fill Stope	98%
Ore Development*	100%

Note: * Assumed unrecovered ore development tonnes are expected to be recovered during stoping.



16 Mining Methods

Grey Fox will be a combination of two underground operations accessed from two separate portals: Grey Fox North, to be excavated from the existing Gibson Ramp and Portal, which includes the Whiskey Jack, Gibson, Contact, and 147NE zones, and Grey Fox South, which includes the 147 and South zones. The zones are portrayed in Figure 16-1.

The Fox Complex is a group of gold properties in Ontario extending eastward from Timmins along Provincial Highway 101. Located 11 km east of Matheson is the Eastern Fox Complex, which include Black Fox open pit mine and the Froome underground mine. Grey Fox is located approximately 3 km southeast of the Black Fox Mine which is approximately 9 km east of Matheson in Hislop Township. Access to the site is via Tamarack Road off Highway 101 East.

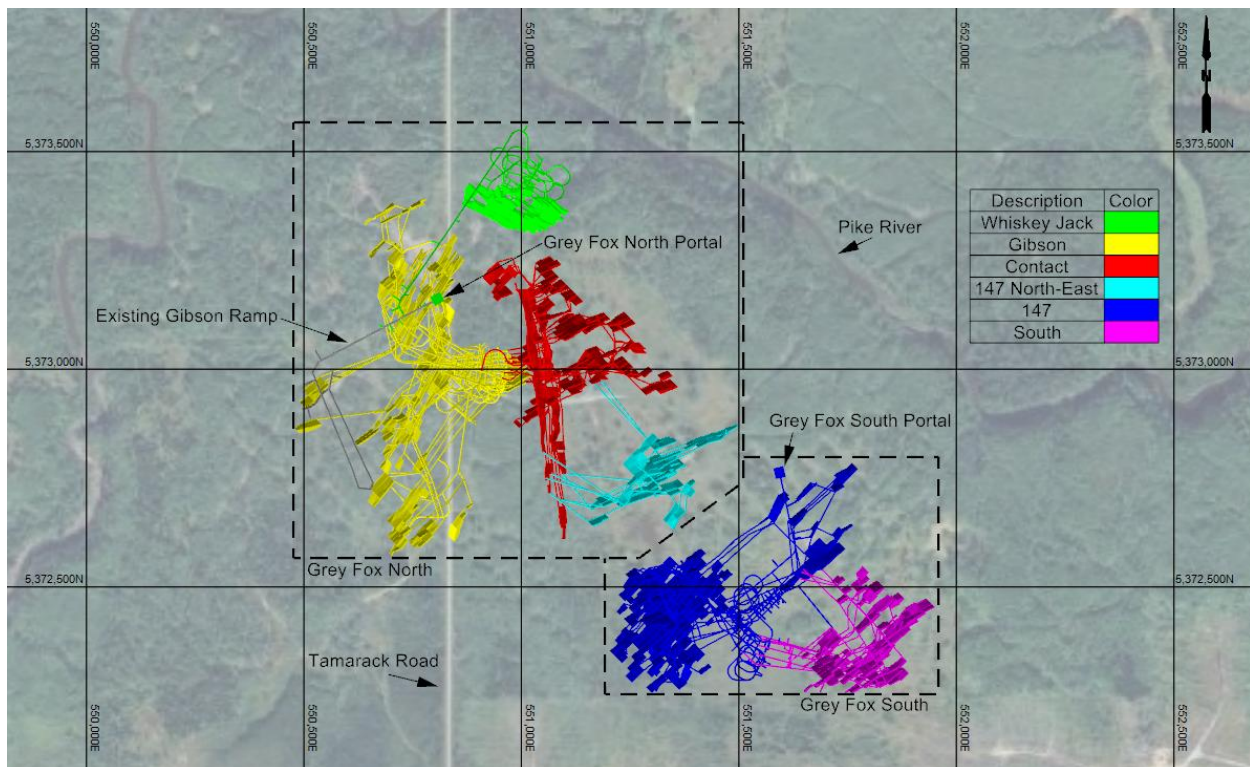


Figure 16-1: Grey Fox Mine Zones in Plan View with Google Maps Surface

These zones will be mined using a longitudinal retreat longhole mining method supplemented with overhand cut-and-fill mining. Ore will be hauled on surface to the existing Srock Mill at McEwen's Stock Property as part of a "hub and spoke" asset strategy for the McEwen Assets along Highway 101.

Project Period development will initiate at the existing Grey Fox North portal in August 2027 where the existing Gibson ramp will be rehabilitated and expanded to 4.0 m W x 4.3 m H approximately 75 m down the ramp. From the Gibson ramp, a new 4.0 m W x 4.3 m H ramp will be excavated northwards towards



Whiskey Jack which will begin mining at a target of 650 t/d. The existing Gibson ramp will be rehabilitated to approximately 130 m in length total concurrent with development towards Whiskey Jack.

The Whiskey Jack Zone has 9 levels, split into two blocks, at 30 m vertical intervals (except the upper two levels which are 25 m). Figure 16-2 presents a longitudinal projection of the Whiskey Jack Zone.

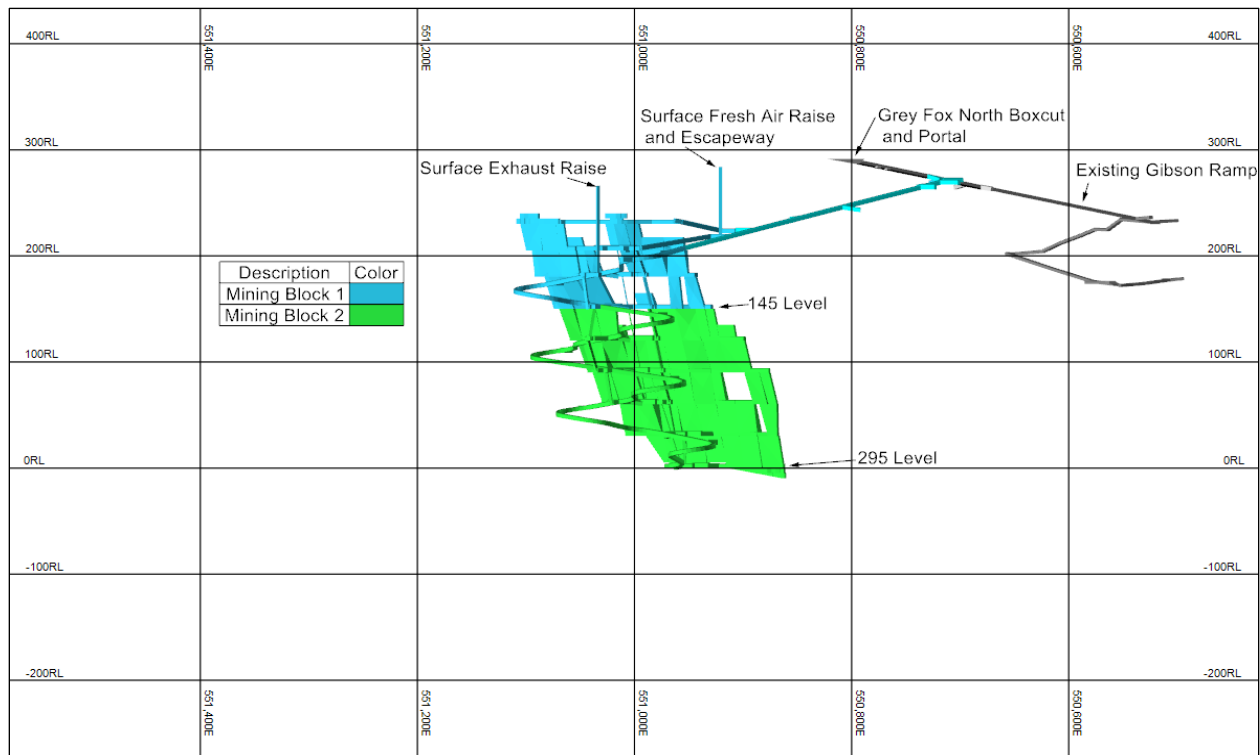


Figure 16-2: Whiskey Jack Zone looking South

In August 2028, as capital development comes to completion in Whiskey Jack, a second, new 4.0 m W x 4.3 m H ramp will be excavated southwards from the existing Gibson ramp towards the remaining Grey Fox North zones (Gibson, Contact, and 147NE). In January 2030, there will be multiple mining fronts in the Grey Fox North first mining block to support a target of 1,500 t/d marking the end of the project period and beginning of the operating period.

The remaining Grey Fox North has a total of 17 levels at 25 m vertical intervals and will be split into four blocks. Figure 16-3 presents a longitudinal projection of Grey Fox North excluding the Whiskey Jack Zone. The mine life of Grey Fox North will be approximately 14.5 years, including an initial ramp up period of approximately 2.5 years.

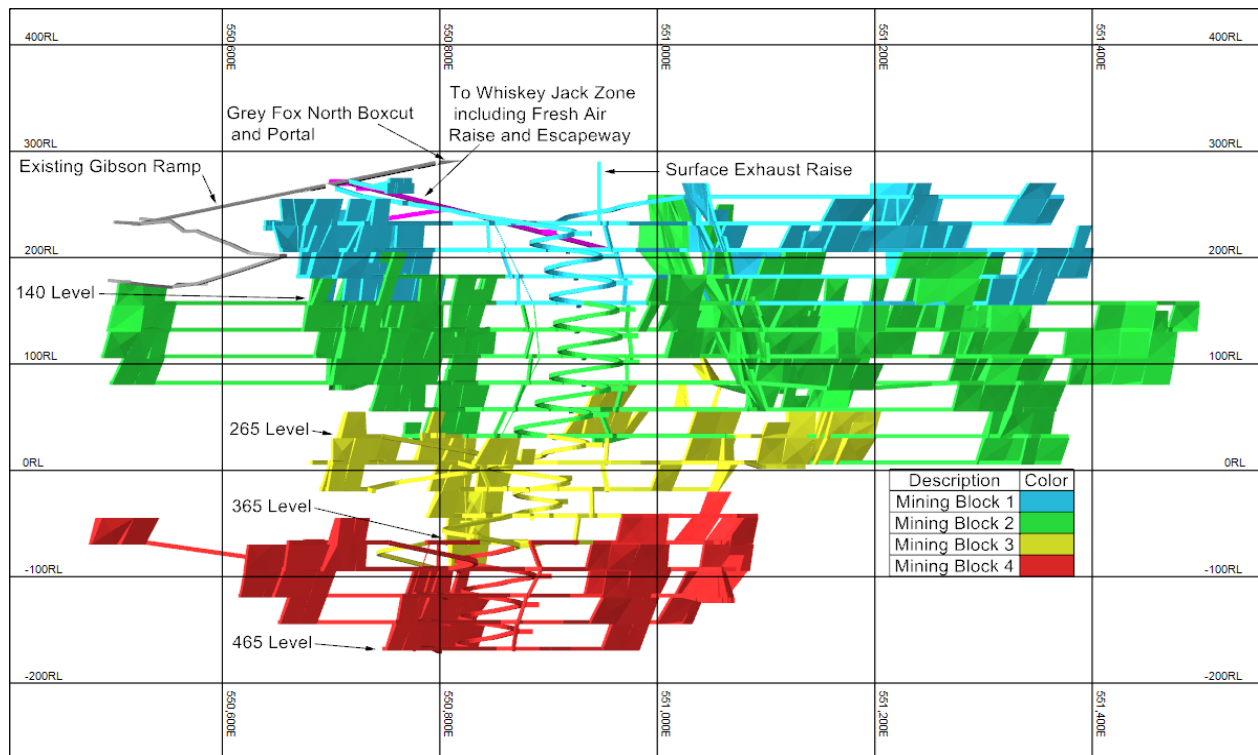


Figure 16-3: Grey Fox North looking North

As mining at Grey Fox North progresses into the second mining block, in April 2033, Grey Fox South underground development commences to establish the infrastructure for the first mining block in preparation for the first Grey Fox South stope in January 2035. In January 2035, Grey Fox South mining will proceed concurrently with Grey Fox North at a target of 2,700 t/d until 2040, when the ramp down period starts until December 2041.

The Grey Fox South has 12 levels at 25 m intervals and will be split into three blocks. The mine life of Grey Fox South will be approximately 8.5 years, including an initial ramp up period of 1.5 years. Figure 16-4 presents a longitudinal projection of Grey Fox South.

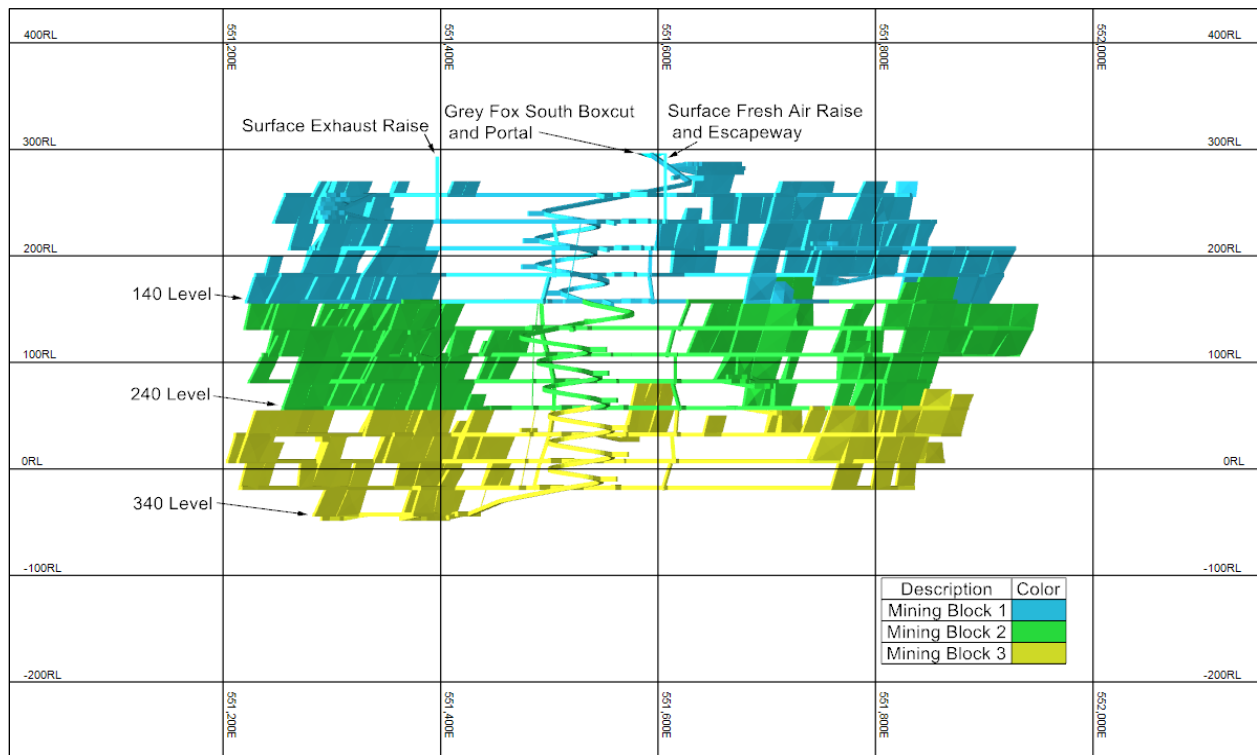


Figure 16-4: Grey Fox South Looking North

To achieve these phased production targets, the mine plan includes production from multiple mining blocks, with multiple stopes available per block. The aforementioned overall production profile and tonnage targets are expected to be complimentary with production from other assets, notably the stock mine, in maximizing the throughput of the Stock Mill.

Conventional trackless mining equipment will be used for the lateral development required to access the zones. Longhole stopes will be drilled using a combination of ITH (primarily for slot drilling) and top hammer longhole drills, blasted using ANFO with some packaged emulsion explosives, and mucked using conventional LHD vehicles. Ore will be hauled to a primary crusher on surface using trucks and waste rock will be hauled either to a surface waste dump or to an active production level for use as stope backfill.

Mine dewatering will be achieved using a multi-level dirty water system. Main sump and pump stations for Grey Fox North will be located on the ramp at 40 Level, 145 and 265 Levels in the Whiskey Jack Zone, and on the 140, 265, 365, and 465 Levels in the Gibson Zone. An additional sump will be placed at the portal entrance to collect surface run-off water that enters the mine via the ramp or portal. For Grey Fox South, main sump and pump stations will be located on the 140, 240, and 340 Levels. A network of borehole sumps and level sumps fed to these main sumps. All sumps will collect any groundwater and any service water used during mining activities. The main sumps are staged up to the next main sump in that zone to surface. Any residual solids in the sumps will be mucked using an LHD and placed into a nearby stope for disposal.

The stopes will be backfilled with a combination of cemented rock fill and unconsolidated rock fill to reduce the use of cement while still minimizing backfill dilution in adjacent stopes. For uppers stopes, backfilling is



not planned and rib pillars may be left in place during mining to support the voids. A cement slurry will be hopper fed from surface through boreholes to an underground slurry receiving station. The slurry receiving station will be used to distribute the slurry to each level, and spray bars will be used to add the slurry to waste rock in LHD buckets for placement in open stopes. When possible, waste rock will be sourced directly from active development underground.

Design criteria and parameters specific to each aspect of the mining method are presented in subsequent sections. The following were considered when determining the criteria and parameters during the mine design process:

- Health and safety for the workers, local communities, and the environment
- Develop and design the mine to achieve regulatory approvals
- Achieve buy-in from local stakeholders
- Company standards and specifications (or industry best practices where company standards and specifications were not available)
- Prevention through design concepts
- Minimizing risk to production
- Use of proven industry technology, equipment, and processes
- Operational flexibility
- Minimizing operating costs
- Mineral resource recovery (extraction rate)

16.1 Geomechanical

16.1.1 Geology

Grey Fox is characterized by complex geology featuring stacked epithermal gold veins, with similarities to other major Canadian Shield gold deposits. Individual vein lens lengths are most commonly 200 m to 300 m long in strike, and the total deposit exists along an approximate 750 m in strike length. The typical thickness HW to FW is 2 m to 10 m. The deposit typically extends from surface to approximately 300 m deep below ground surface (BGS) from the current topography at the Grey Fox North portal, the maximum depth BGS is approximately 800 m, for indicated material in the Gibson Zone.

16.1.2 Data

The 2025 drill core logging records provided Rock Quality Designation (RQD) data for Grey Fox, where RQD is generally regarded as excellent with a majority of measured RQD, weighted by core run length, exceeding 90% (Figure 16-5). Data indicates the first quartile of the cumulative RQD distribution is approximately 88%, and the median and third quartiles are closer to 95%. The first quartile RQD value is used for geotechnical design in this study, for 75% reliability.



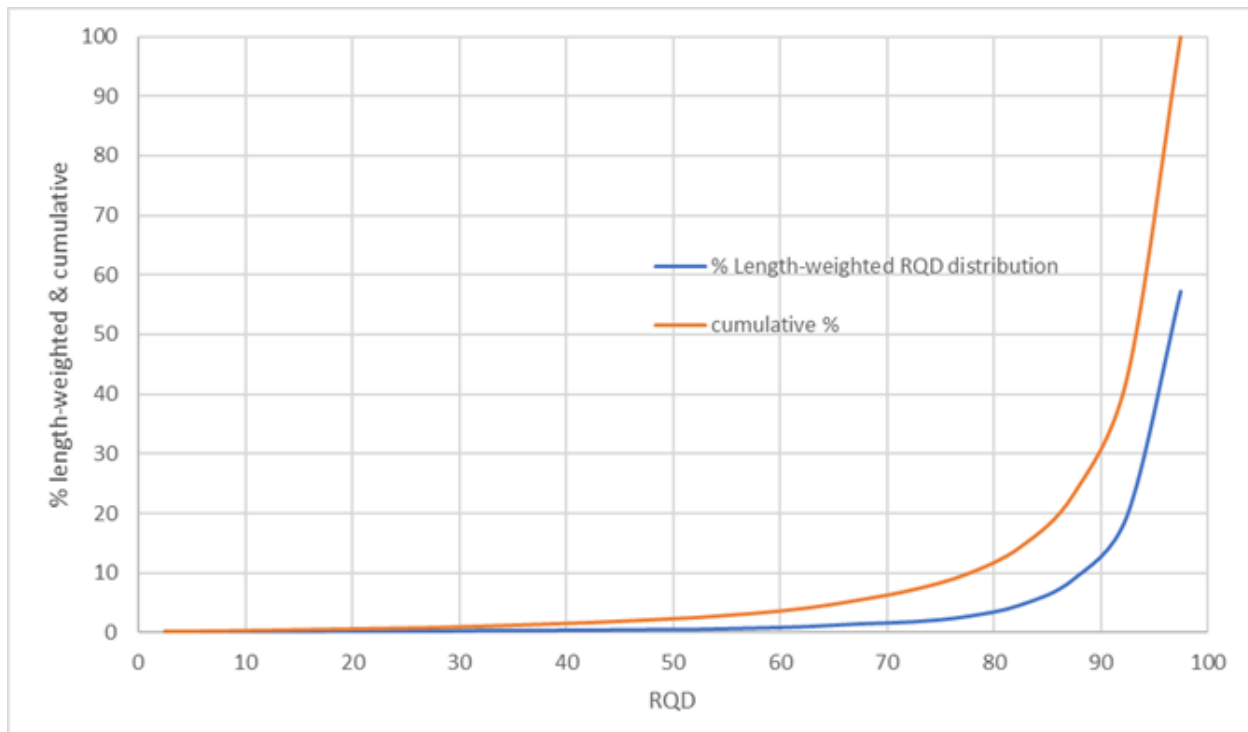


Figure 16-5: Grey Fox Length-Weighted and Cumulative RQD Distribution

Grey Fox 2025 drill core samples were selected and brought to Queen's University for rock strength testing, including unconfined compressive strength (UCS), intact elastic modulus (E_i), Poisson's Ratio (ν), and specific gravity (G_s).

Additional data for rock mass characterization is provided by previous geotechnical studies for the nearby Froome project pre-scoping studies, by Mine Design Engineering (MDEng) and Golder Associates. Froome data is assumed applicable to Grey Fox due to proximity and geologic similarity.

Grey Fox 2025 oriented drill core logging records provide structural orientation data of discrete and systemic geologic structure.

The regional strike of the deposit is N44°W and the dip is 72.8°SW (azimuth 136° Right Hand Rule). The mine plan proposed stope dimensions are 20 m length along the deposit strike with a vertical height or sublevel standoff of 25 m and a width of 5 m. Backs on the drill/extraction levels are flat and end walls are vertical. Stope FW and HW dips are 63° for Whiskey Jack and 60° for the remainder of the Grey Foxes deposit.

Core logging and testing data were used for Q-System rock mass classification. The rock mass quality Q is the product of the rock mass inherent strength and applied stress:

$$Q = (RQD / J_n) \times (J_r / J_a) \times (J_w / SRF)$$

The joint property J_n is the joint density and RQD / J_n is a measure of block size. Grey Fox 2025 oriented core logging is summarized (Figure 16-6) and indicates two main joint sets plus random joint orientations,



so $J_n = 6$. The critical joint $197^\circ/65^\circ$ is identified and plotted in relation to mineralized orientations at Grey Fox.

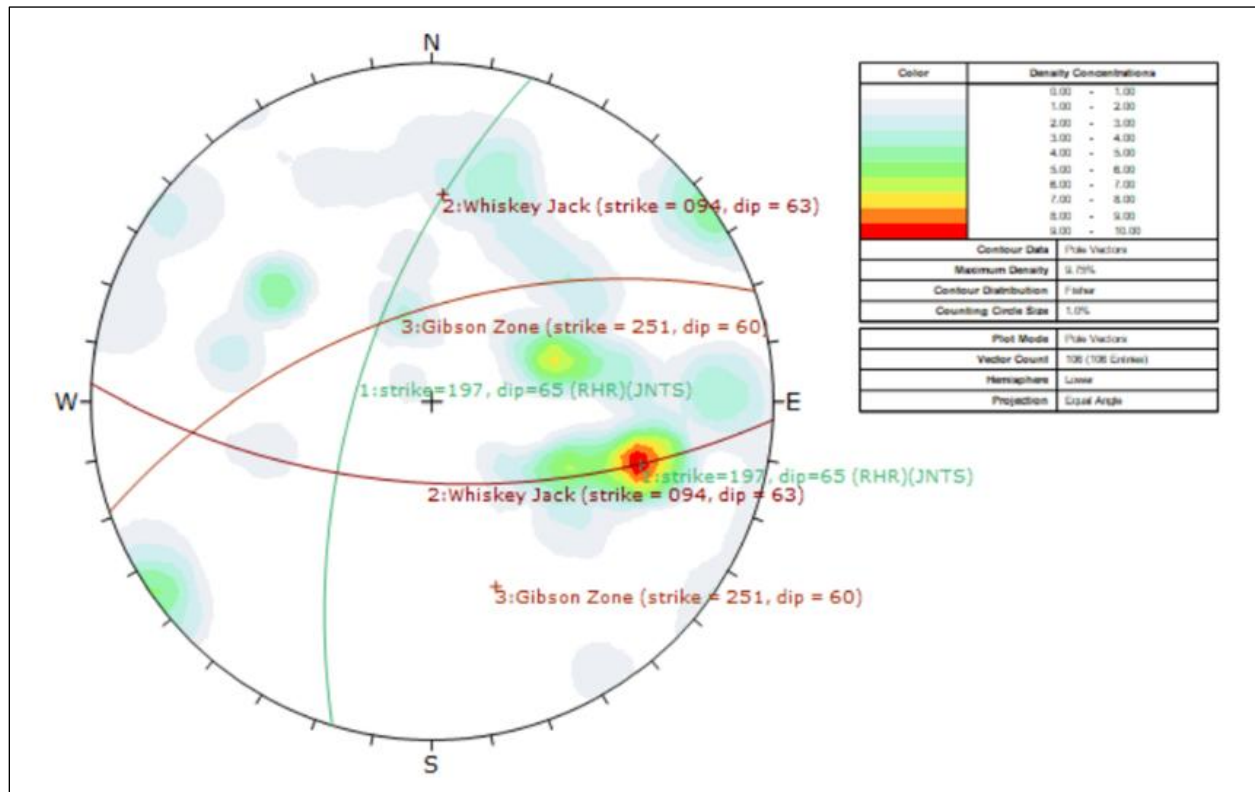


Figure 16-6: Grey Fox Joint Orientations and Critical Joint

Joint surface roughness is J_r , and joint alteration is J_a . The ratio J_r / J_a is a measure of joint surface strength and stiffness. Golder Associate's core logging for the 2017 pre-scoping studies indicate J_r / J_a is typically 1.

The joint water factor J_w represents hydraulic pressure due to rates of water flow within the rock mass and the strength reduction factor SRF accounts for loosening of the rock mass at excavation boundaries during and after mining, due to stress, strength, and elasticity of the rock. The modified rock mass quality Q' considers inherent (logged) properties of the rock mass only and isolated from the impacts of stress, where the terms J_w and SRF drop out:

$$Q' = (RQD / J_n) \times (J_r / J_a)$$

16.1.3 Analysis and Results

Stability analysis of open stopes and arrays of adjacent open stopes follows the Modified (Mathews-Potvin) Stability Graph Method. This method considers the inherent competence of the rock mass due to joint frequency, orientations, and properties, represented by the modified stability number N' . Stability is determined by plotting the excavation's face hydraulic radius versus N' and evaluated in terms of where the excavation face's hydraulic radius versus N' plots in relation to the empirically derived graph stability curves.



Hydraulic Radius (HR) is a shape factor representing the exposure of the excavation face, such as the back, end walls, or FW/HW, for individual stopes or arrays of adjacent open stopes. HR is defined as the area to perimeter ratio of the exposed face.

Base stope geometries for Grey Fox are longitudinal 20 m long stopes following the strike of the deposit, minimum 3 m horizontal width and 25-30 m vertical sublevels. The approximate FW/HW dip angles are 63° for Whiskey Jack and 60° for the remainder of Grey Fox. Stope backs are assumed horizontal, and end walls vertical (Table 16-1).

Table 16-1: Grey Fox Stope HR

Zone	HW and FW	Back	End Walls
Whiskey Jack	5.8	1.3	1.4
Remainder of Grey Fox	5.9	1.3	1.4

N' is based on Q' combined with terms representing stress acting on the rock mass (A), joint orientation relative to the excavation surface (B), and gravity adjustment due to excavation geometry (C):

$$N' = Q' \times A \times B \times C$$

Combining a 25th percentile (75% reliability) RQD of 88% (90% "typical") with $J_n = 6$ and $J_r / J_a = 1$, Q' of the rock mass is 14.67.

To estimate A, it is assumed in-situ ground stress is due to gravity self-loading, and vertical stress σ_v is the product of rock/overburden G_s and depth BGS. The average G_s is 2.75 and the maximum (worst case in terms of stress) depth of each zone's indicated material is approximately 300 m BGS for Whiskey Jack, and 800 m BGS for the remainder of Grey Fox. The stress ratio κ defines horizontal stress σ_h and is a function of E_i and depth BGS:

$$\kappa = \sigma_h : \sigma_v = 0.25 + 7 \times E_i \times (0.001 + 1/\text{depth(m)})$$

Due to the resolution of σ_h and σ_v stress trajectories around excavation boundaries, stress magnitudes acting directly at the stope faces are determined by applying a methodology to estimate stress magnitudes at open stope boundaries. These stresses along with intact UCS of the rock are used to determine A (Figure 16-7 and Table 16-2).



Rock Stress Factor A

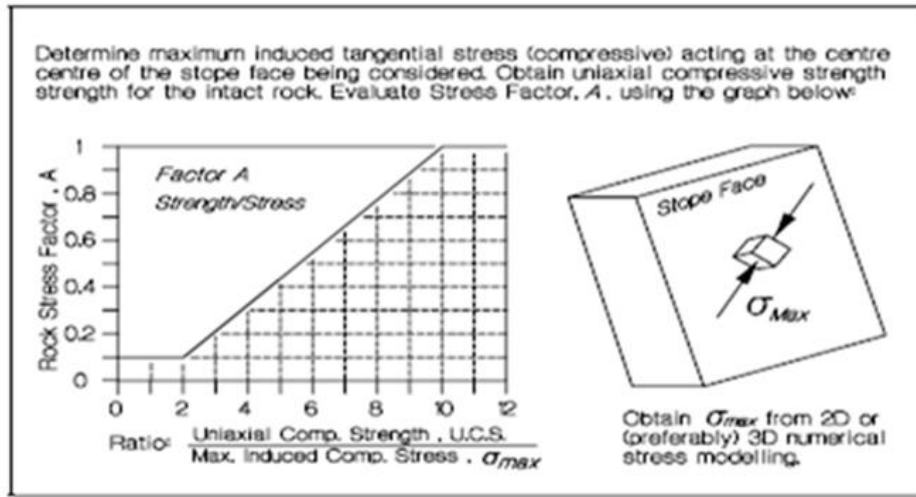


Figure 16-7: Rock stress factor “A”

Table 16-2: Grey Fox “A” factors

	UCS _{avg} (MPa)	Depth (m)	κ	σ_v (MPa)	σ_h (MPa)	A _{sidewall}	A _{back}	A _{endwall}
Whiskey Jack	118.48	300	1.26	8.1	10.2	1.00	0.44	1.00
Remaining Grey Fox	118.48	800	0.78	21.6	16.8	1.00	0.18	1.00

The factor “B” is determined by identifying the critical joint, which is most likely to impact the stability of an exposed stope face, then measuring the angle α between the critical joint orientation and the stope face orientation (Figure 16-8). The most critical stope face is the HW, which is typically parallel to mineralization for longitudinal stoping methods.

Joint Orientation Factor, B

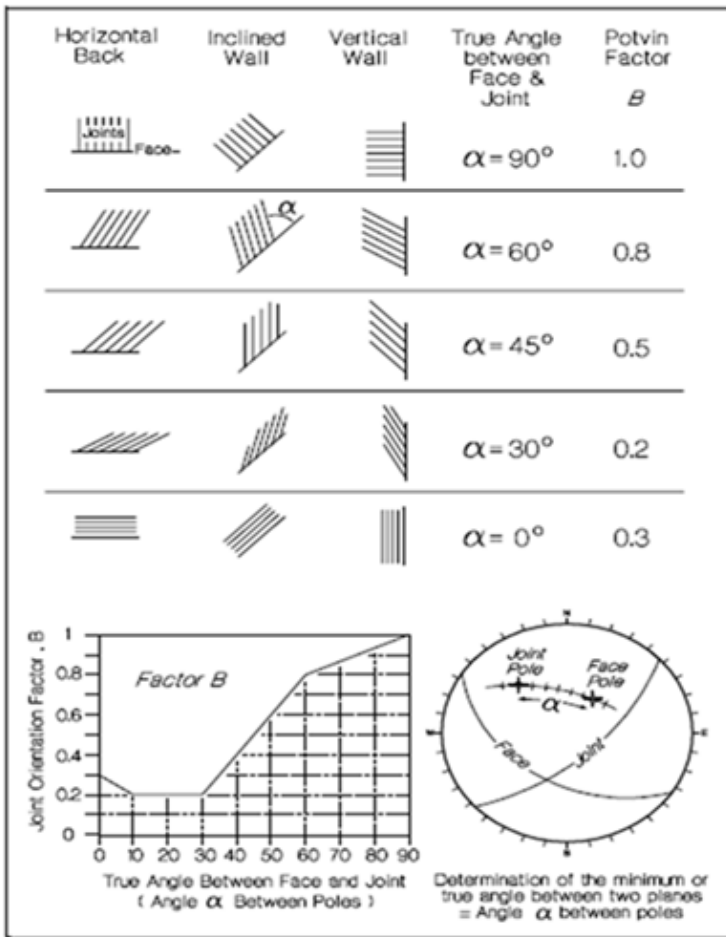


Figure 16-8: Joint Orientation Factor "B"

Factor "C" is the gravity adjustment factor, which depends on the likely failure mechanism at each stope face. Kinematic analysis identifies the failure mechanism is statistically most likely to impact stope faces, based on the orientations in relation to critical geologic structure, and is based on cumulative kinematic failure probability of each failure mechanism. Kinematic analysis of critical joint $197^\circ/65^\circ$, versus stope face orientations, indicate gravity failure is the most likely mechanism of each stope face. The HW is presumed parallel to mineralization dip. For Whiskey Jack (63° dip), C is 5.28 and for the remaining Grey Fox (60° dip), C is 5.0 (Figure 16-9).

Gravity Adjustment Factor, C

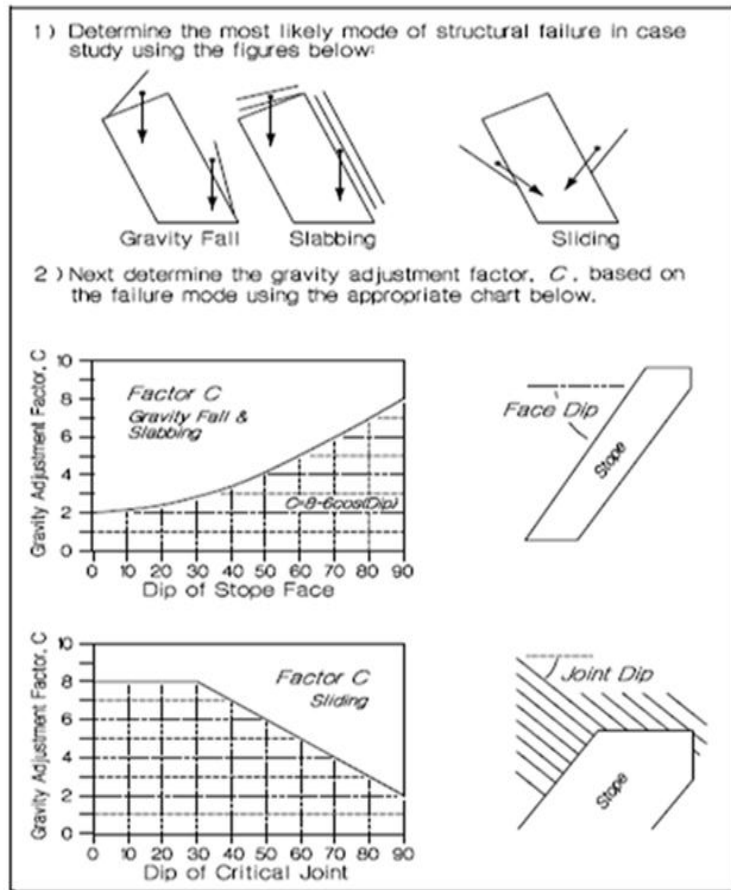


Figure 16-9: Gravity Adjustment Factor “C”

Stope design HW/FW, back, and end wall HR are plotted with respective N' of the HW/FW, backs, and end walls on the stability graph for Whiskey Jack (Figure 16-10) and remaining Grey Fox (Figure 16-11), indicating at which exposed HR ground support is required or where stopes must be expediently backfilled. For the proposed stope dimensions and deposit dip, stope faces are regarded as “stable-unsupported.” Up to six open longitudinal stopes, adjacent along strike, are possible at Whiskey Jack, and for the deeper Grey Fox deposit (e.g., accounting for deepest stoping in Grey Fox), before the HW/FW is regarded as “transitional-unsupported,” where HW support or very expedient stope cycling and dilution tolerance are required.

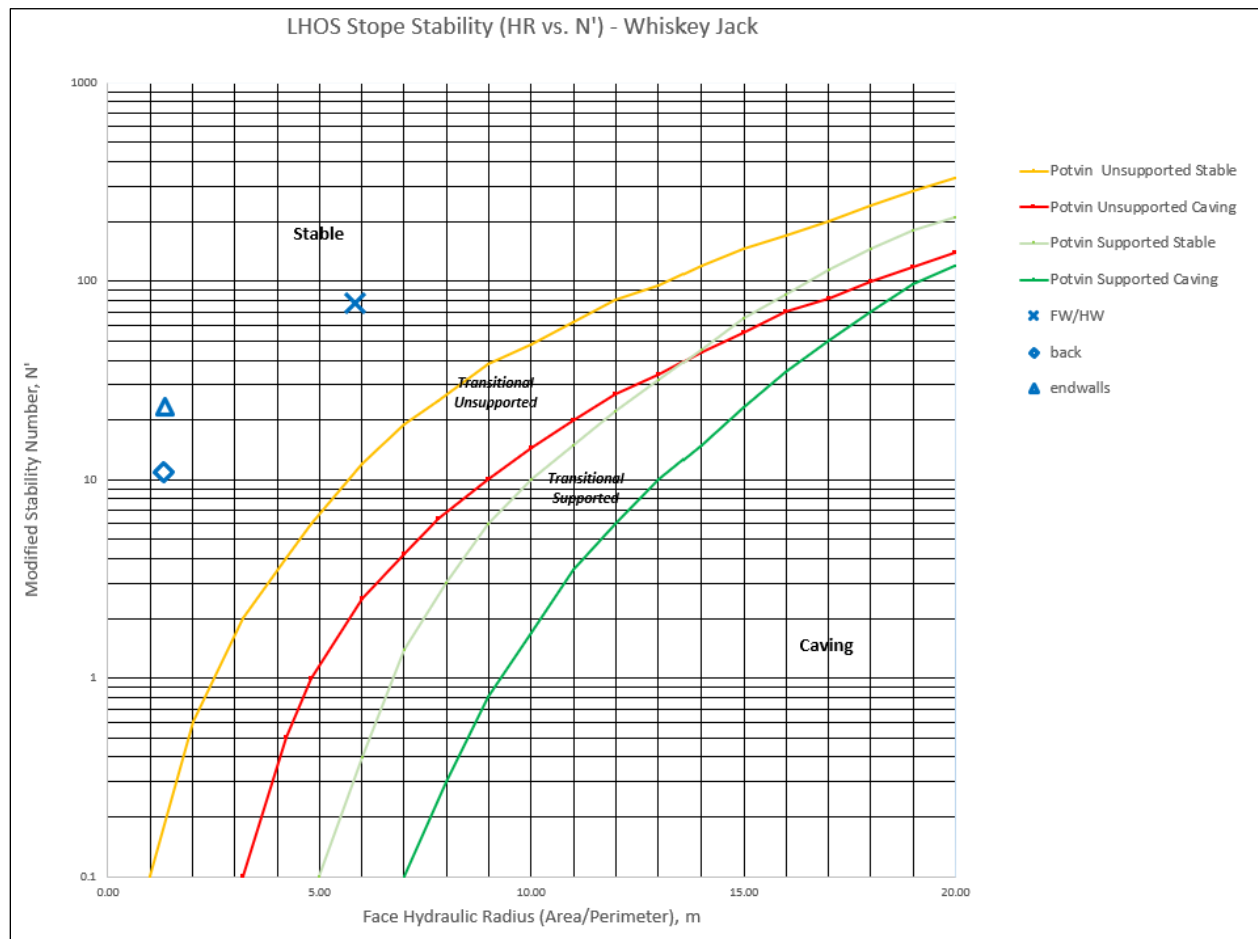


Figure 16-10: Whiskey Jack Stability Graph

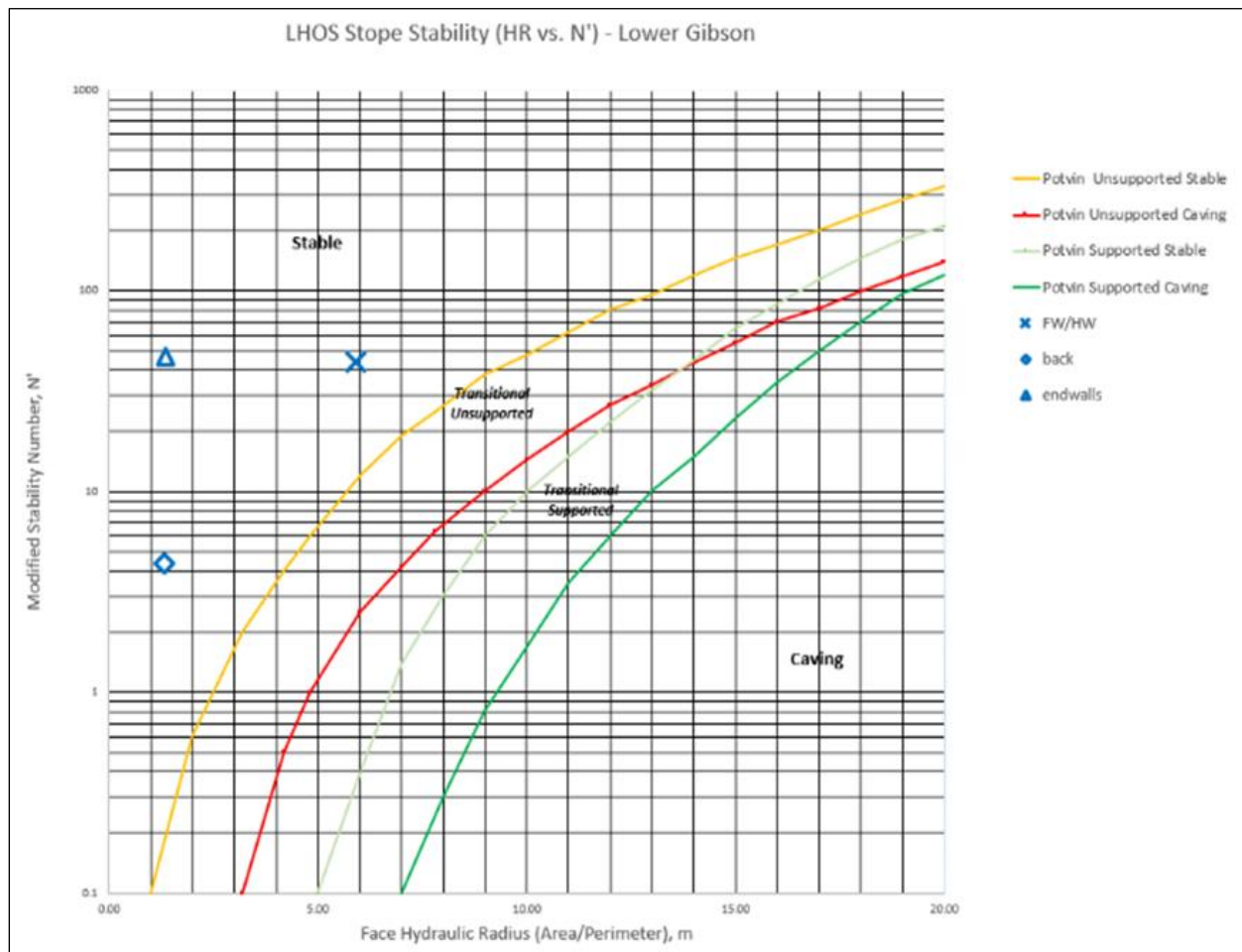


Figure 16-11: Remainder of Grey Fox Stability Graph

16.1.4 Waste Pillar Stability

Mineralization, particularly at Whiskey Jack, places recoverable grades of the deposit along the FW/HW closer to the host rock contact, with lesser grades in the center. This may result in stoping along two parallel headings along strike, with lower grade “waste” material left in situ between each heading, resulting in a “waste pillar.” A required waste pillar thickness W_p at increasing depth BGS is calculated, to maintain a pillar factor of safety F_s of 1.3.

The proposed waste pillar thickness for the mine design is 3 m minimum required pillar thicknesses at increasing depth, maintaining a 1.3 F_s , are determined using Lunder-Pakalnis pillar strength methods:

$$PS = (K \times UCS) \times (C_1 + C_2 \times \kappa)$$

Where PS is the pillar strength in MPa, intact rock strength UCS is in MPa, and K , C_1 , and C_2 are empirical constants from a case history database and equal to 0.44, 0.68, and 0.52 respectively. The average pillar confinement C_{pav} is a function of pillar geometry and is used to determine κ :

$$C_{pav} = 0.46 \times \{\log[(W_p / H) + 0.75]\}^{1.4 / (W_p / H)}$$



$$kappa = \tan\{\arcsin[(1-C_{pav})/(1+C_{pav})]\}$$

Where W_p and the pillar height H are in metres. A geometric correction factor σ_{ec} accounts for the effects of pillar inclination θ , since pillars are not vertical but governed by the dip of the deposit:

$$\sigma_{ec} = (1 - \sin\phi) / (1 - \sin(\phi - \arctan(\tan(\theta/2(1+\gamma))))))$$

Where ϕ is the internal friction angle of an intact rock sample and γ is Poisson's Ratio. Inclined PS is generally less than that of vertical pillars since shear strength has a greater role in the overall PS.

Backfill reinforces adjacent pillars through the “arching” effect, where backfill, as it settles, exerts a normal force on pillars which translates to frictional forces, along the backfill/pillar contact surface, providing additional pillar support. The vertical and horizontal components of pillar stress reduction due to arching are functions of cemented rock fill material properties and volume:

$$\sigma_{vh} = (\rho_{bf} B' / 2 K_{bf} \tan\delta') \times \{1 - \exp(-2 K_{bf} D / B' \tan\delta')\}$$

$$\sigma_{hh} = K_{bf} \sigma_{vh}$$

$$K_{bf} = 1 - \sin\phi_{bf}$$

$$\delta \approx (2/3) \times \phi_{bf}$$

Where:

σ_{vh} = vertical stress component of the arching effect (MPa)

σ_{hh} = horizontal stress component of the arching effect (MPa)

K_{bf} = horizontal to vertical stress ratio of the backfill

ϕ_{bf} = backfill internal friction angle, assumed 30°

δ' = effective friction angle between the pillar and the backfill

B' = stope width (m)

D = depth below ground surface (m)

ρ_{bf} = backfill density, assumed 1.95 t/m³ = 0.01911 MN/m³

The analysis assumes that the stope on the FW side of the deposit is mined first, so “B” is the width of the stope on the FW side. If mining and backfilling the HW side first, the waste pillar support must be considered, particularly at lower elevations.

For inclined pillars, pillar stress reductions due to arching are calculated based on arching components σ_{vh} and σ_{hh} in relation to the deposit dip angle (or pillar inclination), and the total stress reduction due to arching is assumed intermediate to σ_{vh} and σ_{hh} , with frictional support coplanar to the pillar surface.

The pillar F_s is expressed as a ratio of PS to total pillar stress:

$$F_s = PS / (\sigma_{ind} - \sigma_{red})$$



Where the induced pillar stress σ_{ind} is the product of the rock supported rock unit weight and the pillar's tributary area, concentrated on the pillar area, and σ_{red} are pillar stress reductions due to counteracting backfill frictional support from arching.

Required W_p at increasing depths BGS, at $F_s = 1.3$, are presented for Whiskey Jack (Figure 16-12) and the rest of Grey Fox (Figure 16-13). The maximum depth of mineral resource at Whiskey Jack is approximately 300 m BGS, and 800 m BGS at Gibson. At these depths, a 3 m W_p is possible at Whiskey Jack, the main area of concern. For the remaining Grey Fox, 3 m for W_p is possible up to 375 m BGS, though at greater depths the pillars must be supported to maintain a 3 m W_p at $F_s = 1.3$, particularly if mining along the HW side first.

Cable bolts installed in a fan pattern into the waste pillar, from one of the ore drives at drilling or mucking levels, may provide the necessary waste pillar support.

Additional arching pillar support from backfill is negligible at Grey Fox. The stope geometries dictate low backfill volume, generating insufficient normal and frictional forces due to arching against the pillars to provide any additional support.

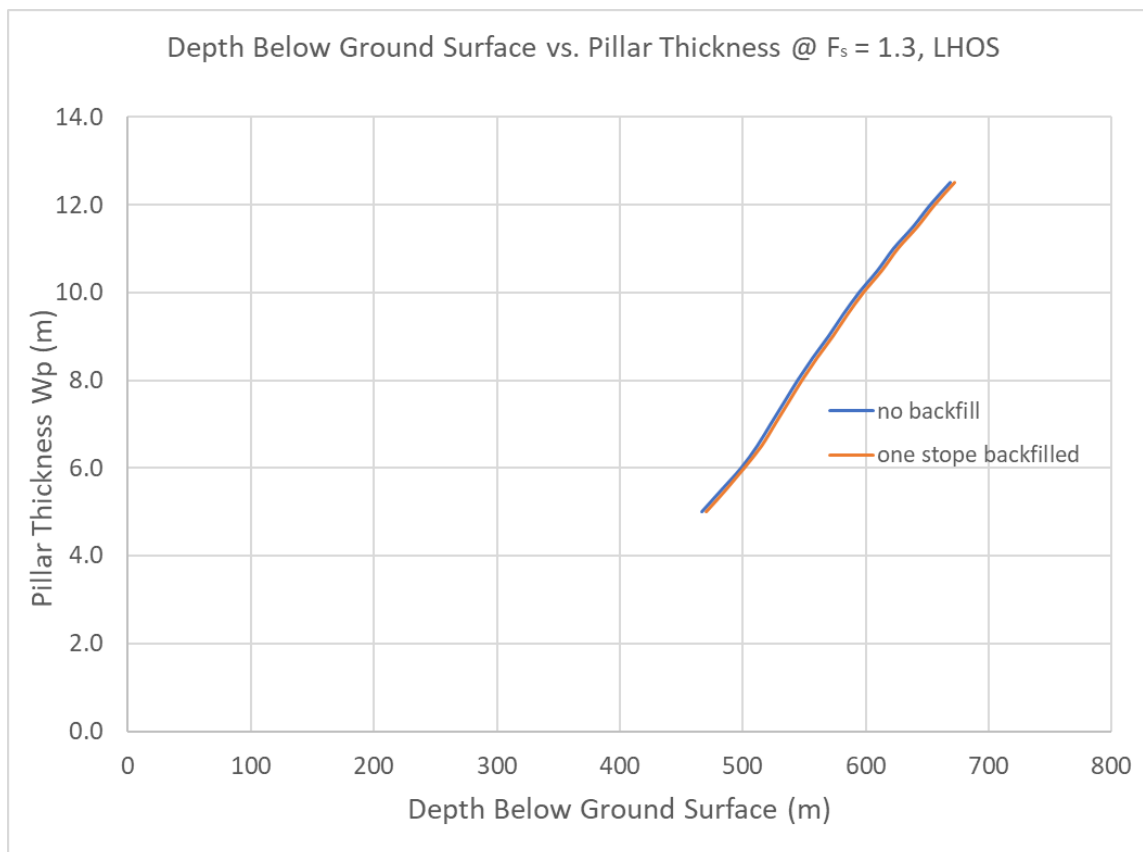


Figure 16-12: W_p versus depth BGS, Whiskey Jack



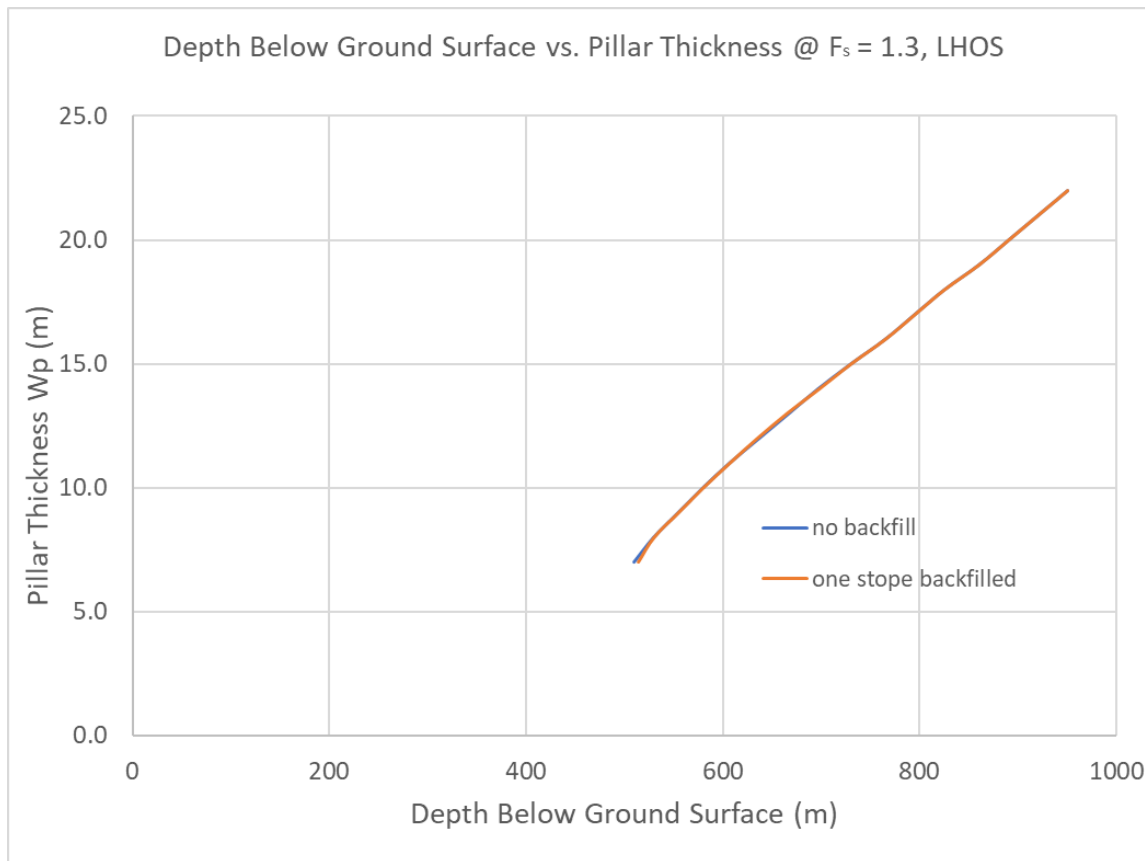


Figure 16-13: W_p versus depth BGS, Remainder of Grey Fox

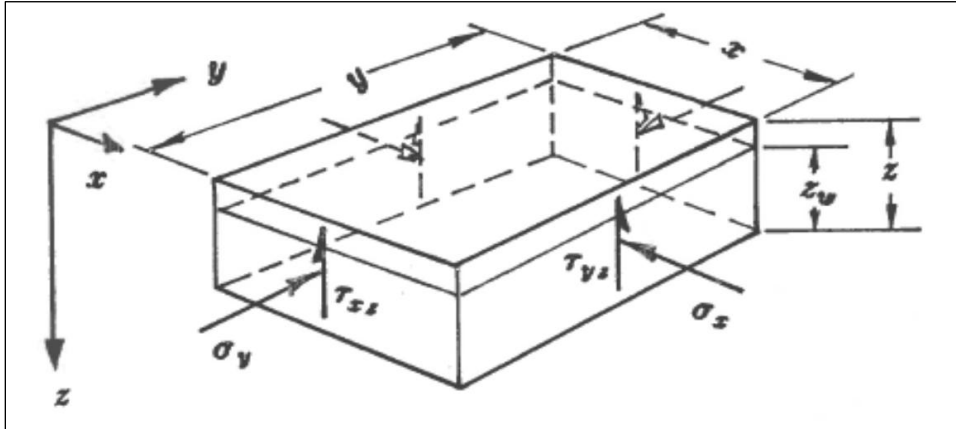
16.1.5 Sill Pillar

A proposed 4 vertical metre thick sill pillar is evaluated for stability, between stoping areas at Grey Fox. Limit equilibrium methods described by Hoek are used. The sill pillar factor of safety $F_{s,sil}$ is expressed as:

$$F_{s,sil} = (2/\rho) ([\tau_{xz}/y] + [\tau_{yz}/x])$$

Where ρ is the rock density constituting the sill pillar. Figure 16-14 illustrates sill pillar dimensions, resisting shear forces (τ_{xz} and τ_{yz}), and normal in-situ stresses (σ_x and σ_y).





Source: (Hoek, 1989)

Figure 16-14: Sill Pillar Dimensions, Resisting Stress and Normal Stress Vectors

Resisting shear stresses are assumed equal in the xz and yz planes ($\tau_{xz} = \tau_{yz} = \tau$), and are defined by the rock mass Unconfined Compressive Strength UCS_{rm} , the rock mass angle of internal friction ϕ_{rm} (32.678°), and the Hoek-Brown parameter m :

$$\tau = ([\cot\phi_{rm} - \cos\phi_{rm}] m UCS) / 8$$

Where:

$$\phi_{rm} = \arctan(1/((4h_{xz}\cos^2\theta_{xz}) - 1)^{0.5})$$

$$\theta_{xz} = (1/3)(90^\circ + \arctan(1/((h_{xz}^3 - 1)^{0.5})))$$

$$h_{xz} = 1 + (16(m\sigma_h + s UCS)/(3 m^2 UCS))$$

$$m = m_i \cdot \exp([GSI - 100] / 14)$$

$$s = \exp([GSI - 100] / [9 - 3D])$$

Assume:

$$GSI = 9 \ln(Q') + 44 \text{ where RQD is assumed } 88\%, J_n \text{ is } 6, \text{ an } J_r / J_a \text{ is } 1$$

$$m_i = 19.0 \text{ for quartz/quartz-carbonate veins}$$

The disturbance factor D is zero, assuming good blasting practice.

Due to high rock strength properties in the mineralized zone and the narrowness of the stope designs, frictional forces supporting a small sill pillar are much greater than the weight of the pillar and a 4 m surcharge of cemented rock fill on top of the sill pillar, resulting in elevated $F_{s,sill}$ for Grey Fox. Sill pillars are evaluated for 3 m horizontal widths versus 20 m length on strike, then for 9 m horizontal widths for scenarios where a 3 m waste pillar between two parallel stopes is mined.

The waste rock or ore sill pillar could be eliminated entirely, with sill construction by placement of cemented rock fill with a higher cement content into an empty stope. A cemented rock fill pilot program consisting of

mixing and testing trial cemented rock fill batches will identify cemented rock fill batch specifications for creating a cemented rock fill sill which may be safely mined underneath.

16.1.6 Backfill

Backfill will consist of cemented rock fill or unconsolidated rock fill. Due to relatively small stope sizes and backfill volumes, very limited arching support is anticipated so cemented rock fill and rock fill are expected to behave similarly for taking weight from the waste pillars. One disadvantage with rock fill is it will run if a waste pillar fails.

For twin stopes where the interim waste pillar W_p is 3 m, cemented rock fill is placed on the HW side if this side is mined first, with rock fill on FW side of the waste pillar. If the FW side is mined first, rock fill may be placed on both sides of the waste pillar. Refer to Figure 16-31.

A cemented rock fill pilot program consisting of mixing and testing trial cemented rock fill batches will identify cemented rock fill batch specifications which satisfy general and specialized strength requirements. For the cemented rock fill strength, Mitchell's Method is used to identify strength requirements based on cemented rock fill vertical exposure, then cement contents are determined based on strength testing trial batches of cemented rock fill at different strength contents and adjusting the cement content to get to the target strength identified by Mitchell's Method.

16.1.7 Crown Pillar

The crown pillar provides ground surface stability over an underground mine. Crown pillar stability analysis considers open stopes or other large underground voids, in proximity to the surface. An appropriate crown pillar thickness determination is based on ground conditions, crown pillar geometry, purpose/expectancy, and risk tolerance.

The Carter Method for scaled span analysis is the application of a scaling factor to the proposed design dimensions to account for geometric influences and ground conditions and applied to sizing / modifying the design span. This method assumes a caving or unraveling failure mechanism, with design geometry and rock parameters as input to determine reliability of pillar integrity at decreasing thickness and increasing span due to mining. Caving is an appropriate assumption since it takes weathering of the bedrock near the surface into account. The method assumes backfilling is not used underground.

The critical span S_c is the maximum self-supporting span due to rock mass conditions:

$$S_c = 3.3 (Q^{0.43}) \times \sinh^{0.0016} (Q)$$

Where Q is the product of Q' and (J_w/SRF) , J_w is assumed 1.0 (dry), and SRF of the rock is a function of κ , UCS, and crown pillar thickness or depth in metres BGS:

$$SRF = 0.244(depth/UCS)^{1.322} + 0.176(UCS/depth)^{1.413}$$

The Scaled Span C_s is the product of the design span and a scaling factor accounting for geometric influences and ground conditions:



$$C_s = S \{G_s / [T (1+SR) \cdot (1 - 0.4\cos\theta_{cp})]\}^{0.5}$$

S = crown pillar design span (m)

θ_{cp} = orientation of the crown pillar-wall interface (assumed equal to deposit dip)

T = crown pillar thickness (m)

SR = span ratio = S/L

L = crown pillar strike length (m)

The Probability of Failure ($P_{F,crown}$) of the crown pillar is a function of the crown pillar Factor of Safety ($F_{S,crown}$):

$$P_{F,crown} = 1 - \text{erf} [(2.9 F_{S,crown} - 1) / 4]$$

$$F_{S,crown} = S_c / C_s$$

With $P_{F,crown}$ and the equivalent $F_{S,crown}$ of the crown pillar known, the crown pillar may be assigned to a risk class which provides additional guidance on the expected service life of the crown pillar, accessibility and level of concern post-mine closure, and degree of stability monitoring. Table 16-3 presents Carter's Crown Pillar Acceptable Risk Exposure Guidelines.

Table 16-3: Crown Pillar Acceptable Risk Exposure Guidelines

Class	Probability of Failure %	Minimum Factor of Safety	Maximum Scaled Span, $C_s (= S_c)$	ESR (Barton et al. 1974)	Design Guidelines for Pillar Acceptability/Serviceable Life of Crown Pillar				
					Expectancy	Years	Public Access	Regulatory position on closure	Operating Surveillance Required
A	50 – 100	<1	$11.31Q^{0.44}$	>5	Effectively zero	< 0.5	Forbidden	Totally unacceptable	Ineffective
B	20 – 50	1.0	$3.58Q^{0.44}$	3	Very, very short-term (temporary mining purposes only ; unacceptable risk of failure for temporary civil tunnel portals)	1.0	Forcibly Prevented	Not acceptable	Continuous sophisticated monitoring
C	10 – 20	1.2	$2.74Q^{0.44}$	1.6	Very short-term (quasi-temporary stope crowns ; undesirable risk of failure for temporary civil works)	2 – 5	Actively prevented	High level of concern	Continuous monitoring with instruments
D	5 – 10	1.5	$2.33Q^{0.44}$	1.4	Short-term (semi-temporary crowns, e.g. under non-sensitive mine infrastructure)	5 – 10	Prevented	Moderate level of concern	Continuous simple monitoring
E	1.5 – 5	1.8	$1.84Q^{0.44}$	1.3	Medium-term (semi-permanent crowns, possibly under structures)	15–20	Discouraged	Low to moderate level of concern	Conscious superficial monitoring
F	0.5 – 1.5	2	$1.12Q^{0.44}$	1	Long-term (quasi-permanent crowns, civil portals, near-surface sewer tunnels)	50–100	Allowed	Of limited concern	Incidental superficial monitoring
G	<0.5	>>2	$0.69Q^{0.44}$	0.8	Very long-term (permanent crowns over civil tunnels)	>100	Free	Of no concern	None required

The crown pillar consists of approximately 31 m of rock from the back of the Whiskey Jack 65 Level top cut to the bedrock/soil overburden contact, then 22 m of overburden to the surface. The soil overburden is assumed glacial till. Rock is assumed to have a minimum RQD of 10%. At less than 10%, particle sizes are at a scale that the laws of soil mechanics apply. Assuming a worst-case scenario, soils may be represented with a Q value equal to zero. For the rock, Q is the product of Q' (=15 at 88% RQD for 75% reliability) and strength reductions due to hydraulic stress, rock strength, and depth ($=J_w / \text{SRF}$), where J_w is assumed 1.0



for dry conditions or negligible inflows. The rock G_s is 2.75 and the glacial till G_s is assumed 2.6, and the crown pillar G_s and Q at increasing total thickness are a weighted average of the glacial till and rock components, with $F_{s,crown}$ (Figure 16-15) and $P_{f,crown}$ (Figure 16-16) calculated at increasing total thickness, for a 9 m W x 60 m L crown pillar footprint, or three sets of 20 m long parallel stopes with the central barrier pillar removed and mined along strike.

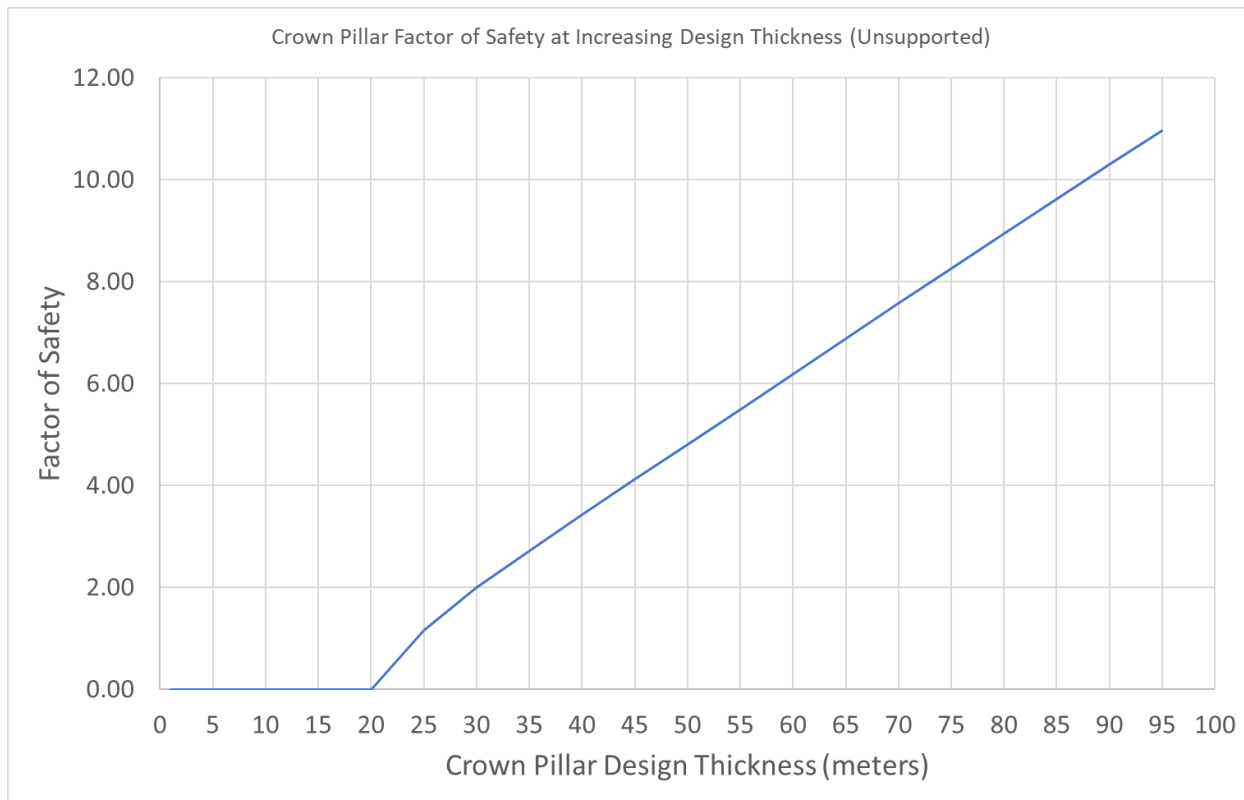


Figure 16-15: $F_{s,crown}$ at increasing Total Thickness

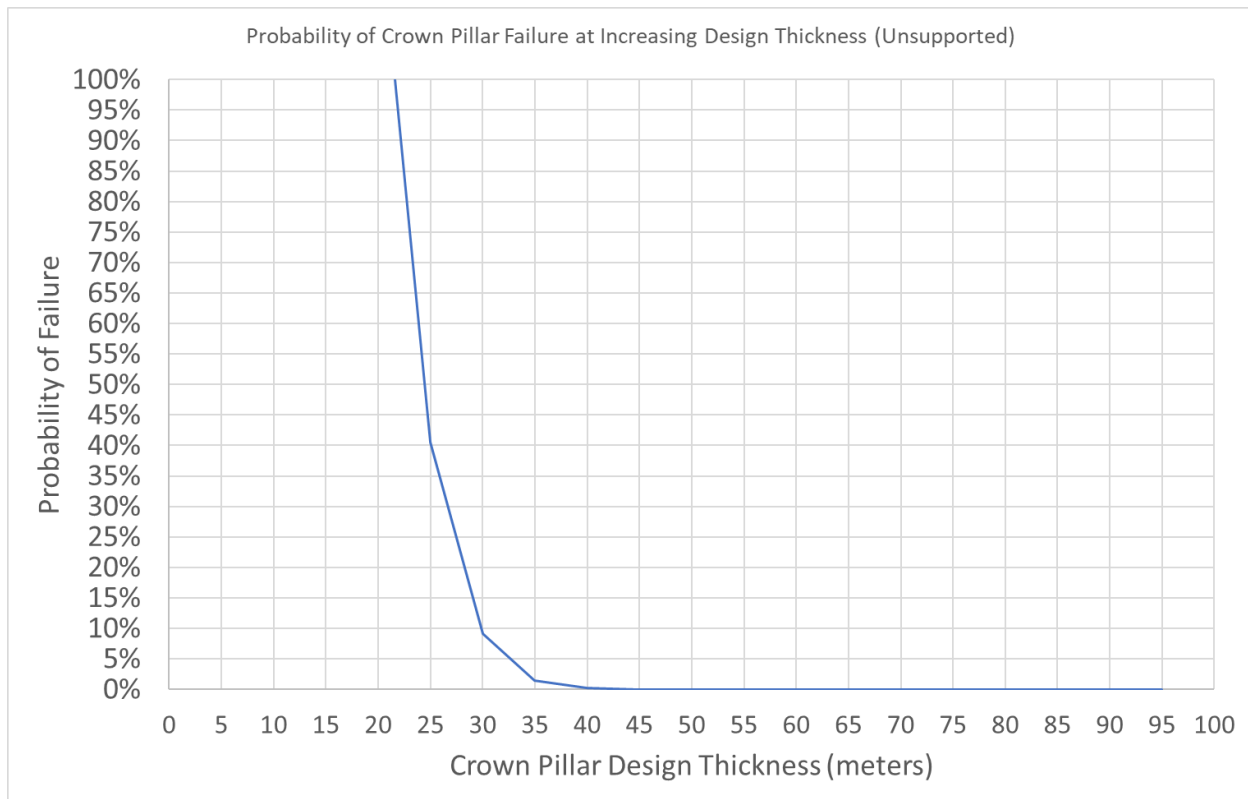


Figure 16-16: $P_{F,crown}$ at increasing Total Thickness

Surface elevation directly over the Whiskey Jack 65 Level is 288 m above sea level, and the total crown pillar thickness is 53 m, consisting of 22 m overburden and 31 m rock, given a 5 m vertical top cut at 65 Level. At this total thickness, $F_{s,crown}$ is 5.22 and $P_{f,crown}$ is virtually 0%. This classifies the crown pillar as Class G, with a very long-term to permanent service life, minimal post-closure stability concerns, and no anticipated monitoring requirements.

16.1.8 Ground Support Standards

First-pass pattern (primary) support for Grey Fox consists of 1.5 m long #6 grouted rebar bolts, on a 1.5 m “dice” pattern, where the nearest distance of the patterned bolts to the “5-spot” bolt is 1 m. The bolts pin 3.3 m x 1.8 m welded wire mesh sheets to the ribs and back for “skin” support. The welded wire mesh sheets are 6 gauge, 10 cm aperture wire mesh sheets, overlapping 3 “squares.” Shotcrete and heavy support may be added, to be determined by encountered ground conditions. Long (secondary) support is added in the back at slashes and intersections (Table 16-4 and Table 16-5). Support classes are defined by the rock mass quality Q or equivalent Rock Mass Rating ($RMR, = 9\ln Q + 44$).

Bolting typically does not extend below the survey reference line (approximately 1.5 m to 1.8 m above the floor) to protect mobile equipment from tire damage. Galvanized bolts and welded wire mesh are recommended for ramp and level development, and non-galvanized for development in ore.



Table 16-4: Ground Support Standards, Ramp and Level Development

Support Class	<i>criteria:</i> Rock Mass Class	<i>criteria:</i> Profile Dimensions	Bolting and Mesh	Shotcrete	Steel Sets
RA	I - Very good rock <i>RMR: 81-100</i> <i>(Q ≥ 61)</i>	4.3 m high x 4 m wide	1.5 m long #6 grouted rebar on 1.5 m "dice" pattern. 3.3 m x 1.8 m 6 gauge 10 cm aperture wire mesh sheets, overlapping 3 "squares". Bolts and mesh galvanized.	None	None
RB	II - Good rock <i>RMR: 61-80</i> <i>(Q = 6.6 - 60)</i>				
RC	III - Fair rock <i>RMR: 41-60</i> <i>(Q = 0.7 - 6.5)</i>			5 cm to 10 cm across the back and 5 cm on the ribs where required.	3.7 m PM24 inflatable bolts. Light to medium steel or shotcrete arches spaced 1.5 m if required.
RD	IV - Poor rock <i>RMR: 21-40</i> <i>(Q = 0.08 - 0.6)</i>			10 cm to 15 cm across the back and 10 cm on the ribs.	
RE	V - Very poor rock <i>RMR: < 20</i> <i>(Q < 0.08)</i>			15 cm minimum across the back and 15 cm on the ribs.	
RF	VI - Intersections, Slashes	4.3 m high x (4.0 m to 8.0 m) wide	1.5 m long #6 grouted rebar on 1.5 m "dice" pattern. 3.3 m x 1.8 m 6 gauge 10 cm aperture wire mesh sheets, overlapping 3 "squares". Minimum length 6.1 m (embedded) single-strand 7-wire 1.5 cm diameter cable bolts on 1.8 m centers, 5 tonnes minimum tension. Galvanized bolts and mesh.	Where required based on Q versus excavation dimensions, 5 cm to 10 cm across the back and 5 cm on the ribs.	Avoid placing slashes and intersections in Class RE, RF ground



Table 16-5: Ground Support Standards, Ore Development

Support Class	<i>criteria:</i> Rock Mass Class	<i>criteria:</i> Profile Dimensions	Bolting and Mesh	Long Support	Notes
SA	I - Very good rock <i>RMR:</i> 81-100 <i>(Q ≥ 61)</i>	5.0 m high x 4.5 m wide	1.5 m long #6 grouted rebar on 1.5 m "dice" pattern. 3.3 m x 1.8 m 6 gauge 10 cm aperture wire mesh sheets, overlapping 3 "squares". Non-galvanized bolts and mesh.	None	N/A
SB	II - Good rock <i>RMR:</i> 61-80 <i>(Q = 6.6 - 60)</i>				
SC	III - Fair rock <i>RMR:</i> 41-60 <i>(Q = 0.7 - 6.5)</i>			3.7 m PM24 coated inflatable bolts.	
SD*	IV - Poor rock <i>RMR:</i> 21-40 <i>(Q = 0.08 - 0.6)</i>			Minimum length 6.1 m (embedded) single-strand 7-wire 1.5 cm diameter cable bolts on 1.8 m centers, 5 tonnes minimum tension	Reduce stope size or select alternative mining method.
SE	V - Very poor rock <i>RMR:</i> < 20 <i>(Q < 0.08)</i>	N/A			Stoping not possible, select alternative mining method.

At 75% reliability in design, 75% of mine development may be in Classes RC/SC or better (blocky to massive), based on a 25th RQD percentile of 88%, $J_n = 6$, J_r / J_a and J_w assumed 1.0, and SRF assumed 2.5 for competent, mainly massive rock under low stress and near the surface.

Support requirements are not anticipated for stope HW's at the majority of Grey Fox. At depth greater than 375 m BGS, waste pillar support may be needed to maintain a 3 m horizontal waste pillar. This may consist of variable length cable bolts, connectable premium inflatable bolts, or connectable hollow bar injectable resin bolts, installed in a fan pattern from the drilling or mucking levels.

16.2 Hydrogeological Considerations

A hydrogeological model was prepared in 2022 by Wood. An updated model was prepared in 2025 by WSP. The memo describes modifications made to the previously developed (Wood 2022) site groundwater flow model to accommodate the updated mine plan and presents results of new predictive simulations of potential groundwater inflows and impacts to the shallow groundwater flow system from mine dewatering. At the time the model was updated, an open pit was planned for the Grey Fox Project. For this technical report the open pit was eliminated as an option. The following section summarizes the hydrogeological model updates assuming the open pit was still in the plan, although it has been assumed the underground inflows will not be impacted by eliminating the open pit.

16.2.1 Model Predicted Seepage Rates

The model predicted groundwater inflow rate to the open pit/underground mine workings (UMWs) for the Base Case model input parameters is 930 m³/d, which represents an increase of 260 m³/d relative to the 2022 model predictions. Model simulation results presented in 2022 indicated that the model predicted



groundwater seepage rates were most sensitive to changes in the assumed properties of the glacial till unit (sensitivity variant 3, Wood 2022) and also to the bedrock hydraulic conductivity (sensitivity variant 6, Wood 2022). To provide upper estimates on the predicted groundwater seepage rate into the open pit/UMW's for the current model, these sensitivity variants were also run for the current model build. Model predicted groundwater seepage rates for these runs are presented below in Table 16-6.

Table 16-6: Predicted Groundwater Seepage into Open Pit and Underground Mine Workings

Predicted Groundwater Seepage Rate (m³/d)		
Model Variant	2022 Model Predicted Seepage rate into UMWs	2025 Model Update Predicted Seepage rate into UMWs and Open Pit
Base Case	670	930
Till k x3	735	1,000
Bedrock k x3	1,500	1,700

16.2.2 Model Limitations

Limitations associated with the development and application of this numerical groundwater model for this Project site include:

- **Steady-state conditions:** The groundwater was simulated assuming steady-state (i.e., long-term, constant) conditions, which represents the long-term dewatering rate after the initial drawdown period. It is expected that short-term dewatering rates during the initial drawdown period and seasonal inflows following the spring freshet will be greater than the long-term rate as storage is released from the surrounding overburden.
- **Data Gaps:** Several assumptions were made during model development due to the site-specific data availability:
 - *Equivalent Porous Media:* This approach was implanted which is deemed sufficient for characterizing the overall groundwater flow regime at the scale of this analysis. Local faults may be indicated by a bedrock low that passes northwest through the proposed underground workings to the Pike River; however, as there is a considerable thickness of overburden sediments above the bedrock low even where it passes under the river, the prospect of significant groundwater and surface water interaction appears to be low.
 - *Pike River:* The riverbed is assumed to be located within the glaciolacustrine deposits across most of the model domain, except at a few locations where bedrock outcrops have been noted from the available surficial geological mapping. Bedrock outcrops along the Pike River may act as “windows” which allow greater hydraulic connection between the bedrock and surficial water features. Presence/absence of bedrock outcrops along the Pike River, within the predicted Zone of Influence is recommended.
 - *Average annual conditions:* The model simulates average annual conditions, and not seasonal conditions. Seasonal variations in stream flows, which may result in small tributaries being dry during late summer and late winter conditions, will not be captured by the model.



16.2.3 Conclusions

The calibrated 3D groundwater flow model developed for the Grey Fox 2022 model build was modified with the updated open pit mine plan and then used in predictive mode to provide new estimates of groundwater seepage inflow rate to the previously proposed open pit and underground mine workings as well as the Zone of Influence relating from mine dewatering and to estimate changes in groundwater-surface water interaction near the mine site. This open pit model was used as a conservative proxy for the drawdown of results for this underground only mining method across a similar footprint.

For the Base Case model input parameters the simulation results indicate the simulated groundwater inflow rate into the fully developed open pit and underground mine workings is 930 m³/d and the Zone of Influence is predicted to extend approximately 2,800 m from the centre of the previously proposed open pit to the south and 1,300 m to the north, east and west of the mine site, which are limited by the presence of Pike River. Model sensitivity results indicate that the long-term groundwater seepage rates into the open pit and UMW's may be 1,700 m³/d. The model results indicated that the Pike River and the three tributaries near the mine site experience an overall reduction in baseflow contribution, with the baseflow reduction to the Pike River constituting approximately 1.3% of the 7Q20 value for the Pike River.

For the Grey Fox Project, the assumed underground inflows are an average of 670 m³/d and a peak of 1500 m³/day per portal. It is recommended to update the model to reflect the current proposed Grey Fox Project mine plan, excluding the previously planned open pit and its influence on the model.

16.3 Mine Design and Mining Methods

16.3.1 Supporting Assumptions and Background Data

16.3.1.1 Design Codes, Regulations, and Standards

The health and safety of workers will be at the forefront of the mine design. The design will consider industry best practices and will comply with safety codes and regulations.

The following practices and codes will be followed:

- Minimize risk to worker health and safety, including prevention through design concepts.
- Minimize the impact to the environment and local communities.
- Occupational Health and Safety Act, R.R.O. 1990, Regulation 854, Mines and Mining Plants.
- Minimize risk to production. Use proven industry technology, equipment, and processes suitable for worker skill sets.

16.3.1.2 Units of Measure

The SI system will be used except where specifically indicated otherwise.



16.3.1.3 Rock Densities

The ore and waste rock densities are included in the resource model. For calculations where data is not directly taken from the model, average in-situ and blasted densities will be used and are presented in Table 16-7.

Table 16-7: Rock Densities

Zone	In-situ Density	Swell Factor (broken)	Broken Density
Gibson	2.76 t/m ³	40%	1.97 t/m ³
All other Zones	2.88 t/m ³	40%	2.06 t/m ³

16.3.2 Development

16.3.2.1 Overbreak and Design Allowance

Overbreak and design allowance factors will be applied to the neat quantities from the design solids. Overbreak accounts for the excavation area exceeding the planned dimensions. The design allowance accounts for development that will be required but will not be included in the details of the mine design (intersection slashes, safety bays, etc.). Table 16-8 lists design factors that will be applied to the neat quantities.

Table 16-8: Development Allowances

Item	Allowance
Design Allowance	
Ramp and lateral capital development	10%
Lateral operating development	0%
Raise development	5%
Overbreak Allowance	
Ramp and lateral development in waste	10%
Lateral development in ore	0%
Raisebored raises	0%
Other raises	10%

16.3.2.2 Lateral Development

Lateral development includes ramp and level development. The development equipment will include:

- Drill – 2 boom electric hydraulic jumbo
- Blast – mobile explosives loader
- Muck – 7-t class LHD
- Ground Support – mechanized bolter



- Truck Haul – 20-t class underground haul truck

The drift profiles will have a flat back with 1 m arch corners.

16.3.2.2.1 Mine Access

The primary access to Grey Fox North will be excavated from the existing Gibson portal and ramp, which will be rehabilitated and expanded to 4.0 m W x 4.3 m H with the portal boxcut to suit, to be known as the Grey Fox North portal, hereafter. Two separate access ramps for Whiskey Jack and the remaining Grey Fox North (Gibson, Contact, and 147NE zones) will be developed from the existing Gibson Ramp. A separate portal will be developed for Grey Fox South with a single access ramp throughout.

The design criteria for the ramps are summarized in Table 16-9.

Table 16-9: Ramp Design Criteria

Item	Criteria
Profile	4.0 m W x 4.3 m H to accommodate: Maximum 6 m/s ventilation velocity a 1.1 m (42") diameter or twin 0.91 m (36") diameter, semi-rigid ventilation ducting during development 150 mm roadbed ballast 20 t Haul Truck Service water and dewatering pipes Compressed air pipe Electrical and communications cables
Gradient	Maximum 17%
Turning Radius	Minimum 20.0 m
Remuck Bays	Distance between Remuck Bays: 150 m Dimensions: 4.0 m W x 5.0 m H x 15.0 m L
Truck Loadouts	Distance between Truck Loadouts: 150 m (with remucks) Dimensions: Take Down Back in ramp 4.0 m W x 5.0 m H x 8.0 m L at the remuck area
Safety Bays	1.5 m W x 2.0 m H x 0.6 m L, spaced minimum 30 m apart

16.3.2.2.2 Typical Level Development

Levels will be named according to their depth from surface (surface assumed at 295 m elevation). In practice, level development will be at 1% to 2% gradient as required for water drainage but will be designed at 0% gradient for the PFS. Unless otherwise stated, drift size will be 4.0 m W x 4.3 m H with 1 m arch corners for capital development and 3.5 m W x 4.0 m H with 1m arch corners for operating development. Development excavations will target a 4:1 pillar to excavation ratio with a minimum of 3:1 where required. Capital development will be designed in waste rock outside of the mineralization where possible to avoid sterilizing potential future stoping. The criteria for typical level development are summarized in Table 16-10.



Table 16-10: Typical Level Development Drift Profiles and Criteria

Drift Description	Criteria
Level Access	4.0 m W × 4.3 m H
Ventilation Access	4.0 m W × 4.3 m H
Remucks	4.0 m W × 5.0 m H x 15 m L
Stope Access	4.0 m W × 4.3 m H
Stope Sills	3.0 m - 5.0 m W x 4.0 m H
Backfill Access	4.0 m W × 5.0 m H x 15/25 m L
Electrical Cut-out	5.0 m W × 4.3 m H x 10/20 m L
Sump	4.0 m W × 4.3 m H x 10 m L, max. Gradient -20%
Explosive Storage	7.0 m W x 4.3 m H x 20 m L
Detonator Storage	5.0 m W x 4.3 m H x 10 m L
Underground Workshop	6.5 m W x 4.3 m H x 44 m L
Satellite Fuel Bay	5.0 m W x 4.3 m H x 23 m L
Wash Bay	5.0 m W × 4.3 m H x 20 m L
Permanent Refuge Station (12-person)	4.0 m W × 4.3 m H x 18 m L

Levels will be developed every 30 vertical metres in the Whiskey Jack Zone (except the first two levels are 25 m). Whiskey Jack Zone is relatively small and uniform throughout. Figure 16-17 presents the Whiskey Jack Zone typical level development.



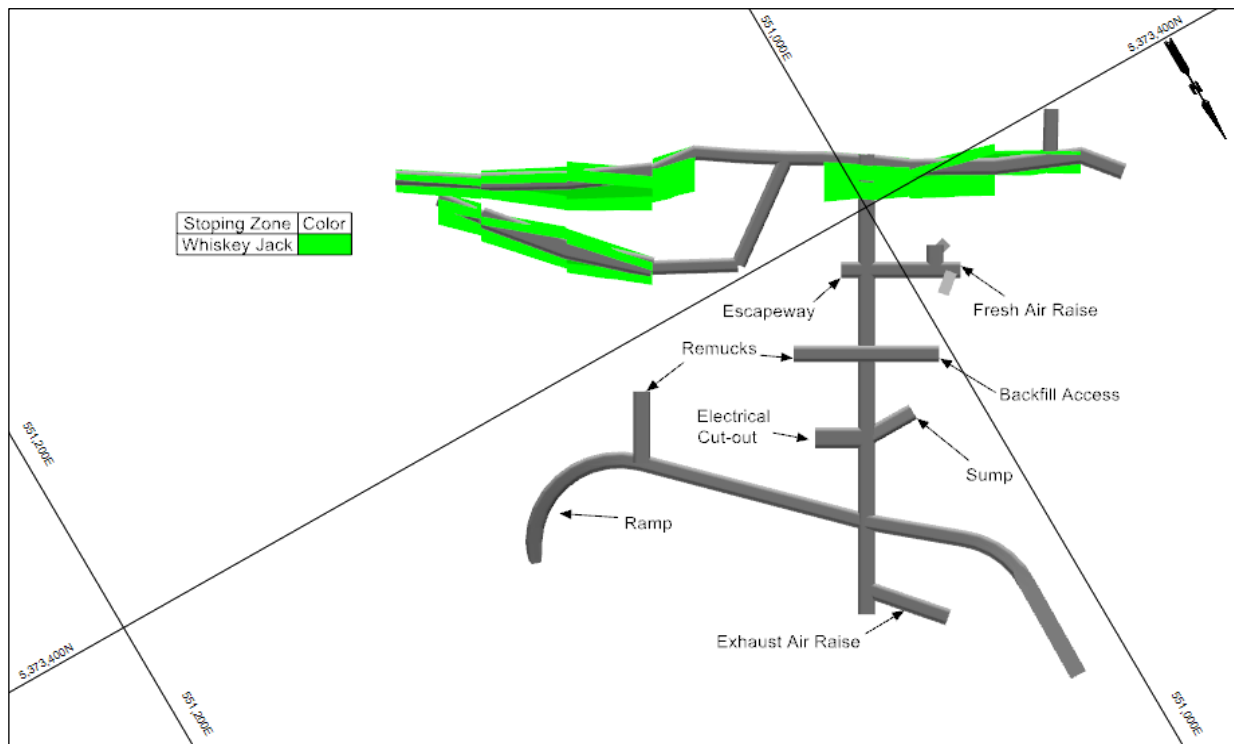


Figure 16-17: Typical Level Development Whiskey Jack Zone 115 Level

The remaining levels in Grey Fox North and Grey Fox South will be developed every 25 m. These zones are more spread out and will have more variance in development level-to-level.

Figure 16-18 presents the typical level development in the second block of Grey Fox North excluding Whiskey Jack Zone. Figure 16-19 presents the typical level development in the first block of Grey Fox South.

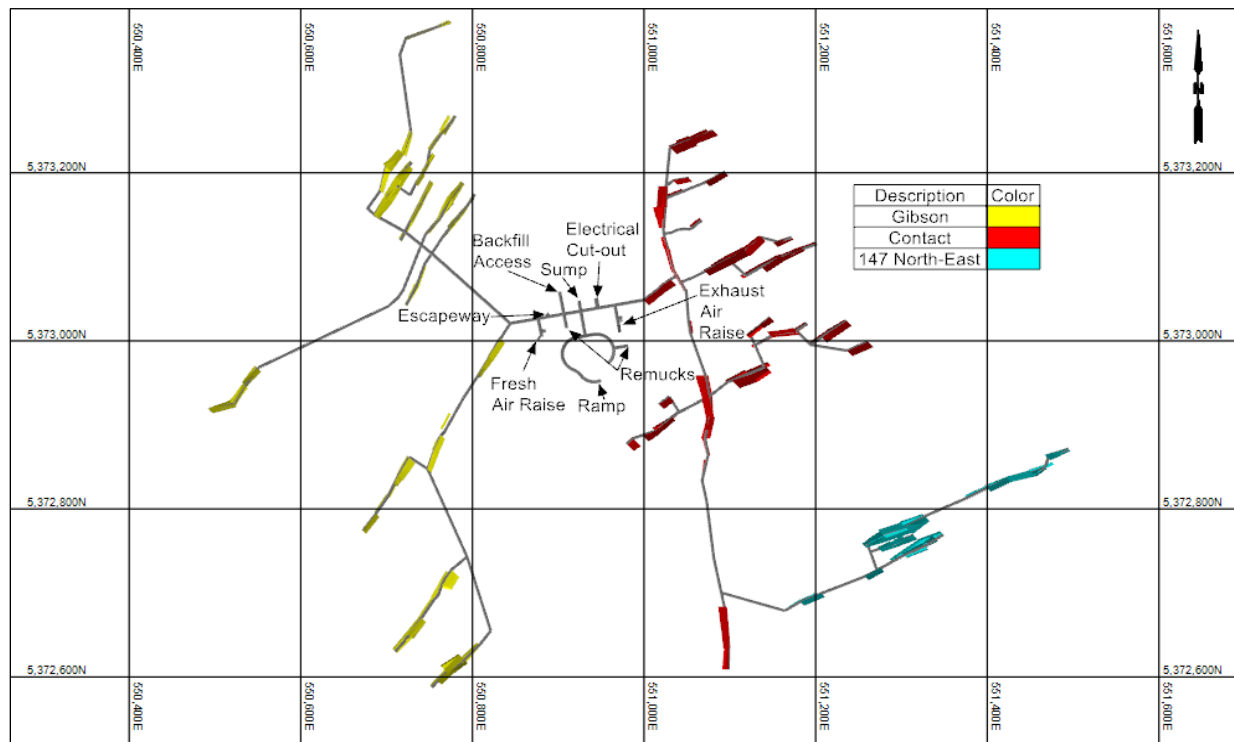


Figure 16-18: Typical Level Development Grey Fox North Zone Block 2 165 Level

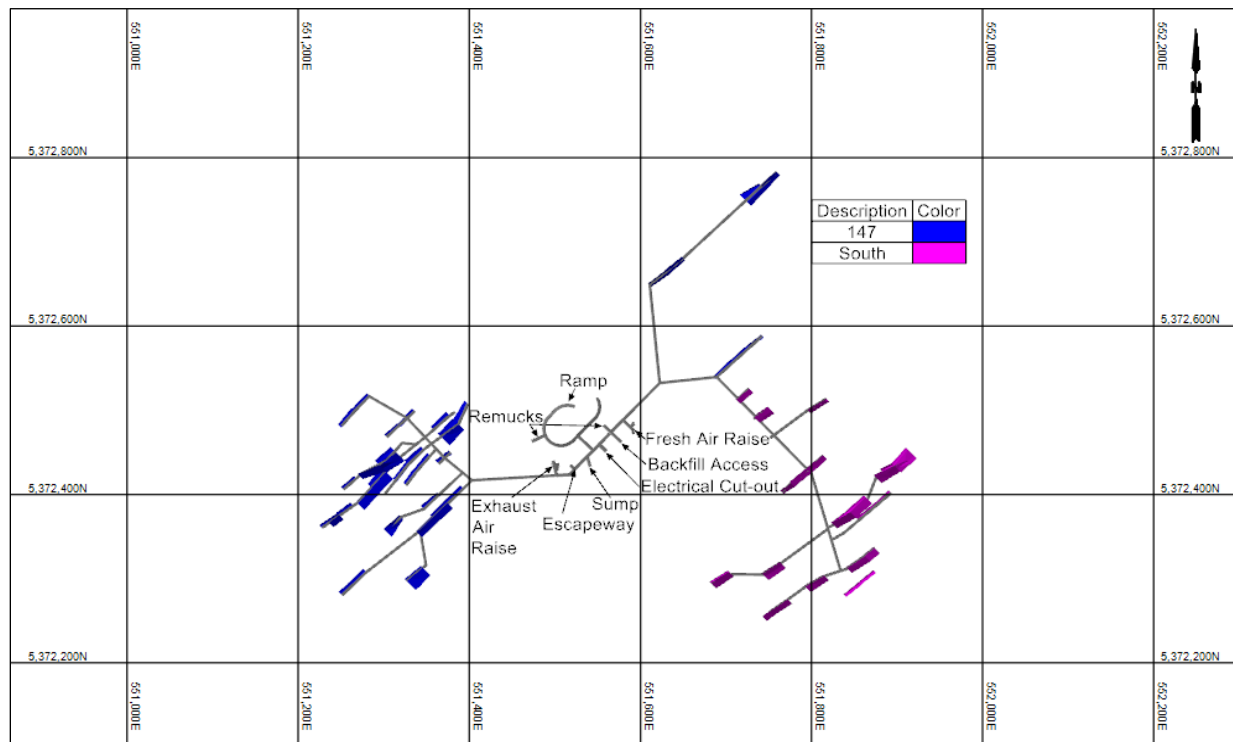


Figure 16-19: Typical Level Development Grey Fox South Block 1 115 Level

16.3.2.2.3 Lateral Development Quantities

All lateral development quantities are expressed in equivalent lateral metres to the dimensions as per below:

- Existing Gibson Ramp Capital – Rehabilitation and Expansion to 4.0 m W x 4.3 m H
- Capital Development – 4.0 m W x 4.3 m H
- Whiskey Jack Operating Development – 4.0 m W x 4.0 m H
- Remaining Operating Development – 3.5 m W x 4.0 m H

The lateral development quantities include cut-and-fill stope access but not cut-and-fill stoping. The lateral development quantities are present in Table 16-11.

Table 16-11: Lateral Development Quantities

Item	Grey Fox North (m)	Grey Fox South (m)	Total (m)
Capital – Existing Gibson Expansion and Rehabilitation	156	-	156
Capital – Ramp and Ramp Remucks	5,791	2,686	8,477
Capital – Level and Infrastructure Development	8,668	4,485	13,153
Operating – Stope Access and Sill Development	40,532	26,310	66,842
Total	55,147	33,481	88,628



16.3.2.3 Vertical Development and Quantities

Vertical development assumptions are summarized in Table 16-12.

Table 16-12: Vertical Development

Raise Type	Raise Size	Dip	Excavation Method
Ventilation Raise (to Surface and Underground)	2.7 m - 3.5 m square (>30 m length)	Vertical (to Surface) minimum 70° (UG)	Alimak
Ventilation Raise (Underground)	2.7 m - 3.5 m square (20-30 m length)	minimum 60°	Drop Raise
Dedicated Escapeway Raise (Underground)	1.2 m diameter (20-90 m length)	minimum 60°	Raisebore

The vertical development quantities are present in Table 16-13.

Table 16-13: Vertical Development Quantities

Item	Grey Fox North (m)	Grey Fox South (m)	Total (m)
Escapeway Raise - Raisebore	604	249	853
Ventilation Raises - Alimak	211	127	338
Ventilation Raises - Drop Raise	1,115	482	1,597
Total	1,930	858	2,788

16.3.3 Primary Development and Intersection Ground Support

The primary ground support for lateral excavations is summarized in Table 16-14.

Table 16-14: Primary Ground Support

Excavation Surface	Bolt	Screen
Back (<5m span)	1.5 m long #6 rebar bolts 1.5 m x 1.5 m dice pattern	#6 Gauge Screen
Back (5-7m span)	1.8 - 2.4 m long #6 rebar bolts 1.2 m x 1.2 m dice pattern	#6 Gauge Screen
Walls	1.5 m long split sets 1.5 m x 1.5 m dice pattern	#6 Gauge Screen Down walls from back to within 2.1 m of the floor

The larger spans at intersections will require secondary ground support. The intersection ground support is summarized in Table 16-15.



Table 16-15: Intersection Ground Support

Rock Type	Support
Good Ground (spans > 7 m)	Center 9 cables - 3.65 m long spin cable 1.5 m spacing Additional cables – 3.0 m long spin cable 1.5 m spacing

16.3.4 Production Stopping

For production mining, longitudinal longhole stoping and overhand cut-and-fill methods will be used.

Longhole stoping requires developing mining levels to access the stopes and facilitate drilling, blasting, and extraction of the material. Once extraction of material within a stope is completed, the stope will be filled with a combination of cemented rock fill or unconsolidated rock fill as required. Longhole is a non-entry mining method (i.e., during mining, personnel will be prohibited from entering the open portion of a stope). Material from within the stope will be mucked remotely using LHDs to reduce worker exposure to potential ground hazards.

The longhole stoping equipment will include:

- Drill - Slot Drill Rig (with V30 Reamer) and Production Drill Rig
- Blast - Explosive Loader
- Muck - 10 t class LHD
- Fill - 10 or 7 t class LHD

Cut-and-fill requires dividing targeted mineral resource into stacked development drifts. The bottom most drift will be extracted first followed by the next drift above once the previous lower drift is filled with cemented rock fill. The final cut will not require any filing. The cut-and-fill mining method follows the same principles as conventional lateral development otherwise. The cut-and-fill equipment will be the same equipment used for development. Table 16-16 presents the recovered ore tonnes by mining method. Most of the mineral resource will be extracted via the longitudinal longhole method.

Table 16-16: Ore Tonnes by Mining Method

Mining Method	Grey Fox North Recovered Ore Tonnes	Grey Fox South Recovered Ore Tonnes	Total Recovered Ore Tonnes
Longitudinal Longhole	4,713,651	3,009,295	7,722,946
Cut and Fill	55,165	62,819	117,984
Development	921,239	643,660	1,564,899
Total	5,690,055	3,715,774	9,405,829

16.3.4.1 Mining Blocks

To achieve the planned production rate and allow flexibility in the schedule, production will be required concurrently from multiple fronts. To achieve this, and limit congestion, the mine will be split into mining blocks, and where possible the infrastructure will be grouped by mining block. The mining block divisions



attempt to minimize the occurrence of mining under backfill, and the divisions are not strictly from one level to the next but change per ore lens based on the local geometry. Each mining block will use a bottom-up approach where possible.

The main vertical mining blocks are as follows and presented in Figure 16-20, Figure 16-21, and Figure 16-22 for the Whiskey Jack Zone, the remaining Grey Fox North, and Grey Fox South, respectively:

- Whiskey Jack Zone Mining Block 1: 145 Level to 65 Level
- Whiskey Jack Zone Mining Block 2: 295 Level to 145 Level
- Grey Fox North Mining Block 1: 140 Level to 40 Level
- Grey Fox North Mining Block 2: 265 Level to 140 Level
- Grey Fox North Mining Block 3: 365 Level to 265 Level
- Grey Fox North Mining Block 4: 465 Level to 365 Level
- Grey Fox South Mining Block 1: 140 Level to 40 Level
- Grey Fox South Mining Block 2: 240 Level to 140 Level
- Grey Fox South Mining Block 3: 340 Level to 240 Level

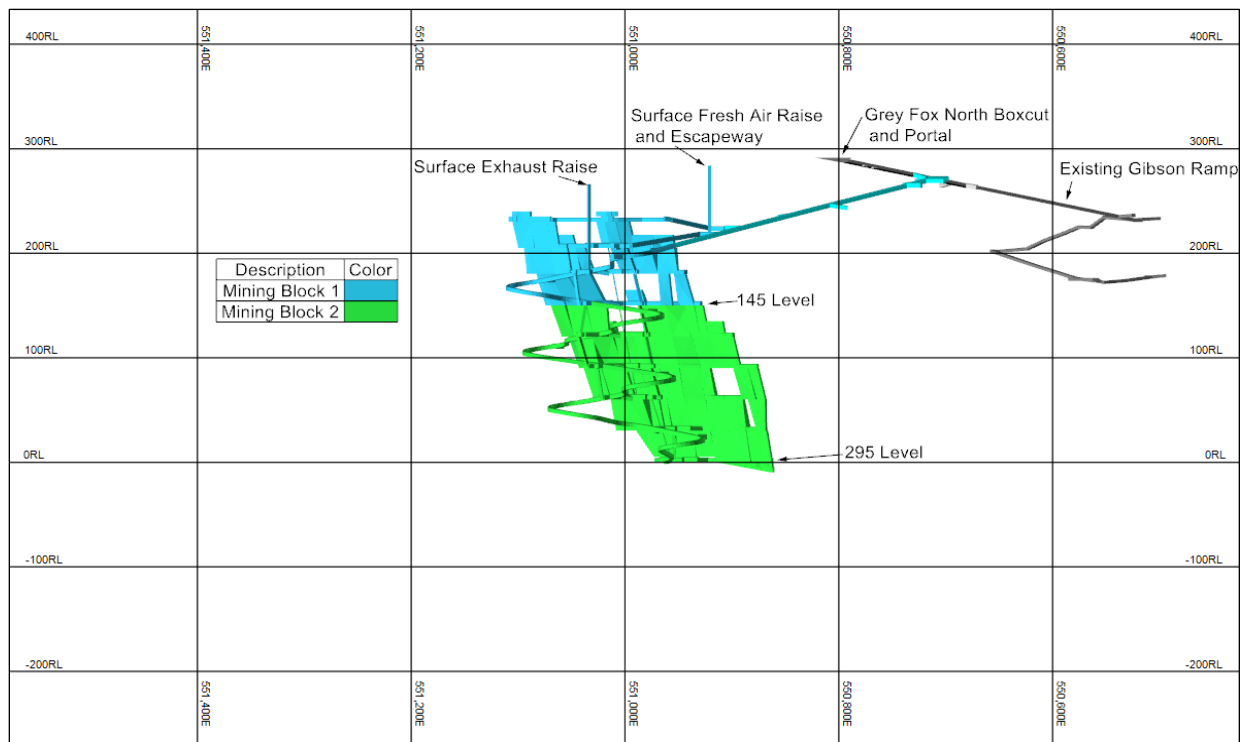


Figure 16-20: Whiskey Jack Zone Mining Blocks Looking South



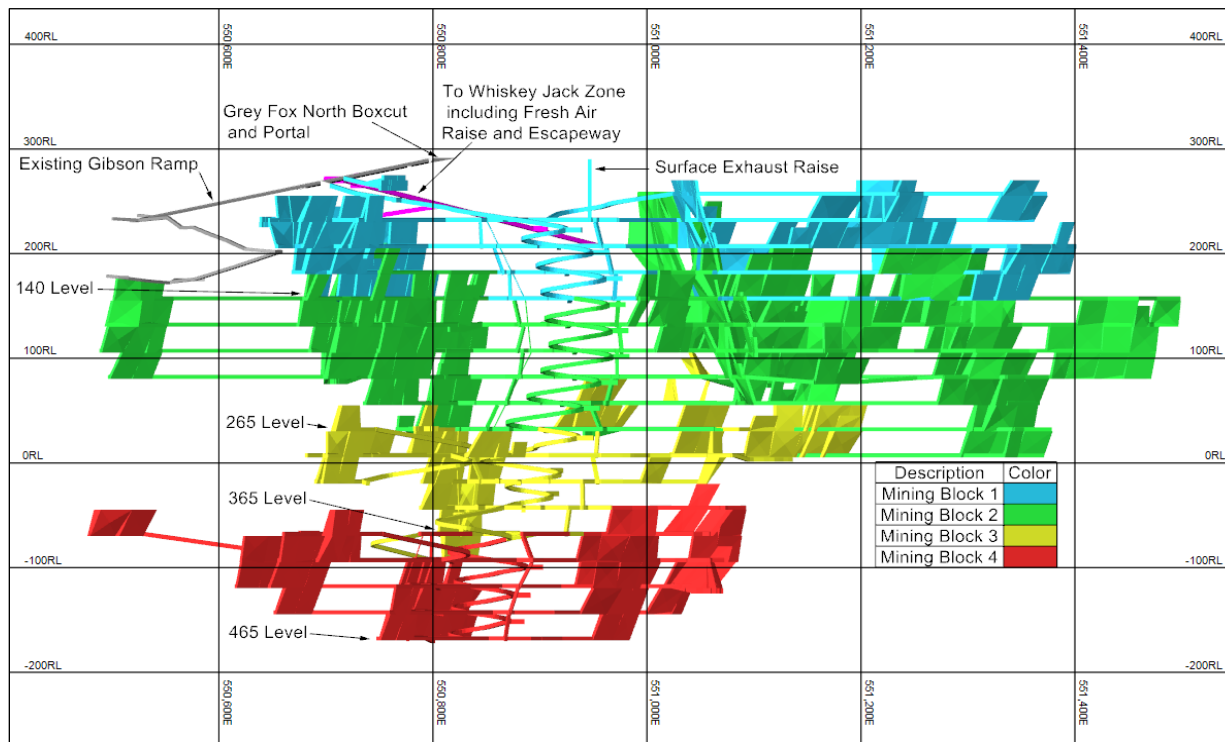


Figure 16-21: Remaining Grey Fox North Mining Blocks Looking North

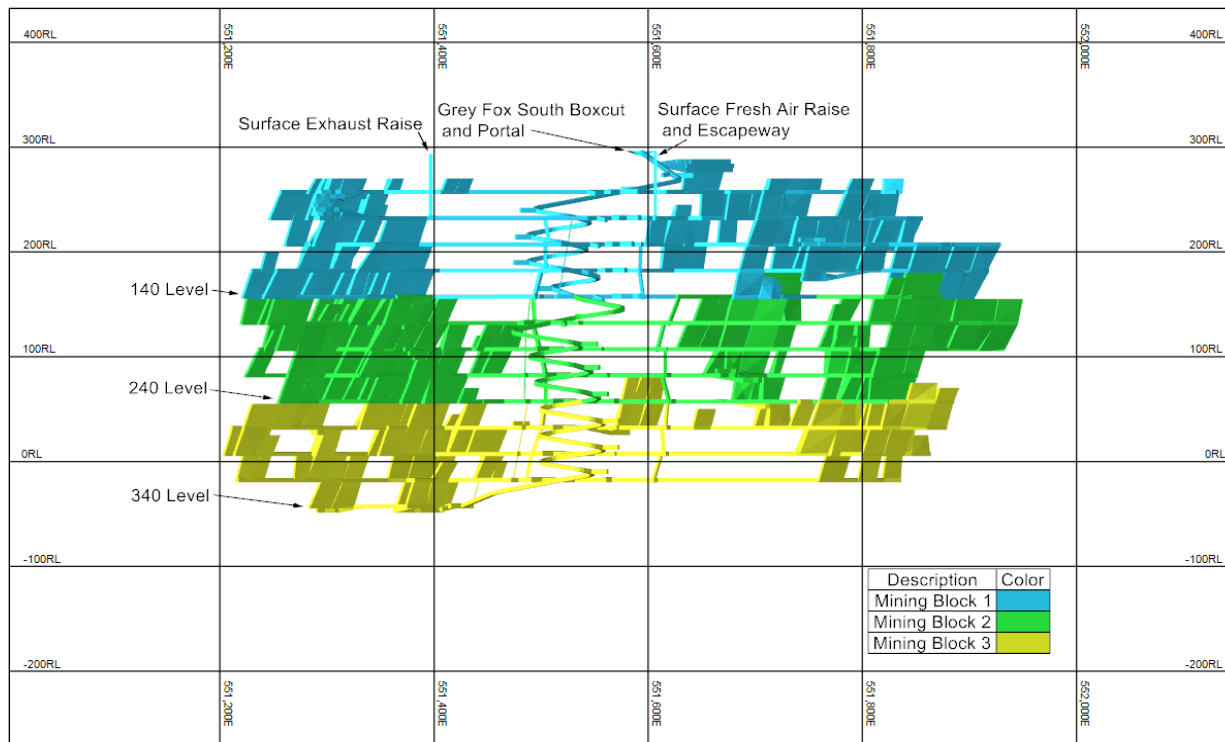


Figure 16-22: Grey Fox South Mining Blocks Looking North



16.3.5 Longhole and Cut-and-Fill Stopping Design

Longitudinal longhole stopping will be the primary recovery method. Some stopes will be mined with up-holes only when a drill overcut is not suitable or unavailable. Where suitable, shorter length sub-stopess are merged with adjacent stopess not exceeding 40 m. The longhole stope design criteria are summarized in Table 16-17.

Table 16-17: Longhole Stope Design Criteria

Description	Whiskey Jack Base	All Other Zones Base	Range
Level Interval (floor to floor)	30 m	25 m	25 to 30 m
Stope Length (along strike)	20 m	25 m	10 to 40 m
Stope Width (HW to FW)	5.0 m	3.5 m	Minimum 3.0 m
Stope Footwall dip	70°	70°	Minimum 45°
Minimum Pillar Width from filled stopess (between longitudinal lenses, footwall to hanging wall)	Minimum 3 m <300 m below surface, 6 m otherwise		
Minimum Pillar Width from unfilled uphole stopess (between longitudinal lenses, footwall to hanging wall)	Minimum 10 m <300 m below surface, 15 m otherwise		

Overhand cut-and-fill will be used to supplement the longhole stopess with up to 5 m H cuts. The cut-and-fill stope design criteria are presented in Table 16-18.

Table 16-18: Cut-and-Fill Stope Design Criteria

Description	Base	Range
Cut Width (HW to FW)	3.0 m	3.0 to 5.0 m
Cut Height (floor to floor)	5.0 m	3.0 to 5.0 m
Number of Cuts per Sequence	2	1 to 3

16.3.5.1 Stope Sequencing

Stopess will be sequenced to minimize equipment interaction between stopess and maintain overall ground stability. Constraints will be placed on the sequence when two stopess share an access (e.g., the drill horizon for one stope and mucking horizon for the stope above) such that activities do not occur concurrently. In the cases of multiple lenses, there will be a preference to mine the footwall stope prior to the hanging wall stope for ground control purposes. Where possible, up-hole stopess will be sequenced last as these will not be backfilled.

The longitudinal (L) stopess will be sequenced using a bottom-up retreat approach back to an access. Stopess in a longitudinal sequence are dependent on the stopess adjacent and below. Stopess are accessed via a center access (Figure 16-23) or an access on one of the strike ends (Figure 16-24).



Cut-and-fill stopes will be mined overhand (bottom-up) where multiple cuts are mined in sequence via one attack ramp.

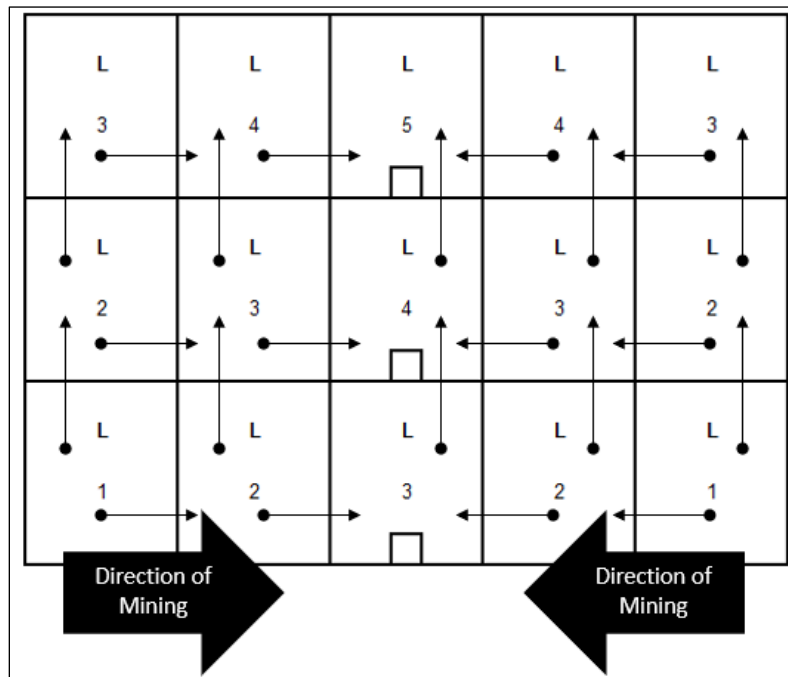


Figure 16-23: Typical Longitudinal Stope Sequence with Center Access (Long Section View)

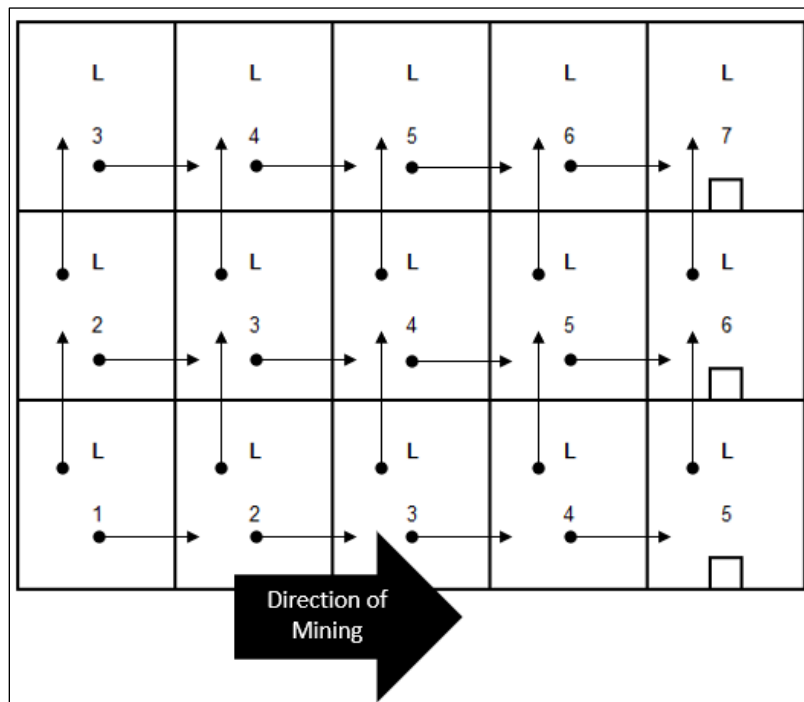


Figure 16-24: Typical Longitudinal Stope Sequence with End Access (Long Section View)



16.4 Productivities

16.4.1 Annual Working Days and Shift Schedule

The annual working days and shift schedule are listed in Table 16-19.

Table 16-19: Working Days and Shift Schedule

Item	Value
Days per Year	360
Project Period Shift Schedule	2 × 12 hours
Operating Period Shift Schedule	2 × 10 hours

16.4.2 Effective Hours

The underground operations will consist of two 12-hour shifts per day, seven days per week for contractor personnel and two 10-hour shifts per day, seven days a week for the McEwen personnel. Time worked during these shifts is expected to fall into one of two categories: effective or non-effective.

Non-effective time may result from necessary parts of the job (e.g., travel to the workplace, equipment refueling, and shift safety meetings) during which workers are not actively conducting mining work.

The effective time calculation presented in Table 16-20 represents the available time underground that a worker will be able to complete tasks.

Table 16-20: Effective Time Calculations

Item	Zone					Unit
	Priority Ramp	Whiskey Jack	Remaining Grey Fox	Whiskey Jack	Remaining Grey Fox	
Personnel	Contractor	Contractor	Contractor	McEwen	McEwen	
Shifts per Day	2	2	2	2	2	shift/day
Work Week	7	7	7	7	7	days
Planned Hours per Shift	12	12	12	10	10	hour/shift
Planned Minutes per Shift	720	720	720	600	600	min/shift
Rotation	7 on / 7 off	7 on / 7 off	7 on / 7 off	7 on / 7 off	7 on / 7 off	day
Non-Productive Activities						
Shift Line-up	15.0	15.0	15.0	15.0	15.0	min/shift
Daily Safety Toolbox Huddle	5.0	5.0	5.0	5.0	5.0	min/shift
Monthly Safety Meeting (60 min/mo)	2.1	2.1	2.1	2.1	2.1	min/shift



Item	Zone					Unit
Transportation to Work Area	14.8	19.4	18.9	19.4	18.9	min/shift
Field Level Risk Assessment / Worksite and Equipment Inspection	10.0	10.0	10.0	10.0	10.0	min/shift
Equipment Servicing/Refuelling	15.0	15.0	15.0	15.0	15.0	min/shift
Break 1	0.0	0.0	0.0	15.0	15.0	min/shift
Lunch	0.0	30.0	30.0	30.0	30.0	min/shift
Break 2	0.0	0.0	0.0	15.0	15.0	min/shift
Supervisory/Survey/Engineering Visits	15.0	15.0	15.0	15.0	15.0	min/shift
Transportation from Work Area	15.5	24.1	23.1	24.1	23.1	min/shift
Miscellaneous Delays	5.0	5.0	5.0	5.0	5.0	min/shift
Blast Delays	20.0	30.0	30.0	5.0	5.0	min/shift
Total Non-Productive Time per Shift	117.4	170.6	169.1	175.6	174.1	min/shift
Total Effective time per shift	10.04	9.16	9.18	7.07	7.10	hours

16.4.3 Labour

The majority of the underground development and construction will be completed by McEwen personnel. Contractors will be mostly utilized for the initial year of development activities, underground infrastructure construction and maintenance and replaced by McEwen personnel where required. Contractors will also be used for ongoing Alimak raising and raise bored escapeways throughout the LOM.

The underground labour will peak at 327 personnel on-site payroll. The total underground related labour (peak) by year is presented in Table 16-21.



Table 16-21: Total On-site Labour

Labour Type	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Contractor															
Integrated Project Execution Team	16	17	17	1											
Mine Management		1	1	1	1	1									
Mine Operations	26	50													
Mine Maintenance	20	30													
McEwen															
Integrated Project Execution Team	25	29	29	2											
Mine Management		5	5	29	31	33	38	38	40	39	39	38	35	26	15
Mine Operations	4	27	85	97	109	113	147	177	209	205	209	215	207	185	37
Mine Maintenance	2	28	40	50	52	52	60	70	78	78	74	74	73	64	20
Total On-Site	93	187	177	180	193	199	245	285	327	322	322	327	315	275	72

Note: Numbers may not sum due to rounding.



16.4.4 Lateral Development

The lateral development advance rates were divided into the components of the drill-blast-muck-bolt cycle and estimated from first principles. Advance rates were developed for single and multiple heading scenarios. The rates reflect the advance each drill jumbo crew and its associated gear will achieve over extended periods of operation. The rates reflect long-term averages and include an efficiency allowance. The efficiency allowance accounts for interferences from other activities and conflicting priorities that occur during the operating period.

The priority ramp development rate is estimated with all breaks or interruptions to affect secondary headings and blast at will practices shall be employed before production begins. It is only used while establishing the main ramp and initial levels of the Whiskey Jack Zone with contractors in the first year of development.

The Gibson Ramp rehabilitation will be handled within the first year of development with only contractors on site. Different ore development rates were estimated for the Whiskey Jack Zone as there is generally wider development. The estimated development productivities used in the schedule are summarized in Table 16-22.

Table 16-22: Lateral Development Productivity

Item					Unit
Personnel	Contractor	Contractor	McEwen	McEwen	
Development Type	Single Heading	Multi-Heading	Single Heading	Multi-Heading	
Priority Ramp Development	6.8	9.1	N/A	N/A	m/d
Gibson Ramp Rehabilitation	8.2	13.5	N/A	N/A	m/d
Ramp Development	5.5	9.1	4.2	7.0	m/d
Lateral Waste Development	5.4	8.9	4.2	6.9	m/d
Ore Development Whiskey Jack	5.7	9.4	4.4	7.3	m/d
Ore Development	6.1	10.1	4.7	7.8	m/d

The time for each element of the development cycle (4.0 m average round length) for a 4.0 m W × 4.3 m H ramp heading is presented in Table 16-23.

Table 16-23: Development Cycle for the Ramp (4.0 m W × 4.3 m H)

Item	Value	Unit
Drill	2.9	hours
Load/Blast	1.2	hours
Muck	2.1	hours
Ground Support	3.4	hours
Services Install (Critical Path)	0.2	hours
Total Cycle Time	9.8	hours



16.4.1 Vertical Development

The vertical development advance rate using the drop raise method includes setup and teardown, pilot and reaming of a V30 762mm (30"), and drilling and blasting of the raise. All drop raise activities will be completed by McEwen personnel. Table 16-24 presents drop raise productivities.

Table 16-24: Drop Raise Development Productivity

Drop Raise Dimensions	Rate	Unit
3.1 m x 3.1 m	1.8	m/d
2.7 m x 2.7 m	2.2	m/d

The vertical development advance rate using the raise boring method includes setup and teardown, pilot hole drilling, and reaming of the raise. The only planned raisebore development is for the dedicated escapeways. All raisebore activities will be completed by a contractor. Table 16-25 presents the raise bore productivity.

Table 16-25: Raise Bore Development Productivity

Raise Bore Dimensions	Rate	Unit
1.1 m (42") Diameter	6.3	m/d

The vertical development advance rate using the Alimak method includes drilling, blasting, mucking, bolting, and installation of rails. Set-up and teardown are a single lag of 10 days in schedule. The 3.5 m x 3.5 m raises will require two passes. All Alimak raise activities will be completed by a contractor.

The raise development rates are summarized in Table 16-26.

Table 16-26: Alimak Raise Development Productivity

Alimak Raise Dimensions	Rate	Unit
2.7 m x 2.7 m	3.6	m/d
3.1 m x 3.1 m	3.2	m/d
3.5 m x 3.5 m	1.9	m/d

16.4.2 Stope Productivity

Stope production rates were divided into the components of the drill-blast-muck and backfill cycle and estimated from first principles. Stope cycles were estimated for the average longitudinal stope sizes. The cycle includes slot raise drilling, production drilling, blasting, mucking, and backfill. Slot raise drilling will be performed by a drill rig equipped with a V30 762 mm (30") reamer bit and includes a pilot hole and reaming. Production drilling utilizes 89 mm (3.5 in) holes. All stoping activities are to be completed by McEwen personnel. The stope productivities are summarized in Table 16-27.



Table 16-27: Longhole Stope Productivities

Item	Whiskey Jack	All Other Zones	Unit
Slot Raise	3.5	4.0	m/d
Production Drilling	129	148	m/d
Blasting	2	2	days
Mucking (including loading trucks)	493	463	t/d
Backfilling	571	432	t/d
Average Stope Size	8,100	5,300	t
Average Total Tonnes per Day	167	141	t/d
Average Total Tonnes per Day Excluding Backfill	210	189	t/d

The stope average cycle times are summarized in Table 16-28.

Table 16-28: Longhole Stope Average Cycle Times

Item	Whiskey Jack	All Other Zones	Unit
Slot Raise	8	6	days
Production Drilling	12	9	days
Blasting	2	2	days
Mucking (including loading trucks)	17	12	days
Backfilling	10	10	days
Average Stope Size	8,100	5,300	t
Average Cycle Time	49	38	days
Average Cycle Time Excluding Backfill	39	28	days

16.5 Mine Production Schedule

16.5.1 Underground Development Scheduling

All mine development and production scheduling has been prepared using Deswik scheduling software. The schedule is interactively linked to the 3D mine model. All development and production scheduled is based on dependencies linked within the mine model. All data are contained within the mine model and schedule.

The primary development focus will be the ramp to Whiskey Jack Zone and establish the ventilation system. Whiskey Jack is the first focus for production as it is of the highest-grade resource and requires less development per ore tonne. On each level, priority will be to establish the sump, electrical cut-out, and ventilation access followed by the rest of the infrastructure while ore development will be planned as late as possible without delaying stoping activities. The total lateral development is separated into Capital and Operating and summarized annually in Table 16-29, and Figure 16-25 presents a graph of this data



Table 16-29: Summary of Annual Lateral Development Metres Profile

Development Type	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Operating Lateral	14	2,095	3,725	5,124	3,943	4,871	4,813	8,938	10,383	4,048	6,452	5,629	6,075	730	-
Capital Lateral	1,417	3,756	1,294	94	1,387	441	2,954	1,843	172	3,245	1,501	2,380	1,178	125	-
Total Lateral	1,431	5,851	5,020	5,218	5,330	5,313	7,767	10,781	10,555	7,293	7,953	8,009	7,253	855	-

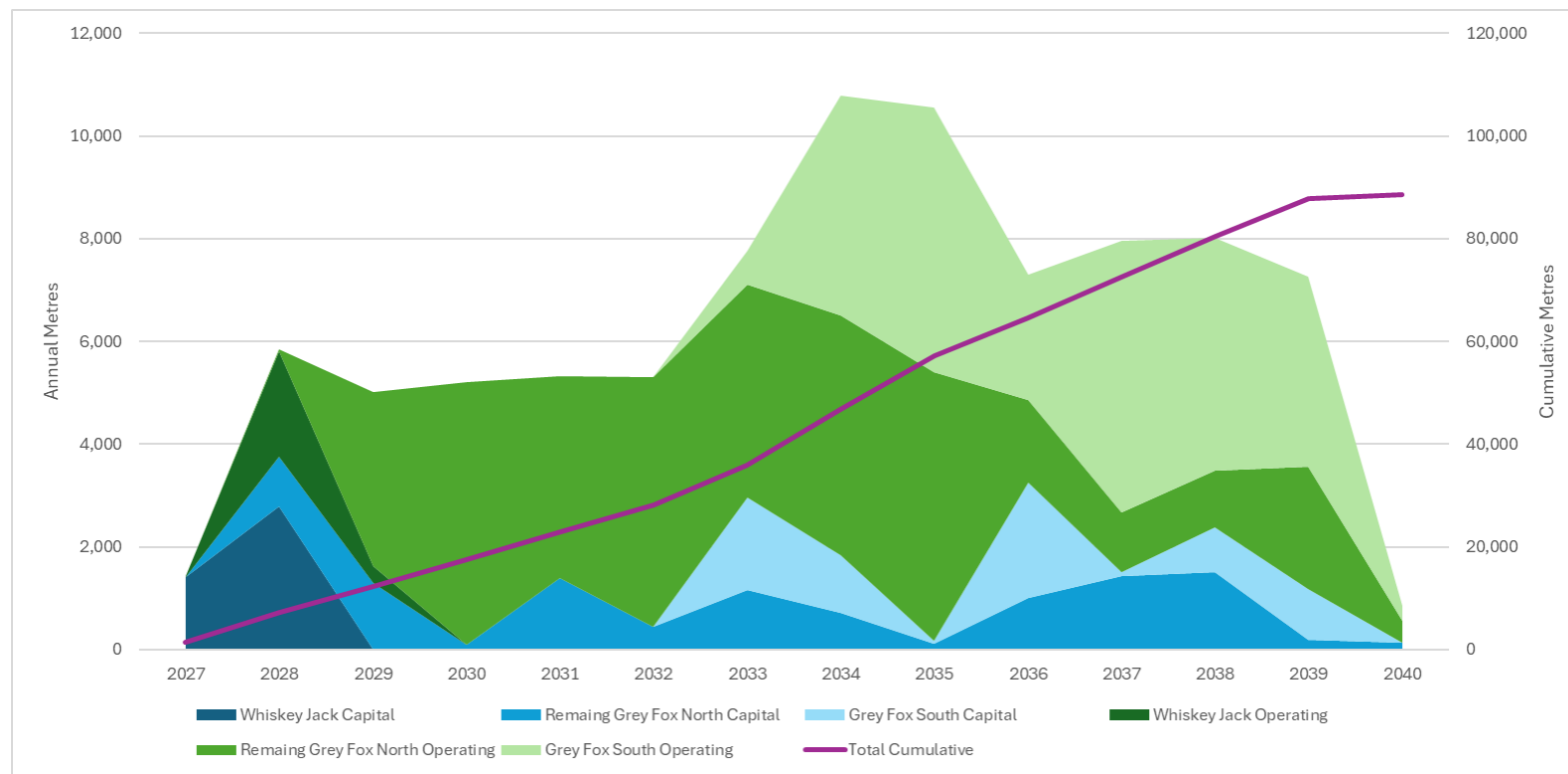


Figure 16-25: Annual Lateral Development Metres Profile



16.5.2 Underground Production Scheduling

The focus at Grey Fox will be to extract the higher-grade Whiskey Jack Zone first. Mining will then ramp-up to 1,500 t/d with the rest of Grey Fox North and then to 2,700 t/d with the addition of Grey Fox South in 2035. The process plant is located on McEwen's Stock Property. The overall production profile and tonnage targets aforementioned are expected to be complimentary with production from other assets, notably the Stock mine, in maximizing the throughput of the Stock Mill. This is part of a "hub and spoke" asset strategy for the McEwen Assets along Highway 101.

16.5.1 Production Profile

Table 16-30 and Figure 16-26 present the LOM production tonnes and profile.



Table 16-30: Summary of Production Profile by Mining Method

Ore Type	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Stoping Ore Tonnes	-	63,378	230,630	375,495	385,333	399,603	407,400	297,073	652,813	761,885	674,422	731,355	773,894	731,689	327,171
Development Ore Tonnes	-	65,079	85,918	111,472	90,869	122,508	104,410	196,130	238,568	82,550	178,309	135,939	135,207	17,941	-
Cut and Fill Ore Tonnes	-	8,083	10,082	3,995	2,048	8,444	5,058	3,552	19,758	28,920	9,765	1,137	15,101	2,041	-
Marginal Stopping Ore Tonnes	-	-	27,534	38,266	83,375	27,826	43,532	52,901	72,357	108,139	121,820	119,798	50,124	95,471	69,661
Total Ore Tonnes	-	136,540	354,164	529,227	561,625	558,380	560,400	549,657	983,496	981,494	984,316	988,229	974,325	847,143	396,832
Average t/d	-	374	970	1,450	1,539	1,530	1,535	1,506	2,695	2,689	2,697	2,707	2,669	2,321	1,087

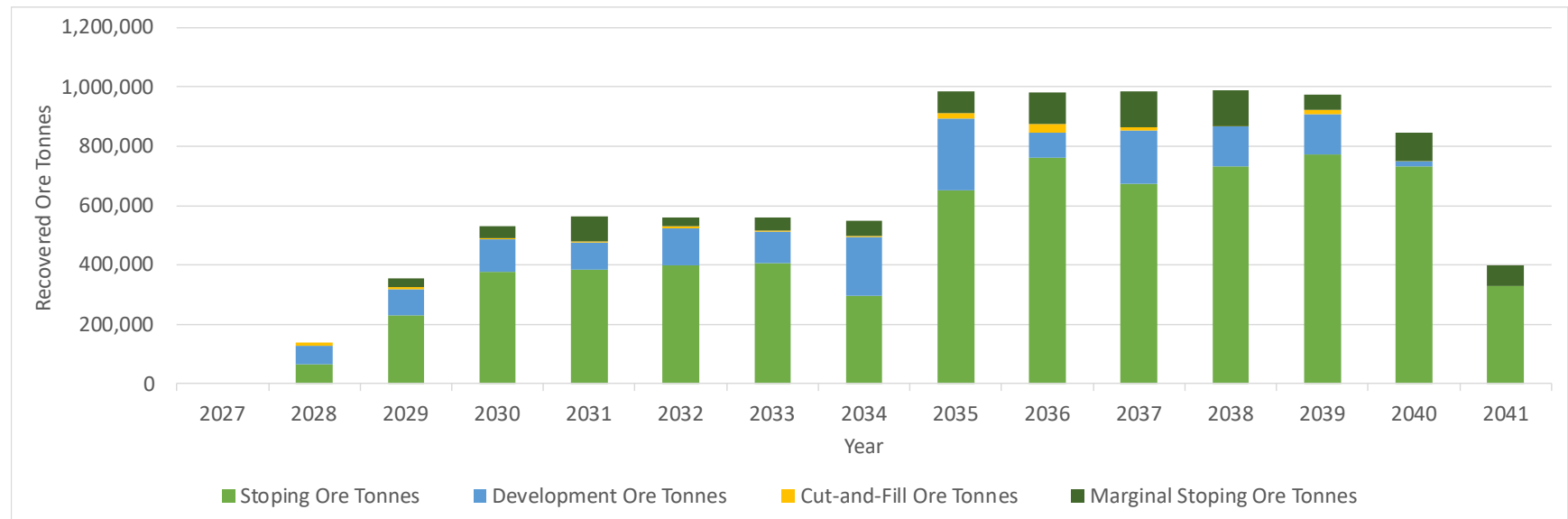


Figure 16-26: Production Profile by Mining Method



16.6 Blasting

16.6.1 Development

Development rounds will be loaded using a mechanical explosives loader. ANFO products will be used for all primary development blasting. Specialty blasting such as lifters will be performed with package emulsion products as required.

16.6.2 Stopping

ANFO and packaged emulsion products will be used for all longitudinal stope blasting. Non-electronic detonators will be used to control blast timing.

Cut-and-fill stopping will be loaded using a mechanical explosives loader. ANFO products will be used for all cut-and-fill blasting.

16.7 Mobile Equipment Selection

Grey Fox will be developed using a high degree of automated equipment. The main production equipment will be capable of remote operation. Mobile equipment was selected that will be appropriate for the mining methods and to support the production rate.

Equipment fleet costs and performance are supported by information from the original equipment manufacturers (OEMs) and only commercially available products were considered. The mine development contractor will use and maintain equipment provided by McEwen and will return the equipment to McEwen in proper working condition. The Alimak and raise boring raising equipment is excluded from the equipment lists as they will be provided by contractors.

Key elements considered in the ultimate equipment selection included:

- Mine design
- Mining methods
- Productivities
- Fleet standardization of key components
- Maintainability
- Economics

The diesel equipment fleet includes the main development and production gear as well as auxiliary gear to support operations.

The makes and models in the following equipment lists were used for pricing estimates at this time, similar equipment could be used.

Table 16-31 presents the complete equipment list including type, function, make, model, and total peak units for Grey Fox North.



Table 16-31: Grey Fox North Underground Mobile Equipment List

Equipment Type	Primary Function	Units
10 t LHD	Transferring muck from draw point to remuck or remuck to truck. Transferring backfill to stopes.	5
7 t LHD	Transferring muck from development headings or raise undercuts to remuck or remuck to truck or to open stopes.	2
20 t Truck	Transport material to surface or remuck	6
2-Boom Jumbo	Development drilling and face ground support installation	2
Mechanized Bolter	Ground support installation	3
Explosive Loader	Development and production explosive loading	1
Scissor Lift	Construction support	2
Production Drill Rig	Production drilling, miscellaneous boreholes	2
Slot Drill Rig	Stope Slot Production Drilling, Drop Raises, miscellaneous boreholes	2
Forklift/Tractor	Material Handling	3
Boom Truck	Material Handling and Transport	2
Motor Grader	Ramp maintenance	1
Small Personnel Carrier	Crew / personnel transportation	3
Small Carrier (Mech/Elec)	Maintenance personnel access to equipment outside of the shop	2
Fuel and Lubrication Truck	Provide fuel and lubricants to production equipment	1

Table 16-32 presents the complete equipment list including type, function, make, model, and total peak units for Grey Fox South.

Table 16-32: Grey Fox South Underground Mobile Equipment List

Equipment Type	Primary Function	Units
10 t LHD	Transferring muck from draw point to remuck or remuck to truck. Transferring backfill to stopes.	5
7 t LHD	Transferring muck from development headings or raise undercuts to remuck or remuck to truck or to open stopes.	2
20 t Truck	Transport material to surface or remuck	6
2-Boom Jumbo	Development drilling and face ground support installation	2
Mechanized Bolter	Ground support installation	3
Explosive Loader	Development and Production explosive loading	1
Scissor Lift	Construction support	2
Production Drill Rig	Production drilling, miscellaneous boreholes	3
Slot Drill Rig	Stope Slot Production Drilling, Drop Raises, miscellaneous boreholes	2
Forklift/Tractor	Material Handling	3
Boom Truck	Material Handling and Transport	2



Equipment Type	Primary Function	Units
Motor Grader	Ramp maintenance	1
Small Personnel Carrier	Crew / personnel transportation	3
Small Carrier (Mech/Elec)	Maintenance personnel access to equipment outside of the shop	2
Fuel and Lubrication Truck	Provide fuel and lubricants to production equipment	1

For Grey Fox North, beginning in August 2027, a single crew will advance the main ramp down to Whiskey Jack and establish the fresh air ventilation. Once flow through ventilation is established in January 2028, a second development crew will be introduced. This second crew will remain until January 2036, and the remainder of mine life will require only one crew to support mining operations.

For Grey Fox South, beginning April 2033, a single crew will advance the main ramp to establish the flow through ventilation. Once established in October 2033, a second development crew will be introduced. This second crew will remain until January 2040, and the remainder of mine life will require only one crew to support mining operations.

Each development crew will have a two-boom drill jumbo for face drilling and installing ground support, access to a shared ANFO loader for development headings, a 7-t LHD to muck the face and load trucks, and a scissor lift for installing services. The development crew will be supported by the haulage truck fleet.

Vertical development will be a combination of drop raises, Alimak raises and raise boring. Drop raising will be completed using the slot drill unit as required. Alimak and escapeway raise boring will be completed by contractors providing the equipment as required.

Production will be largely completed by longitudinal longhole mining. Longhole ITH and top hammer drills will be used to drill the blastholes, the blastholes will be loaded using a shared explosive loader, and an LHD will muck the stope to the nearest remuck or to a waiting truck.

The fleet for each of Grey Fox North and Grey Fox South includes three mechanical bolters for cable bolt drilling and other secondary ground support installation when required.

For loading and transport, Grey Fox North and Grey Fox South, each include up to six haulage trucks with one unit being shared during peak production years for a total of 11 units. These will be accompanied by up to five 10-t LHDs for stoping, for Grey Fox North and Grey Fox South, with one unit being shared between them for a total of nine units.

There will be a peak of 37 and 38 pieces of equipment operating in Grey Fox North and Grey Fox South, respectively. This results in an overall peak of 68 total pieces of equipment operating on site, as some units are shared or permanently transferred between the two ramp networks.

The number of jumbos used for the overall peak (3) is different then the jumbo peak (4), as the overall peak does not coincide jumbo peak. There are six shared or transferred units which are one LHD, one truck, one mechanical bolter, one grader, one fuel truck, and one slot drill rig.

The mobile equipment fleet operating each year is summarized in Table 16-33.



Table 16-33: Total Operating Equipment by Year for Grey Fox Site

Equipment Type	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
10-t LHD	0	1	3	5	5	5	5	5	9	9	9	9	9	9	2
7-t LHD	1	2	2	2	2	2	3	4	4	4	4	4	4	3	1
20-t Truck	1	3	3	4	4	5	7	8	9	10	11	11	11	11	4
2-Boom Jumbo	1	2	2	2	2	2	3	4	4	3	3	3	3	2	0
Mechanized Bolter	2	3	3	3	3	3	5	5	5	5	5	5	5	4	0
Explosive Loader	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2
Scissor Lift	1	2	2	2	2	2	3	4	4	4	4	4	4	4	1
Production Drill Rig	0	1	2	2	2	2	2	3	4	4	5	5	5	5	2
Slot Drill Rig	0	1	1	1	2	2	2	3	3	3	3	3	3	3	2
Forklift/Tractor	1	3	3	3	3	3	4	6	6	6	6	6	6	5	2
Boom Truck	0	2	2	2	2	2	2	4	4	4	4	4	4	4	1
Motor Grader	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Small Personnel Carrier	1	3	3	3	3	3	4	6	6	6	6	6	6	6	3
Small Carrier (Mech/Elec)	0	2	2	2	2	2	2	4	4	4	4	4	4	4	1
Fuel and Lubrication Truck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	9	28	31	34	35	36	46	60	66	66	68	68	68	64	23



Equipment operating hours have been estimated from first principles. The requirement for equipment specific to development or production will fluctuate with changes to annual throughput, while some equipment will operate at a certain rate regardless of the tonnage or metres being developed.

A mobile equipment rebuilds and replacements schedule was developed based on the operating hours required and the available equipment. These rebuilds and replacements are determined by the life of the engine on the units. The total number of rebuild and replacement quantities are presented in Table 16-34.

Table 16-34: Total Number of Rebuilds and Replacements by Equipment Type

Equipment Type	Rebuilds	Replacements
10 t LHD	14	7
7 t LHD	6	5
20 t Truck	12	5
2-Boom Jumbo	0	0
Mechanized Bolter	0	0
Explosive Loader	1	1
Scissor Lift	3	2
Production Drill Rig	0	0
Slot Drill Rig	0	0
Forklift/Tractor	6	3
Boom Truck	4	2
Motor Grader	1	1
Small Personnel Carrier	3	0
Small Carrier (Mech/Elec)	2	0
Fuel and Lubrication Truck	1	0
Total	53	26

16.8 Ventilation

The underground mine ventilation system is designed to be a “push-pull” system, with fans at the top of the fresh air raise and at the portals pushing fresh air into the mine and fans at the top of the return air raise pulling exhaust and contaminated air out of the mine. Heaters will be operated as required on surface at the fresh air raise in conjunction with the fresh air fans to prevent freezing in winter.

The intake ventilation fans will have a maximum design capacity of 50 m³/s each with parallel fans providing up to 250m³/s of intake air at the intake raise and portal, and the exhaust fans will provide 250 m³/s at each exhaust fan station to support development and production activities for the LOM. This ventilation capacity is based on the requirement to provide airflow for diesel particulate matter dilution, dust control, and to maintain adequate airflow through all active areas.



16.8.1 Airflow Requirements

To determine the ventilation requirements for the operating equipment at Grey Fox North and Grey Fox South, full production activities were considered with Whiskey Jack and Gibson/Contact/147NE evaluated separately due to a difference in the mobile fleet requirements and therefore the ventilation requirement in those zones.

A utilization factor was applied to estimate the operating engine power for each type of equipment, and to estimate the ventilation required (0.063 m³/s of ventilation air was provided for each kW of engine power operating). The full production ventilation required for Whiskey Jack, Grey Fox North, and Grey Fox South are provided in *Table 16-35*, *Table 16-36*, and *Table 16-37*, respectively. This is calculated to be 131 m³/s for Whiskey Jack, and ~237 m³/s for each of the Grey Fox North and Grey Fox South. With identical ventilation infrastructure assumed, due to the similar ventilation requirements.

Table 16-35: Whiskey Jack Full Production Ventilation Required

Equipment Type	Units	Utilization (%)	Engine Rating (kW/unit)	Vent Required (m ³ /s)
10 t LHD	2	80%	202	20
7 t LHD	2	80%	117	12
20 t Truck	2	80%	242	24
2-Boom Jumbo	2	15%	120	2
Mechanized Bolter	3	15%	119	3
Explosive Loader	1	40%	99	2
Scissor Lift	2	40%	110	6
Production Drill Rig	2	15%	80	2
Slot Drill Rig	1	15%	100	1
Forklift/Tractor	3	40%	74	6
Boom Truck	2	40%	110	6
Motor Grader	1	40%	97	2
Fuel and Lubrication Truck	1	80%	110	6
Small Personnel Carrier	3	40%	74	6
Small Carrier (Mech/Elec)	2	40%	74	4
Sub-Total Equipment Ventilation				101
Leakage / Contingency		30%		30
Total Vent Requirements		29		131



Table 16-36: Grey Fox North Full Production Ventilation Required

Equipment Type	Units	Utilization (%)	Engine Rating (kW/unit)	Vent Required (m³/s)
10 t LHD	5	80%	202	51
7 t LHD	2	80%	117	12
20 t Truck	6	80%	242	73
2-Boom Jumbo	2	15%	120	2
Mechanized Bolter	3	15%	119	3
Explosive Loader	1	40%	99	2
Scissor Lift	2	40%	110	6
Production Drill Rig	2	15%	80	2
Slot Drill Rig	2	15%	100	2
Forklift/Tractor	3	40%	74	6
Boom Truck	2	40%	110	6
Motor Grader	1	40%	97	2
Fuel and Lubrication Truck	1	80%	110	6
Small Personnel Carrier	3	40%	74	6
Small Carrier (Mech/Elec)	2	40%	74	4
Sub-Total Equipment Ventilation				181
Leakage / Contingency		30%		54
Total Vent Requirements		37		236

Table 16-37: Grey Fox South Full Production Ventilation Required

Equipment Type	Units	Utilization (%)	Engine Rating (kW/unit)	Vent Required (m³/s)
10 t LHD	5	80%	202	51
7 t LHD	2	80%	117	12
20 t Truck	6	80%	242	73
2-Boom Jumbo	2	15%	120	2
Mechanized Bolter	3	15%	119	3
Explosive Loader	1	40%	99	2
Scissor Lift	2	40%	110	6
Production Drill Rig	3	15%	80	3
Slot Drill Rig	2	15%	100	2
Forklift/Tractor	3	40%	74	6



Equipment Type	Units	Utilization (%)	Engine Rating (kW/unit)	Vent Required (m ³ /s)
Boom Truck	2	40%	110	6
Motor Grader	1	40%	97	2
Fuel and Lubrication Truck	1	80%	110	6
Small Personnel Carrier	3	40%	74	6
Small Carrier (Mech/Elec)	2	40%	74	4
Sub-Total Equipment Ventilation				182
Leakage / Contingency		30%		55
Total Vent Requirements		38		237

16.8.2 Overall Ventilation

16.8.2.1 Stage 1: Initial Development

The initial development for Whiskey Jack (Grey Fox North) and Grey Fox South from the respective portals will be supported with twin semi-rigid 0.9 m (36 ft) oval ducting from surface, until the first raise is established. Two development fans in series will be installed in each duct to support approximately 700 m of development (with the second series fan required at the midway mark). Each fan is rated at 112 kW (150 HP) and will provide 47 m³/s at the length to the first fresh air raise. To prevent freezing in the winter months, a rental 11 MMBTU/h heater will be installed at the portal, with the twin fans/ducts installed downstream of the heater. This heater will be maintained through the LOM. A schematic outlining the ventilation system for the initial ramp development at Whiskey Jack is presented in Figure 16-27.

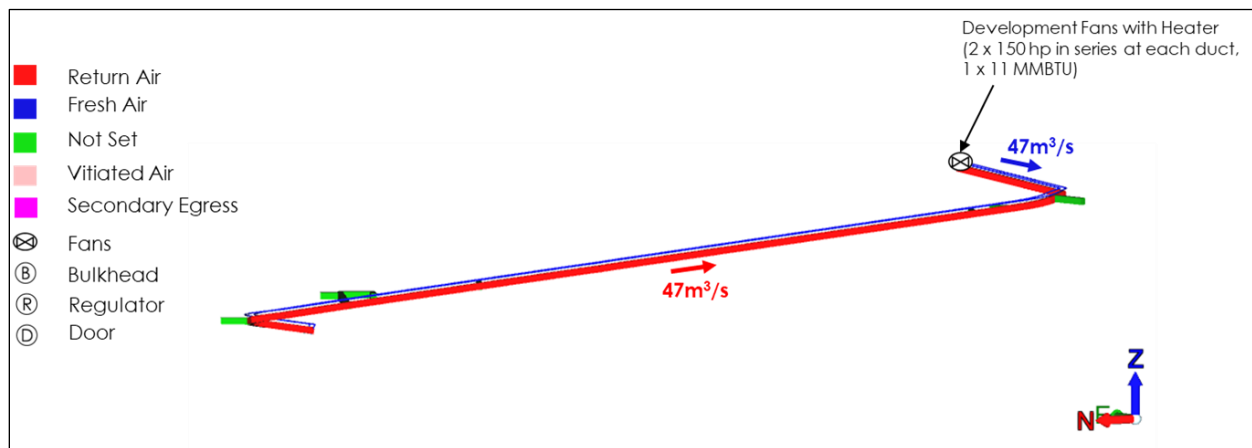


Figure 16-27: Ventilation Schematic Initial Ramp Development

Once the first fresh air ventilation raise is established at Whiskey Jack, the existing 20 MMBTU/hr heater from Froome will be relocated to the top of the raise along with 2 x 112 kW (150 HP) intake fans pushing a

maximum of 100 m³/s to support production and development activities. This will allow for flow through ventilation to be established, with fresh air through the raises and return air through the ramp. A schematic of this stage is provided in Figure 16-28.

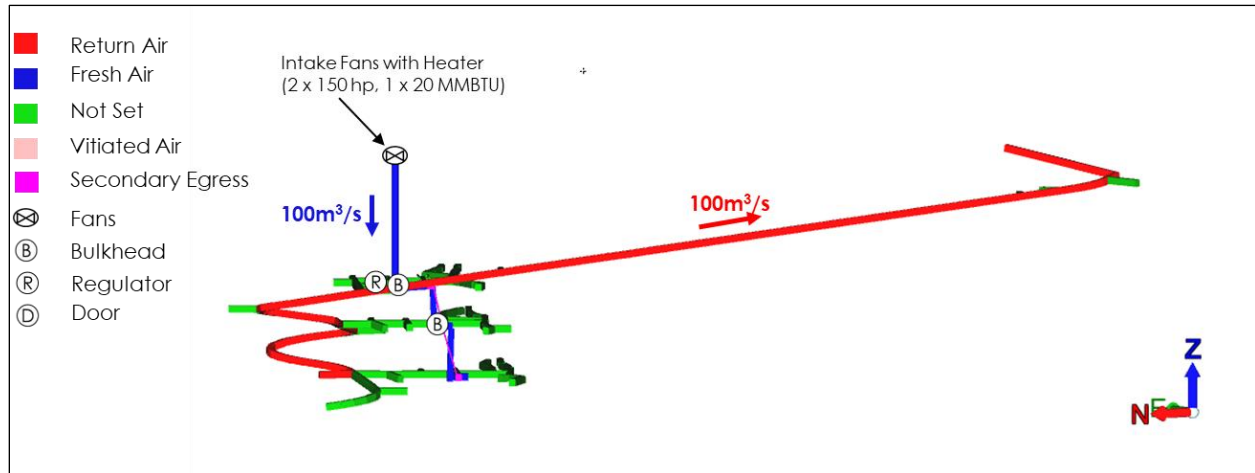


Figure 16-28: Whiskey Jack Initial Flowthrough Ventilation Established

Production development fans will be installed in the fresh air bulkheads, with the duct running to the headings providing ventilation air to support the production activities.

16.8.2.1 Stage 2: Transition from Whiskey Jack to Gibson

The full production ventilation system for Whiskey Jack features a second raise, for exhaust, to maintain velocity limits in the ramp and to aid in smoke clearance after blasting. The exhaust fan installed on surface on top of the exhaust raise will be a single 373 kW (500 HP) fan. Once the exhaust fan is commissioned, a second heater identical to the one installed in Stage 1 will be installed in parallel at the surface intake raise collar, with an additional two 112 kW fans.

Similar to the previous stage, bulkheads will be installed on each production level and production fans will be installed in the bulkheads to support the production activities. A schematic of the ventilation system is provided in Figure 16-29.

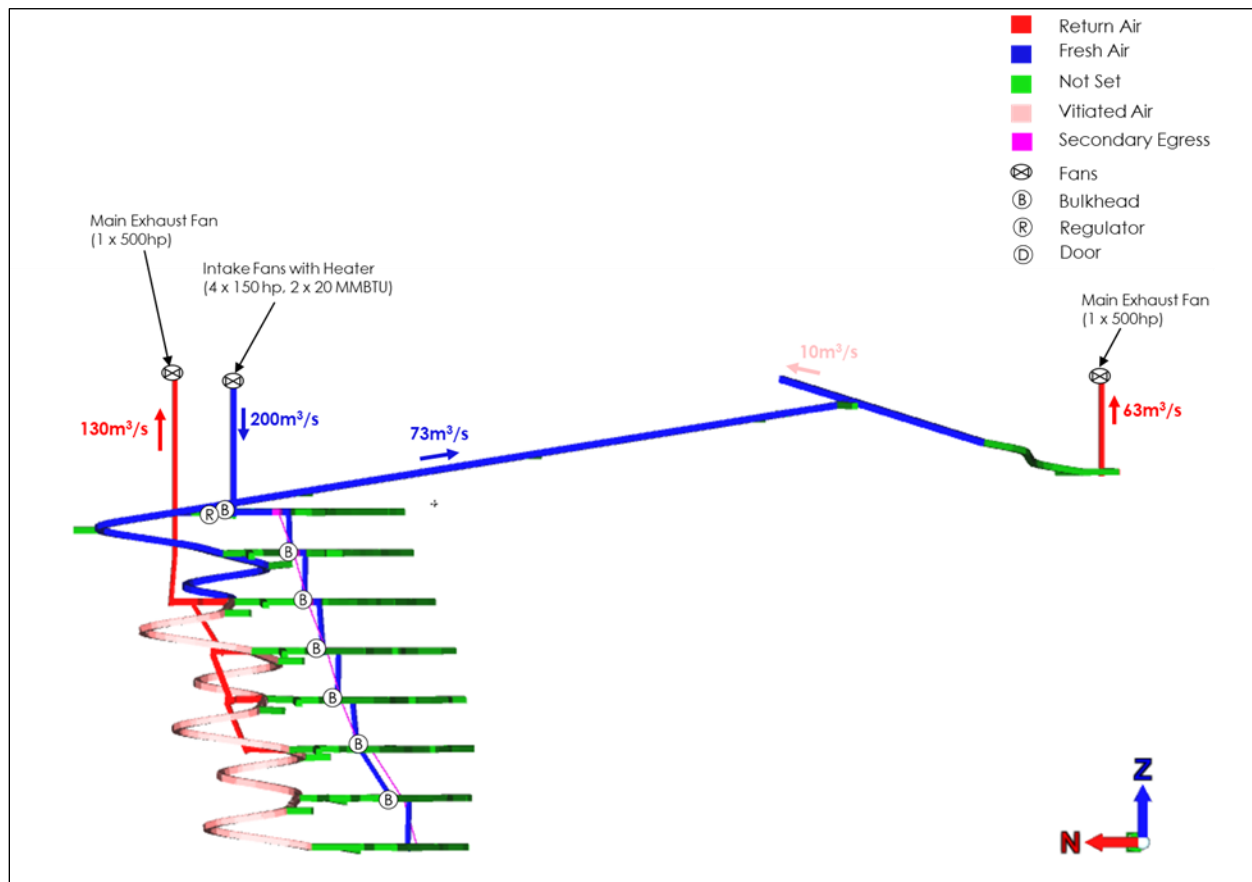


Figure 16-29: Whiskey Jack Full Production Ventilation System

This stage also indicates the exhaust raise at Gibson is established to provide flow through ventilation, with the development to the raise supported by the flowthrough ventilation in the ramp. For the exhaust raise, fans identical to the Whiskey Jack exhaust fan will be installed at Gibson, with the fan from Whiskey Jack being relocated to Gibson as production in Whiskey Jack is complete.

16.8.2.2 Stage 3: Life of Mine

Once production in Whiskey Jack is complete, the ramp to Whiskey Jack will be dedicated to transfer the fresh air to Gibson. The exhaust fan at Whiskey Jack, as previously noted will be moved at the Gibson exhaust raise, and Gibson will operate with 3 x 373 kW (500 HP) fans at its peak. A 10 MMBTU heater with a 112 kW (150 HP) intake fan will be installed at the ramp to provide additional intake ventilation with the heater air ducted to the portal entrance. Bulkheads and regulators will be used to push air to the bottom of Gibson. Production fans will be ducted from the closest flowthrough location on the level.

A similar ventilation system is planned for Grey Fox South, with two raises: one for fresh air and the other for exhaust air with initial development until the first raise is established and supported by the portal development fans/heaters. A schematic of the ventilation system for Gibson and Grey Fox South is outlined in Figure 16-30.

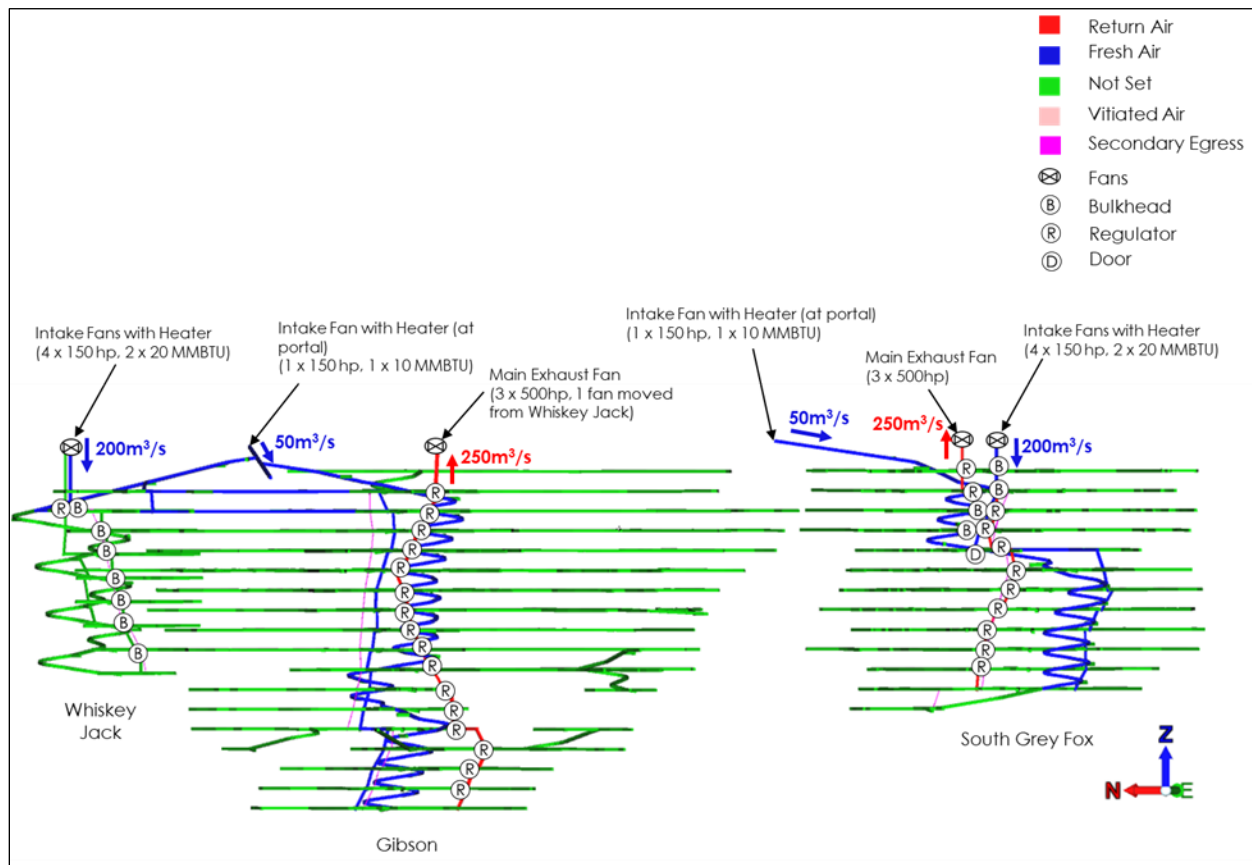


Figure 16-30: Grey Fox North and Grey Fox South at Full Production Ventilation System

16.8.3 Ventilation Infrastructure

16.8.3.1 Main Fans

The main fan motor power requirements were estimated from the ventilation model and system capabilities that are summarized in Table 16-38. These are to be verified by the vendor. The exhaust fans are to be equipped with variable frequency drives (VFDs) and will be installed as a single fan for Whiskey Jack and in a trifurcated arrangement at Gibson/Grey Fox South. The intake fans will be installed in parallel, downstream of the heaters, with ducting to the raise house which will allow for access to the secondary escapeway. All main fans are to be a four-fan arrangement and equipped with dampers to prevent re-circulation in the event of one or more fans not being in operation.

Table 16-38: Main Fan Ventilation Requirements

Location	Installation	Type	No. Fans	Airflow per Fan (m³/s)	Collar Total Pressure per Fan (Pa)	Estimated Power per Fan (kW)
Whiskey Jack	Surface	Intake	4 (at raise)	50	1,050	112
Whiskey Jack	Surface	Intake	1 (at portal)	50	1,050	112



Location	Installation	Type	No. Fans	Airflow per Fan (m ³ /s)	Collar Total Pressure per Fan (Pa)	Estimated Power per Fan (kW)
Whiskey Jack	Surface	Exhaust	1	130	685	373
Gibson	Surface	Exhaust	3*	92	2,600	373
Grey Fox South	Surface	Intake	4 (at raise)	50	1,050	112
Grey Fox South	Surface	Intake	1 (at portal)	50	1,050	112
Grey Fox South	Surface	Exhaust	3	92	2,600	373

Note: * The second of three fans will be relocated from Whiskey Jack.

16.8.3.1 Auxiliary Ventilation

The auxiliary fan systems were sized to provide sufficient airflow taking into consideration mobile equipment use and minimum velocity thresholds. Table 16-39 outlines the fans' requirements for the development and production auxiliary ventilation based on the development and production activities in the mining schedule.

Table 16-39: Auxiliary Fan Ventilation Requirements

Description	Quantity	Airflow per Fan (m ³ /s)	Fan Total Pressure (Pa)	Estimated Fan Power (kW)	Fan Size Diameter (m)
Development Fans	8	25	2,500	112	1.07 (42")
Production Fans (Short Levels)	10	19	1,900	75	0.91 (36")
Production Fans (Long Levels)	20	19	2,900	112	0.91 (36")

Short development or production headings less than 300 m long will use flexible ducting (or layflat), while longer development or production headings will use semi-rigid PVC ducting.

16.8.3.2 Ventilation Controls

Ventilation controls will be used to control the flow throughout the mine, to optimize the ventilation system performance, and reduce health and safety risks from recirculation and air short circuiting. These controls will primarily include bulkheads (including bulkheads with doors) and drop-board regulators.

Bulkheads will be used to isolate areas of the mine to prevent short-circuiting, and several types of bulkheads are planned to be utilized. Installations such as plain bulkhead with fan inserts, drop-board regulator bulkheads, and a mobile access/personnel door will be used.

For the ventilation emergency alert system, ethyl mercaptan is planned to be used as a stench gas; this will be used together with R134a as a propellant to form a gas and then introduced into the ventilation stream. The system will be located at the portal intakes and other strategic locations to ensure timely warning to all personnel in remote working areas. It will have redundancy injection and a cylinder package complete with enclosures, and the system will be sized to a capacity suitable for the air flow intake and required concentrations through the mine workings.



16.9 Ore and Waste Handling System

The ore and waste handling system will use truck haulage consisting of up to six 20-t trucks to be loaded at designated remucks at both Grey Fox North and Grey Fox South. One truck will be shared for a total of 11 units. These will be loaded with up to five 10-t LHDs and two 7-t LHDs in each of Grey Fox North and Grey Fox South. Waste will be stockpiled on surface during initial development. During production, waste will be deposited into the stopes along with the surface stockpile and supplemented with waste rock from McEwen's Black Fox site.

16.10 Mine Dewatering

The underground dewatering facilities at Grey Fox North and Grey Fox South are designed to receive water from two sources: groundwater seepage and underground operations. The planned backfill activities will not require flush water, nor will there be any decant water from backfill.

Groundwater seepage will contribute approximately 62.5 m³/h to each mine. While it is likely that most of this water will report to the upper levels of the mine, for the purposes of designing the mine dewatering infrastructure, it was assumed that this flow could report at depth. Service water from operations which includes drilling, dust control, washing equipment, etc. will produce an average of 19.6 m³/h of water. Water will be directed to the sumps located in the underground mine workings for a total combined average rate of 82.1 m³/h as presented in Table 16-40. Daily volumes will vary depending on the level of activities, mining sequence, etc.

Table 16-40: Estimate Water Source Flows

Mine Source	Average Flow (m ³ /h)
Groundwater	62.5
Service Water	20
Backfill Flush/ Decant	0
Total	82

For each mine, a target pumping rate of 136 m³/h (600 gpm, 2,270 L/min) was selected taking into consideration the average flowrates, peak flows, and a reasonable water velocity in the pipe to maintain solids in suspension during pumping.

The dewatering system will be a cascading design that includes borehole sumps, "pocket" or "hole in the wall" sumps and decline sumps. Borehole sumps will collect and drain water to the next level below, offering no storage capacity and not equipped with a pump. The hole in wall sumps will be short decline sumps with minimal storage capacity and equipped with a single 22 to 37 kW (30 to 50 HP) submersible pump which will deliver water to another sump on the same level, or to the next level above. The decline sumps will be the major pumping stations, offering some water storage capacity (nominal 35 m³ of live capacity, depending on pump selection), and equipped with operating and standby 112 kW (150 HP) submersible pumps accessed via steel walkway and monorail/trolley,



At Grey Fox North, main pump stations (decline sumps) will be located on 465, 365, 265, and 140 Levels in Gibson and 265 and 145 Levels in Whiskey Jack. Hole in the wall sumps will be located 315 and 215 Levels in Gibson, 295 Level in Whiskey Jack, and one located inside the portal to capture immediate ground water. Borehole sumps will be located on all other mining levels; some levels may have more than one borehole sump, depending on strike length and mine layout. Ultimately, all water will report through the main decline sump located on 115 Level and be pumped to the holding pond on surface.

At Grey Fox South, main pump stations will be located on 340, 240, 140, and 40 Levels. The uppermost main station (40 Level) will pump water to the portal and overland (approximately 2,000 m) to the collection pond on surface. Borehole sumps will be used on other mine levels to drain water to the next main station. There will also be a small sump located inside the portal to capture immediate groundwater.

16.11 Backfill Distribution

The stopes will be backfilled with a combination of unconsolidated rock fill plus cemented rock fill to prevent backfill dilution in subsequent stopes while also minimizing the use of cement.

Primary method will be cemented or uncemented rock fill.

1. Slurry delivery - Slurry will be mixed on surface in a hopper and distributed underground via borehole to an underground receiving station.
2. Waste Rock Mixing for cemented rock fill – The underground receiving station will be connected to a pump and spray curtain. An LHD will drive under the curtain with a load of waste rock and slurry will be sprayed over the rock. The slurry loading cutout on the levels will be developed slightly down gradient to collect any spillage.
3. Stope Placement – The LHD will tram and dump the cemented rock fill into the stope.

The backfill placement is presented in Figure 16-31.



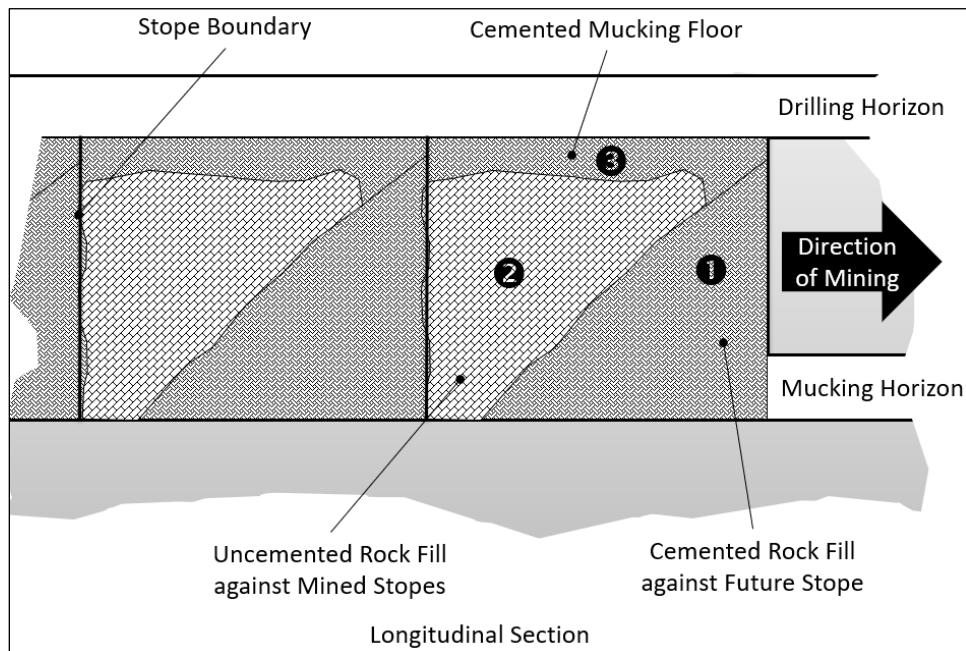


Figure 16-31: Backfill Placement

The cemented rock fill against future stope placed first, uncemented rock fill placed second, and cemented mucking floor placed last. If stopes were being mined below there will need to be complete coverage of cemented backfill on the floor. If stopes are not being mined adjacent to or above the stope it could be filled completely with uncemented rock fill. If stopes were being mined above but not adjacent a cemented mucking floor will still be required. If stopes are not planned above a cemented muck floor will not be required. The cure time used to work on top of cemented rock fill will be 7 days and to blast a stope adjacent to cemented rock fill a 21-day cure time will be required.

The amount of cemented rock fill used in a stope compared to uncemented rock fill is summarized Table 16-41. The annual fill schedule is presented in Table 16-42.

Table 16-41: Cemented Rock Fill Backfill Factors

Stope Description	Cemented Rock Backfill %
Longhole Stope with Stope Adjacent and/or Below and Above	65%
Longhole Stope with Stope Adjacent Only	55%
Longhole Stope with Stope Above Only	10%
Longhole Stope with No Stope above, below or adjacent	0%
Uphole Stope	0%
Cut and Fill Stope	100%

Table 16-42: Underground Backfill Schedule

Year	Waste Produced Underground - From Deswik Schedule	Fill Required - From Deswik Schedule	Development Waste Stays Underground as Fill	Surface Waste Stockpile Back Hauled Underground as Fill	Additional Waste Required from Black Fox as Fill (Surface Haulage)
2027	71,581	0	0	0	0
2028	238,009	45,415	45,415	0	0
2029	147,150	189,588	147,150	42,438	0
2030	110,303	297,440	110,303	187,137	0
2031	156,252	322,485	156,252	34,600	131,633
2032	101,199	304,097	101,199	0	202,898
2033	265,146	329,126	265,146	0	63,980
2034	291,467	259,530	259,530	0	0
2035	209,397	495,057	209,397	31,937	253,723
2036	270,102	569,525	270,102	0	299,423
2037	166,025	535,390	166,025	0	369,365
2038	226,084	511,162	226,084	0	285,078
2039	185,047	462,944	185,047	0	277,897
2040	18,344	490,223	18,344	0	471,879
2041	0	176,497	0	0	176,497
Total	2,456,106	4,988,479	2,159,994	296,112	2,532,373

16.12 General Underground Infrastructure

There are numerous underground facilities proposed for the Grey Fox Project. The facilities are described in this section, the total quantities of facility is presented in Table 16-43.

Table 16-43: Underground Infrastructure Matrix

Item	Qty
Underground Facilities	Wash Bay
	2
	Fuel and Lube Storage
	5
	Satellite Garage
Refuge Station	5
	Detonator Storage
	7
	Explosive Storage
	7
Refuge Station	Refuge Station/Lunchroom (permanent - 12 person)
	4
	Refuge Station (portable - 12 Person)
	8
	Latrine
	12



Item		Qty
Power	Typical Electrical Room Type 1	12
	Typical Electrical Room Type 2	27
Rockfill	Slurry Surface Hopper	3
	Slurry Distribution Station	5
	Slurry Receiving Station	28
Mine Water	Hole in Wall Sump (with Pump)	8
	Borehole Sump/ Slash	31
	Decline Sump	11
	Pressure Regulating Valve Station	21

16.12.1 Personnel and Material Movement

All personnel and materials will be transferred to and from the underground workings via a mine ramp.

During full mine production an additional means of emergency egress from the mine will be a ladder system located in the fresh air raise to surface of each mine exiting through an airlock in the fan ducting on surface. A series of dedicated internal escapeways will lead to this emergency egress through both mines. For Grey Fox North, personnel will have to travel across 65 Level in the Gibson Zone and down the Whiskey Jack Ramp to the fresh air raise.

16.12.1.1 Personnel Movement

Personnel carriers will be used to move workers between surface and the underground workplaces at the start and end of each shift. Workers that operate mobile equipment will park in a designated area (in some cases on surface) at the end of each shift.

16.12.1.2 Material Movement

Consumable materials, equipment, and maintenance parts will be delivered to designated underground storage locations via the mine ramp using boom trucks.

16.12.2 Refuge Stations

Refuge stations will be located throughout the mine in the fresh air circuit, and the vertical interval between stations will not exceed 100 m. To achieve this a refuge station will be placed every third level at a minimum. Therefore, every working level will have access to a refuge either on that level or up/down a single vertical level.

The refuge stations will serve the development, production, maintenance, and construction crews working in the area. Portable refuge stations will primarily be used and will be moved when areas of the mine cease production. Permanent Refuge Stations will accompany the underground maintenance shops except in the



Whiskey Jack Zone and serve dual purpose as lunchrooms. A latrine or portable sanitary system will be provided near each refuge station location.

16.12.2.1 Permanent Refuge Stations

Permanent refuge stations will be used primarily as an employee lunchroom but are designed for use during emergency situations located on the same levels as the underground workshops except in the Whiskey Jack Zone. Whiskey Jack will not have a permanent refuge station due to its relatively short production life.

Each refuge station will be constructed with materials that have at least a one-hour fire resistance rating, and will be equipped with the following:

- Boot and hand-washing facilities for use prior to entering the refuge station
- Racks to hang outer work gear prior to entering the refuge station
- Thirty-six hours of potable water and bottled respirable air
- Emergency food
- Lights
- First aid supplies
- Sanitation facilities
- Service water line
- Suitable firefighting equipment
- Sufficient seating for each worker
- Multiple means of communication with the surface
- A double door that creates an air lock when closed
- Means of tight sealing the door to prevent gases from entering

16.12.2.2 Portable Refuge Stations

Portable refuge stations will be strategically purchased throughout the LOM. As mine development and production advances into new mining areas, these portable refuge stations will be placed in appropriate existing cutouts. As mining ceases in areas the stations will be moved to alternate locations.

16.12.3 Explosives and Detonators Storage Facilities

Underground storage magazines for explosives, detonators, and blasting accessories will be located on the 90 Level in the Whiskey Jack Zone, 140, 265, and 390 Levels in the Gibson Zone, and 140 and 240 Levels in Grey Fox South. These will be set a safe distance from the underground infrastructure and work areas. There will be separate magazines for explosives and detonators.

Explosive products used for development and production mining will be securely handled and stored in the magazines, including ANFO, packaged emulsion explosives, and detonators.

All explosives will be stored, stacked, and labeled to facilitate a first-in / first-out inventory control system. Each magazine will be designed with a locking gate. The location of the explosive or detonator facility will be at least 25 m from the main travel way.



Explosive and detonator materials will be transported via truck from the surface via the ramp to the underground magazines. Special trucks operated by trained and authorized individuals will be used to transport explosive materials from the underground magazines to the workplace.

16.12.4 Maintenance Shop

The main workshop for the mine will be located on surface and mobile equipment will be driven up ramp for maintenance and repair. Underground workshops will be constructed on the 175 Level in the Whiskey Jack Zone, 140 and 315 Levels in the Gibson Zone, and 140 and 240 Levels in Grey Fox South.

These underground workshops will be used for minor repairs and maintenance to supplement the main workshop on surface. The underground workshops will be a single-bay facility that can service up to two pieces of mobile equipment.

16.12.5 Fuel and Lubricants

The underground mine will have a fuel and lube storage accompanying each of the underground workshops on 175 Level in the Whiskey Jack Zone, 140 and 315 Levels in the Gibson Zone, and 140 and 240 levels in Grey Fox South. A fuel and lube truck will be the primary delivery vehicle.

Fuel bladders will be transported via truck from surface via the mine ramp to the fuel bays. Lubricants will be transported in bladders and pails via truck from surface via the mine ramp to the lube storage bays.

A fire door will be installed at the entrance to the fuel and lubricant bay. This door will close in the event of a fire to prevent any potential fires and noxious gases from spreading into the mine. Fire detection and suppression equipment that interfaces with the mine's emergency alarm system will be included in the fuel and lubricant storage bays. All mobile equipment will have onboard fire suppression systems.

16.13 General Underground Services

16.13.1 Compressed Air

Permanent compressors will be located in a building on surface at each portal for Grey Fox North and Grey Fox South to provide compressed air for the mine development and operating phases. Each compressor house will have three compressors installed, with two normally operating and the third typically on standby, but occasionally able to provide additional capacity if required.

Compressed air will be used by the drills (production and jumbos), bolters, face pumps, miscellaneous hand tools, garage facilities, Alimaks, and a source of emergency air for the refuge stations. Compressed air usage underground was estimated as approximately 1.1 m³/s or 2,350 cubic feet per minute (cfm), with peak flows of approximately 1.4 m³/s (3,000 cfm).

The compressors are sized to deliver nominally 0.7 m³/s (1,500 cfm) each, with motor size of 220 to 260 kW (300 to 350 HP), final flow and related motor size will depend on the compressor manufacturer's standard offerings. An air receiver will be located at each compressor plant. At each mine, a DN150 (6" diameter)



pipe will be installed from the air receiver to the portal and in the ramp. Branches to the mine levels will be DN100 (4" diameter) pipe.

16.13.2 Service Water

Service water will be required for use in exploration, development, production drilling, dust control, washing floors and equipment, preparing shotcrete and concrete mixes, fire protection, and other miscellaneous processes. The underground water distribution system will be supplied from a surface holding pond (the same holding pond to which the underground mine dewatering pumps report).

Pumps will be installed in a pumphouse near the surface holding pond to provide service water to each underground mine. The pumps will use variable frequency drives and/or a pressure tank to maintain pressure and flow under varying conditions. Table 16-44 outlines the primary service water usage underground at each mine (common fleets, quantities, and usage profiles at each).

Table 16-44: Service Water required by Equipment Type

Equipment	Qty	Water Usage (L/min)	Utilization per Shift	Operating Hours per Shift	Total (L/Shift)	Peak (L/min)
2-Boom Jumbo	2	200	50%	7.5	90,000	400
Mechanized Bolter	3	100	15%	7.5	20,250	300
Production Drill Rig	3	49	80%	7.5	53,352	148
Slot Drill Rig	1	35	80%	4.0	6,739	35
Dust Control	4	30	5%	2.0	720	120
Handheld Drill	4	28	25%	6.0	10,080	112
Diamond Drill	1	130	50%	8.5	33,150	130
Subtotal					214,291	1,245
Allowance – leaks, misc.	10%				21,429	125
Average (L/min)					327	
Peak (L/min)						1,370

At each mine, the average flow is estimated to be 327 L/min, with peaks to 1,370 L/min. The distribution pipe will be sized to deliver peak flows. A DN150 (6 in diameter) pipe will be installed in each main ramp with smaller branches on each level. Pressure reducing stations will be installed every second level to maintain the pressure suitable for use.

16.13.3 Potable Water

Potable water will be delivered underground via large plastic bottles. Potable water will be provided in underground workshops, permanent refuge stations, and dedicated bottle filling stations.



Personnel must fill appropriate water containers on surface or at a designated bottle filling station and carry a water supply to the work areas.

16.13.4 Fire Water

Underground fire-related services are required to meet National Fire Protection Association requirements including fire water, fire detection and suppression systems, and dedicated reservoirs.

Fire water must be available for underground fire suppression, hose reels, and sprinkler systems.

Fire detection and suppression systems will interface with the emergency alarm system and will be included in areas at high risk of fires. These areas include the underground workshops. Individual operating systems, such self-contained fuel and lube units and hydraulic power units, will be equipped with integral fire-suppression and will not require fire water.

Dedicated reservoirs will be excavated a level above the underground workshops, approximately 12 m long to contain sufficient volumes of water to provide sufficient water for the fire suppression requirements.

16.14 Underground Electrical and Power Distribution

The power distribution system for Grey Fox North will be four feeders from the surface substation to the Grey Fox North portal and down the ramp. Power will be distributed from the ramp electrical room via the ramp to the levels. Power will be distributed throughout the mines via ramp and boreholes, maximizing the use of boreholes wherever possible to reduce the cable lengths and voltage drop. Power feeders will also alternate powering levels in Gibson to provide redundancy.

The power distribution system for Grey Fox South will be four feeders from the surface substation to the Grey Fox South portal and down the ramp. Power will be distributed from the 40 Level electrical room via ramp and boreholes, maximizing the use of boreholes wherever possible to reduce the cable lengths and voltage drop. Power feeders will also alternate powering levels in the to provide redundancy.

16.14.1 Underground Electrical Distribution

Power for the Grey Fox North mine will be provided via four 5 kV feeders. Feeders will land in the ramp electrical room. On the secondary side of the switchgear, one feeder will provide power to Whiskey Jack and four feeders will provide power to the remaining Grey Fox South areas to support mining activities, as well as development, underground infrastructure, dewatering and ventilation.

Power for the Grey Fox South mine will be provided via four 5 kV feeders. Feeders will land in the 40 Level electrical room, then distribute to support mining activities, as well as development, underground infrastructure, dewatering and ventilation.

The main ventilation fans for both mines are powered on the surface from separate feeders from the surface substation.



The power distribution system is designed to have a main (Type 1) electrical room every three to four levels to provide power to three to four smaller (Type 2) electrical rooms. From the electrical room on each level, 5 kV power will be fed to development and production equipment, while a 600 V power distribution panel or splitter will feed power to infrastructure facilities (ventilation, dewatering, shops, etc.).

Boreholes will be used to route both 5 kV and 600 V power between levels to minimize voltage drop and to protect cabling.

Figure 16-32 presents the underground power distribution system and the locations of the types of electrical rooms.



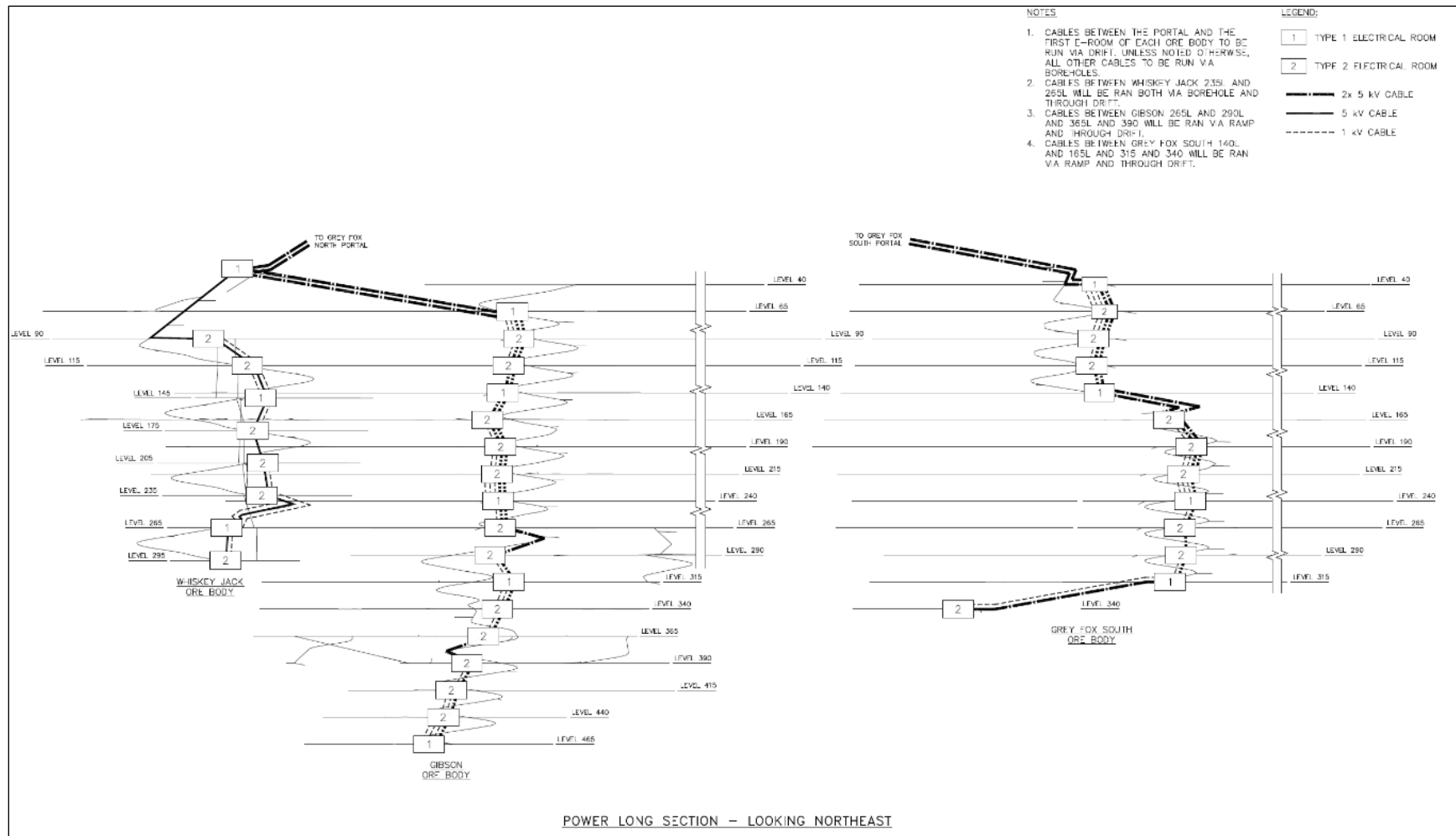


Figure 16-32: Underground Power Distribution Routing



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Figure 16-33: Type 1 Electrical Room Layout

Figure 16-34 presents the layout of a typical Type 2 electrical room. Type 2 electrical rooms will consist of a 600 V power distribution panel or splitters, lighting transformer and panel, remote I/O cabinet (as required), network cabinet and medium and/or low voltage junction boxes (as required). The Type 2 electrical rooms are designed to have the space for an underground substation (a mine power centre) for mine development purposes.

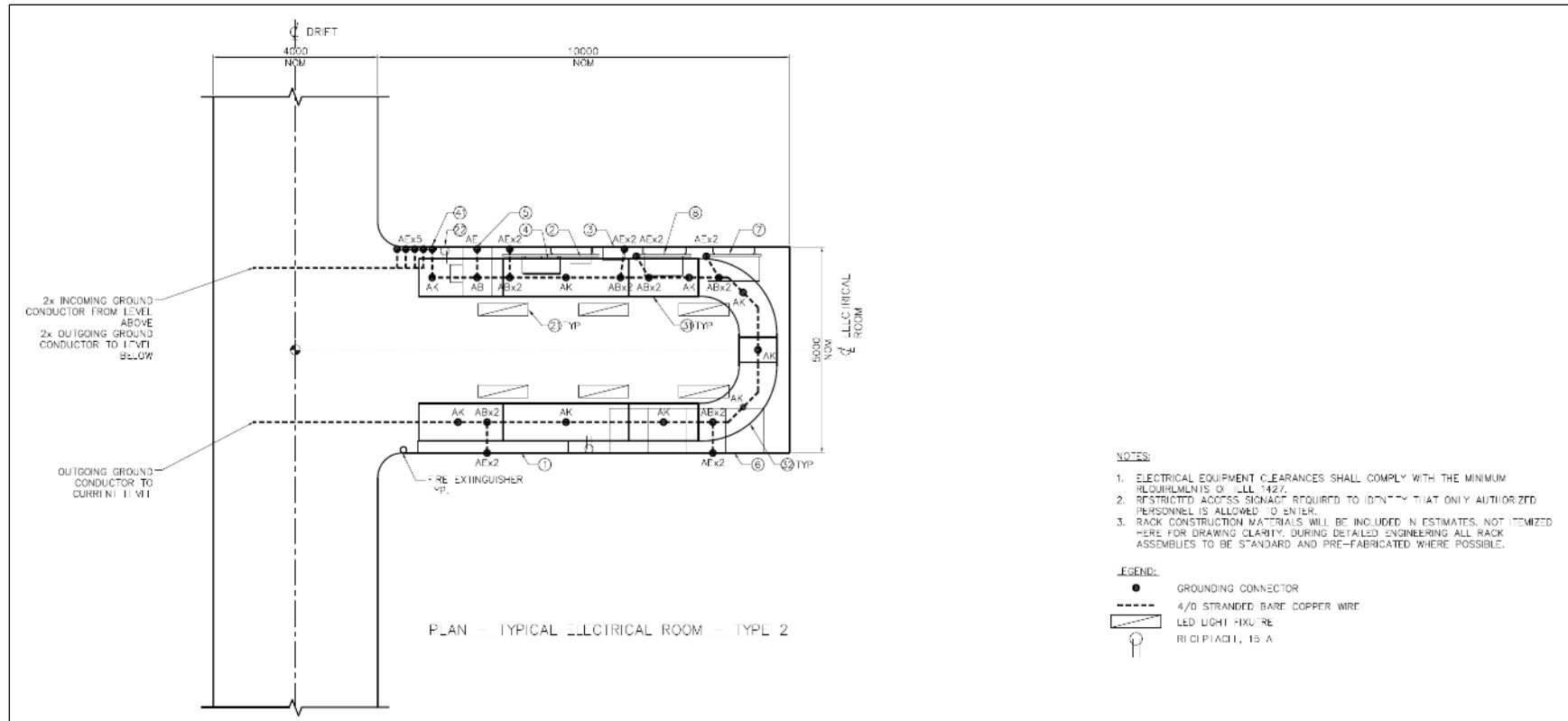


Figure 16-34: Type 2 Electrical Room Layout



16.14.2 Load List and Schedule for Mining Facilities

Table 16-45 provides a load summary by area for underground and surface mining facilities, while Table 16-46 and Table 16-47 provide power schedules broken down by quarter and by year, respectively.

Table 16-45: Grey Fox Mining Load Summary (for peak year – 2038)

Area	Demand Load (kVA)	% of Total
Surface Infrastructure	1,091	5%
Surface Dewatering	590	3%
Surface Ventilation	4,295	21%
Surface Compressors	1,746	9%
Underground Mining Equipment	926	5%
Underground Infrastructure	3,921	19%
Underground Dewatering	2,362	11%
Underground Ventilation	5,658	27%
Total	20,589	

Table 16-46: Power Schedule by Quarter

	2027				2028			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Load (kVA)	0	0	2,118	2,825	5,008	7,496	8,029	8,446

Table 16-47: Power Schedule by Year

	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Load (kVA)	9,751	9,891	10,689	10,916	15,489	18,124	19,645	19,975	20,117	20,589	20,135	18,958	17,670

16.15 Underground Communication and IT

The underground communication between levels will be routed using a combination of ramp, drift and/or boreholes (for fiber optic and analog telephone systems) to minimize cable lengths or routed in the ramps (Leaky Feeder system) to ensure coverage to every underground area. Communication cables installed in boreholes will be kept separate from power cables.

The following systems are planned as part of the underground communication and IT package:

- Fibre optic network
- Underground wireless network (5G)
- Leaky feeder system (radio)
- Analog telephone



16.15.1 Fibre Optic Network

The underground fiber optic network will be used for transmitting information for business networks, process control networks, and other dedicated systems (as required). The design for the fiber optic network will be a star topology, with the backbone connecting all underground Type 1 network cabinets, located in each electrical room.

16.15.2 Underground Wireless Network (5G)

The underground wireless network will be 5G using dedicated fibers from the fiber optic backbone to connect hardware throughout the mine. The 5G network will be the primary communication system underground to be used with telephones, tablets, and other peripheral devices. The 5G network will also be used for data collection (e.g., mobile equipment telemetry, equipment/personnel tracking), and tele-remote operation.

16.15.3 Leaky Feeder System (Radio Communication)

The leaky feeder system will be the primary method of communication in the ramp during development activities until the fiber optic backbone and 5G system is installed. The system's head end will be in the surface substation, and it will be connected to the local network cabinet.

Underground, the leaky feeder system will be routed along the main ramp and will split into the main level access drift of each level.

16.15.4 Analog Telephone

Analog telephones will be used for underground communications and will be in each electrical room and refuge station.



17 Recovery Methods

The Project involves the processing of Grey Fox underground material through the existing Stock Mill. Ore is sourced from underground mining operations only, crushed to -120 mm at the Grey Fox portals using a new modular jaw crushing plant, and then trucked to stockpiles at the Stock Mill. The crushing plant at Grey Fox and the trucking of crushed rock to the Stock Mill operates for 8 hours daily during daylight hours only.

Primary crushed material is loaded from stockpiles at the Stock Mill through the existing load-in to the two-stage crushing circuit. The -12 mm product from the crushing circuit is stored in the fine ore bin and is nominally 80% passing approximately 8 mm. Finely crushed ore is reclaimed from the fine ore bin and fed to the existing two-stage grinding circuit, which comprises a primary ball mill followed by two parallel ball mills. Ore is ground to a P_{80} of 75 μm and then pumped to the existing thickener for densification ahead of treatment in the existing leach- carbon-in-leach (CIL) circuit.

The existing 3-t elution circuit and smelting operation are used to produce doré bars. Tailings are deposited in the existing Stock Mill tailings management facility.

Metallurgical testwork demonstrated variable ore characteristics, with whole-ore cyanidation recoveries ranging from 70% to 95%, with an average of approximately 88%. Mineralogical analysis confirmed that gold is primarily associated with pyrite.

Testwork also indicated that flotation followed by fine grinding of the flotation concentrate could improve overall recoveries for certain portions of the orebody. However, retrofitting flotation and regrinding into the existing Stock Mill flowsheet would be costly and operationally disruptive and was therefore not considered economical. As a result, the existing whole-ore cyanidation flowsheet was adopted unchanged for the Project.

Comminution testing indicated a hard and competent ore requiring significant grinding energy. Consequently, the installed grinding power available at the Stock Mill represents the primary processing bottleneck and determines the overall achievable throughput.

The process flowsheet incorporates the following main unit operations:

- Primary crushing (at Grey Fox)
- Transportation to Stock Mill
- Secondary and tertiary crushing
- Two-stage grinding and classification
- Densification
- Cyanidation
- Recovery of gold onto activated carbon
- Elution, electrowinning and smelting to produce doré

Detailed descriptions of each unit operation are provided in Section 17.4.



17.1 Process Flowsheet

The Grey Fox process flowsheet was developed based on metallurgical testwork results and industry best practice guidelines. The proposed flowsheet, as shown in Figure 17-1 below, incorporates crushing, grinding, and cyanidation stages tailored to the ore characteristics determined during testwork.

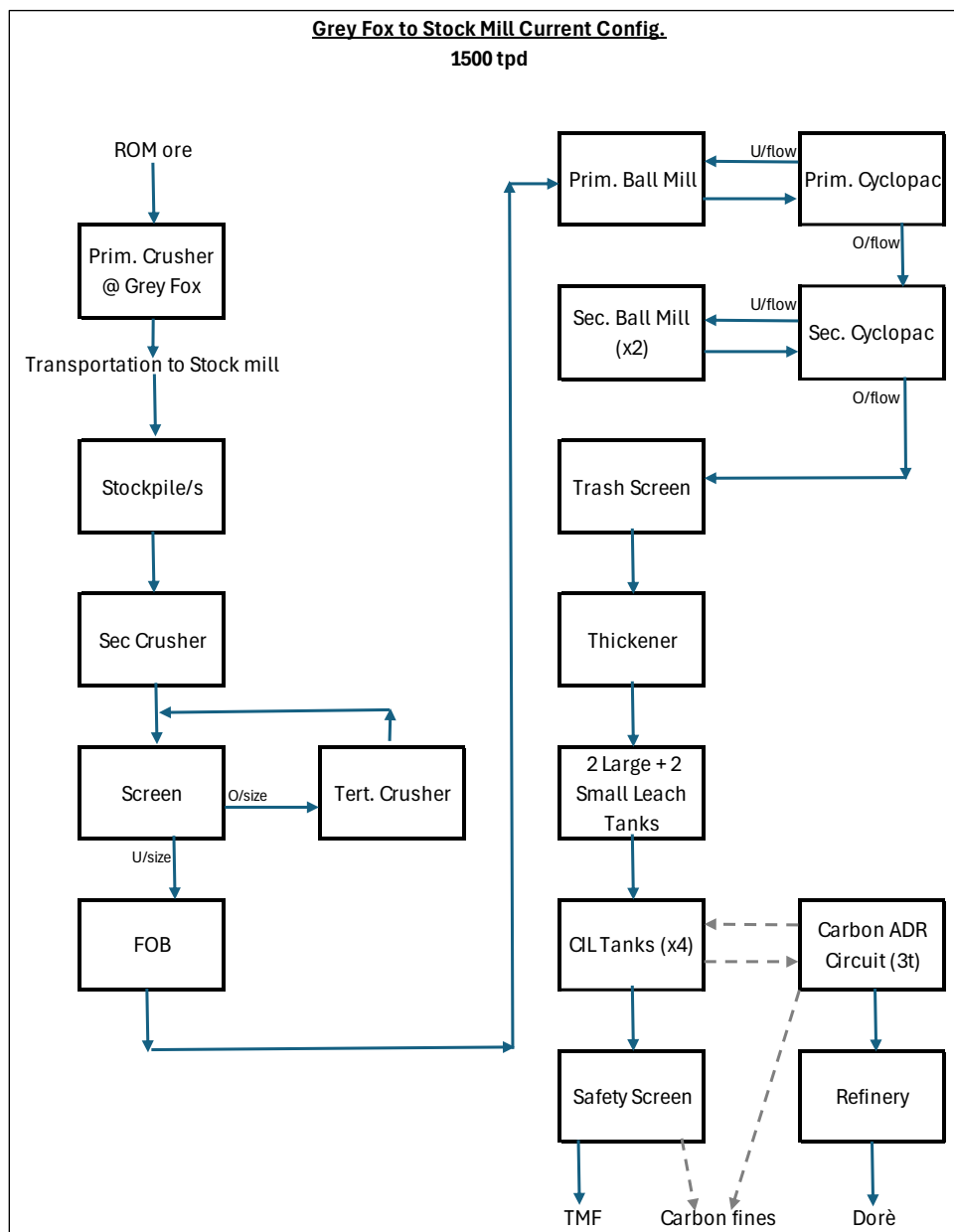


Figure 17-1: Block Flow Diagram for the Treatment of Grey Fox Ore

17.2 Process Design Criteria

Table 17-1 below summarises the key design criteria for the treatment of Grey Fox ore from underground mining operations through the existing Stock Mill. Note that the “Nominal” and “Design” cases refer to the average (50th percentile) and design (85th percentile) ore hardnesses, respectively. The expected average throughput for each case reflects the maximum throughput at which the installed grinding equipment can deliver the targeted grind size of 80% passing 75 µm.

Table 17-1: Process Design Criteria

Description	Units	Nominal	Design
General:			
Expected throughput	t/d	1,470	1,123
ROM Feed Grade	g Au/t	3.42	3.42
Gold Extraction (Overall)	%	88	88
ROM Feed Specific Gravity		2.91	2.91
JKTech Axb Parameter (50th and 85th Percentiles)		33	30
Bond Ball Mill Work Index (50th and 85th Percentiles)	kWh/t	23.2	27.2
Primary Crushing:			
Primary Crushing Section Utilization	%	33.3	33.3
ROM Feed Grizzly Square Aperture	mm	350	350
Primary Crushed Product P ₈₀	mm	100	100
Moisture	%	3.0	3.0
Throughput Rate	tph	183.8	140.4
Secondary and Tertiary Crushing:			
Sec and Tertiary Crushing Section Utilization	%	70	70
Tertiary Crushing Circuit Product P ₈₀	mm	7.3	7.3
Throughput Rate	tph	87.5	66.8
Grinding:			
Grinding Section Utilization	%	92	92
Throughput Rate	tph	66.6	50.9
Grinding Section Product P ₈₀	µm	75	75
Total power available (3 motors' nameplate)	kW	2,034	2,034
Total pinion power available (after 8% losses)	kW	1,871	1,871
Total pinion power required	kW	1,871	1,871
Leaching and Adsorption:			
NaCN consumption (Testwork)	kg/t	0.42	0.42
Hydrated Lime Consumption (Testwork)	kg/t	2.35	2.35



Description	Units	Nominal	Design
Number of large Leach tanks	#	2	2
Number of CIL tanks used for leaching	#	2	2
Number of CIL tanks used as CIL tanks	#	4	4
Slurry Residence Time Required	h	34	34
Slurry Residence Time Provided	h	45	57
Targeted Solution Tails Concentration	mg/L	0.01	0.01
Carbon Concentration	g/L	10	10
Loaded Carbon Grade	g/t	3,700	3,700
Eluted Carbon Grade	g/t	100	100
Cyanide Detoxification:			
Total Residence Time Required	h	2.5	2.5
SO ₂ : WAD requirement	kg/kg	4.6	4.6
Cu: WAD requirement	kg/kg	0.1	0.1
CIL terminal WAD CN	ppm	100	100
Reclaim Return Solution WAD CN conc.	ppm	50	50
Target WAD CN concentration	ppm	0.1	0.1
Elution and Regeneration:			
Available carbon batch size	t	3	3
Expected number of strips (calculated)	per week	2.5	1.9
Max Strips per Day	#	2	2
Carbon regeneration kiln energy	kWh/kg	2	2
Elution heater benchmarked energy cons.	kW/t carbon	230	230
Carbon regeneration kiln run time	h/d	20	20

Note: WAD CN = Weak Acid Dissociable cyanide

17.3 Process Description

17.3.1 Crushing

The proposed Grey Fox underground material will be processed through the existing Stock Mill facility at an average rate of 536,550 dry tonnes per year (1,470 t/d), with a ROM ore head grade of 3.42 g/t Au. When treating ore blocks of above-average hardness, the average throughput is expected to decrease to as low as 1,123 t/d at the 85th percentile of hardness.

The primary crushing section at the Grey Fox site is staffed for 12 h/d but operates for 8 h/d. This operating schedule avoids nighttime transportation of ore and allows 4 h/d for delays and maintenance. Ore is fed to a modular primary crushing plant consisting of a ROM feed bin, a vibrating grizzly feeder, a fixed rockbreaker, a C106 jaw crusher, and a crushed product conveyor. Crushed rock is loaded by front-end loader onto road



trucks and transported to the Stock Mill, where it is dumped onto stockpiles. This primary crushing front end consists of new equipment that has been specifically purchased for the Project. All subsequent processing is conducted through the existing facilities at the Stock Mill, as described below.

Primary crushed ore is reclaimed from stockpiles by front-end loader and fed into the load-in bin. The material is conveyed via a dedicated conveyor to an open-circuit Sandvik CH430 secondary cone crusher. Secondary crushed product (-40 mm) is conveyed by two conveyors to a double-deck banana screen measuring 6.1 m in length by 2.05 m in width. Screen oversize (+10 mm) reports to a Metso Nordberg HP300 tertiary cone crusher. Screen undersize (-10 mm) is conveyed to the fine ore bin. Tertiary crushed product is recycled via the screen feed conveyor back to the banana screen. The fine ore bin has a capacity of 1,360 tonnes, providing approximately one day of surge capacity between the three-stage crushing front end and the downstream grinding circuit.

17.3.2 Grinding

Two reclaim belt feeders extract finely crushed ore from the fine ore bin and discharge onto the primary ball mill feed conveyor.

The primary ball mill has a diameter of 3.66 m and an effective grinding length (EGL) of 5.49 m and is driven by a 1,250 kW motor. The mill discharges into a dedicated cyclone feed pumpbox, from which slurry is pumped to a dedicated 4-way cyclopack fitted with 380 mm diameter hydrocyclones. Cyclone underflow returns to the primary ball mill feed chute, while cyclone overflow reports to the secondary grinding cyclone feed pumpbox.

The primary ball mill circuit product is subjected to further grinding in two parallel ball mills, designated as Secondary Ball Mill #1 and Secondary Ball Mill #2. The two secondary ball mills are similarly sized but not identical. Secondary Ball Mill #1 has a diameter of 2.89 m and an EGL of 3.66 m, while Secondary Ball Mill #2 has a diameter of 2.74 m and an EGL of 3.48 m. These mills are driven by 448 kW and 336 kW motors, respectively.

The secondary ball mills operate in closed circuit with a common secondary grinding cyclone cluster, which ensures that coarse particles are returned to the mills, while fine slurry proceeds to downstream processing. The cyclone cluster consists of six ports, with three 250 mm diameter cyclones in operation, two installed standbys, and one blanked port.

Cyclone overflow is directed to a trash screen to remove wood chips, plastics, and similar foreign objects. Screen underflow flows by gravity to the thickener feed pumpbox, from which it is pumped to the 19.8 m diameter high-rate thickener. Cyclone underflow is split into two streams that recycle back to the feed chutes of the two secondary ball mills.

17.3.3 Sedimentation, Leaching and Carbon-In-Leach

Flocculant is added at a rate of approximately 15 g/t solids to promote settling and clarification in the thickener. Thickener underflow, at a density of approximately 50% w/w solids, is pumped to the first of two leach tanks, while thickener overflow flows by gravity to the process water storage tank for re-use.



Sodium cyanide and lime are added to extract gold from the finely ground ore. Most of the sodium cyanide and lime, approximately 0.42 kg/t NaCN and 2.35 kg/t lime, are added into the first leach tank. The two large leach tanks have a live capacity of 1,166 m³ each, providing approximately 27 h of leach residence time at the nominal throughput rate. Additional lime and cyanide are added downstream to maintain a minimum level of free cyanide concentration and to ensure the slurry pH remains above a minimum of 10.7. Compressed air is bubbled through all leach and CIL tanks to provide oxygen for the oxidation reactions occurring during cyanidation.

Slurry from the second leach tank flows by gravity to the six CIL tanks (277 m³ live capacity each). The first two tanks operate as leach tanks only, with no carbon addition. The remaining four tanks are equipped with carbon advance pumps and carbon retention screens, allowing them to operate as CIL tanks.

The combined residence time for the two leach tanks and six CIL tanks is 45 h, which exceeds the required residence time of 34 h by a significant margin.

Activated carbon is added to the four CIL tanks to adsorb dissolved gold to a target solution concentration of approximately 0.01 mg/L. A minimum carbon concentration of 10 g/L is maintained in the four CIL tanks to limit the amount of pumping required to move 1.24 t/d of carbon. At this advancement rate, the loaded carbon has a grade of approximately 3,700 g/t Au.

A carbon safety screen is installed as a final safeguard against gold loss. This vibrating screen intercepts any gold-bearing carbon that may pass through the carbon retention screen or broken carbon particles, preventing loss of gold to the tailings management facility.

17.3.4 Acid-Washing, Elution and Regeneration

Loaded carbon, extracted from the first CIL tank, is stripped using the pressure Zadra process in 3-t batches. With the 3-t circuit and an expected gold loading of 3,700 g/t, approximately 2.5 strips per week would be required under nominal operating conditions. This is again well below the typical industry practice, which generally ranges from 5 to 6 strips per week.

Following washing on the loaded carbon screen, the washed carbon flows by gravity from the screen to the acid wash column, where it is contacted with dilute nitric acid at a concentration of 3% w/w. The purpose of the acid wash step is to remove any calcite buildup from the carbon in preparation for the subsequent elution step. The acid wash cycle employs sequential soak and rinse steps, using 0.60 bed volumes of dilute acid followed by rinsing with two bed volumes of clean water. The total acid wash cycle duration is typically 10 h.

Following acid washing and neutralization, the carbon is transferred to the elution column. The elution cycle lasts approximately 15 h and includes carbon transfer from the acid wash, solution preheating, elution, carbon cooling, and transfer from the elution column to regeneration. The resultant loaded solution (eluate) is processed by electrowinning to produce a gold sludge for smelting.

Stripped carbon from the elution process is first dewatered on a screen and then flows by gravity into a storage bin, from which it is fed at a controlled rate into the regeneration kiln.



Carbon regeneration is performed in a horizontal, electrically heated kiln with a minimum power rating of 143 kW and a capacity of 100 kg/h. The kiln operates at temperatures ranging from 650°C to 700°C. Reactivated carbon discharges into a quench tank, with the discharge pipe terminating below the water surface to prevent combustion of the carbon. From the quench tank, regenerated carbon is pumped to a horizontal vibrating sizing screen for removal of carbon fines before being returned to the adsorption circuit.

17.3.5 Smelting

The pregnant solution (gold-rich eluate) is collected in a dedicated pregnant solution tank and pumped to two operating electrowinning cells. Each electrowinning cell contains 30 stainless steel cathodes onto which gold is plated. Once loaded, the metals sludge is dislodged from the cathodes using a high-pressure washer and then dewatered using a small pressure filter. The filtered sludge is dried in an oven and smelted in a furnace to produce doré bars.

On average, approximately one smelt is required per week. The furnace has significant spare capacity to accommodate short-term grade peaks.

17.3.6 Tailings Detoxification and Disposal

CIP tailings are directed through the carbon safety screen to a tailings pumpbox, from which they are pumped to the tailings management facility. A dam lift is planned for the 2026-2027 period to expand tailings management facility capacity as required for continued deposition. Supernatant solution is reclaimed and returned to the Stock Mill to replenish the process water system. Prior to reuse or discharge, the reclaimed solution is treated in dedicated detoxification tanks to destroy residual CN_{WAD} .

Chemical oxidation using the SO_2 /air process is employed to convert cyanide into environmentally safe compounds. Sodium metabisulphite is used as the sulphur dioxide source, while hydrated lime is added to neutralize acid generated during the detoxification reactions. Air is bubbled through the tanks to supply oxygen for the oxidation process.

17.3.7 Reagent Systems

Reagents used in the process plant include hydrated lime, sodium cyanide (NaCN), activated carbon, caustic soda (NaOH), nitric acid (HNO_3), flocculant, smelting fluxes (borax, silica sand, and nitre), antiscalant, copper sulphate ($CuSO_4$), and sulphur dioxide (SO_2).

Reagent mixing is carried out in a dedicated area of the plant specifically designed with appropriate safety features, including air extraction systems and bunded containment equipped with dedicated sump pumps. Reagents are typically prepared in batches. Once mixed, each batch is transferred from the mixing tank to a storage tank, from which it is delivered to end-use points at controlled rates. Dosing is generally achieved using positive displacement pumps to ensure accuracy and consistency.

One of the decommissioned CIP tanks is used for hydrated lime mixing.



17.3.8 Water Services

Reclaimed water is recovered from the tailings management facility supernatant pond and is detoxified prior to being returned to replenish the Stock Mill process water inventory. Due to the hardness of the Grey Fox ore, throughput is reduced and all water services are therefore operated at reduced capacity.

17.3.9 Air Services

Existing compressors are used to supply compressed air for leaching and detoxification aeration, as well as for operation of filter presses and all pneumatic equipment.

17.4 Conclusions

Due to the hardness of the Grey Fox ore, the Stock Mill will continue to operate at a reduced throughput relative to its current capacity. This provides the benefit of longer cyanidation residence time, which has been shown to be required for some of the samples tested.



18 Project Infrastructure

The Grey Fox infrastructure (Figure 18-1) will include ore and waste stockpiles, primary crushing facility, water collection ponds, two underground portals (North and South Portals) with associated ventilation plants, and supporting buildings including administration and dry facilities, gatehouse, compressor facilities, and maintenance shop.

Site access is provided via Tamarack Road, a public road connected to Ontario Highway 101 near the town of Matheson. Ore produced at the site will be trucked via Highway 101 to the Stock Mill, located approximately 34 km west. Electrical power will be supplied through a tie-in to the existing grid infrastructure along Tamarack Road.

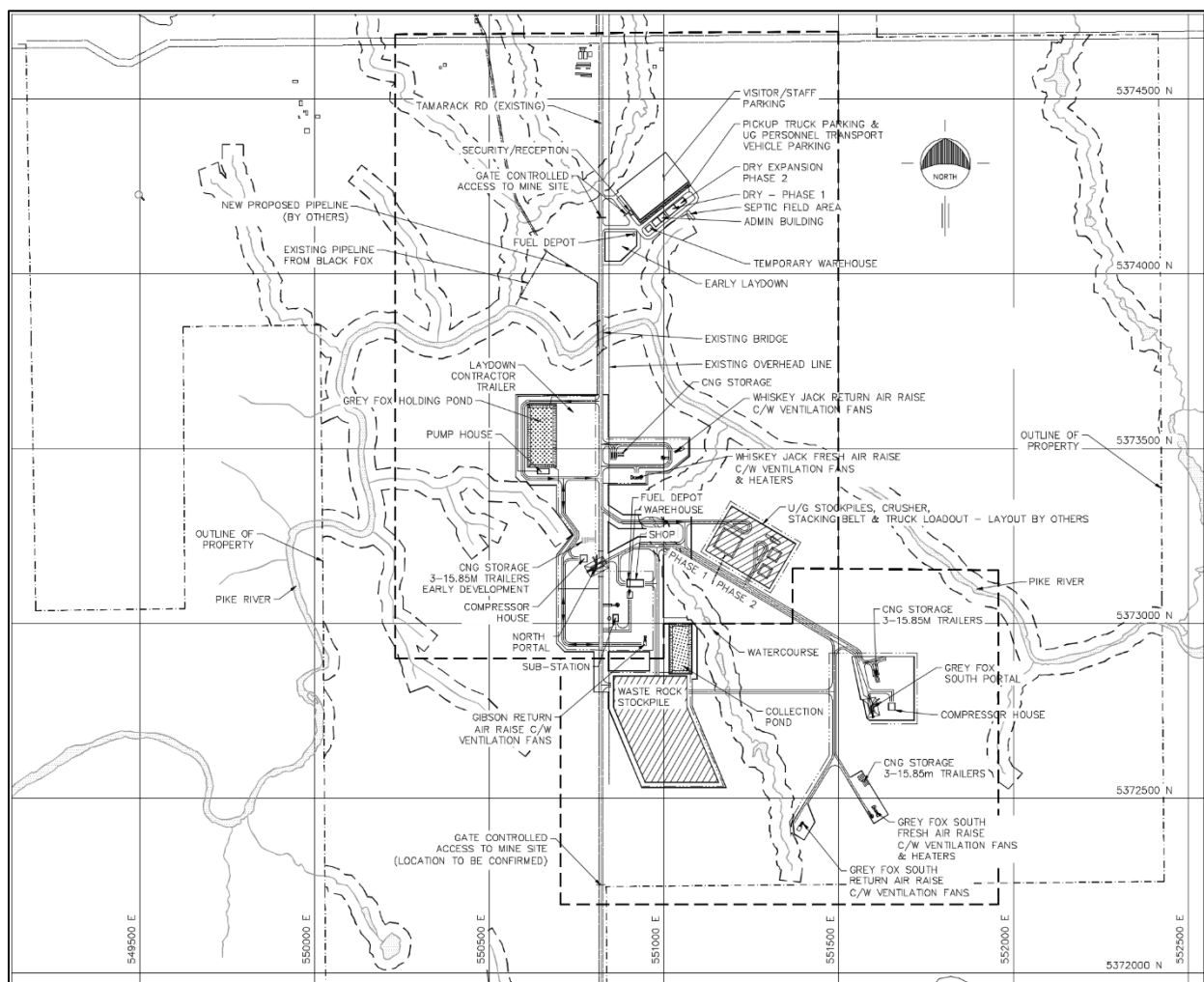


Figure 18-1: Grey Fox Site Plan



18.1 Existing Infrastructure

The Grey Fox Project is located approximately 3 km south of the Black Fox site, which includes the currently operating Froome underground mine. As a result, the Project can leverage selected existing infrastructure at the Black Fox site to support early development and construction activities at Grey Fox (Figure 18-2).

Facilities that can be used to support early works include:

- 27.6 kV Grid Electricity
- Security office and parking
- Office and dry facilities
- Surface maintenance shop
- Fuel storage tanks
- Water collection pond
- Clean waste rock stockpiles
- Available capacity within existing overburden stockpiles



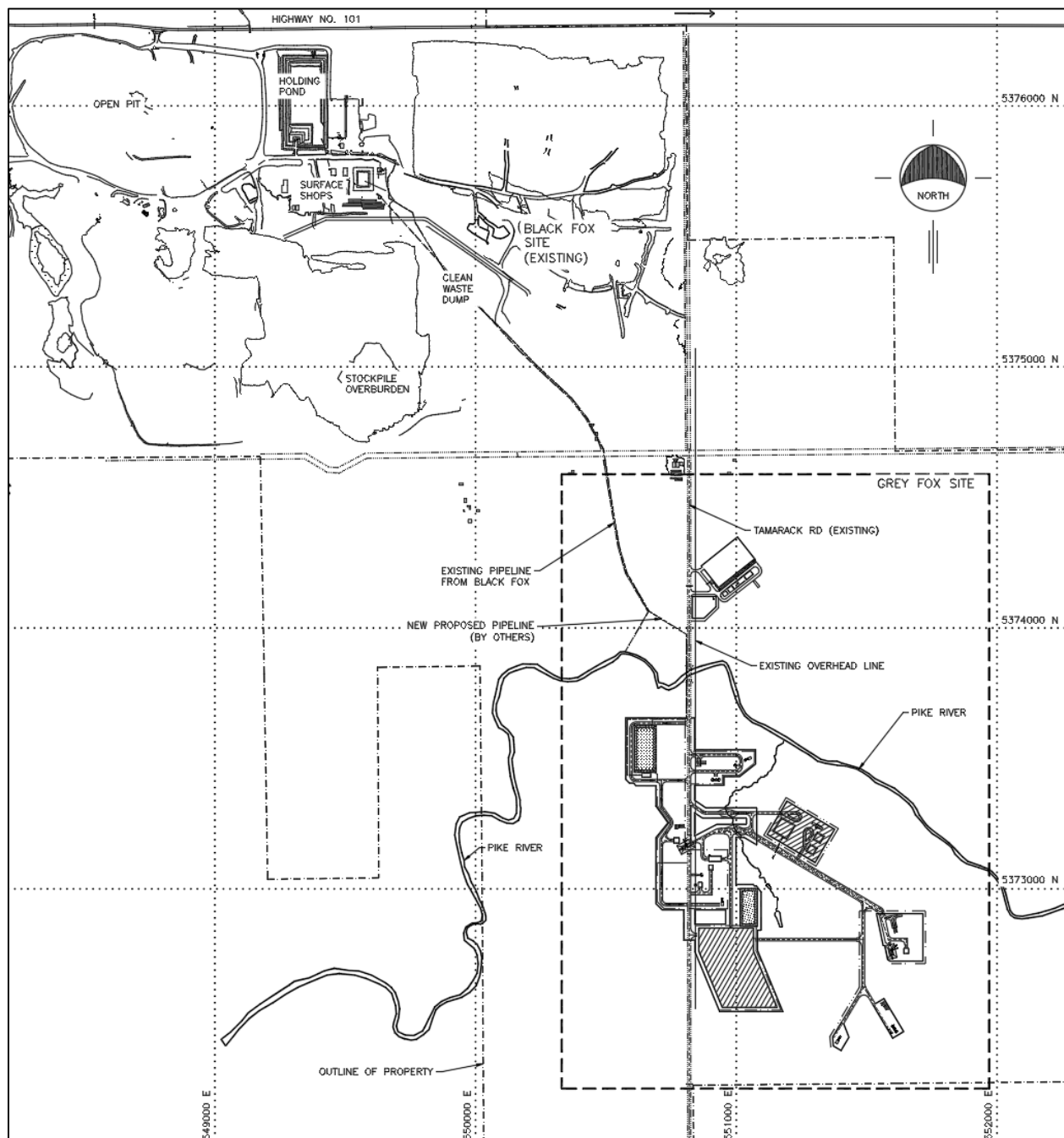


Figure 18-2: Grey Fox and Existing Black Fox Site Plan

18.2 Project Site Preparation

The Grey Fox site layout is presented in Figure 18-1. The site is designed to be compact and centralized around the two mine portals. Prior to construction, a significant portion of the site, particularly around the Grey Fox North Portal and areas north of the river designated for the office and dry area, will be cleared and grubbed. Soil and organics will be stockpiled for future surface reclamation. The remainder of the site will be cleared prior to the development of the Grey Fox South Portal and related surface facilities.

Contractor and mine laydown areas will be established and utilized for site establishment activities. The existing maintenance shop at the Black Fox site will be used for early equipment commissioning and maintenance until the Grey Fox surface maintenance shop is constructed.

Clearing and grubbing activities will include disposal of brush, roots, and stumps. Cutting and disposal of trees have already been completed. The area surrounding the Grey Fox North Portal will be roughly graded prior to construction.

18.2.1 Soil Storage

Topsoil removed from the site will be stored in the designated soil storage area for reclamation at Black Fox site. The actual volume of stripped topsoil stored for reclamation will be determined with a complete site investigation and geotechnical assessment.

18.3 Project Access

18.3.1 Mine Road

The existing gravel road, Tamarack Road, provides access south of Highway 101 and will be used for transportation of mine personnel and general supplies, as well as for hauling ore to the Stock Mill (Figure 18-3). The road is in good condition and will require minimal maintenance. An existing bridge crossing the Pike River is located approximately 2.5 km south of Highway 101 and will require a field inspection prior to commencement of underground construction to confirm suitability for its intended use without restriction. McEwen Inc. has begun discussions with the Township of Black River-Matheson regarding the care and custody of this infrastructure.





Source: (Google Earth, 2026; Site Visit 2025)

Figure 18-3: Grey Fox Site Access Road

18.3.2 Highway 101

Traffic on public Highway 101, including the segment near the Tamarack Road intersection and the town of Matheson, is under the jurisdiction of the Government of Ontario and is maintained accordingly, including winter snow removal. New turn-in and turn-out lanes, along with road lighting, are recommended at the intersection of Tamarack Road and Highway 101 to improve visibility and safety, given the anticipated increase in traffic movements during nighttime operations.

18.3.3 Existing Tamarack Road Bridge

The Tamarack Road Bridge is a 42 m long single-span, 30° skewed steel plate girder structure with a cast-in-place concrete deck supported on elastomeric bearings. It carries a single lane of two-way traffic.

Using the Canadian Highway Bridge Design Code (CSA S6:19), Section 14, a desktop assessment was completed of the bridge superstructure under permanent loads, including steel, deck and barriers, and asphalt wearing surface. Transitory loads were assessed as well, including normal traffic loading (CL1, CL2, CL3 625 Ontario truck and lane loads) and permit bulk mining trucks, including CAT777G, CAT773G, CAT745C, and CAT770.

Assessment results indicate the steel girders have adequate capacity for both normal traffic loading and the assessed permit bulk mining vehicles; however, the existing concrete deck does not. Load posting is not

recommended for normal traffic, provided the bridge continues to exhibit no signs of distress. Annual visual inspections are advised to confirm ongoing deck performance.

To enable mining operations involving heavier vehicles in a future mining scenario, deck strengthening would be required. Options include bonded fibre reinforced polymer sheets applied to the deck surface, or partial-depth deck replacement with modern stainless-steel reinforcement and steel traffic barriers, which would provide sufficient capacity for CAT745C and CAT770 vehicles or equivalent. Full-depth deck replacement and thickening, using either cast-in-place or precast panels, would be required to accommodate larger CAT777G and CAT773G trucks, as well as CAT745C and CAT770 vehicles.

If deck replacement proceeds, ancillary rehabilitation works should also be integrated, including replacement or elimination of expansion joints, assessment of elastomeric bearings for condition and capacity, development of a bearing replacement strategy if needed, and upgrading of approach slabs to accommodate heavier mining loads.

Further hydrotechnical and geotechnical investigations are needed to confirm the capacity and performance of the bridge substructure.

18.3.4 Mine Service Roads and Site Parking

A light vehicle parking lot will be located adjacent to the administration and dry complex. The parking area will be located inside the gatehouse perimeter and separated from haul truck traffic. The parking lot will be equipped with receptacles for engine block heaters.

18.4 Buildings and Ancillary Items

18.4.1 Temporary Construction Facilities

Mobilization, demobilization, temporary construction facilities, and selected mobile equipment required during the construction phase will be contractor supplied. McEwen will provide underground mobile equipment, power, and water.

18.4.2 Surface Maintenance Shop

The surface maintenance shop will be designed to maintain the underground and surface fleet, and will be housed in a pre-engineered building measuring 22 m wide x 30 m long x 12 m high. The building will be complete with metal cladding, insulation, HVAC, and snow guards. The ground floor will contain four (4) bays with overhead doors designed to accommodate mining mobile equipment. The remaining ground floor area will include an electrical bay, tool rooms, welding stations, vehicle wash area, and warehouse space for spare parts.

A 10-tonne crane will be installed above the vehicle maintenance bays.

The pre-engineered building will be installed on a concrete perimeter wall with footings under each structural column. The foundation will be backfilled and compacted on both the interior and exterior of the building. The slab-on-grade, which will be sloped in the maintenance area, is designed to contain and direct



spills toward in-ground sumps equipped with pumps. The wash area will be isolated by low curb walls and will include dedicated in-floor sump pumps fitted with an oil and water separator.

A Layout of the truck shop and warehouse are presented in Figure 18-4.

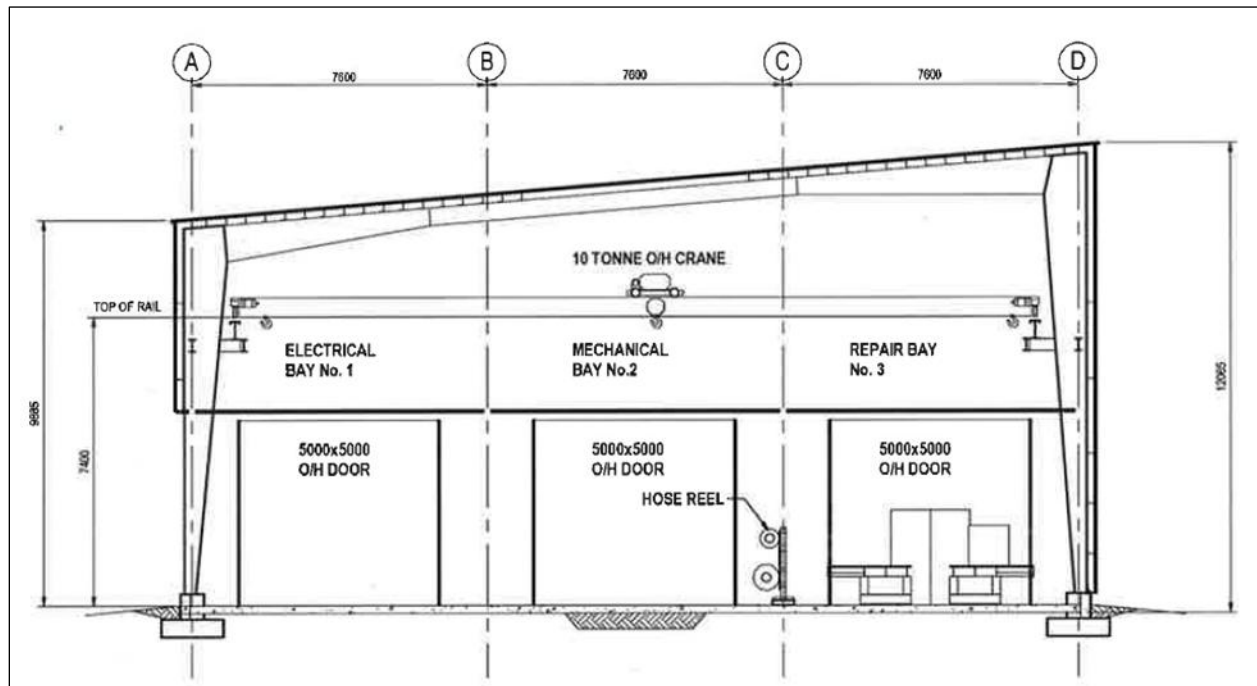


Figure 18-4: Surface Maintenance Shop – Section View

18.4.3 Warehouse

Two warehouses will be installed for Grey Fox, one near the administration office and mine dry, and a second installed near the surface maintenance shop. The warehouses will be new manufactured fabric buildings, each approximately 20 m wide x 25 m long, and will be equipped with power and HVAC. Figure 18-5 provides an example of a typical warehouse.



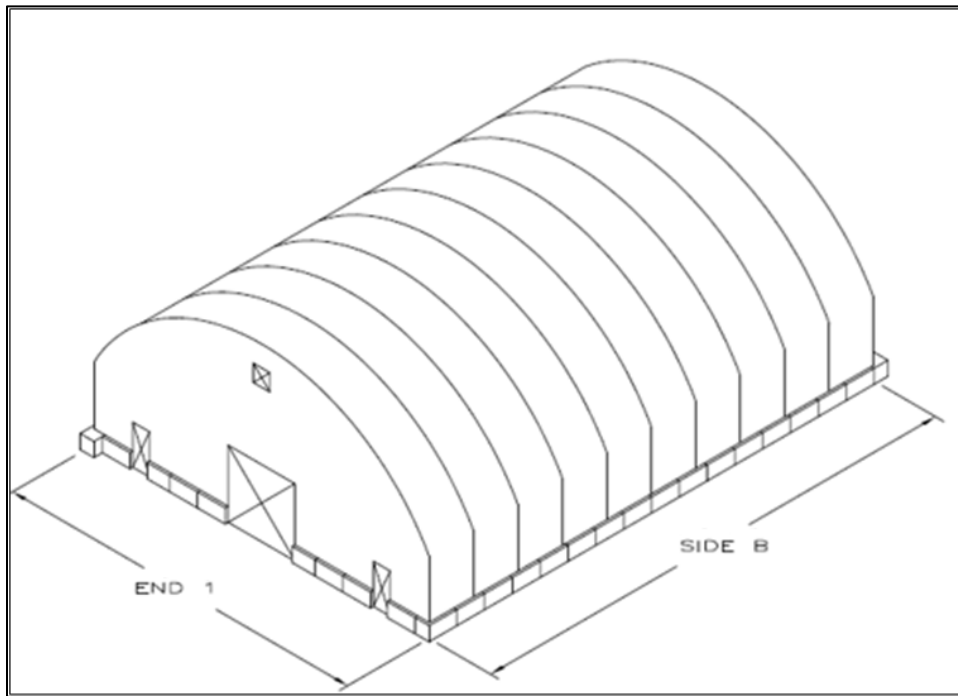


Figure 18-5: Example of Typical Warehouse

18.4.4 Administration Office and Mine Dry

18.4.4.1 Mine Dry

The mine dry will be a single-story, prefabricated building composed of multiple connected trailers. Separate female and male dry areas will be provided and sized to accommodate approximately 350 workers. The trailers will be fitted with lockers, washrooms, and shower facilities.

18.4.4.2 Administration Office

The administration office will consist of multiple prefabricated building trailers set on cribbing foundations and combined to form a single complex. The facility will include office and cubicle spaces, two meeting rooms, training room, storage areas, washrooms, and a kitchen and lunchroom. A mine rescue office will be incorporated into the administration office.

The Mine Dry and Administration Office will come furnished with the following:

- HVAC units and internal ducting
- Interior finishes including flooring, painting, lighting and emergency lighting, internal partition walls, and interior and exterior doors and glazing
- Fire detection and protection systems
- Internal plumbing and water filtration systems
- Electrical and communications wiring

18.4.5 Security/Reception

A security gatehouse will be located at the entrance to the mine site and will control access to and from the site. All personnel, including operations staff, management, and visitors will be required to report to security prior to accessing the site.

A first aid office will be incorporated into the mine dry complex.

18.4.6 Fuel Storage and Distribution

Fuel storage and distribution facilities will be located adjacent to the security / reception and near the surface maintenance shop. The facility near the security / reception will store gasoline and diesel, while the facility near the maintenance shop will store diesel and low-sulphur diesel.

Storage and dispensing equipment for low-sulphur diesel will be constructed for use by underground mobile equipment. Low-sulphur diesel will be delivered to site via tanker trucks. The storage facility will be equipped with a high-flow pump to fill the service truck used to refuel underground mobile equipment.

Storage and dispensing equipment for gasoline and regular diesel will be constructed for use by surface generators, water pumps, and mobile surface equipment.

18.4.7 Waste Disposal

Site waste including plastic, paper, cardboard, food scraps, and rubber will be collected and stored in a designated garbage disposal area. Waste will be removed on a set schedule and transported offsite for disposal.

Hazardous materials such as spent oil, batteries, paint, compressed gas cylinders, and corrosive products will be managed and disposed of in accordance with McEwen's Standard Operating Procedures. McEwen's Health and Safety Department will ensure that appropriate signage, safety data sheets, and standard operating procedures are available and kept current for all hazardous materials used on site.

Hazardous materials will be transported to and from the site by qualified contractors.

18.4.8 Mine Laydown Area

Mine laydown areas will be constructed of compacted granular material. A laydown area will be located near the administration office and mine dry complex. Additional laydown areas will be constructed near the North Portal and will serve as the construction staging area during the surface infrastructure and portal construction phases.

18.4.9 Concrete/Shotcrete Receiving Stations

Multiple concrete / shotcrete receiving station will be constructed for the Grey Fox Project. The receiving stations will be located near the Whiskey Jack ventilation raises, near the Gibson Return Air Fan and near the Grey Fox South ventilation raises.



Concrete/shotcrete will be delivered to site as wet ready-mix.

The receiving stations will include the following:

- a cased borehole from surface to the initial production level at each station
- weatherproof enclosure
- borehole inserts on surface
- receiving hopper
- washout and pumping equipment and
- related electrical, automation and IT equipment.

The underground receiving stations are defined in Section 16.

18.4.10 Surface Pumphouse

A surface pumphouse will be located adjacent to the surface collection pond.

The pumphouse will be equipped with a pumping system to deliver recycled service water to Grey Fox North and Grey Fox South portals. The system will consist of both operating and standby pumps equipped with variable speed drives.

The pumphouse will also include an operating pump and standby pump to deliver excess water to the Black Fox site holding pond for treatment.

18.5 Ventilation Collars

The Grey Fox Project will include multiple ventilation collars: two near the Whiskey Jack deposit, one at the existing Gibson portal, and two in the South Portal area.

Geotechnical assessments at Whiskey Jack indicate that the permanent vent raise collars intersect unconsolidated soils above bedrock. To mitigate this risk and ensure safe, stable access, vertical collars will be constructed. This approach will require crane mobilization and preparation of a properly compacted subgrade.

Each collar will have a finished inside diameter of 4.3 m and extend approximately 30 m below grade. Construction will proceed in controlled 2 m lifts using formed concrete sections with 400 mm wall thickness. Each lift will incorporate a reinforced rebar mat with dowels extending into the subsequent pour. Following curing, material beneath each section will be excavated, allowing the collar to be advanced incrementally to the next elevation. This method will continue until bedrock is reached.

At bedrock, perimeter anchors will be installed to secure the structure, followed by a socket pour to control groundwater ingress. If required, supplementary well drilling may be implemented to manage groundwater conditions. Upon completion and final curing, the vertical development contractor will break into the collar to commence raise excavation.

The remaining collars are located in areas with minimal overburden. These will be constructed by removing overburden, installing a concrete pad, and completing raise excavation from underground using an Alimak



system in a two-pass sequence. All raises will maintain the same finished diameter for consistency across the project.

18.6 Power Distribution

The initial power distribution system for the Grey Fox surface substation will be supplied from an existing 27.6 kV distribution line located on the eastern side of Tamarack Road. The final power configuration will require updates to the overall incoming utilities, to be finalized by McEwen. In both cases, a 27.6 kV tap will feed a new 35 kV disconnect switch and two 27.6 kV to 4.16 kV transformers located outside the substation. From the substation, 4.16 kV and 600 V power will be distributed to the remainder of the Grey Fox site through new switchgear.

To distribute power to the various surface facilities, 5 kV and 1 kV cables will be buried in trenches and routed where required. This is intended to minimize the risk of encounters with overhead power lines. The substation switchgear will provide 4.16 kV power to the ventilation facilities, compressor houses, administration office, crusher, pumphouse, and the portals for Grey Fox North and Grey Fox South.

The remaining surface facilities will be supplied with 600 V power via a 4.16 kV to 600 V transformer and 600 V power distribution panel located inside either the substation, administration office, or the Grey Fox South compressor house. Figure 18-6 illustrates the surface routing of buried electrical cables.

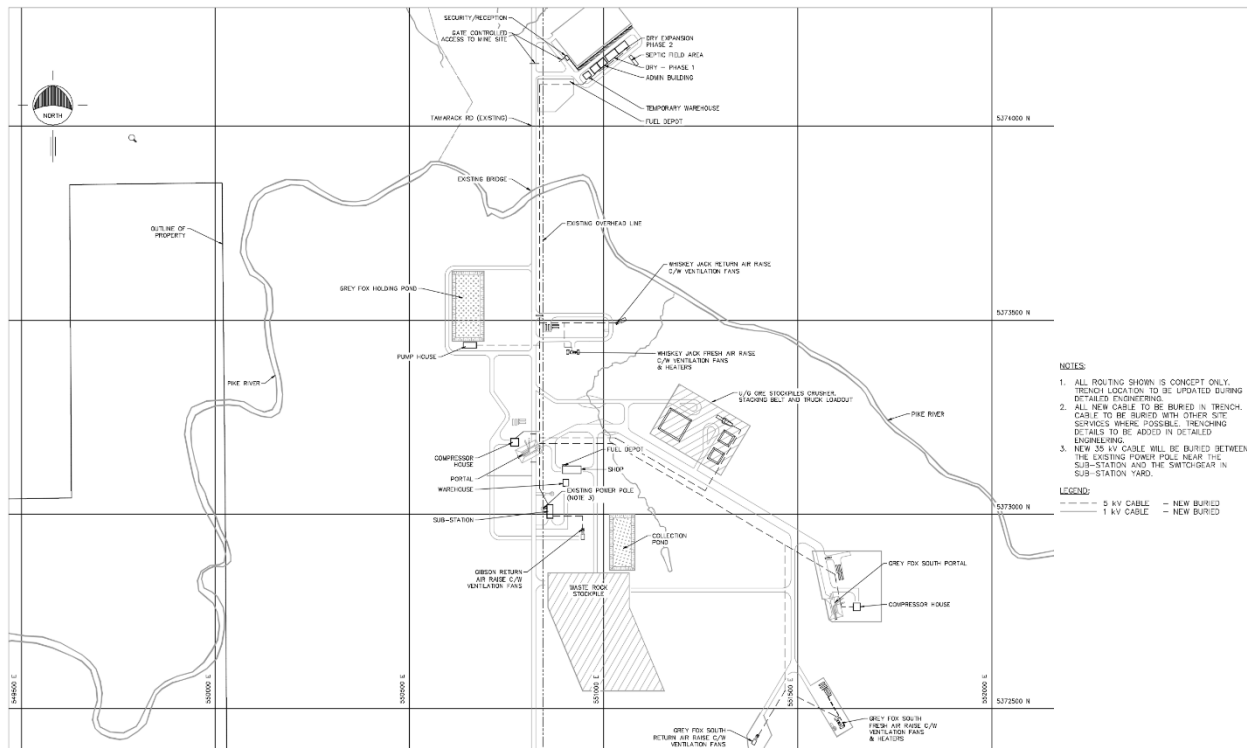


Figure 18-6: Surface Routing of Buried Cables



In the event of a utility outage, the generator located at the substation will provide sufficient power for all critical surface infrastructure, including ventilation systems and compressor house. The generator will be configured in a main-tie-main-tie-main arrangement with the two 27.6 kV to 4.16 kV transformers.

18.7 Communications and IT

Communication between surface facilities will be routed using buried conduits to ensure coverage to every area of the surface site. Communication cables installed in conduit will be kept separate from power cables.

The following systems are part of the planned surface communication and IT package:

- Fibre optic network
- Radio communications

For surface facilities, communication infrastructure will be installed in each building using either Type 1 or Type 2 network cabinets. Servers and core network switches will be located within the administration office.

18.7.1 Fibre Optic Network

The surface fibre optic network will be used for transmitting information for business networks, process control networks, and other dedicated systems as required, including fire alarm and power management systems.

The fibre optic network will connect with the 5G network infrastructure through network cabinets and will support systems such as tele-remote equipment operation and wireless network access. It will also connect to VoIP phones, which will be installed in surface facilities, as required.

18.7.2 Radio Communications

To allow for site-wide radio communications, repeater units will be installed throughout the Grey Fox site. These repeaters will have an overlapping radius and communicate with the existing Black Fox radio system. Figure 18-7 outlines the radio repeater locations across the Grey Fox site to achieve site-wide surface radio communication.



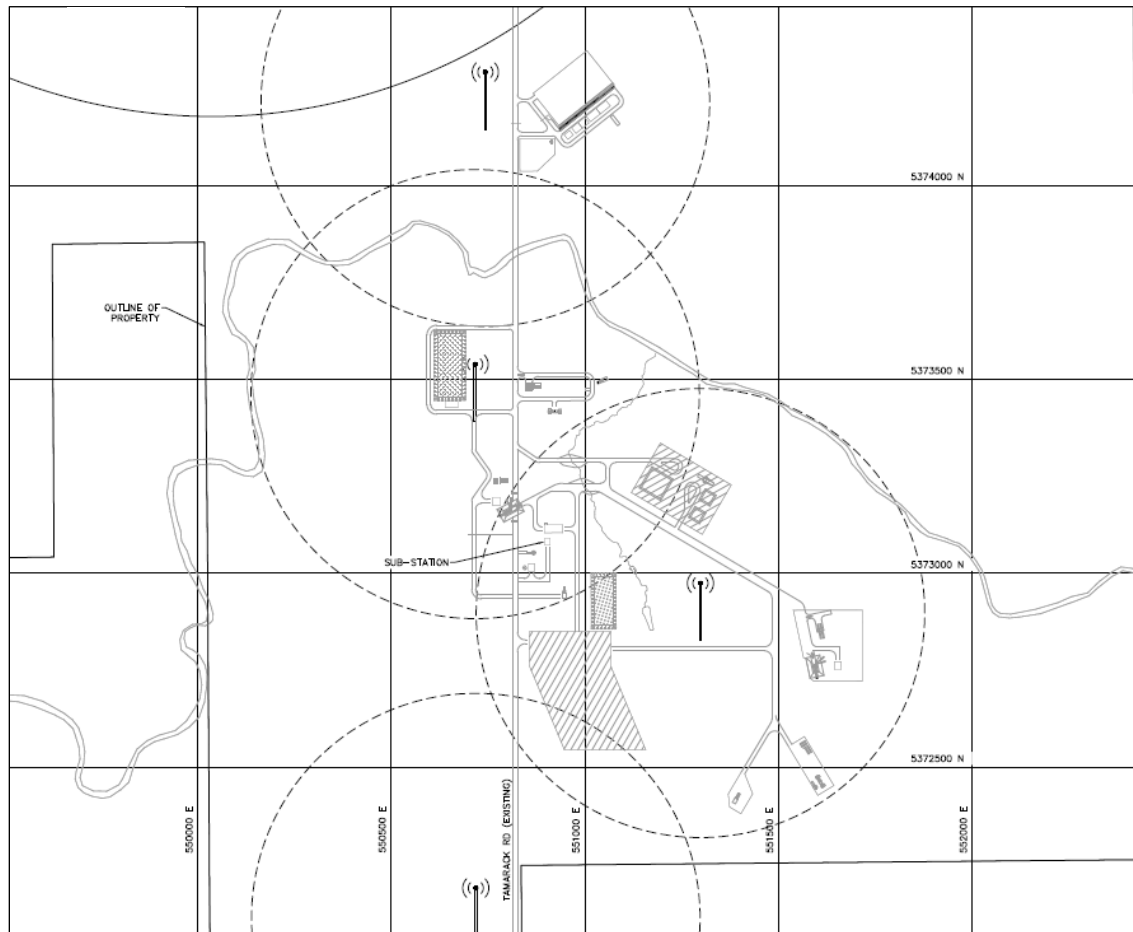


Figure 18-7: Locations of Radio Repeaters at the Grey Fox Site

18.8 Ore and Waste Rock Management

Separate ore and waste pads will be established between the Grey Fox North and South portals and will be constructed of compacted granular material.

A primary crusher with a stacking conveyor will be located on the ore pad. ROM ore will be trucked from either the North or South Portal and deposited near the primary crusher. A loader will be used to feed the crusher. Once crushed, the ore will be loaded into surface haulage trucks and hauled to the Stock Mill for processing.

The waste pad will be designed to accommodate approximately 300,000 tonnes of waste at peak accumulation prior to the start of underground production. Once production commences, stopes will be backfilled with ROM waste, and supplemented as required with material reclaimed from the surface waste stockpile. Waste rock from the Black Fox site will also be hauled to the Grey Fox waste stockpile as backfill demand exceeds the waste produced from Grey Fox underground operations.

18.9 Water Management

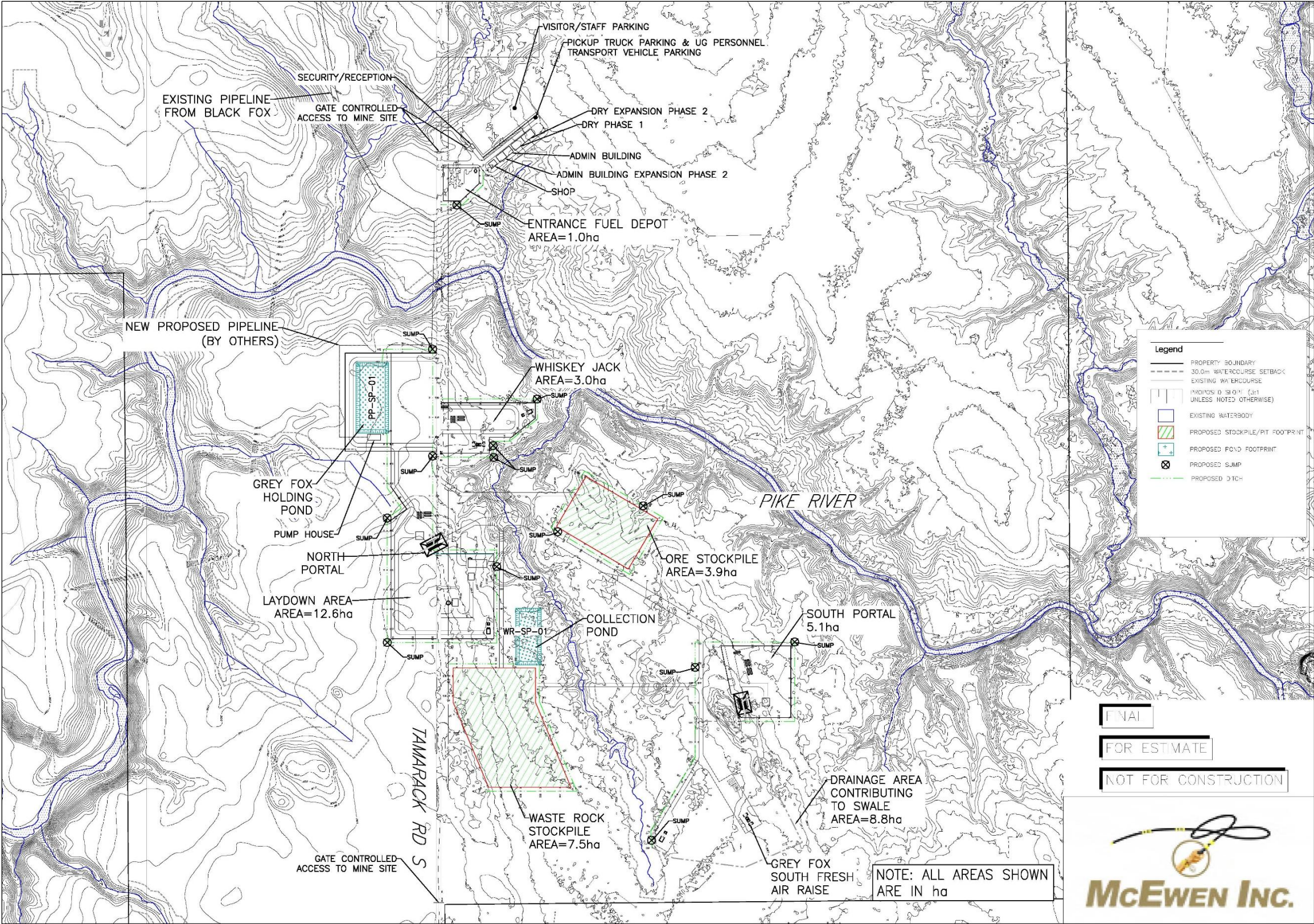
The water management plan for the Grey Fox site is to collect and store on-site contact water and then transfer that contact water to the Black Fox mine for treatment and discharge. To reduce the mine water inventory, non-contact runoff at Grey Fox will be diverted using perimeter berms, allowing runoff to drain naturally offsite. Existing watersheds within the Grey Fox site include tributaries that drain to the Pike River, which bisects the Grey Fox property boundary west to east.

Two water management ponds will be constructed at Grey Fox to contain contact water from the existing 1:100-year annual exceedance probability (AEP) storm event. Each pond will have an emergency spillway sized to attenuate the 1:100-year AEP storm with climate change while maintaining a minimum freeboard of 0.5 m. The ponds will target a minimum length-to-width ratio of 2:1 to reduce short-circuiting. The Grey Fox holding pond will have a permanent pool to promote settling and provide extended detention and storage of settled material.

Contact water collection ditches will drain by gravity where practicable to reduce pumping requirements. In areas where gravity drainage to a sedimentation pond is not feasible due to site topography or stockpile geometry, excavated sumps and pumps will be installed to direct water to the water management ponds. Electric pumps will be used in areas where electrical service is available, with diesel pumps used elsewhere as required. Water discharged from excavated sumps will be released into the same perimeter ditch further downstream once gravity drainage toward a pond or closest downstream sump resumes.

The conceptual water management arrangement is provided in Figure 18-8 and detailed in Section 18.9 below.





Source: (Stantec, 2026)
 Figure 18-8: Grey Fox Conceptual Water Management General Arrangement



18.9.1 Water Management Overview

Contact water management will be required for the waste rock stockpile and ore stockpile areas, dewatering from the underground workings, and areas where fuel storage or handling occurs, including the north fuel depot, laydown area, North and South Portal areas, and fuel depot. A minimum 15 m setback from watercourses and waterbodies will be maintained to limit potential interactions with the natural environment.

Runoff and seepage generated from the Project will be managed through a stockpile collection pond and a holding pond. The stockpile collection pond will receive contact water from the waste rock stockpile and South Portal area, and will be designed to convey collected water to the Grey Fox holding pond, with drawdown occurring over an estimated period of five days following the 1:100 AEP, 24-hour precipitation event.

The Grey Fox holding pond will be designed to receive direct runoff from the ore stockpile, North Portal area, south fuel depot, and laydown area, in addition to pumped inflows from underground dewatering and the stockpile pond. The Grey Fox holding pond will also serve as a source of water for underground service water requirements and surface site needs, such as dust suppression.

The Grey Fox holding pond will function as a throughput pond and pumping station for Project infrastructure located south of the Pike River, conveying water via pipeline at a maximum rate of approximately 130 L/s to the Black Fox water management system for treatment. The holding pond will be sized to manage inflows associated with the 1:100 AEP, 24-hour precipitation event, with discharge occurring over an estimated period of five days.

Water collected from the administration fuel depot located north of the Pike River will be managed separately and conveyed to the main discharge pipeline at an approximate rate of 0.5 L/s for transfer to the Black Fox site for treatment.

18.9.2 Water Balance

A site-wide water balance was completed to estimate the quantity of mine site contact water expected to be managed during the operational phase of the Project to support water management design and feasibility studies. As presented in Figure 18-9, the Grey Fox water balance accounts for precipitation, runoff, evaporation, and infiltration flows at each identified mine site component. The water balance represents the mine site components at full development during operation.

To reduce the mine water inventory, non-contact runoff is proposed to be diverted using perimeter berming and ditching to allow runoff to naturally flow offsite. Contact water collected on site will be pumped via pipelines to the Black Fox site.

The site-wide water balance was developed using an average climate scenario based on Climate Normals precipitation data for the 1981-2010 period (Class A climate data) from the Environment and Climate Change Canada (ECCC) Victor M. Power Airport climate station (Station ID 6078285). A portion of precipitation during colder months, from October to March, was assumed to be stored as snow with partial



melt occurring in October, November, and March, and a larger snowmelt event occurring in April during the spring freshet.

The percentage of precipitation that results in runoff from the stockpile areas was accounted for in the water balance model by balancing precipitation less evaporation and net infiltration losses for each catchment. Net infiltration represents the sum of groundwater infiltration and toe seepage, less any soil moisture losses. This proportion of net infiltration reports to perimeter seepage collection ditches and is diverted to the water management ponds.

Figure 18-9 presents the water balance transfer rates, identified in cubic metres per day (m^3/d), to the Black Fox site under average climate conditions. Actual instantaneous flow rates are expected to vary seasonally and annually depending on climate conditions. The water balance model was used to analyze the volume of excess water that will be transferred via pipeline to the Black Fox site.

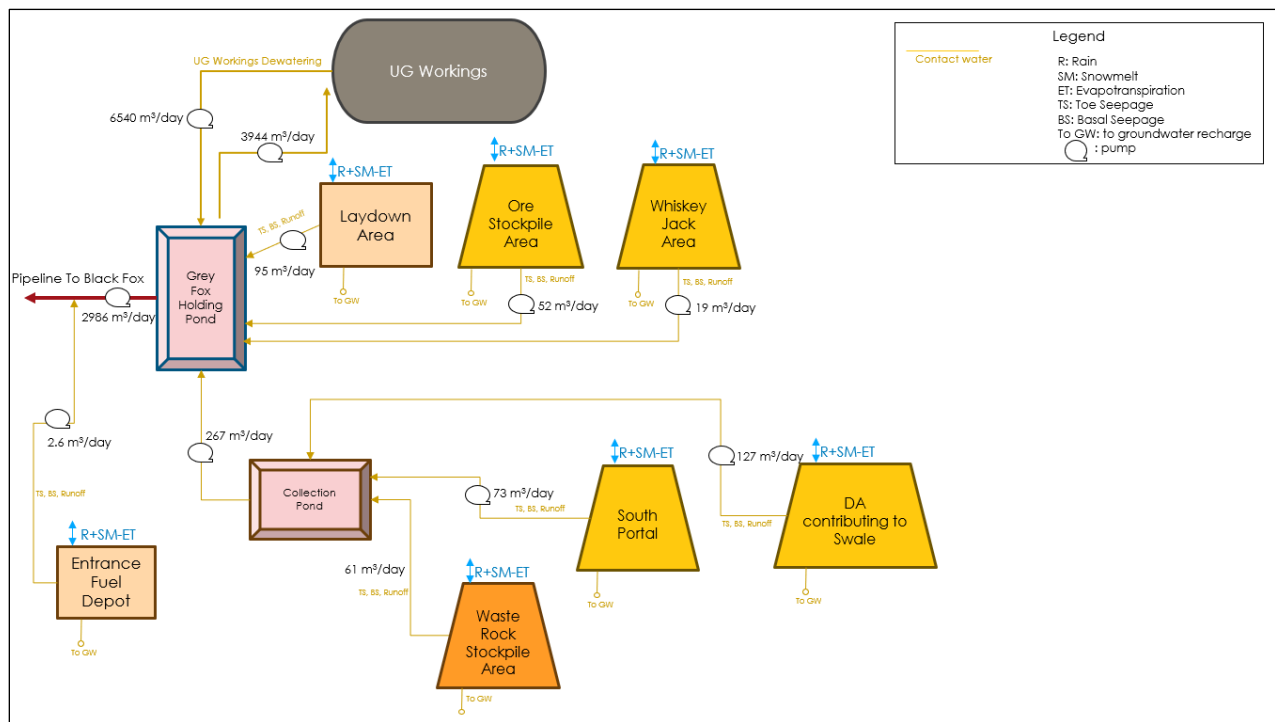


Figure 18-9: Conceptual Water Balance Schematic

18.9.3 Mine Dewatering

The underground mine dewatering system is described in Section 16 of this report. The peak underground net flow at full mine development is expected to be approximately 3,270 m^3/d from each underground area, for a combined total of approximately 6,540 m^3/d .

18.9.4 Sewage Handling

Domestic wastewater from the administration office and mine dry, and process washroom facilities will be discharged to an on-site septic bed located to the northwest of the facilities.

18.9.5 Domestic Water

A fully enclosed and climate-controlled domestic water treatment plant will be installed to treat water from an on-site well for non-potable uses (showers, toilets, and sinks in the mine dry, and office complexes). Potable water for consumption will be provided as bottled water.

18.10 Tailings Storage Facilities

18.10.1 Stock Mill Complex Tailings Management Area

The McEwen Stock Mill complex is located approximately 34 km west of the Black Fox mine and proposed Grey Fox mine, along Highway 101 (Figure 18-10). Ore from Grey Fox will be transported to the existing Stock Mill for processing, and the produced tailings will be stored within the existing tailings facility and a planned western expansion of the Stock Mill Tailings Management Area (TMA), currently referred to as the Western Expansion Area (WEA). Haul distance from Grey Fox to the Stock Mill is approximately 39 km.

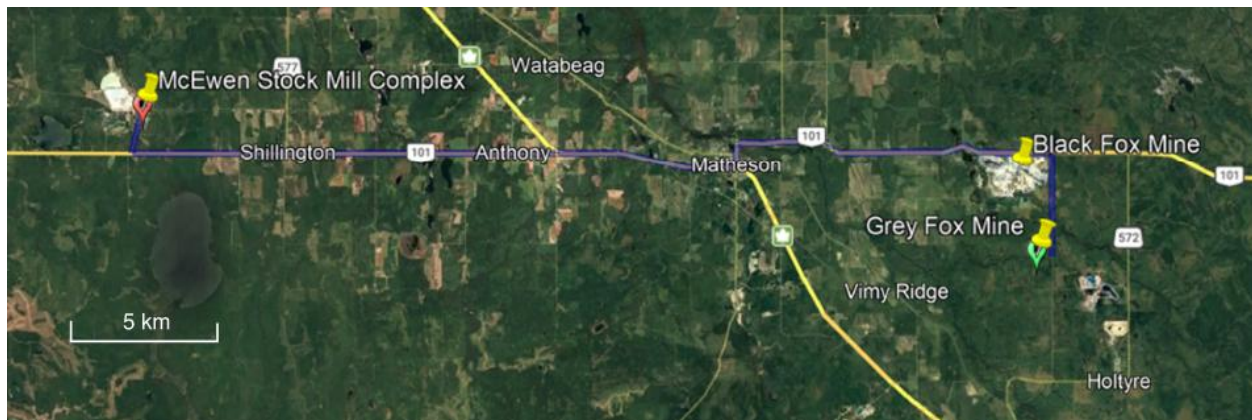


Figure 18-10: Grey Fox to Stock Mill Haul Route

As of April 2026, the Stock Mill has a currently permitted design capacity of 3,200 t/d, with current mill production ranging from approximately 900 t/d to 1,500 t/d. At the start of Grey Fox mining, mill production is planned to comprise approximately 50% from Grey Fox and 50% from the Stock West underground (located at the Stock Mill site) and other sources (e.g., custom milling). As mining progresses, the Grey Fox feed will gradually ramp up to approximately 100% of the Stock Mill feed.

18.10.2 Stock Mill Complex TMA Western Expansion

McEwen-leased lands to the west and north of the existing Stock Mill TMA and northern water management pond provide room for expansion of the facility. A planned western expansion of the Stock Mill complex



TMA (i.e., the WEA) is projected to provide approximately 3 million m³ (4.2 million tonnes) of additional tailings storage capacity. This estimate assumes an in-place tailings density of approximately 1.38 t/m³ based on historical operational data for slurry tailings placement, at approximately 39-40% solids by weight, and a full buildout dam crest elevation similar to the Stock Mill TMA of 285.0 m. Similar to the existing Stock Mill TMA, decant water and stormwater from the expansion will be routed through the water management pond for recirculation and treatment, as needed.

At a planned tailings placement rate of 3,000 t/d, the WEA is expected to provide approximately 3.8 years of operational storage capacity, necessitating additional future expansions to accommodate the full projected tailings volumes from Grey Fox Mine and other tailings sources. The footprint of the WEA is limited to less than 50% of the existing TMA footprint in accordance with SOR/2019-285, Schedule (Section 2), part 19(c) of the Government of Canada Physical Activities Regulations.

The projected operational life and capacity of the TMA expansion is currently less than the total projected mine life and tailings storage needs for Grey Fox. However, additional McEwen-leased lands located to the northwest and north of the currently planned expansion area can accommodate additional expansion stages. Alternate tailings storage methods are also actively being explored, including dry stacking, to largely eliminate the need for a dam and allow for additional storage capacity within a similar footprint, and tailings storage within the open pit at the Black Fox Mine site, located immediately north of Grey Fox. These alternatives are being further evaluated as the Project progresses toward Feasibility.

The current expansion concept (Figure 18-11) includes approximately 2,600 linear meters of dam construction comprising a 2H:1V upstream slope, 3H:1V downstream slope, and 4-m top width (similar to the current TMA). With an average dam height of approximately 9.8 m (reaching a maximum height of approximately 14 m), the total embankment volume is estimated at approximately 730,000 m³. The dam would be raised in stages to allow for consolidation of the underlying soft clays in between raises, with a currently estimated one-third of the required borrow material to be sourced from the upper firm silty clay layer within the expansion footprint. This borrow volume is therefore added to the available tailings storage capacity estimate.



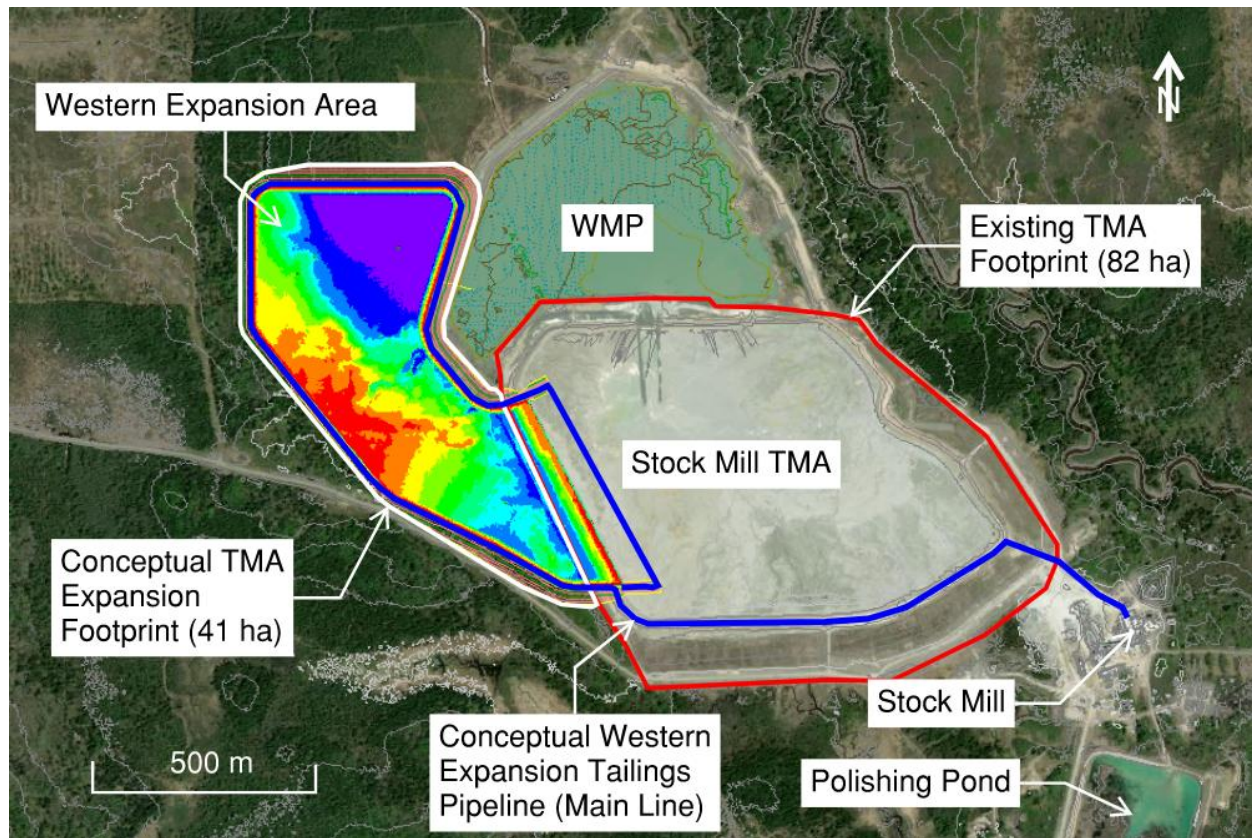


Figure 18-11: Stock Mill TMA and Conceptual Western Expansion Area

Tailings generated at the existing milling facility will be pumped to the TMA expansion via extension of the tailings pipeline servicing the existing Stock Mill TMA. Based on historical operational data, the tailings slurry is characterized by an average solids content of 39.5% by weight, with a solids specific gravity of 2.89 t/m³. At a maximum permitted production rate for the Stock Mill of 3,200 t/d, the system will be designed to manage a nominal slurry flow rate of approximately 250 m³/h and is projected to comprise approximately 4,500 m (total length) of 8-inch (219-mm) diameter SDR 11 HDPE fusion-welded pipeline.

A distribution system will be designed to promote uniform beach development and geotechnical stability through controlled deposition. At this conceptual stage, the main line is planned to run the perimeter of the TMA expansion, with tee-offs spaced at 20-m intervals extending into the impoundment area. Similar to the current Stock Mill operation, for summer operations, each tee is planned to be equipped with a removable manifold consisting of three 3-inch (90-mm) diameter cyclone spigot lines to facilitate thin-layer beaching. In winter months, the spigot lines would be disconnected to allow for direct end-of-pipe discharge, mitigating the risk of freezing. A series of isolation and bypass valves will be installed at each tee location to allow operators to effectively cycle discharge points and manage the tailings pond elevation and beach development.

18.10.3 Water Management and Recirculation

The Stock Mill utilizes a recirculating water management system designed to prioritize the reuse of process water. Tailings decant water is directed to the existing on-site water management pond for recirculation to the mill. To maintain a sustainable site water balance, treated volumes in excess of mill requirements are directed to a dedicated polishing pond for final clarification. This treated water is subsequently discharged to the environment in compliance with the site's Environmental Compliance Approval (ECA).

A new engineered spillway will be constructed along the adjoining edge of the new expansion dam and the existing water management pond to convey decant water from the TMA expansion into the water management pond. From there, water will be recirculated for treatment and mill use or discharged to the environment using the Stock Mill's existing recirculation system.

18.10.4 Subsurface Conditions and Foundation Design

Geotechnical investigations within the TMA expansion footprint (Golder, 2015) indicate a typical Northern Ontario glaciolacustrine profile.

The stratigraphy consists of:

- 0 to 0.15 m BGS: Topsoil/Peat (to be removed prior to dam construction; thicker in low-lying areas)
- 0.15 m to 3.0 m BGS: Firm silty clay
- 3.0 m to 18.0 m BGS: Very soft varved clays and silts
- 18.0 m+ BGS: Very dense glacial till (silty sand to sandy silt)

Due to the thickness of the underlying soft clay unit, the dam design is projected to include an additional 15% overbuild volume to address settlement during its operational life. Borrow material for dam construction will be sourced on-site, primarily utilizing the upper firm silty clay layer and waste rock generated from the on-site underground mining operation.

18.10.5 Seepage Control and Stabilization

As a component of the WEA dam construction, a seepage cutoff wall will be constructed along the dam alignment, approximately 2 m into the underlying varved clay (i.e., layered lenses of clay and silty sand that represent seasonal deposition in the historical lake that once covered the area), to cut off near-surface varves (i.e., the silty sand lenses) from lateral seepage out of the impoundment.

Additionally, because the expansion will share an adjoining dam with the existing TMA, a waste rock toe berm will be included to mitigate phreatic surface mounding in the existing dam's outer toe during new tailings placement. The design of the new dam will also incorporate internal chimney and blanket drains, as needed, to mitigate the risk of piping. Additionally, due to the soft varved clays underlying the WEA, wick drains will be installed under the dam footprint to accelerate pore-pressure dissipation and foundation consolidation.



18.10.6 Future Tailings Storage Facility Expansion

The Stock Mill Complex has existing access to leased lands immediately adjacent to the existing tailings facility that are estimated to provide at least 193 ha of additional tailings containment, conceivably accommodating up to approximately 24 Mt of additional storage capacity. Following the initial western expansion of the Stock Mill TMA by 41 ha and 4 Mt (comprising Stock Mill and Grey Fox tailings), Grey Fox is anticipated to provide an additional 7.4 Mt of tailings to the Stock Mill site.

Additionally, the existing open pit at Black Fox mine (located immediately north of Grey Fox) provides a potential opportunity for future storage of Grey Fox tailings, with an anticipated lower environmental-risk profile and lower cost than Stock Mill. This alternative is currently under evaluation, in addition to alternative methods of tailings management at Stock Mill, as the project approaches feasibility.

18.10.7 Geotechnical and Operational Risks

The primary risk to the conceptual layout of the TMA expansion and its maximum tailings storage capacity is the interplay between the underlying soft clay stratigraphy and the fixed regulatory footprint.

Capacity Reduction Risk: Because the footprint is limited by the 50% regulatory objective, any side-slope flattening required to address geotechnical concerns would need to be accommodated within the approved footprint, reducing the estimated storage capacity. Mitigation measures include using prefabricated vertical drains (PVDs) or wick drains to improve foundation pore-pressure dissipation and strength gain.

Settlement and Staged Construction: The underlying soft clay is expected to cause notable settlement and elevated pore pressures during construction and operation, depending on the rate and magnitude of dam raises and corresponding tailings placement. Accordingly, a staged construction approach will be used to allow pore-pressure dissipation and strength gain in between raises, which will impact the timing of available storage. Methods and rates of pore-pressure dissipation and construction staging will be further evaluated as the Project approaches feasibility.

Foundation Seepage Management: To accelerate the timeline for subgrade consolidation and dam construction, the design includes installation of PVDs and a granular drainage blanket to increase the rate of pore-pressure dissipation and strength gain in the foundation soils. Additionally, to reduce the potential for infiltration of tailings pore waters into near-surface silt lenses (varves) in the underlying varved clay, which could provide a horizontal seepage pathway out of the impoundment, a slurry or in-situ stabilization cutoff wall is envisioned to disrupt the near-surface varves. The design of the PVDs, drainage blanket, and the seepage-cutoff wall will be explored in greater detail as the Project approaches Feasibility, in addition to exploring alternative tailings placement methods (e.g., dry-stack tailings) that could potentially reduce cost and risk.

Regulated TMA Footprint: The current ECA for the Stock Mill (ECA #4371-B2KHZT) lists a footprint area of 56 ha. However, aerial imagery of the site and past design documents show an outer footprint of approximately 82 ha. The actual regulated footprint of the existing TMA will need to be verified with the Ministry of Environment and Mines (MEM) and the Ministry of the Environment, Conservation and Parks (MECP) prior to detailed design of the western expansion, as the 56 ha listed in the current ECA appears to either be in error or representative of a condition other than the outermost TMA footprint. Additionally, it has



been confirmed verbally with the MEM that future expansions of the TMA would be held to a similar rule based on the active footprint of the facility at that time, allowing additional future expansions to proceed similarly, without the need for a full federal environmental assessment, assuming adherence to the '50% rule'. Changes to this interpretation could alter the timeline and feasibility of future facility expansions.

18.10.8 Supporting Information

The following tables summarize the geometric parameters and characteristics associated with the conceptual layout of the WEA storage facility. The grading design for the conceptual layout was developed in Autodesk Civil 3D, with dam and tailings volumes calculated as volumetric comparisons between the conceptual-grading design surfaces and an existing-grade surface represented by 2025 aerial drone LiDAR data gathered by Arcadis combined with the 2016 COOP 2-m DEM for areas outside of the modified portions of the Stock Mill tailings storage facility.

Table 18-1 presents the design parameters for the conceptual WEA layout used in preliminary tailings storage cost estimation. Table 18-2 and Table 18-3 provide WEA storage capacity and operational life, and tailings slurry design parameters, respectively.

Table 18-1: WEA Geometric Design Criteria

Dam Component	Value	Unit	Source/Notes
Total Dam Length	2,632	m	Concept Layout
Upstream Slope	2H:1V	Horizontal:Vertical	Existing Dams
Downstream Slope	3H:1V	Horizontal:Vertical	Planning Assumption
Crest Width	4.0	m	Existing Dams
Average Dam Height	9.2	m	Local Topography
Maximum Dam Height	14.0	m	Local Topography
Dam Embankment Footprint	131,269	m ²	Concept Layout
Foundation Key-in Depth	2.0 to 3.0	m	Historical Borings
Spillway Type	Sheet pile weir with rock-lined downchute channel	-	Existing Dams
Maximum Footprint	41	ha	50% of Existing TMA (82 ha)
Key-in Depth (slurry or in-situ stabilization wall)	4 to 5	m	Subsurface Stratigraphy
Additional Stabilization of Adjoining Dam	37,772	m ³	Concept Grading Design (approx. 10H:1V buttress)



Table 18-2: WEA Storage Capacity and Operational Life

Dam Component	Value	Unit	Source/Notes
Total Available Storage Volume	3.0	Million m ³	Concept Grading Design
Average Deposited Dry Density	1.38	tonnes/m ³	Historical Data
Total Storage Tonnage	4.2	Million tonnes	Combined Grey Fox tailings (comprising 2.1M tonnes), Stock West, and other facilities
Daily Tailings Production	3,000	t/d	Mine plan
Annual Tailings Production	1.10	Million tonnes	Calculation
Projected Life for the Western Expansion of the Stock Mill TMA	3.8+	years	Calculation; additional life to be provided by future expansions
Supplemental Capacity	Additional Expansion at Stock Mill, Alternate tailings placement methods, Black Fox Open Pit	-	Viable options based on current land uses

Table 18-3: Tailings Slurry Design Parameters

Parameter	Value	Unit	Source/Notes
Maximum Throughput	3,200	t/d	Peak permitted capacity of the Stock Mill
Solids Content by Weight	39.5	%	Historical Data
Density (Solids)	2.89	t/m ³	Historical Data
Slurry Flow Rate (Nominal)	250.4	m ³ /h	Calculated
Pipeline Material	HDPE SDR 11	-	Fusion-welded
Pipeline Diameter (OD)	8 (219)	in (mm)	Recommended
Pipeline Length	4,500	m	Site Layout
Static Elevation Head	+13	m	Survey Data



18.11 Sitewide Water and Mass Balance Model

Arcadis constructed a water and solute mass balance model to evaluate flow rates and water quality for mine water potentially requiring water treatment as part of development of the Grey Fox mine. Since water generated at the Grey Fox mine is to be sent to Black Fox for handling and treatment prior to discharge, the water balance model includes the Grey Fox underground mining features, the Black Fox Pit, and surrounding water-management facilities.

The water/mass balance model was constructed using the GoldSim model platform. The GoldSim model represents an expansion and refinement of the previous water balance developed for Black Fox Mine by Wood (2022) to include the simulation of Black Fox pit filling and Grey Fox underground mining. Model development assumptions and results are provided below.

18.11.1 Model Development and Assumptions

The GoldSim model was constructed to couple water flows and water quality and allow for evaluation of water management scenarios. A schematic of the overall water balance is provided in Figure 18-12.

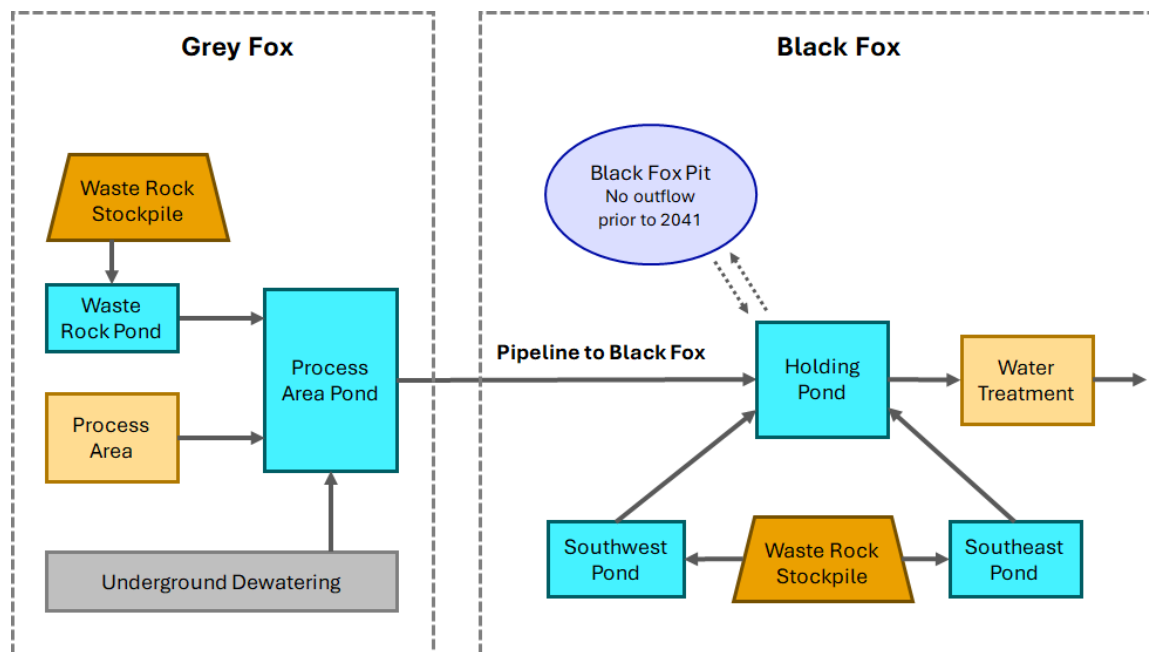


Figure 18-12: Water Balance Schematic

A brief overview of key model components and assumptions is provided below.

- The site-wide water and mass balance model uses historical daily precipitation for past periods and long-term monthly average precipitation from the Timmins Climate Station for future periods to represent seasonal conditions at the site. With the use of monthly-average precipitation, it was effectively run for a climate normal year, and since the model usage was for a relatively short period of 15 years during mine operation, potential climate change effects were not considered. Only the climate normal model results (without wet or dry years) is provided here, but buffering of



wet year flows is discussed further below. Open-water evaporation losses are included using monthly evaporation rates from the Timmins Victor Power A Weather Station and exposed water surface areas for each modeled subunit

- Surface runoff is estimated using monthly precipitation, catchment areas, and representative runoff coefficients for key mine features and disturbed areas at both Black Fox and Grey Fox. The model also accounts for snow accumulation, sublimation, and snowmelt, with rainfall contributing immediate runoff while snow is carried forward and released according to monthly melt assumptions.
- The Black Fox portion of the model includes major mine water features such as the open pit, southwest pond, southeast pond, holding pond, and treatment pond. Black Fox pit filling is controlled by direct precipitation, runoff, and groundwater inflow, with the largest groundwater contribution coming from overburden and an additional smaller contribution from bedrock. Model simulations indicate that following Black Fox mine operations, the pit will fill with water and has water storage capacity until at least 2043. As such, water flows from the Black Fox Pit do not enter the Grey Fox water balance during the project period from 2027 through 2041. As a conservative assumption, the model simulation does not include any other potential mine reclamation and closure activities that could occur at Black Fox before 2043.
- The Grey Fox portion of the model includes the underground mine, process area, process area pond, waste rock stockpile, and waste rock pond, as described further in Section 18.9. During Grey Fox operations, runoff and underground dewatering water are collected in the Grey Fox Process Area Pond and then pumped to the Black Fox holding pond, where Grey Fox and Black Fox waters are blended for treatment and discharge.

Solute mass balance was added to the water balance model for the following analytes of interest:

- Major cations: Calcium, magnesium, potassium, and sodium
- Major and minor anions: Alkalinity, sulphate, chloride, nitrate
- Metals, oxyanions, and other water quality constituents: Arsenic, antimony, cobalt, copper, iron, lead, nickel, zinc, selenium, molybdenum, vanadium, cadmium, and ammonia.

Cyanide was not included as a model parameter, since the Grey Fox and Black Fox units do not include mill tailings contact water. Water management associated with the Stock Mill Complex is summarized in Section 18.10.3.

Although pH is not directly modeled in GoldSim, pH was calculated based on the mass balance of acidity and alkalinity. For calculation of pH, carbon dioxide and calcite are both assumed to sustain a slight supersaturation, with an assumed $\log(p\text{CO}_2)$ of -3.0 and a calcite saturation index of 0.3.

Development of the solute mass balance required determination of water quality for each flow input to the model. Table 18-4 summarizes the water quality data used in developing the geochemical water quality source terms for each flow term in the model. Additional details on the available test data for Grey Fox used in the development of geochemical source terms are provided in Section 20.2.



Table 18-4: Summary of Water Quality Geochemical Source Terms and Data Sources

Water Flow	Description/Source
Direct precipitation	Major ion contents for typical rainwater in North America; concentrations assumed zero for metals.
Natural runoff	Water quality from the Pike River monitoring station, median of data collected from 01/2020 through 10/2020.
West Waste Rock Stockpile drainage	Average of 2024 monthly water quality results from West Waste Rock Stockpile monitoring location.
East Waste Rock Stockpile drainage	Average of 2024 monthly water quality results from East Waste Rock Stockpile monitoring location.
Black Fox Pit groundwater inflow	Average of 2024 quarterly water quality results from Interceptor Well 147125.
Black Fox Pit wall runoff	Calculated as flow-weighted difference between Black Fox Pit/underground mine sump water quality and groundwater inflow.
Grey Fox waste rock stockpile	Developed from Grey Fox Week 0 humidity cell test results (see Section 20.2), scaled based on comparison with Black Fox Waste Rock Stockpile drainage.
Grey Fox Underground Mine dewatering	Assumed the same chemistry as the Grey Fox waste rock stockpile source term, based on similar geologic rock types intersected during mining.

18.11.2 Model Scenario and Results

The water and solute mass balance model was used to simulate future water management associated with Grey Fox underground mining with water management at Black Fox.

The model was set to run from January 1, 2027 through December 31, 2041. The Grey Fox underground dewatering rate was assumed to increase to 1,500 m³/day within the first year, up to an operational rate of 1,938 m³/day by the second year. For simplicity, the full operational footprint of the Grey Fox site features, including the process area and waste rock stockpile, were assumed to be present at the start of the modelling period. Model outputs include flow rates and estimated water quality over the modeled timeframe. This evaluation informs the water treatment requirements as described in Section 18.12.

The model simulation results indicate that with pumping of Grey Fox contact water to Black Fox, the average flow to the holding pond varies seasonally between 2,200 and 4,700 m³/day with an annual average of approximately 3,300 m³/day which was the rate used in sizing of a potential water treatment system in Section 18. Importantly, this value is an annual average, and the actual flow at any given time will likely exceed this average. Although the various holding ponds at Black Fox and Grey Fox were designed to buffer short-term storm and runoff events, the ponds are not large enough to retain water during the spring snowmelt period (average flow rates of 4,700 m³/day) for a water treatment plant designed to handle the annual average flow rate (3,300 m³/day). However, for the purposes of this evaluation, it is assumed that treatment of annual-average flow rates based on average annual precipitation is reasonable. In practice, it is likely that the Black Fox open pit itself can be used for seasonal water storage to buffer flows to the holding pond.



The holding pond/treatment pond water quality was compared to water quality standards to understand potential treatment requirements. For this evaluation, simulated water quality is compared to the existing ECA values issued in 2019, understanding that these values may no longer be applicable when the ECA is revised following incorporation of Grey Fox. Specifically, the values included in the 2019 ECA were based on criteria in Ontario Regulation (O. Reg.) 560/94, which McEwen understands was revoked in June 2021 with O. Reg. 462/21. Updated effluent criteria will need to be developed based on MECP Procedure B-1-5¹. This development will likely include consideration of the more recent Canadian Metal and Diamond Mining Effluent Regulations (MDMER)², Canadian Council of Ministers of the Environment Canadian Environmental Quality Guidelines for the Protection of Aquatic Life³, Federal Environmental Quality Guidelines for freshwater⁴, and British Columbia Ministry of Environment and Parks Water Quality Guidelines⁵. However, Procedure B-1-5 specifically refers to the use of Ontario PWQO⁶ values for surface water. For preliminary identification of parameters of potential concern and evaluation of potential water treatment requirements, the water quality predictions are compared with PWQOs. A comparison to MDMER monthly mean concentrations is also provided purely for comparison purposes.

Results for key water quality constituents, including those exceeding potentially relevant water quality standards, are summarized in Table 18-5. Parameters of potential concern include arsenic, cobalt, copper, iron, nickel, molybdenum, vanadium, and zinc. Antimony, cadmium, and lead were evaluated based on their elevated levels in specific geochemical sources, but simulation results do not predict an exceedance of PWQOs in the holding pond. Observations from the analysis relevant to water treatment demands include the following:

- The alkalinity of the water reporting to the holding pond is simulated to be between 130 and 150 mg/L as calcium carbonate (CaCO_3). The pH of the water in the holding pond is predicted to be between 7.9 and 8.1, with a tendency to precipitate calcite with carbon dioxide off-gassing.
- Arsenic, cobalt, copper, iron, nickel, molybdenum, and zinc are predicted to exceed PWQO values throughout the year.
- There is a chance that vanadium may slightly exceed the PWQO seasonally.

¹ <https://www.ontario.ca/page/b-1-5-deriving-receiving-water-based-point-source-effluent-requirements-ontario-waters>; last accessed on 7/2/2026.

² <https://www.canada.ca/en/environment-climate-change/services/managing-pollution/sources-industry/mining-effluent/metal-diamond-mining-effluent/metal-diamond-mining-effluent-regulation.html>; last accessed on 4/9/2026.

³ <https://ccme.ca/en/resources/water-aquatic-life>; last accessed on 7/2/2026.

⁴ <https://www.canada.ca/en/health-canada/services/chemical-substances/fact-sheets/federal-environmental-quality-guidelines.html>; last accessed on 7/2/2026.

⁵ <https://www2.gov.bc.ca/gov/content/environment/air-land-water/water/water-quality/water-quality-guidelines/approved-water-quality-guidelines>; last accessed on 7/2/2026.

⁶ <https://www.ontario.ca/page/water-management-policies-guidelines-provincial-water-quality-objectives>; last accessed on 4/9/2026.



Table 18-5: Holding Pond Water Quality during the Project Period

Analyte	Result			2019 ECA (Monthly Concentration Limit; mg/L)	PWQO (mg/L)	MDMER (Monthly Mean Concentration; mg/L)
	Min	Max	Average			
Arsenic	<u>0.026</u>	0.173	0.108	0.5	0.005	0.10
Antimony	0.0053	0.0091	0.0072	-	0.02	-
Cadmium	0.00018	0.00028	0.00023	-	0.0001-0.0005 ^a	-
Cobalt	<u>0.106</u>	<u>0.206</u>	<u>0.156</u>	-	0.0009	-
Copper	<u>0.0094</u>	<u>0.0167</u>	<u>0.0131</u>	0.3	0.005 ^a	0.10
Lead	0.00052	0.00083	0.00068	0.2	0.003-0.005 ^a	0.08
Iron	<u>0.56</u>	<u>0.70</u>	<u>0.63</u>	1.0	0.3	-
Nickel	<u>0.072</u>	<u>0.122</u>	<u>0.097</u>	0.5	0.025	0.25
Molybdenum	<u>0.108</u>	<u>0.134</u>	<u>0.121</u>	-	0.04	-
Vanadium	0.0050	<u>0.0073</u>	<u>0.0062</u>	-	0.006	-
Zinc	<u>0.046</u>	<u>0.087</u>	<u>0.066</u>	0.5	0.02	0.40

Notes:

BOLD – exceeds the MDMER Monthly Mean Concentration

Underline – exceeds the PWQO

^a PWQOs for cadmium, copper, and lead are hardness based. Water quality simulations predict an average hardness of 146 mg/L as CaCO₃ (seasonal range of 45 to 194 mg/L). For copper, the PWQO of 0.005 mg/L (applicable above 20 mg/L hardness) is assumed to apply throughout the year. For cadmium and lead, the ranges in values applicable within the simulated hardness range are shown.

18.12 Water Treatment Plant Infrastructure

The Grey Fox water treatment requirements are informed by the water and mass balance evaluation as described in Section 18.11. The annual average discharge flow rate and water quality were used as the basis for the conceptual water treatment process design, capital cost estimate, and annual operating cost estimate.

18.12.1 Flow Rate

The Black Fox holding pond outflow is predicted to vary between 2,200 and 4,700 m³/day with an annual average of approximately 3,300 m³/day. The nominal hydraulic capacity of the water treatment plant would be based on this annual average flow rate. Over the course of a year, variability in water inputs to the site is largely dependent on precipitation, evaporation, and snowmelt levels. Although the available water storage capacity within the holding and treatment ponds is designed for wet weather events and not anticipated to be sufficient to accommodate storage during extended high-water input periods (e.g., during snowmelt), the excess during spring snowmelt can be accommodated in the Black Fox pit. As such, the average discharge rate of 3,300 m³/day can be maintained throughout the year.



18.12.2 Target Constituents

Target constituents for treatment were identified in the water and mass balance evaluation as described in Section 25. Water quality predictions were compared with potentially relevant water quality standards to determine potential exceedances. The following constituents were identified as potentially requiring treatment based on comparison of average and maximum annual values against the site's 2019 ECA and PWQOs.

- Arsenic
- Cobalt
- Copper
- Iron
- Nickel
- Molybdenum
- Vanadium
- Zinc

The average and maximum concentrations in the holding pond water are summarized in Table 18-5 above.

All of the listed constituents exceed at least one of the applicable standards listed above. In addition to the constituents listed above, discharge water quality is expected to be subject to limits for total suspended solids and pH. The conceptual water treatment system design will consider these additional limits in configuring the treatment process.

Any changes to the regulatory framework that impact the target constituent list or discharge requirements would be reflected in the water treatment process design as it advances through design and permitting.

18.12.3 Conceptual Design

The conceptual water treatment process consists of two primary stages to address the target constituents listed above. The proposed treatment technology is based on precipitation, primarily to remove iron and associated constituents through a two-step process as described below.

Step 1 will consist of iron oxidation and greensand filtration. This step would oxidize any ferrous iron to ferric iron to induce precipitation at the circumneutral pH. Minor amounts of arsenic and molybdenum may co-precipitate with iron during this step. The precipitated solids would be removed using a greensand filter, producing a low total suspended solids (e.g., < 10 mg/L). This would require continuous injection of an oxidizer, typically sodium hypochlorite or oxygen, into the feed stream to oxidize the reduced iron and other metals. The filters would require periodic backwash to remove accumulated solids and maintain permeability. Backwash solids would require thickening and dewatering prior to final disposal.

Step 2 will consist of ion exchange to remove the residual low-level metals and meet discharge requirements. Given the range of potential constituents, the ion exchange step would include a strong base anion (SBA) resin to remove the oxyanion species (arsenic, molybdenum, and vanadium) and a strong acid cation (SAC) resin to remove trace metals (copper, cobalt, iron, nickel, and zinc). Both resin types may be regenerated using a sodium chloride brine, which represents the primary input to this stage of the treatment



process. Brine may be made up directly from solid sodium chloride (NaCl) salt at the site. Spent regenerant solution would require treatment or offsite disposal as a waste brine. As an alternative, non-regenerable specialty resins could be used, in which case the primary residual would be exhausted resin rather than spent regenerant. At this stage of the conceptual process development, it is assumed that spent brine will be disposed of offsite. If onsite treatment is required due to cost or limited availability of offsite brine treatment, it may be evaluated at a later time.

These two steps are not anticipated to substantially affect discharge pH or total suspended solids. If additional conditioning is required to meet applicable discharge requirements, additional steps such as pH adjustment and alkalization may be included.

Waste residuals collected from steps 1 and 2 would require temporary storage prior to onsite treatment or offsite disposal. Solids are expected to be thickened and dewatered prior to disposal. Sludge will first be thickened using a high-rate plate clarifier, followed by dewatering using a filter press. Dewatered sludge can then be disposed at a designated land repository at the site subject to assessment, design considerations, and approval, as may be required.

And overall conceptual design is provided in Figure 18-13.

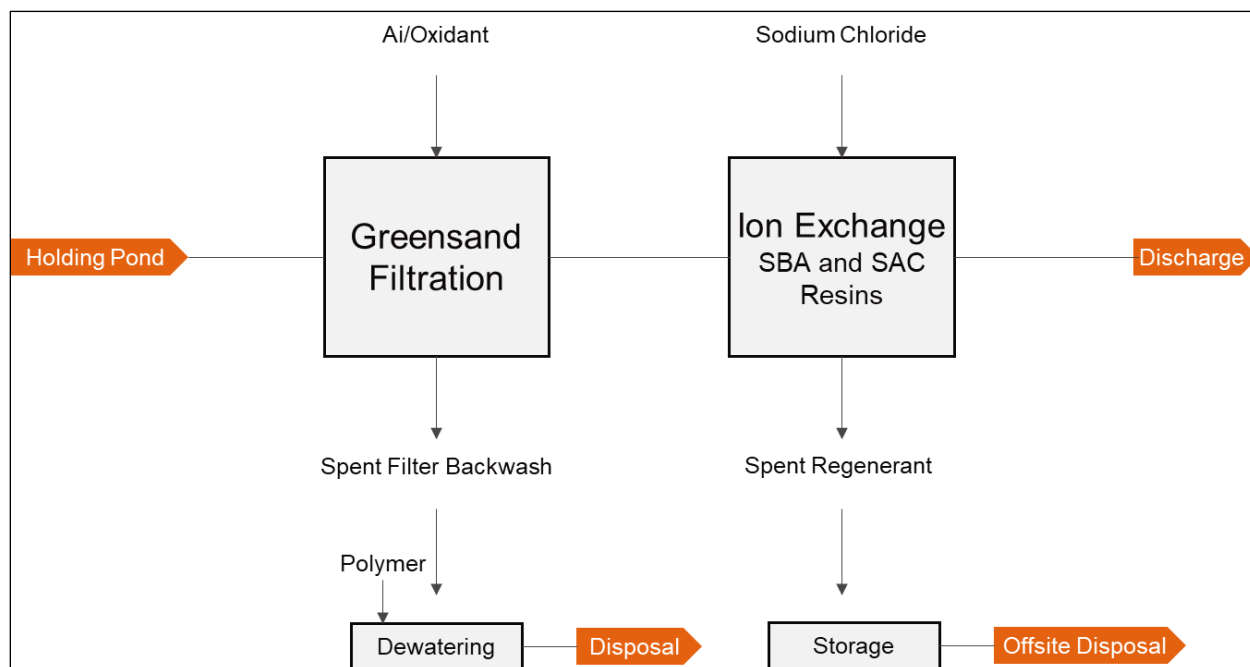


Figure 18-13: Water Treatment Plant Conceptual Design

18.12.4 Implementation and Operations

The water treatment plant will operate under a series of treatment trains.

The water treatment plant will consist of two parallel treatment trains for greensand filtration followed by ion exchange. Each train will have a capacity of approximately 1,650 m³/day to accommodate the design flow



rate of 3,300 m³/day. Constructing the system in two 1,650 m³/day trains would allow for more flexible operation during low flow periods.

Filtration will consist of pressurized filter vessels with the appropriate chemical storage and injection pumps, as well as water pumps for backwashing. Backwash solids will be separated from the backwash water in a dedicated tank using a dewatering polymer to aid solids separation. Supernatant liquid may be returned to the head of the filtration units, and solids will be dewatered using either a screw press or filter press. Dewatered solids are expected to consist primarily of iron solids and may be disposed of in a suitable on-site or offsite repository.

The ion exchange step would be completed through a series of pressurized SBA and SAC resin beds. The ion exchange system would include a brine solution makeup tank and supply pump to regenerate the resin beds. Spent regenerant brine would be stored in a separate tank for offsite disposal. A single storage tank would be used to store brine from both treatment trains.

Each train will consist of a dedicated containerized treatment system for each step in the treatment process (i.e., one container for greensand filtration and a second for ion exchange). Each container will house the primary filters as well as the chemical injection, backwash, and resin regeneration equipment. Treatment residuals will be managed, processed, and temporarily stored either in separate containers or in an external insulated and heat-traced tank.

Containerized treatment systems offer the advantage of off-site fabrication and delivery of complete systems, requiring minimal on-site mechanical and electrical assembly beyond service connections for water and power. Each individual container can be climate-controlled to account for temperature extremes at the site, if required; though, a climate-controlled building enclosure has been assumed for conceptual design purposes. Additional heat tracing and insulation of connection lines and external infrastructure (e.g., brine storage tank) would be completed on-site, as needed.

Ongoing operations would require the following inputs:

- Sodium hypochlorite – continuous injection to concentration of approximately 10 mg/L based on the influent iron concentration.
- Sodium chloride – used as a regenerant for the SBA and SAC resins.
- Dewatering polymer – used to thicken solids removed from the greensand filters during backwashing.
- Power (480 V - 3-phase service) – Process and backwash water pumps, chemical feed pumps and mixers, dewatering equipment, building HVAC and lighting, heat trace, instrumentation and controls.



19 Market Studies and Contracts

19.1 Market Studies

Gold is traded freely at widely known prices, consequently, the prospects for the sale of any production of the metal are virtually assured. A market study for the saleability of gold products or for the expected price of gold was not undertaken.

Based on monthly closing data from April 2024 to April 2026, the approximate two-year trailing average gold price is **US\$3,000 per ounce**. This period reflects a rapid rise in gold prices, starting from approximately \$2,285 in April 2024 and exceeding \$4,700 by April 2026.

Note: The calculation is based on monthly spot closing prices available as of April 2026.

19.2 Contracts

The terms contained within any future sales contracts are expected to be typical and consistent with standard industry practice and contracts for the supply of doré. Currently there are no contracts in place for the Grey Fox Project.

The selling costs related to the refining of the doré to produce saleable gold and to the doré shipping cost estimates used in this analysis were benchmarked based on existing McEwen contracts in the same region as Grey Fox. Refer to Table 19-1.

Table 19-1: Selling Costs for Gold Doré

Description	Unit	Value
Refining Cost (Au)	US\$/oz	\$0.41
Gold Payable	%	99.95
Doré Shipping Charge	US\$/oz	\$2

19.3 QP Opinion on Markets and Contracts

The QPs note the following:

- McEwen can market the doré products produced from the Grey Fox Project; and
- The terms contained within the sales contracts are based on existing McEwen contracts.



20 Environmental Studies, Permitting and Social or Community Impact

20.1 Existing Environment

20.1.1 Location and Land Use

The Eastern Properties of the Fox Complex is located in Hislop Township, approximately 10 km east of Matheson, Ontario, within the District of Timiskaming. The site falls within the geological setting of the Porcupine-Destor fault zone, a region recognized for its gold deposits with considerable current and historical mineral exploration and mining activity.

Land use is governed by the Township of Black River-Matheson Official Plan (MMM Group, 2017), which zones the broader area as “rural” with the Black Fox Mine site designated as an active mining area. The Froome and Grey Fox deposits fall within a mineral aggregate resource area centred on Highway 101. Prior to the 1990s, predominant land uses were hay production, pasturing, and forestry. The site lies within the Abitibi River Forest (Management Unit No. 110) and Wildlife Management Unit 28.

20.1.2 Climate

The site climate is typical of the northern boreal forest. Extended sub-zero temperatures occur from November through March; July is the warmest month and January the coldest. Maximum rainfall occurs in July and maximum snowfall in December. Prevailing winds are from the south and northwest in winter, and from the south and west in summer. Mean annual precipitation is approximately 840 mm/yr based on long-term records (1956-2019) from the Timmins Climate Station (Wood, 2022a), with roughly equal contributions from rain and snow.

20.1.3 Soils and Surficial Geology

The region is characterized by flat glaciolacustrine clay plains interspersed with rock knobs, ancient raised beaches, eskers, and other outwash deposits. Overburden in the Fox Complex area ranges from 10 to 60 m in thickness (averaging approximately 20 m) and consists of silty clay at surface, underlain by variable silty sand and till, with a basal till resting on weathered bedrock. Flat topography, low permeability, and cool climate have contributed to the formation of an overlying peat layer over much of the clay plain east of the site. A series of north-south trending esker complexes, composed of sorted sands, gravels, cobbles, and boulders, are present throughout the region; an esker formation intersected during open pit mining at Black Fox is a significant contributor to groundwater inflow to the pit.

20.1.4 Surface Water

The Fox Complex sits at the northern end of the Black River watershed (total drainage area 1,372 km²). Two sub-catchments are directly associated with the Fox Complex: the Pike River (westward flow through lacustrine plains and an esker ridge) and Salve Creek (southern portion of the Fox Mine property). Surface



water quality is monitored at multiple stations in accordance with site ECAs and MDMER; treated effluent currently meets all applicable discharge requirements. Long-term background data is available for the Pike River, Froome Lake, Lawlar Lake, Leach Lake, and several tributary stations.

20.1.5 Groundwater

The hydrogeological setting of the Fox Complex is dominated by the overburden sequence described in Section 20.1.3 (clay, silt, sand/till, basal till) and underlying weathered and competent bedrock. Hydraulic conductivities are lowest in the silty clay unit and highest in the esker / alluvial formation. Regional groundwater flow is controlled by topography, with flow generally towards local surface water features. Active dewatering of the Black Fox open pit and underground workings has modified local groundwater gradients; monitoring indicates that nearby Froome Lake is hydraulically separated from the mine workings by low-permeability basal sediments.

A comprehensive groundwater monitoring program (monitoring wells at Black Fox, Froome, and Grey Fox) confirms all groundwater quality consistently below permitted limits with no adverse temporal trends identified. Local groundwater users within approximately 1 km of mine workings have been identified; none rely on groundwater from within the modelled zone of mining influence.

20.1.6 Ecological Setting

Vegetation and wildlife communities at the Fox Complex are typical of the mixed-boreal forest region of northeastern Ontario. Upland communities include deciduous, mixed-wood, and coniferous forest in various stages of succession, with trembling aspen and balsam poplar as dominant species. Wetland communities include swamp, fen, and marsh-type habitats. The majority of flora and fauna species are rated common to very common (provincial and global rarity rank 4 or 5, per NatureServe) within their respective habitats.

Baseline surveys documented 88 breeding bird species typical of the southern boreal forest. Thirteen avian Species at Risk are known to occur within the broader Kirkland Lake District; no species listed under the federal Species at Risk Act were detected during field surveys at the time. Candidate significant wildlife habitats present at the project include bat hibernacula, bat maternity colony roost habitat, amphibian breeding habitats (wetlands), mast-producing areas, and amphibian movement corridors. The surrounding landscape supports moose, black bear, white-tailed deer, beaver, and common furbearers. Appropriate surveys and registration / mitigation measures will be completed prior to development.

20.1.7 Aquatic Setting

The Fox Complex discharges surface effluent to the Pike River through a 2.4 km pipeline, with the river serving as the receiving aquatic environment. The Pike River is a naturally mineral-rich, turbid, moderately hard, circumneutral northern watercourse with elevated background aluminium, ammonia, and iron, and sediments naturally above the LEL for chromium, iron, manganese, and nickel across both upstream and downstream areas. Effluent disperses rapidly—an estimated 2.4% of end-of-pipe concentration within 250 m downstream—and six phases of MDMER Environmental Effects Monitoring have found no ecologically significant mine-related effects on water quality, sediment, the benthic community, or fish, with arsenic the



only constituent showing a modest, localized exposure-area signal that remains below applicable guidelines.

20.2 Site Water Quality and Geochemistry

Static and kinetic tests were conducted to evaluate the chemistry of Grey Fox mine materials and the potential for release of metals and acidity from development rock. Shake flask tests were conducted as part of a geochemical characterization static testing program conducted on Grey Fox drill cuttings (Wood 2022b, references therein). Although the method used was not specified, this test typically involves reacting solids in deionized water in a 3-parts water to 1-part solid ratio for 24 hours, per Mine Environment Neutral Drainage (MEND) guidance (Price, 2009). Tests were conducted on 84 samples covering the seven major geologic rock types encountered at Grey Fox. The sample results were averaged within geologic unit, and then a geologic-unit-weighted average was calculated to represent mixed/ROM rock at Grey Fox.

In addition to the shake flask tests, humidity cell tests were conducted on Grey Fox drill cuttings collected from exploration boreholes (Wood, 2022b; WSP, 2024). A total of 11 humidity cell tests were conducted representing the four most abundant geologic rock types, including Massive Mafic Volcanic, Pillowed Mafic Volcanic (PMV), Variolitic Mafic Volcanic, and Sedimentary (SED).

One of the key objectives of the humidity cell tests was to evaluate potential for mine materials to become acid generating. Of the geologic rock types at the site, acid-base accounting results indicated that the presence of potentially acid-generating (PAG) material was particularly associated with SED and PMV geologic units, although all units primarily contained non-PAG (NPAG) material (Wood 2022b). For this reason, the humidity cell tests for SED and PMV material included both PAG and NPAG material to evaluate the potential for PAG materials to go acidic over time. The humidity cell tests were run for between 84 and 131 weeks. Over this time, two humidity cells (SED cell HC-02 and PMV cell HC-03) went acidic; however, based on a comparison of acidity generated in these two cells relative to the alkalinity generated in other cells, coupled with a material mass balance based on acid-base accounting results and anticipated waste rock tonnages for different geologic units, it was determined that the mixed ROM material would maintain net-alkaline. Further, the humidity cell test results indicated that all materials regardless of their potential to go acidic exhibited similar constituent leaching behavior (particularly for oxyanions including arsenic and molybdenum).

Per MEND guidance (Price 2009), humidity cell tests are typically run until constituent concentrations are stable for at least five weeks, noting that at least 40 weeks may be required. The humidity cell tests conducted for Grey Fox (between 84 and 131 weeks) were run until constituent concentrations stabilized. This long timeframe is often required to evaluate depletion of acid neutralization potential, noting that many sulfidic rock types start out acid neutralizing but become acid generating after acid neutralization potential is depleted. Because of this, concentrations of several constituents (including acid soluble metals) can increase as humidity cells progress, and the long-term stabilized concentrations are used to develop geochemical source terms. However, for the rock types at Grey Fox, several constituents were higher in the first few weeks of the humidity cell tests due to the washout of soluble constituents from freshly exposed rock surfaces. This “first flush” of constituents is anticipated to be an important influence on Grey Fox contact water quality during operations, and therefore both the Week 0 and average of post-Week 30



humidity cell test concentrations were considered in developing geochemical source terms. Additional details are provided in Section 18.11.1.

20.3 Environmental Setting

Environmental baseline (pre-development) investigations were completed for a number of the Fox Complex properties prior to mine development and provide a basis for future closure comparison. These studies establish background conditions for surface water, groundwater, aquatic ecology, and terrestrial ecology. Key baseline investigations completed include the following:

- Pre-development environmental baseline investigations were completed for all Fox Complex properties covering surface water, groundwater, aquatic ecology, and terrestrial ecology (AMEC, 2008, 2012, 2014; Golder, 2015; BZ Environmental, 2006; supplementary data ongoing at Fox West properties). These studies establish background conditions for future closure comparison. All relevant baseline references are listed in Section 27.

As required by existing permits and approvals, McEwen operates a comprehensive environmental monitoring program at the Fox Complex. Information collected supports current compliance and future permitting application requirements. Currently, all McEwen Fox Complex properties are in compliance with all applicable regulatory requirements, including Ontario Regulation 35/24 – Mine Development and Closure and the Mine Rehabilitation Code of Ontario. Financial assurance, as set out in Part VII of the Mining Act and Ontario Regulation 35/24, has been provided for required closure activities. Refer to Section 20.5 for further details on closure cost estimates and financial assurance.

20.4 Regulatory Framework

20.4.1 Impact / Environmental Assessment

Mining projects in Canada can be reviewed under one or more Environmental Assessment (EA) or Impact Assessment processes, whereby design choices, environmental impacts, and proposed mitigation measures are reviewed to determine how best to proceed through the approvals and permitting process. Entities involved in the review process normally include government agencies, Indigenous communities, interested parties, and the general public.

Based on proposed activities at the Grey Fox, Stock, and Western Properties, a Federal Impact Assessment process under the Impact Assessment Act is not required as per Sections 18 and 19 (Mine and Metal Mills) of the Physical Activities Regulations (SOR/2019-285). The Ontario Environmental Assessment Act regulates the environmental assessment process for mining projects in Ontario and requires that potential environmental impacts be assessed before a project begins. Mining projects and infrastructure developments on Crown land or using Crown resources may be required to complete a Class EA assessment with the Ontario Ministry of Mines.



20.4.2 Environmental Approvals

Federal environmental approvals associated with mine site developments generally correlate to the Fisheries Act and the Canadian Navigable Waters Act. Based on current information, federal environmental approvals under these Acts are not expected to be required. Environment and Climate Change Canada regulates the mine effluent under the federal MDMER under the Fisheries Act.

A large number of active environmental approvals are currently held for individual Western Properties. A comprehensive listing of existing active approvals is provided in Table 20-1. Amendments to existing approvals, or new approvals, will be required depending on final project designs for Grey Fox and associated infrastructure. Table 20-2 provides a preliminary listing of applicable approvals and amendments that may be required as the Grey Fox, Stock, and Western Properties progress.

Table 20-1: Existing Environmental Approvals

Site	Regulatory Agency	Reference No.	Type	Issue Date	Expiration	Status / Notes
Black Fox	MECP	0328-8XVPXT	ECA-ISW	January 23, 2019	N/A	Active – Amendment required for effluent transfer to Black Fox holding pond for Grey Fox mine water
Black Fox	MECP	2126-DRUJZW	ECA-Air	April 27, 2026	N/A	Active – Amendment completed for Grey Fox Operations
Black Fox	MECP	6824-9VCNXB	PTTW	October 26, 2020	October 26, 2030	Active – No change anticipated
Black Fox	MEM	N/A	Closure Plan	June 2018	N/A	Active – Amendment required for Grey Fox closure costs (Black Fox Closure Plan Amendment 2026 <i>filed</i>)
Grey Fox	MECP	Pending	PTTW	Pending	TBD	Pending – New permit required for underground mine dewatering
Stock Mill	MECP	4371B2KHZT	ECA-ISW	August 2, 2018	N/A	Active
Stock Mill	MECP	1357-A4KKFS	ECA-Air	December 11, 2015	N/A	Active
Stock Mill	MECP	6377-88UJE5	PTTW	September 18, 2018	September 18, 2028	Active



Site	Regulatory Agency	Reference No.	Type	Issue Date	Expiration	Status / Notes
Stock Mill	NDMNRF	N/A	Closure Plan	June 2018	N/A	Active

Notes: ECA-ISW: Environmental Compliance Approval – Industrial Sewage Works; ECA-Air: Environmental Compliance Approval – Air and Noise; MECP: Ministry of Environment, Conservation and Parks; MEM: Ministry of Energy and Mines; PTTW: Permit to Take Water; N/A: not applicable (no expiry date); NP: not provided.

Table 20-2: Anticipated Provincial Approval/Amendment Requirements

Approval	Responsible Ministry	Description
Closure Plan Amendment	MEM	Mine construction / production and closure for Grey Fox / Stock, including updated financial assurance. Offline dams above the high-water mark of local watercourses (e.g., TMA).
Environmental Compliance Approval(s) – Air and Noise	MECP	Release of air emissions and noise from haul trucks, process plant, refuelling / oil / lubrication areas, administration complex, and onsite laboratory.
Environmental Compliance Approval – Waste Disposal Site / Waste Management	MECP	Operation of a waste transfer site; transportation of waste materials offsite and onto Provincial highways.
Environmental Compliance Approval – Domestic Sewage	MECP	Domestic sewage treatment plant. Not expected to be required if pre-approved mobile sewage treatment systems are used.
Environmental Compliance Approval(s) – Industrial Sewage Works	MECP	Construction of mine / mill water treatment system(s) discharging to the environment; includes TMA, ponds, mine water, stormwater, stockpile runoff, and fuelling areas.
Permit(s) to Take Water – Ontario Water Resources Act	MECP	Taking of groundwater or surface water (>50,000 L/day) for process water balance and mine dewatering. May also be required for construction dewatering.
Forest Resource License(s) – Crown Forest Sustainability Act	MNRF	Cutting of Crown merchantable timber for site development or transmission line construction.
Work Permit(s) – Lakes and Rivers Improvement Act / Public Lands Act	MNRF	Work / construction on Crown land. Dam construction on any lake or river requires written approval for location and plans and specifications.
Township of Black-River Matheson	Municipality	Construction-related building permits as required.

20.4.3 Permitting Critical Path and Schedule

The critical path for commencement of Grey Fox underground development is anticipated to be the Closure Plan Amendment, the Grey Fox PTTW for dewatering of underground workings and ISW-ECA amendment to the Black Fox mine effluent discharge (0328-8XVPXT) . The Grey Fox PTTW will be a Category 3



application under Ontario Regulation 387/04, requiring a full hydrogeological study prepared by a QP given the scale of anticipated dewatering (estimated at up to 1,938 m³/d, equivalent to approximately 1.94 million L/d) and determination of influence to nearby Pike River and Salve Creek. Category 3 PTTW applications are technically complex and typically require a minimum of 6 to 9 months from application submission to permit issuance, exclusive of the time required for data collection, hydrogeological modelling, and supporting report preparation. The Closure Plan Amendment, to incorporate Grey Fox mine production and associated updated closure costs, requires filing with MEM and confirmation of updated financial assurance—typically a 3-to-6-month review period following submission of a complete application.

ECA-ISW amendments at Black Fox (for effluent transfer, treatment and discharge to handle Grey Fox mine water) will be required prior to commencement of mine operations; this application should be submitted in parallel with the PTTW process. The key requirement for the ISW-ECA related to Grey Fox transfer, treatment and discharge will be the establishment of new effluent criteria for the Black Fox water treatment plant in accordance with MECP Procedure B-1-5 and the development of an acceptable surface and groundwater monitoring plan(s) to MECP. Both these pre-application components must be developed and accepted by MECP before the ISW-ECA application is submitted. MECP's service standard for complete and acceptable ISW-ECA applications is 12 months after submission. McEwen's intent is to progress all key permit applications concurrently and as soon as necessary project design information is available. McEwen has an established regulatory record at the Fox Complex that supports efficient permit processing.

20.5 Environmental Aspects and Sensitivities

20.5.1 Water Discharge

Contact water from the Grey Fox site will be collected and stored on-site and then transferred to the Black Fox mine for treatment and discharge into the Pike River following treatment at the currently permitted discharge point in accordance with an MECP amended industrial sewage works Environmental Compliance Approval. Treatment within the holding pond prior to discharge involves sulphuric acid addition to lower pH, and ferric sulphate dosing to precipitate arsenic. A coagulant (e.g., poly-aluminum chloride) may also be used to enhance the settling of suspended solids in the Holding Pond, if required. Additional treatment options are being considered to meet anticipated approval limits, including arsenic, zinc, nickel, molybdenum and copper. Loading values are dependant on seasonal fluctuations and assimilative capacity of the Pike River. Modelling and discussions with stakeholders is on-going.

To reduce the mine water inventory, non-contact runoff at the Grey Fox site will be diverted using perimeter berms to allow runoff to naturally flow off site. Contact water collection ditches to collect runoff and toe seepage will be constructed around the perimeters of project components, including the waste rock stockpile, ore stockpile, and laydown area. Areas where fuel storage or handling may occur will be isolated to avoid potential cross-contamination of petroleum hydrocarbons. Contact water collection ditches will drain by gravity to the extent practicable to reduce pumping requirements. Excavated sumps and pumps will be used where site topography is not conducive to draining by gravity. The contact water collection ditches will discharge the collected water to water management ponds.



The water management ponds and collection ditches will be designed to contain water from a 1 in 100-year AEP of a 24-h duration storm event, considering snowmelt. The spillways of the ponds will be designed for the climate change adjusted 1 in 100-year annual exceedance probability of a 24-h duration storm event. Two ponds are anticipated to be required for water management, a stockpile pond and a holding pond. The holding pond will have a permanent pool to promote settling and provide extended detention and storage of settled materials. The holding pond will be designed to receive pumped inflows from the underground dewatering in addition to site contact water and will serve as a source of water for service underground water requirements and site surface water needs (e.g., dust suppression).

Ore will be processed, and tailings disposed of at existing processing and tailings storage facilities at the Stock Mill site.

20.5.2 Site Plan Layout

The Grey Fox mine works including waste rock piles and water management infrastructure will be designed to avoid watercourses and waterbodies to address concerns with a Fisheries Act MDMER Schedule 2 prohibition regarding the deposition of mine waste in waterbodies frequented by fish.

20.6 Social and Community

The McEwen Fox Complex is located within territories traditionally used by Indigenous peoples. Consultation and engagement activities with Indigenous Nations regarding the Fox Complex have been ongoing since 2006. Nations consulted include:

- Apitipi Anicinapek Nation (formerly Wahgoshig First Nation) – Reserve located approximately 23 km east of the Fox Complex and the closest Indigenous community to the Site;
- Matachewan First Nation – Reserve located approximately 60 km southwest;
- Métis Nation of Ontario – Timmins Community Council;
- Métis Nation of Ontario – Northern Lights Council; and
- Temiskaming Métis Council.

An Impact Benefit Agreement (IBA) is currently in place with the Apitipi Anicinapek Nation, the Indigenous community in closest proximity to the Black Fox Complex. The IBA includes a pathway for discussing approval amendments, considerations and related potential impacts prior to submission. Concerns or questions raised are addressed within this process. In accordance with the IBA, the filed Closure Plan Amendment for the Black Fox Complex was provided to Apitipi Anicinapek Nation for review prior to submission.

McEwen continues to discuss all projects that require government permits or approvals with First Nations in advance of submission. Details of the agreements and discussions with local Indigenous communities are provided in Section 4.3.2 of this technical report.

Consultation and engagement will also be required with other stakeholders, including regulatory agencies, local communities, and residential and industrial neighbours where development is proposed in proximity to existing properties or operations. Community members will be engaged as appropriate.



20.7 Mine Closure

McEwen has prepared and *filed* a Closure Plan Amendment for the Fox Complex, which was *filed* with the Ministry of Energy and Mines (Formerly Ministry of Northern Development, Mines, Natural Resources and Forestry) in 2026. The Closure Plan Amendment supersedes all previously submitted closure plan amendments and is to be considered in its entirety. The overall objectives of site rehabilitation and closure measures are to stabilize the site and to reduce, mitigate, and/or eliminate risks of hazards and the release of mine-related constituents into the surrounding environment.

Site reclamation work will be fulfilled progressively during active operations as practicable, consistent with industry best practice and the *filed* Closure Plan Amendment. The closure plan addresses three phases: Temporary Suspension, State of Inactivity, and Close Out. The post-closure land use for the site is forest and wildlife habitat, with potential for forestry operations, mineral exploration, agriculture, and recreational uses.

20.7.1 Conceptual Closure

Table 20-3 summarizes preliminary closure concepts for major infrastructure and site features at the Fox Complex per the filed Closure Plan Amendment.

Table 20-3: Closure Approaches

Feature	Concept Summary
Open Pit (Black Fox)	Remove all infrastructure and equipment from within the open pit. Reshape if needed and revegetate overburden slopes to a stable condition. Allow pit to fill naturally with water via seepage and runoff. Ramps will be barricaded and a safety berm established around the perimeter. Construct overflow channel from the open pit to a tributary of Salve Creek to the north. Batch treatment of pit water as appropriate.
Underground Mine (Black Fox, Froome, Grey Fox, Stock, Fuller)	Remove all infrastructure and equipment presenting environmental hazard or with salvage value. Allow workings to flood naturally via groundwater seepage. Seal all entrances to underground workings.
Buildings, Machinery, Equipment and Infrastructure (All Sites)	Salvageable machinery and equipment dismantled and removed from site. Remaining items managed in accordance with regulatory requirements at an approved onsite landfill or licensed offsite facility. Above-grade concrete structures broken to near-grade; rebar cut flush with surface; concrete structures infilled with clean mine rock if required.
Chemicals, Petroleum Products and Explosives (All Sites)	All petroleum products, chemicals, and explosives removed from site and transported to a licensed disposal facility. Contaminated soil handled in accordance with regulatory requirements.
Onsite Infrastructure – Roads and Power (All Sites)	Roads not needed in the long term scarified and seeded. Onsite power distribution lines and associated materials dismantled and deposited in an approved landfill if unable to be sold or re-used.
Tailings Management Area (Stock)	Tailings surface progressively covered with soil and revegetated. Tailings pond drained; water treated prior to discharge.



Feature	Concept Summary
Waste Rock (Black Fox)	Progressive reclamation during operations. Regrading of slopes to 3:1 (overall) where required. Placement of overburden cover followed by revegetation. Pumping of stockpile runoff / seepage to open pit during State of Inactivity.
Waste Rock (Grey Fox, Stock, Fuller)	Regrading of stockpiles to acceptable slopes. Waste rock stockpile surfaces will be covered and revegetated with material that meets the long term cover requirements for reactive material as per the Ontario Mine Rehabilitation Code.

Rehabilitation measures will include:

- The waste rock pile will be built with overall embankment slopes that are adequate for long-term physical stability so that no re-contouring is required at closure. Available stockpiled soil will be used to vegetate waste rock piles as practical to conform to the local landscape. Mine rock types that pose a chemical stability risk will be managed in accordance with the management plan that will be developed as part of the closure plan.
- Final removal of the water management infrastructure will be conducted once no longer required.



21 Capital and Operating Costs

21.1 Methodology

The capital and operating cost estimate meets the criteria to be classified as a Class-3 estimate, as defined by AACE International with an approximate accuracy of +20%/-15%. All costs included in the estimate are reported in Q1 2026 US dollars (US\$).

The cost estimates reflect a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope is described and quantified within material takeoffs in a series of line items. The costs have been compared against project benchmarking, along with McEwen existing operations in the area.

21.2 Total Capital Cost Estimates

The total combined capital cost estimates was prepared by Stantec, DRA, and Arcadis, with contributions from McEwen. The total pre-production capital expenditure (CAPEX) is estimated at \$180.8 million, followed by \$173.8 million in sustaining capital costs (SUSEX), and \$7.4 million for closure. The capital cost estimate is divided into three broad scope areas:

- Mining
- Processing
- Other

A summary of the estimated LOM CAPEX is presented in Table 21-1, with detailed cost breakdowns summarized in the sub-sections that follow.

The project period CAPEX is defined as all costs incurred from 2026 to the end of 2029, while SUSEX is incurred from 2030 through to the end of mine life. Pre-production CAPEX has been divided into the following:

- Engineering and procurement services.
- Non-development mining cost (e.g., equipment, construction, underground infrastructure).
- Mining development cost (including vertical development).
- Surface infrastructure.
- Pre-production operating costs (operating costs that occur during the project period).
- Contingency.

SUSEX are related to the following:

- The initial equipment fleet is planned to be leased by McEwen and included in the operating cost (refer to Section 21.3.1.5). The new equipment required for Grey Fox South, plus the planned rebuild and replacement equipment, is included in the sustaining capital cost.
- Mining development cost (including vertical development).
- Underground infrastructure construction.



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- Surface construction.
- Mine closure.

The cost presented in this technical report outlines an initial pre-production capital cost estimate of \$180.8 million, which includes a contingency of \$27.11 million and pre-production operating costs of \$4.2 million. Estimated LOM capital costs are presented in Table 21-1.

Table 21-1: Capital Cost Estimate (\$ millions)

Description	Project Period Capital Cost	Sustaining Capital Cost	Total Capital Cost
Mining			
Engineering	\$5.8	\$0.0	\$5.8
Procurement	\$4.4	\$48.0	\$52.4
Construction / Excavations	\$98.5	\$125.6	\$224.1
Indirects	\$31.8	\$0.2	\$32.0
Contingency	\$25.9	\$0.0	\$25.9
Total Mining	\$166.4	\$173.7	\$340.1
Surface Crusher Facility			
Engineering	\$0.3	\$0.0	\$0.3
Mechanical Equipment Supply + Install	\$1.7	\$0.0	\$1.7
Commodities	\$1.0	\$0.0	\$1.0
Indirects	\$1.0	\$0.0	\$1.0
Contingency	\$0.5	\$0.0	\$0.5
Total Surface Crusher Facility	\$4.4	\$0.0	\$4.4
Water Treatment Plant			
Engineering	\$0.7	\$0.0	\$0.7
Procurement	\$3.6	\$0.0	\$3.6
Construction	\$0.2	\$0.0	\$0.2
Indirects	\$0.4	\$0.0	\$0.4
Contingency	\$0.7	\$0.0	\$0.7
Total Water Treatment Plant	\$5.7	\$0.0	\$5.7
Total	\$176.5	\$173.7	\$350.3
Pre-Production Costs	\$4.2	\$0.0	\$4.2
Closure	\$0.0	\$7.4	\$7.4
Net Capital Costs	\$180.8	\$181.1	\$361.9

Notes:

- All figures are rounded to reflect the relative accuracy of the estimate.
- Totals may not sum due to rounding as required by reporting guidelines.
- Pre-production costs include operating costs incurred prior to any profits.



21.2.1 Mining Capital Costs

The total project period mining capital of \$166.4 million primarily includes surface construction, lateral and vertical mine development, underground infrastructure construction, as well as the purchase and installation of stationary mine equipment. The cost excludes the pre-production OPEX, estimated at \$4.2 million.

The total mining sustaining capital of \$173.7 million primarily includes lateral and vertical mine development, underground infrastructure construction, and mobile fleet rebuilds and replacement.

The estimated LOM CAPEX and SUSEX for mining-related activities are presented in Table 21-2.

Table 21-2: Mining Preproduction Capital and Sustaining Costs (\$ millions)

Item	Preproduction Capital (\$ millions)	Sustaining Capital (\$ millions)
Detailed Engineering	\$5.8	\$0.0
Procurement		
Contractor Mob/Demob	\$1.4	\$0.1
Mobile Equipment Rental	\$0.6	\$0.0
Mobile Equipment Rebuilds/Replacement	\$0.0	\$47.9
Packaged Equipment and Bulks	\$2.5	\$0.0
Total Procurement	\$4.4	\$48.0
Mine Construction / Excavations		
Underground Mine	\$45.6	\$117.3
Surface	\$42.3	\$6.7
Pre-Commissioning	\$0.8	\$1.1
Contractor Indirects	\$9.8	\$0.5
Total Underground Mine Construction	\$98.5	\$125.6
Construction Indirects		
Logistics and Freight	\$3.0	\$0.0
Capital/Commissioning/Operating Spares	\$2.0	\$0.0
Vendor Support	\$2.0	\$0.0
Total Construction Indirects	\$7.1	\$0.0
Project Indirects		
EPCM	\$24.7	\$0.2
Total Project Indirects	\$24.7	\$0.2
Contingency	\$25.9	\$0.0
Mining Totals	\$166.4	\$173.7



21.2.1.1 Underground Mine Construction / Excavations

Detailed Engineering

Summarizes the cost estimate to complete the detailed engineering for the following:

- Mine design and scheduling
- Underground facilities
- Backfill
- Ventilation
- Surface facilities
- Geomechanical modelling.

Procurement

The procurement includes the costs associated with the contractor indirects, short term equipment rental for early mine development, and package equipment, which equipment that isn't specific to a single construction activity, for example underground subs that would be moved, jumbo cable etc.

The initial mobile equipment fleet is included in the operating cost estimate as a lease. The procurement of fixed plant equipment, such as fans, pumps, compressors, are included in the facility cost under underground or surface construction.

Underground Construction / Excavations

The underground mine construction consists of all project period lateral and vertical excavations, and any underground facilities constructed. The underground lateral development and installations contractor will be responsible for the initial underground ramp, lateral and vertical (raising) development (in waste rock only) starting from the construction of the north access portal to the initial ore access at Whiskey Jack, and construction/installation of stationary mine infrastructure. Once the deposit is accessed at Whiskey Jack, McEwen's labour workforce will assume all underground development and construction activities.

The project period cost for the underground construction and excavations are presented in Table 21-3. See Table 16-43 for list of underground facilities included in the cost estimate.

Table 21-3: Underground Construction and Excavation during Project Period

Item	Cost (\$ millions)
Definition Drilling	\$9.9
Mine Development - Ramp/Lateral	\$17.9
Mine Development - Vertical	\$2.7
Underground Construction	\$15.1
Total	\$45.6



Surface Construction

Surface infrastructure construction capital is estimated to be \$42.3 million, and is exclusive of indirect and contingency costs. The surface construction costs during the project period include a portal, administration building, dry and security facility, surface maintenance shop, ventilation/heating plants, electrical substation, water collection ponds, access roads, and all earth works related to excavating and levelling the site. The project period cost for the surface construction activities are presented in Table 21-4.

Table 21-4: Surface Construction During the Project Period

Item	Cost (\$ millions)
Site Clearing, Roads, Bridges, Ponds, etc.	\$12.7
Surface Facilities	\$15.8
Surface Utilities	\$3.7
Portals with Fans	\$1.3
Ventilation Collars	\$1.4
Main Ventilation Fans	\$3.8
Pipeline from Grey Fox to Black Fox	\$3.7
Total	\$42.3

21.2.1.2 Construction Indirects

Indirect construction costs include freight and logistics for the delivery of equipment and materials to site, spare parts for fixed plant equipment, and vendor commissioning support for fixed plant equipment. Costs are estimated as a percentage of the total fixed plant equipment capital estimate, with the percentages applied as follows:

- Freight – 3%
- Spare parts – 2%
- Vendor support – 2%

21.2.1.3 Project Indirects

Indirect project costs account for all labour and expenses, and required indirect fleet and vehicle operating costs, associated with the execution of the project. The project team organization chart, labour forecasts, and fleet requirements are developed based on the project execution plan, and the costs are then developed from first principles.

21.2.1.4 Mining Sustaining Capital Costs

Sustaining Capital is associated with of the remaining lateral and vertical excavations, underground and surface infrastructure, equipment rebuilds and replacement, along with all indirects costs associated with the work completed during the project period. The annual sustaining capital costs are presented in Table 21-5.



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Table 21-5: Annual Sustaining Capital Costs (\$ million)

Item	Cost	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Procurement													
Contractor Mob/Demob	\$0.1	\$0.1	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0
Mobile Equipment Rebuilds/Repurchases	\$47.9	\$0.0	\$2.4	\$2.6	\$6.1	\$4.9	\$4.4	\$3.3	\$7.3	\$9.9	\$6.9	\$0.0	\$0.0
Underground Construction and Excavations													
Definition Drilling	\$29.7	\$1.0	\$3.1	\$1.2	\$5.1	\$4.4	\$1.0	\$5.2	\$2.8	\$3.9	\$1.9	\$0.1	\$0.0
Mine Development - Ramp/Lateral	\$44.9	\$1.4	\$4.7	\$1.8	\$7.7	\$6.6	\$1.6	\$7.8	\$4.2	\$6.0	\$2.9	\$0.1	\$0.0
Mine Development - Vertical	\$3.5	\$0.1	\$0.4	\$0.1	\$0.6	\$0.5	\$0.1	\$0.6	\$0.3	\$0.5	\$0.2	\$0.0	\$0.0
Underground Construction	\$39.2	\$1.3	\$4.1	\$1.6	\$6.7	\$5.8	\$1.4	\$6.8	\$3.7	\$5.2	\$2.5	\$0.1	\$0.0
Surface Construction													
Ventilation Collars	\$1.4	\$1.4	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0
Main Ventilation Fans	\$5.3	\$5.3	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0
Pre-Commissioning	\$1.1	\$0.0	\$0.2	\$0.0	\$0.2	\$0.2	\$0.0	\$0.2	\$0.1	\$0.1	\$0.1	\$0.0	\$0.0
Contractor Indirects	\$0.5	\$0.1	\$0.1	\$0.1	\$0.1	\$0.1	\$0.1	\$0.1	\$0.1	\$0.1	\$0.1	\$0.0	\$0.0
Project Indirects	\$0.2	\$0.2	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0	\$0.0
Total	\$173.7	\$10.8	\$15.0	\$7.5	\$26.5	\$22.5	\$8.5	\$23.9	\$18.5	\$25.6	\$14.5	\$0.4	\$0.0



21.2.2 Surface Crushing Facility

A primary crusher with a stacking conveyor will be located on the ore pad at the Grey Fox site. ROM ore will be trucked from either the North or South Portal and deposited near the primary crusher. A loader will be used to feed the crusher. Crushed ore will be placed in temporary stockpiles with the stacking conveyor, and then loaded into surface haulage trucks and hauled to the Stock Mill for processing.

Bulk earthworks costs are included with the surface construction costs presented in Table 21-2.

The estimate presented in Table 21-6 includes the cost of crusher, conveyor, and related equipment, engineering, indirect costs, and contingency.

Table 21-6: Surface Crusher Costs (\$ millions)

Item	LOM Total
Mechanical Equipment Supply and Install	\$1.7
Commodities	\$1.0
Contractor Indirects	\$0.4
Freight and Land Transport Cost	\$0.3
Spares and Inventory	\$0.1
EPCM	\$0.3
Commissioning and Vendor Representative	\$0.3
Contingency	\$0.5
Total Capital Costs	\$4.5

21.2.3 Water Treatment Plant Capital Costs

The capital cost estimate for the Grey Fox water treatment plant has been developed based on several key assumptions. The treatment system is assumed to be housed within a new 24 m x 46 m industrial metal featuring insulated metal panels, an HVAC system, and a concrete foundation designed to withstand anticipated site temperature extremes. The process configuration, as described in Section 18, is based on projected holding pond water chemistry and is designed to meet Ontario PWQO discharge limits. Allowances for design and engineering have been estimated at 20% of capital equipment costs, while project management and QA/QC have been estimated at 10% of the combined capital equipment and construction costs. Table 21-7 summarizes the water treatment plan costs.



Table 21-7: Water Treatment Plant Costs

Description	Cost (\$ millions)	Notes
Direct Costs		
Building	\$2.8	80' x 150' Industrial metal building and foundation
Filtration System	\$0.4	Includes initial filtration media
Containerized IX System	\$0.4	Includes initial resin load
Containerized Dewatering System	\$0.1	
Field Installation	\$0.2	Site civil, electrical connection, and piping interconnect
Indirect Costs		
Design and Engineering	\$0.7	See assumptions
Project Management, Procurement, and QA/QC	\$0.4	See assumptions
Contingency	\$0.7	See assumptions
Total	\$5.7	

21.2.4 Contingencies

A contingency factor was applied against all project capital costs at 20%, excluding engineering and indirect labour costs, resulting in a total estimated contingency of \$27.1 million.

21.2.5 Pre-Production Operating Costs

The pre-production costs include operating costs incurred within the project period associated with operating waste development and ore development/production, mine operating indirect costs, fixed plant maintenance, processing, and utilities. The pre-production cost total is \$4.2 million.

21.2.6 Closure

Closure costs have been estimated in the 2026 Closure Plan Amendment. The total estimated cost of closure activities, as described in the Closure Plan Amendment and reflected in the LOM financial model, is \$7.40 million. This estimate includes active care, passive care (long-term monitoring), and provision for periodic in-situ batch treatment of open pit water, post-closure.

It is noted that current closure cost estimates reflect the Black Fox Mine Closure as currently configured, including the Grey Fox deposit at its current advanced exploration footprint (the Grey Fox Advanced Exploration Closure Plan costs are included). This estimate does not yet incorporate the full mine production infrastructure associated with the Grey Fox mine as described in this technical report. A further Closure Plan Amendment will be required once Grey Fox mine design is finalized at future design study



stages, and this amendment will include updated closure cost estimates and a corresponding update to financial assurance.

21.3 Total Operating Cost Estimates

Stantec, DRA, and Arcadis, with contributions from McEwen, developed operating cost estimates to determine the annual production costs. Unit costs are expressed in total LOM and \$/tonne processed. All costs included in the estimate are reported in Q1 2026 US dollars.

Total LOM OPEX is estimated at \$1,398 million, exclusive of pre-production operating costs, refining and shipping and royalties. Expenditures that are non-capital costs that occur during the project period are classified as Pre-production operating costs. Pre-production operating costs incurred in 2028 only have been capitalized in the economic models (refer to Section 21.2.5).

The operating cost estimate is divided into the following broad areas:

- Mining
- Processing
- Tailings
- Wastewater Treatment

The operating costs detailed herein are based on mine and process plant design criteria and engineering and unit cost information derived from various sources.

A summary of the estimated LOM operating costs is presented in Table 21-8 and details of the cost breakdowns are summarized in the sub-sections that follow:

Table 21-8: Total Operating Cost Estimate

Operating Cost Item	LOM Total (\$ millions)	Cost \$/tonne Processed
Mining	\$1,013.3	\$107.7
Processing	\$267.0	\$28.4
Tailings	\$114.6	\$12.2
Water Treatment	\$2.8	\$0.3
Total	\$1,397.7	\$148.6

Note:

- All figures are rounded to reflect the relative accuracy of the estimate.
- Totals may not sum due to rounding as required by reporting guidelines.



21.3.1 Mining – Operating Costs

Mine operating costs have been built up from first principles, using detailed mine development and production schedules together with a variety of unit costs (quoted by suppliers, or using benchmark data and operational experience). The costs have been scheduled by period and built up by type for each basic mining activity (i.e. lateral development, stoping, fixed plant maintenance, utilities and general mine/indirect activities). Mine operating costs in the years prior to the processing of ore have been capitalized. A summary of overall mine OPEX is presented in Table 21-9.

Table 21-9: Summary of Mining OPEX

Operating Cost Item	LOM Total (US\$ millions)	Cost \$/tonne Processed
Underground Production	\$651	\$69.17
Mine Operations Indirect	\$145	\$15.40
Utilities (Power/Gas)	\$122	\$12.94
Fixed Plant Operating	\$22	\$2.34
Leasing - Equipment & Infrastructure	\$62	\$6.62
Project Period Mining Direct	\$12	\$1.26
Total	\$1,013	\$107.73

Notes:

- All figures are rounded to reflect the relative accuracy of the estimate.
- Totals may not sum due to rounding as required by reporting guidelines.

21.3.1.1 Underground Production

The underground production costs have been developed from first principles for all ore development and production and ore haulage related activities. Unit costs for development and production activities, were calculated from first principles and include labor, materials/consumables and mobile equipment operating. The unit cost calculations are provided for direct costs only. Varying unit costs are calculated for both the project and operating periods.

The labour component is based upon the crew size and the calculated performance per day to determine the total hours per unit of measure, which are then applied to the crew labour rates. Materials costs are based on the quantity take offs per unit, derived from the mine design criteria. Equipment operating is based on the calculated performance per day to determine the total hours per unit of measure, which are then applied to the equipment operating costs per hour.

See Table 21-10 and Note: PP = project period; OP = operating period

Table 21-11 for operating development and stoping unit costs during the project period and operating period, respectively .



Table 21-10: Operating Development Unit Costs

Description	Unit Cost (per m)
Contractor	
Stope Access - Single Heading (PP) - 4.0 x 4.3 m (WxH)	\$3,265.68
Stope Access - Multiple Heading (PP) - 4.0 x 4.3 m (WxH)	\$3,201.48
Sill Cut - Single Heading (PP) - 4.0 x 4.0 m (WxH)	\$3,119.31
Sill Cut - Multiple Heading (PP) - 4.0 x 4.0 m (WxH)	\$3,053.77
Sill Cut - Single Heading (PP) - 3.5 x 4.0 m (WxH)	\$2,865.44
Sill Cut - Multiple Heading (PP) - 3.5 x 4.0 m (WxH)	\$2,501.15
McEwen	
Stope Access - Single Heading (OP) - 4.0 x 4.3 m (WxH)	\$3,275.49
Stope Access - Multiple Heading (OP) - 4.0 x 4.3 m (WxH)	\$2,770.69
Sill Cut - Single Heading (OP) - 4.0 x 4.0 m (WxH)	\$3,081.93
Sill Cut - Multiple Heading (OP) - 4.0 x 4.0 m (WxH)	\$2,661.16
Sill Cut - Single Heading (OP) - 3.5 x 4.0 m (WxH)	\$2,902.96
Sill Cut - Multiple Heading (OP) - 3.5 x 4.0 m (WxH)	\$2,481.89

Note: PP = project period; OP = operating period

Table 21-11: Stopping Unit Cost by Deposit

Description	Unit Cost (per tonne)
Underground Mine Stopping - Whiskey Jack	\$52.47
Underground Mine Stopping – Gibson / Grey Fox South	\$64.14

21.3.1.2 Mine Operations Indirect

Owner's Team

The Owner's team operating cost listed in Table 21-9 includes personnel comprised of Management, Technical Services, Operating (non-direct), and Maintenance personnel. The non-direct operating includes underground general foreman, shift bosses, and grader operators, etc.



Indirect Mobile Equipment

The operating cost for the indirect equipment is based on Stantec's internal benchmarking and project experience, along with mining industry published standards aligned with the proposed equipment fleet.

21.3.1.3 Utilities

Power

The total power demand for the underground operations was calculated by considering the connected load and adjusting for utilization (operating hours), demand factor (percentage of nameplate draw), drive efficiency, and system losses. Power for the operating period will be provided by a 27.6 kV power line installed by Ontario Hydro, with an estimated cost of \$0.085/kWh for steady-state power. Details of the power distribution are described in Section 18.6.

The estimated total power consumption for the operating period is 1,379 M kWh, with a total estimated cost of \$86.7 million.

Compressed Natural Gas

The compressed natural gas operating costs are based on the following assumptions:

Heating of mine air takes place over an average period of 8 months per year.

Compressed natural gas supply costs will include the rental of all required infrastructure and delivery charges, and the all-in supply rate is assumed to be \$0.79/standard cubic metre. This rate is based on current database costs for similar projects in Northern Ontario.

21.3.1.4 Fixed Plant Operating

Fixed plant maintenance costs have been estimated based on Stantec's internal benchmarking and project experience, along with mining industry published standards aligned with the scope of work.

21.3.1.5 Leasing

McEwen will finance the initial purchase of all fleet, this leasing cost was carried as an operating cost. The estimate assumes that an interest payment of 6.99% will be applied to the repayment of each fleet unit, and that the repayment schedule will be three years per unit.

21.3.1.6 Project Period Mining Direct

The project period mining costs include all operating development, production and ore haulage related activities that occur during the project period.



21.3.2 Processing – Operating Costs

21.3.2.1 Basis of Estimate

The operating cost estimate for processing Grey Fox material through the Stock Mill has been developed based on defined operating parameters, metallurgical testwork, and industry benchmarks. The estimate assumes an annual plant throughput of 536,550 tonnes at a head grade of 3.4 g/t Au. Reagent consumption rates are based on metallurgical testwork results described in Section 13, supplemented where necessary with standard industry benchmarks in the absence of specific test data (e.g., cyanide detoxification). Comminution circuit power requirements were derived through process modelling using ore hardness parameters established during test programs. The overall process design reflects a combination of testwork outcomes and conventional industry best practices. Unit pricing for reagents and consumables was provided by McEwen based on current delivered costs to the Stock Mill, while costs for wear components such as polyurethane panels, cyclone parts, and screen cloths were estimated using DRA's internal database of comparable projects.

21.3.2.2 Operating Cost Summary

Table 21-12 below summarizes the processing plant OPEX by cost element.

Table 21-12: Process Plant OPEX Summary (\$ millions)

Element	\$ per year	\$ per tonne	Contribution%
Personnel	\$7.1	13.28	46.5%
Electrical power	\$1.4	2.65	9.3%
Reagent consumption	\$2.8	5.31	18.6%
Consumables consumption	\$2.2	4.04	14.1%
Laboratory costs	\$0.4	0.75	2.6%
Contract maintenance	\$0.1	0.23	0.8%
Spare parts and miscellaneous	\$1.2	2.33	8.2%
Total Operating Cost	\$15.3	28.58	100.0%

- Personnel costs include all salaried employees associated with the operating plant including the laboratory staff, water treatment plant operators and maintenance crews.
- Electrical power includes all electrical energy consumed by the site excluding the mining operation.
- Reagent consumption includes all reagents added to the slurry streams in the process plant including smelting fluxes, antiscalant, and water treatment chemicals. Sodium cyanide contributes \$2.30 per tonne of ore treated which is close to half of the total reagent costs.
- Consumables include grinding media, crusher/mill liners, diesel, propane (for heating), compressed natural gas (smelting furnace), lubricants, cyclone wear parts, polyurethane screen panels, crucibles and stainless steel wool.
- Laboratory costs include consumables such as crucibles for fire assays, glass beakers, laboratory chemicals, etc.



- Contract maintenance covers all contract services including annual mill relining, environmental monitoring services, safety audits and inspections, mobile equipment maintenance, etc.
- Spare parts and miscellaneous costs is an allowance for the replacement of equipment, as required, that may fail during operations. It also includes pipe fittings and platework replacement as required.
- A more detailed breakdown of each element is provided in the sections below.

21.3.2.3 Personnel Costs Estimate

Table 21-13 below summarizes the estimated plant personnel costs.

Table 21-13: Personnel Costs Summary

Position	Schedule	Number	Total Costs (\$/year)	Unit Costs (\$/tonne)
Mill Manager	Day	1	191,932	0.36
Mill Superintendent	Day	1	130,240	0.24
Administrative Clerk/Trainer	Day	1	99,136	0.18
Maintenance Superintendent	Day	1	130,240	0.24
Gold Room Supervisor	Day	1	121,166	0.23
Chief Metallurgist	Day	1	150,000	0.28
Junior Metallurgist (EIT)	Day	1	99,136	0.18
Laboratory Supervisor	Day	1	130,240	0.24
Laboratory Technician	Day	6	594,813	1.11
Shift Operations Supervisor	Shift	4	520,960	0.97
Control Room Operator	Shift	4	520,959	0.97
Plant Operator	Shift	24	2,379,253	4.43
Maintenance Planner	Day	1	121,166	0.23
Instrumentation Technician	Shift	4	484,663	0.90
Millwright	Shift	6	726,994	1.35
Electrician	Shift	4	484,663	0.90
Lube Technician	Day	2	242,331	0.45
Totals			7,127,892	13.26

There are six Plant Operators per shift crew covering all areas of the Plant. One or two of these also assist in the gold room with smelting duties (twice weekly) as there are no dedicated refiners in the personnel complement. Provision has not been made for a dedicated Relief Operator position to cover leave and training; the assumption is that the crew size is sufficient to cover these eventualities.



A day shift is based on 2,080 hours per year (52 weeks of 40 hours per week). There are four shift crews which allows for a two-weeks-in and two-weeks-out rotation. Their total hours are 2,190 hours per year (12-hour days for 14 consecutive days each month). The costs include all benefits and bonuses.

21.3.2.4 Electrical Energy Costs Estimate

Table 21-14 summarizes the expected energy consumption costs. The total unit energy consumption is estimated at 44.5 kWh/t.

Table 21-14: Estimated Energy Consumption Costs

Area	Installed Power (kW)	Combined Factors	Consumed (kWh/yr)	Total Costs (\$/year)	Unit Costs (\$/tonne)
Air and water services	349	90	1,993,845	\$118,634	0.22
Primary crushing	55	70	235,550	\$14,015	0.03
Crushed ore handling	48	92	303,895	\$18,082	0.03
Crushed ore reclaim	34	88	224,445	\$13,355	\$0.02
Secondary and tertiary crushing	130	71	787,141	\$46,835	\$0.09
Primary grinding circuit	1,250	91	9,616,446	\$572,179	\$1.07
Secondary grinding circuit	784	91	6,042,543	\$359,531	\$0.67
Leaching and adsorption	405	86	2,337,392	\$139,075	\$0.26
Cyanide destruction and tailings disposal	137	90	927,867	\$55,208	\$0.10
Acid washing, elution and regeneration	255	24	459,422	\$27,336	\$0.05
Refinery	120	37	365,081	\$21,722	\$0.04
Reagents	44	39	87,674	\$5,217	\$0.01
Lime mixing	16	22	24,660	\$1,467	\$0.00
Allowance for Site Overheads (water pumps, lighting, plugs, etc.)	118	53	469,714	\$27,948	\$0.05
Total	3,744		23,875,675	\$1,420,603	\$2.65

Notes:

- Installed power includes all standby motors
- "Combined factors" are the product of the load factor (i.e., how much of the installed power is used when the motor is running), the utilization factor (i.e., the total running hours as a fraction of the total hours in a calendar year), and the inverse of the efficiency factor (i.e., how efficiently is energy drawn from the grid).
- The utilization factor was applied to non-standby motors only so the product of the installed power and the combined factors number will not equate to the energy consumption presented in Table 21-3. The numbers shown in Table 21-3 are summaries of more detailed motor-by-motor calculations in the mechanical equipment list.
- The Allowance for Site Overheads cover all electricity users not pertinently listed in the Major Equipment List (e.g., site stormwater management pumps, sewage system, lighting, plugs).



21.3.2.5 Reagents Costs Estimate

Table 21-15 summarizes the reagent consumption and cost estimates.

Table 21-15: Reagent Consumption and Costs

Description	Consumed (kg/t)	Delivered Price (\$/kg)	Total Costs (\$/year)	Unit Costs (\$/tonne)
Sodium Cyanide Solution (30%w/v)	0.56	\$4.13	\$1,240,933	\$2.31
Activated Carbon	0.017	\$4.57	\$41,694	\$0.08
Hydrated Lime	3.14	\$0.51	\$857,968	\$1.60
Nitric Acid Solution (68%)	0.013	\$0.78	\$5,469	\$0.01
Caustic Soda	0.02	\$2.01	\$18,325	\$0.03
SMBS Solution (38% w/w)	1.02	\$0.73	\$398,382	\$0.74
Copper Sulphate	0.040	\$3.83	\$82,208	\$0.15
Antiscalant	0.02	\$5.08	\$54,460	\$0.10
Allowance for Water Treatment Plant Chemicals			\$110,895	\$0.21
Flocculant	0.015	\$4.60	\$37,014	\$0.07
Smelting Fluxes (combined)			\$352	\$0.00
Total			\$2,847,698	\$5.31

Notes:

- The allowance for water treatment plant chemicals was calculated from an industry benchmark of \$0.40/m³ treated and a preliminary water balance estimate of 277,200 m³ to be treated annually.
- All consumption numbers in the table are per tonne of ROM feed.

21.3.2.6 Consumables Costs Estimate

Table 21-16 summarizes the consumption of consumable items not included in the mechanical equipment or reagents lists.

Table 21-16: Miscellaneous Consumables Costs

Description	Consumed (kg/t)	Delivered Price (\$/kg)	Total Costs (\$/year)	Unit Costs (\$/tonne)
Jaw crusher liners	0.080	\$3.000	\$128,772	\$0.24
Cone crushers liners	0.140	\$3.000	\$225,351	\$0.42
Ball Mills Media	1.560	\$0.805	\$674,134	\$1.26
Ball Mills Liners	0.156	\$6.730	\$563,313	\$1.05
Diesel			\$40,425	\$0.07
Propane (Heating)			\$257,600	\$0.48



Description	Consumed (kg/t)	Delivered Price (\$/kg)	Total Costs (\$/year)	Unit Costs (\$/tonne)
Compressed Natural Gas			\$504	\$0.00
	Sets/annum	USD/set		
Crucibles and SS Wool			\$15,000	\$0.03
Lubricants			\$30,000	\$0.06
Cyclone Body Parts	8	\$17,000	\$136,000	\$0.25
ADR Circuit Screens Poly Panels	1	\$12,000	\$12,000	\$0.02
Scalping Screens Poly Panels	4	\$20,529	\$82,114	\$0.15
Total			\$2,165,214	\$4.04

Notes:

- Media and liner consumption rates for the tumbling mills are based on an assumed abrasion index and in-house empirical correlation calculating steel consumption based on the energy consumed by the mill and the abrasion index of the ore. The total steel consumption is estimated at 1.9 kg/t which aligns well with industry standards.
- Diesel consumption was estimated at 35,000 L/yr covering forklifts, bobcats and light vehicles used in the Plant and a diesel price of CAD 1.65/litre (USD 1.16/L).
- Propane consumption (heating) was benchmarked at USD 0.48/t against similar sites in Ontario. Note that this benchmark also includes the electrical energy consumed for heat tracing.
- An allowance of \$15,000 was added for goldroom crucibles and stainless steel wool.
- An allowance of \$30,000 was added for lubricants.
- For cyclones it was assumed that each operating cyclone (8 of) will be replaced once annually, i.e., 8 sets in total per year.
- Vibrating screen panels was assumed to have a life of 6 months.

21.3.3 Tailings - Operating Costs

Table 21-17 summarizes the costs associated with the Stock Mill WEA tailings storage.



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Table 21-17: Tailings Storage Costs for Stock Mill WEA

Cost Item	Quantity	Unit of measure	Unit Cost	Cost (\$ millions)	Basis of Estimate
Dam Construction	724,350	m ³	\$28.80	\$20.9	Concept Grading Design (2,632 m; 14 m max height; 9.2 m avg height; 4 m top width; 3H:1V ds slope; 2H:1V us slope; firm silty clay sourced from on-site; average 1 km round trip)
Overbuild for Settlement	108,653	m ³	\$28.80	\$3.1	15% Overbuild
Adjoining Dam Stabilization	37,772	m ³	\$9.72	\$0.4	Waste Rock against adjoining West Dam (464 m; 10H:1V; 1 m avg thickness; waste rock available on-site (5 km round trip)
HDPE-lined Emergency Spillway	1	LS	\$720,000	\$0.7	Sheet Pile Weir (5 m deep, 25 m wide, 1 m projecting above grade) and Engineered Riprap-Lined Channel (trapezoidal, 2H:1V sides, 20m base width, 0.5 m deep, D50=10" rock at 0.6 m thick; 50 m length)
Tailings Pipeline (8" SDR11 HDPE & fittings, 3" SDR11 HDPE & fittings)	1	LS	N/A	\$1.2	8" SDR11 HDPE (5 km). 8" fittings (22 tees, 44 shutoff valves) (assumes 10% of piping cost) 3" SDR11 HDPE (1.7 km) 3" fittings (44 tees, 44 elbows) (assumes 30% of piping cost)
Slurry Pump Upgrades	1	LS	\$720,000	\$0.7	High-head slurry pump for 4.5 km 8" pipeline at 250 m ³ /h (1,100 gpm), +13m static head
Slurry Wall Seepage Cutoff	10,528	m ²	\$187	\$2.0	In-situ stabilization slurry cutoff wall (2,632 m x 4 m deep; assumes 2 m average depth of firm brown clay + 2 m penetration into soft gray varved clay)
PVD Installation & Equipment Mobilization	2,147,238	m	\$1.44	\$3.1	PVDs (denser spacing under dam with transition zone beyond toe; 131,269 m ² at 1.2 m spacing + 109,350 m ² at 3.0 m spacing; 18 m depth) Mobilization of standard wick-drain installation rig



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Cost Item	Quantity	Unit of measure	Unit Cost	Cost (\$ millions)	Basis of Estimate
PVD Granular Drainage Blanket	120,310	m ³	\$28.80	\$3.5	Assumes 0.5 m thick granular drainage blanket over full footprint of PVD placement
Geotechnical Instrumentation	1	LS	\$360,000	\$0.4	VWPs/Inclinometers (assumed cost)
Subtotal: Direct Costs				\$35.9	
Engineering and Permitting	18%			\$6.5	~18% of Directs
Indirects and Contingency	20%			\$8.5	20% of Total
Total TMA Expansion Cost				\$50.8	
Initial Tailings Storage Cost (Grey Fox representing ~50% of total tailings storage through 2034)	2,087,619	tonnes		\$12.18	\$/tonne mined
Future Tailings Storage Cost (starting in 2035, Grey Fox represents 100% of total tailings storage)	4,175,238	tonnes		\$12.18	\$/tonne mined

Notes:

- LS = Lump Sum, PVD = Pre-Fabricated Vertical Drain
- Dam Construction and Adjoining Dam Stabilization quantities are based on conceptual earthwork grading designs in Autodesk Civil 3D compared to existing grade elevations that are based on a combination of drone-based aerial LiDAR survey conducted by Arcadis in 2025 and the 2016 COOP 2-m DEM obtained from the COOP GeoHub portal.
- Unit costs for Dam Construction (and overbuild), Adjoining Dam Stabilization, Tailings Pipeline & Distribution, PVD Equipment Mobilization, and PVD Granular Drainage Blanket have been estimated using the Reclaim 7.0 model developed by the Government of the Northwest Territories. These costs have been converted from March 2014 dollars to January 2026 dollars using a conversion rate of 1.32.
- Tailings storage costs assume that the full expansion cost would be recurring for every 4.2 million tonnes of tailings (total) stored at Stock Mill, which, at approximately 3,000 t/d, represents a recurrence interval of approximately every 3.8 years (~46 months).
- In calculating initial and future tailings storage costs, a percentage of the total expansion capital cost proportional to the percentage of Grey Fox tailings stored at the facility is divided by the respective quantity of Grey Fox tailings, resulting in a normalized unit rate of \$12.18 US\$/tonne over the LOM.
- TSF costs are conceptual and may vary as design advances, with potential cost impacts mitigated by contingency and procurement strategies.
- More economical tailings storage options may be available that are either currently under evaluation or have not yet been evaluated (e.g., thickened tailings with central deposition; dry-stack tailings; construction of a new mill at Black Fox to facilitate on-site ore processing and tailings storage in the Black Fox Open Pit).



21.3.4 Water Treatment Plant OPEX

The annual operating cost estimate for the Grey Fox water treatment plant is based on the following assumptions listed below as well as the operating summary described in Section 18.12.4:

- Sodium hypochlorite dosed at 5 mg/L to the influent stream flow.
- 1% of flow through the greensand filter is used for backwash.
- Greensand backwash contains 1% solids.
- The backwash sludge volume was estimated by modelling the iron content as ferric chloride (FeCl_3) assuming all iron in the influent stream, iron added as part of the greensand filtration process, and dewatering polymer, dosed at 5 mg/L, are captured as waste sludge.
- Greensand media replacement is completed every 4 years.
- IX resin replacement is completed every 10 years.
- Spent IX regenerant brine is disposed of at an offsite brine treatment facility.
- Ion exchange resins are regenerated once per week at a sodium chloride dose of 160 g/L.
- Weekly performance sampling for metals, total suspended solids, and pH at a third-party laboratory.

The total annual estimated operating cost for the water treatment plant is \$202,700.



22 Economic Analysis

An economic model was prepared for the Grey Fox Project to estimate annual cash flows and assess the sensitivity of the Project to certain economic parameters.

The Project indicates an after-tax cash flow of \$493.2 million, after-tax NPV (5%) of \$282.1 million, and after-tax IRR of 24.8%. The Project is most sensitive to the head grade and gold price.

The base case discount rate for the Project is 5%. The discounted present values of the cash flows are summed to determine the project's NPV. In addition to the NPV, the IRR and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project from the start of mineral processing.

Tax assumptions were modelled using input from McEwen and its tax advisors.

22.1 Financial Results

A summary of the economic parameters for the Project are illustrated in Table 22-1, Table 22-2, and Figure 22-1.

Table 22-1: Summary of Economic Metrics for the Project

Description	Units	Value
Pre-Tax		
NPV at 5%	\$ million	\$428.6
Internal rate of return	%	30.7%
Payback period	Years	3.9
After-Tax		
NPV at 5%	\$ million	\$282.1
Internal rate of return	%	24.8%
Payback period	Years	4.6

Note: Payback period calculated from start of processing.

Table 22-2: Calculation of After-tax Cash Flow

Description	Units	Value
Gross revenue	\$ million	\$2,573.0
NSR Royalties	\$ million	-\$73.3
Net Revenue	\$ million	\$2,499.7
OPEX & SUSEX	\$ million	-\$1,578.8



Description	Units	Value
Net Operating Income	\$ million	\$920.9
NPI Royalty	\$ million	-\$16.3
EBITDA (Net Revenue)	\$ million	\$902.4
Non Cash Costs	\$ million	-\$144.1
EBIT	\$ million	\$758.2
Taxes	\$ million	-\$228.4
After-Tax Income	\$ million	\$529.9
Non Cash Costs - Added Back in	\$ million	\$144.1
Operating Cash Flow	\$ million	\$674.0
Capital Costs	\$ million	-\$180.8
Total Free Cash Flow (After-Tax)	\$ million	\$493.2

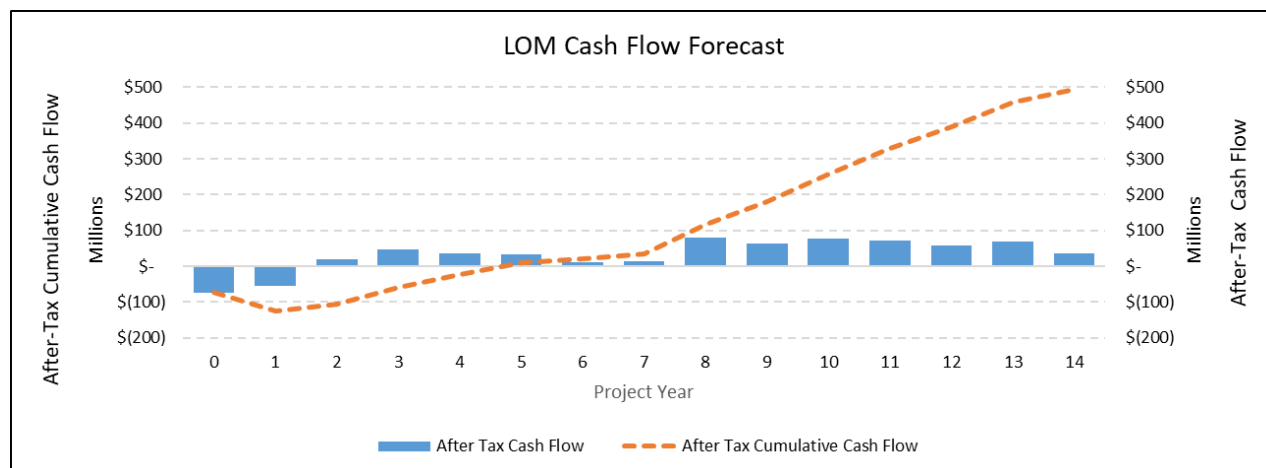


Figure 22-1: Undiscounted After-Tax Cash Flow

22.2 Cautionary Statement

The economic analysis results are based on forward-looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented herein. Forward-looking statements in this section include, but are not limited to, statements with respect to:

- Future gold prices.
- Currency exchange rate fluctuations, primarily the CAD:USD rate.
- Estimation of Mineral Reserves.
- Realization of Mineral Reserve estimates.
- Estimated costs and timing of CAPEX and OPEX, including labour, fuel, materials, consumables, supplies, and services.



22.3 Assumptions

The economic model includes revenue, costs, taxes, and other factors applicable to the Project from the beginning of final design and construction through to the end of mine life. Corporate obligations, financing costs, and corporate-level taxes are excluded.

The model was prepared using mining schedules developed on an annual basis. The financial model is based on the following assumptions:

- All costs are reported in US dollars (US\$) and referenced as "\$" unless otherwise stated.
- 100% equity ownership.
- Escalation and inflation have not been applied.
- Additional project development costs, including feasibility studies and permitting activities, are excluded.
- Annual gross revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year.
- Constant commodity prices.
- A constant exchange rate of US\$1.00 = C\$1.35 (or C\$1.00 = US\$0.74) is applied.

22.4 Economic Model Parameters

22.4.1 Mineral Resource, Mineral Reserve, and Mine Life

The mine plan presented in Section 16 of this report was used as the basis for revenue and cost estimates in the discounted cash flow, and also represents the current Mineral Reserve estimate. The mine plan excludes all material from the Inferred Resource category.

The stated Probable Mineral Reserve is 9.4 Mt at 3.24 g/t Au.

The mine plan considers a 15-year LOM. The mill processing profile is presented in Figure 22-2.



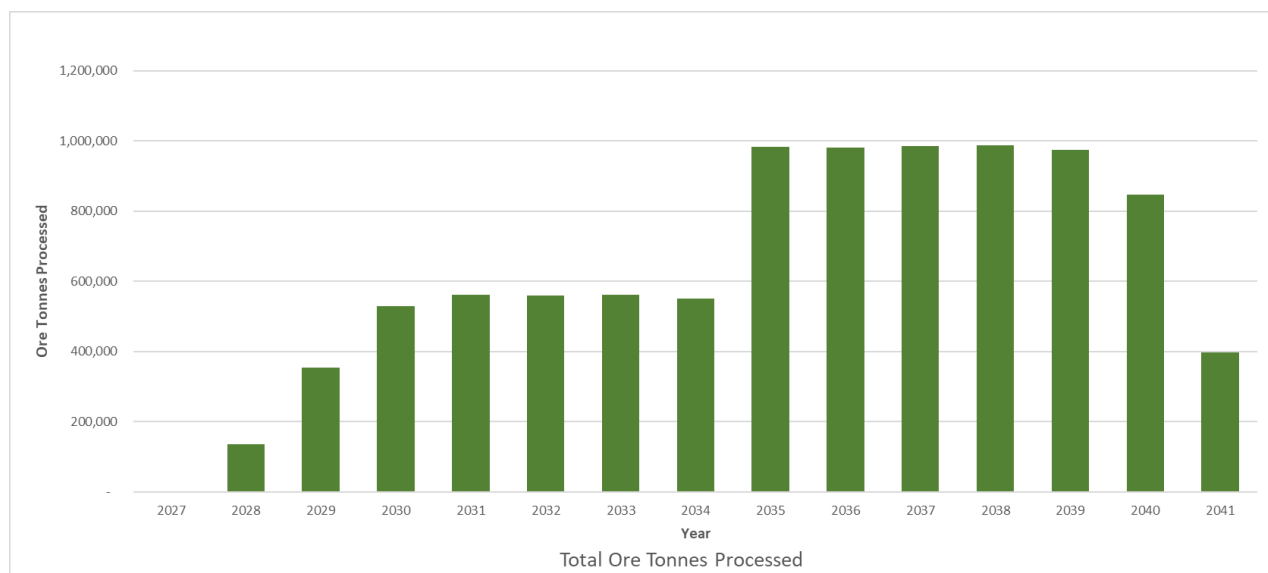


Figure 22-2: Life of Mine Processing Profile (tonnes)

The production profile for payable Au is presented in Figure 22-3.

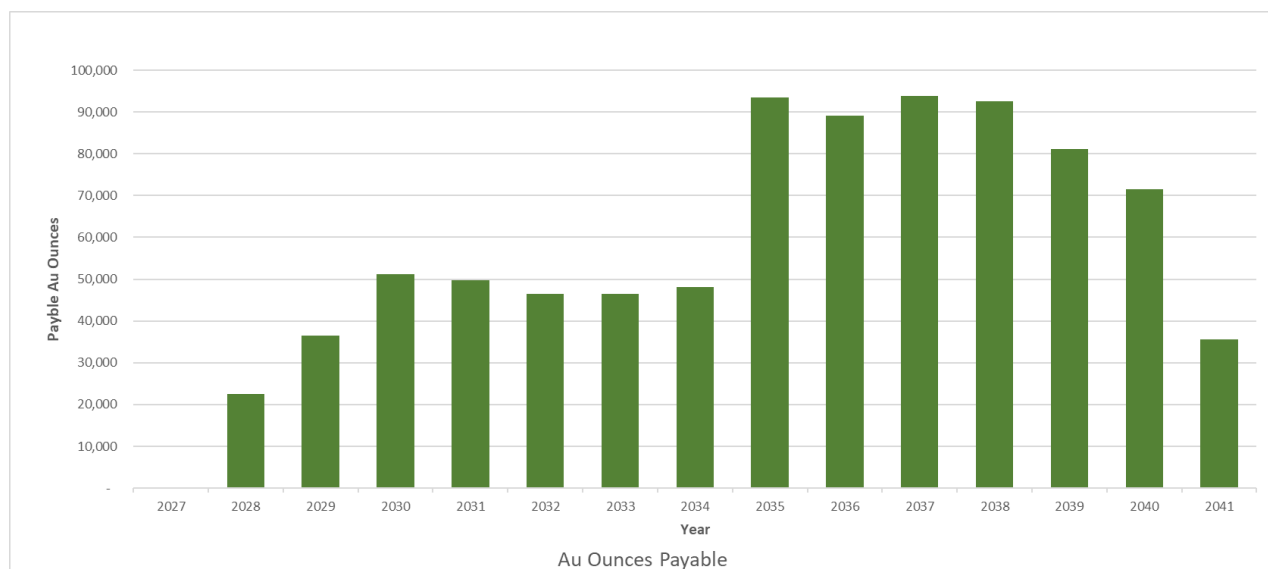


Figure 22-3: Life of Mine Production Profile (Payable Au)

22.4.2 Metallurgical Recoveries

The average metal recoveries for the different ore zones that have been used for the economic analysis are summarized in Table 22-3. Various grade versus recovery formulas were used to calculate recovery metrics on an annual basis within the financial model. A detailed review of metallurgical recoveries is provided in Section 13 of this report.



Table 22-3: Life of Mine Average Metallurgical Recoveries

Zones	Gold
Whiskey Jack	91.8%
Gibson	85.7%
Contact	83.8%
147NE + 147 + South	88.8%
Overall	87.5%

22.4.3 Metal Prices

The metal price for Au used for the economic analysis was based on monthly closing data from April 2024 to April 2026; the approximate two-year trailing average gold price is **US\$3,000 per ounce**. This period reflects a rapid rise in gold prices, starting from approximately \$2,285 in April 2024 and exceeding \$4,700 by April 2026. The metal price used is summarized in Table 22-4.

Table 22-4: Metal Price Assumption

Metal	Units	Price
Gold	US\$ / oz	3,000

Note: The calculation is based on monthly spot closing prices available as of April 2026.

A detailed review of metal prices is provided in Section 19 of this report.

22.4.4 Capital Costs

A summary of pre-production and sustaining capital costs is provided in Table 22-5. A detailed description of the capital costs is provided in Section 21.1 of this report.

Table 22-5: Capital Cost Summary (US\$ millions)

Description	Total
Detailed Engineering	\$6.6
Physical Procurement	\$10.8
Construction	\$98.7
Project Indirects	\$33.3
Capitalized OPEX	\$4.3
Total capital before contingency	\$153.7
Contingency	\$27.1
Sustaining	\$173.7
Closure	\$7.4



Total Capital Costs	\$362.0
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22.4.5 Operating Costs

A summary of LOM operating costs is presented in Table 22-6 and Table 22-7. A detailed description of operating costs is provided in Section 21.3 of this report.

Table 22-6: Life of Mine Operating Cost Summary

Operating Cost Item	Cost (\$/tonne Processed)	Cost (\$/Oz Processed)
Mining	\$108.02	\$1,184.1
Processing	\$28.4	\$311.2
Tailings	\$12.2	\$133.5
Total	\$148.6	\$1,628.8

Note: Mining Costs include WTP

Table 22-7: Life of Mine Cash Costs C1 & All-In Sustaining Costs

Cash Cost	LOM Total	Unit/T_{ore}	Unit/Oz
Royalties	\$90	\$9.53	\$104.43
Mining	\$1,016	\$108.02	\$1,184.09
Processing / Tailings	\$382	\$40.57	\$444.71
Selling	\$22	\$2.33	\$25.50
Production Tax (Gov't Royalty)	\$64	\$6.80	\$74.56
Total C1 Costs	\$1,573	\$167.25	\$1,833.29
DD&A	\$144	\$15.32	\$167.98
Sustaining Capital	\$181	\$19.26	\$211.11
All-in Sustaining Cost	\$1,898	\$201.83	\$2,212.37

22.4.6 Royalties

Three royalties have been included in the economic analysis, each calculated as a percentage of the NSR for portions of the ore zones, as summarized in Table 22-8.

Table 22-8: Royalties as % of NSR & Percent of total Oz per Royalty

Name of Royalty	% of NSR	% of Total Oz
Gray	0.15%	20%
Newmont	2.50%	16%



Name of Royalty	% of NSR	% of Total Oz
Schumacher	3.00%	64%

A fourth royalty, calculated as a percentage of the NPI for portions of the ore zones, is presented in Table 22-9.

Table 22-9: Royalties as % of NPI & Percent of total Oz

Name of Royalty	% of NPI	% of Total Oz
Parsons-Ginn	5.0%	36%

22.4.7 Working Capital

Allowances for working capital beyond metals in inventory at the mine and mill have been excluded for the purposes of this economic analysis.

22.4.8 Taxes

A taxation model has been developed within the economic model with input from McEwen's internal tax experts. Tax calculations for the Project are based on current federal and provincial tax rates and include the existing Mineral Property Tax Pools associated with the Project. McEwen internal tax experts reviewed the final tax calculations.

The tax inputs used in the economic model reflect current conditions; however, the actual taxes payable will depend on several factors, including, but not limited to, the timing of capital expenditures, changes in federal or provincial tax policy, and any investment incentives that may be available to McEwen.

22.4.9 Closure Costs

Closure costs are estimated at \$7.4 million and represent the cost of closure for Grey fox project as well as the contiguous black fox site which is under a single closure plan.

A bonding strategy has not been developed at this stage, and closure costs are assumed to be incurred during the last year of production.

22.4.10 Financing

Project-level financing has not been considered in this economic analysis. 100% equity ownership is applied.

The economic analysis does take into account financing terms offered by the vendor for a portion of the mining equipment fleet and certain surface facilities. These terms are based on a lease-to-own structure with a 15% down payment and an effective financing rate of 0.79% over a 36-month period.



22.5 Sensitivity Analysis

A discounted cash flow calculation was prepared for the Project based on the various cost and revenue inputs. Sensitivity analysis was completed by applying $\pm 30\%$ variations to gold price, head grade, recovery, exchange rate, CAPEX, SUSEX, OPEX, fuel costs, and labour costs. The results are illustrated in Figure 22-4.

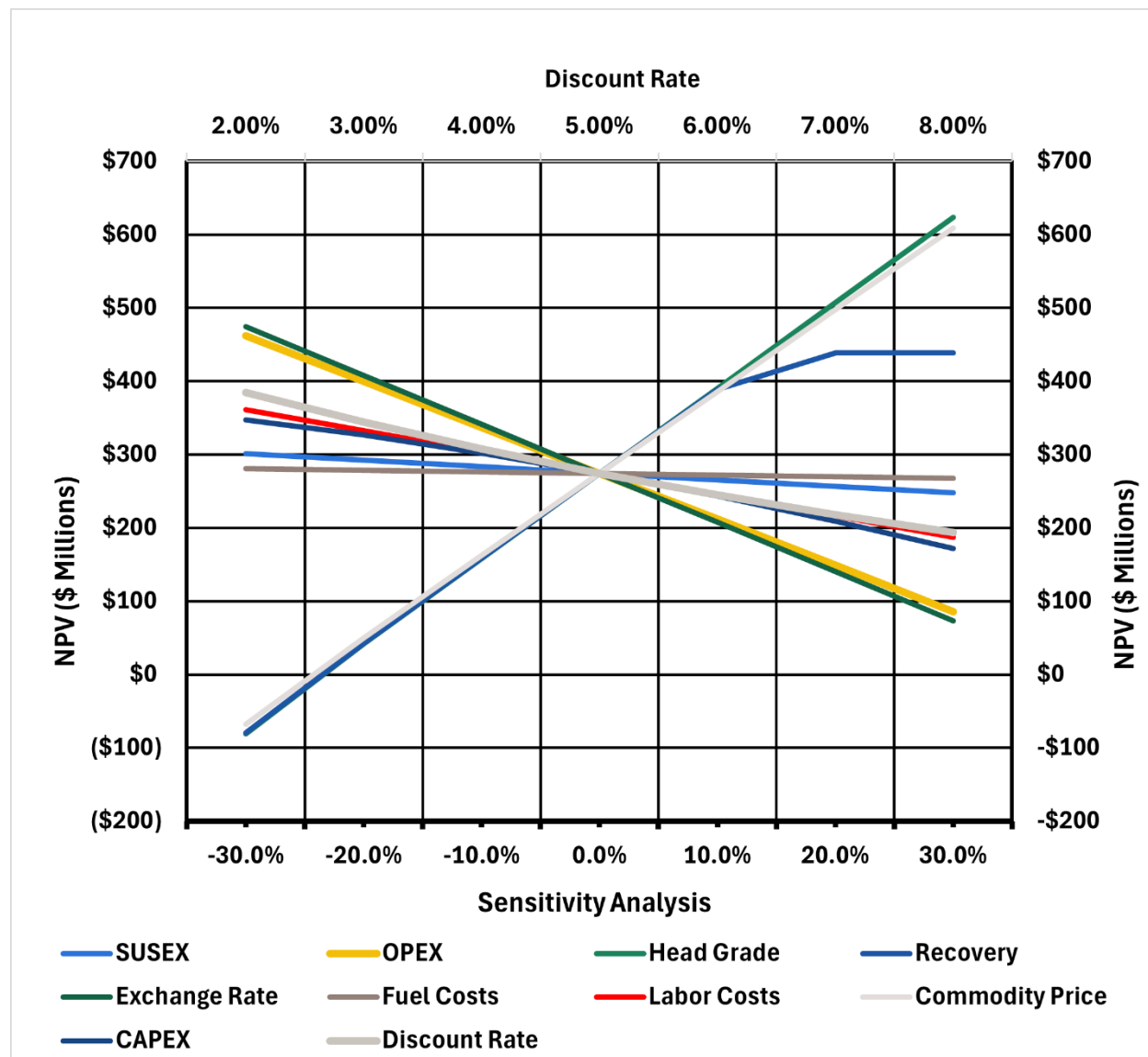


Figure 22-4: Sensitivity Analysis

For the press release Titled “Pre-Feasibility Study for Grey Fox at Fox Complex – High Returns, Manageable Capital, Mine Life Extended to 2041” and dated June 8, 2026 a gold price of \$4500 per ounce (enhanced case) was used to understand the sensitivity of the project to gold price beyond the +30% presented in Figure 22-4. The results of the enhanced case are the following:



- Cash costs and all-in sustaining costs per ounce over the LOM of **\$2,042** and **\$2,421**, respectively, at a gold price of **\$4,500** per ounce.
- Pre-tax NPV (5%) increases to **\$1.25 billion**, with an IRR of **70%**, and payback of **2 years**
- Post-tax NPV (5%) increases to **\$841 million**, with IRR of **55%** and payback of **2.3 years**.



23 Adjacent Properties

The Grey Fox deposit is located proximal to the PDDZ in the Abitibi Greenstone Belt, a region with significant historical and current gold production. The Fox Complex, which includes Grey Fox, is located within an active exploration and mining district, surrounded by claims and leases held by exploration companies and operators including Onyx Gold Inc., STLLR Gold Inc., Mayfair Gold Corp., and Agnico Eagle Mines Ltd.

23.1 Hislop

Located immediately south of Grey Fox is the formerly producing gold mine on the Hislop property, owned by Agnico Eagle Mines Ltd. (Figure 23-1). The Hislop property is also located immediately east and southwest of the Stroud deposit (100% McEwen ownership). McEwen has no ownership interest in the Hislop property, and information concerning this adjacent property has not been independently verified.

Current geological interpretations suggest that mineralization at Grey Fox may extend toward the northern portion of the Hislop property, however this has not been fully verified.

Gold mineralization at the Hislop property occurs in two principal settings associated with a syenite dyke complex:

- To the south, in carbonate and carbonate breccia rocks separating the syenite dykes from the less altered ultramafic volcanic rocks; and
- To the north, in quartz veinlets, stockworks, and fractures within hematite-altered and syenitized mafic metavolcanic rocks (RPA, 2009).

Mineralization north of the syenite dyke may be analogous to mineralization observed at the southern extent of the Grey Fox Project.



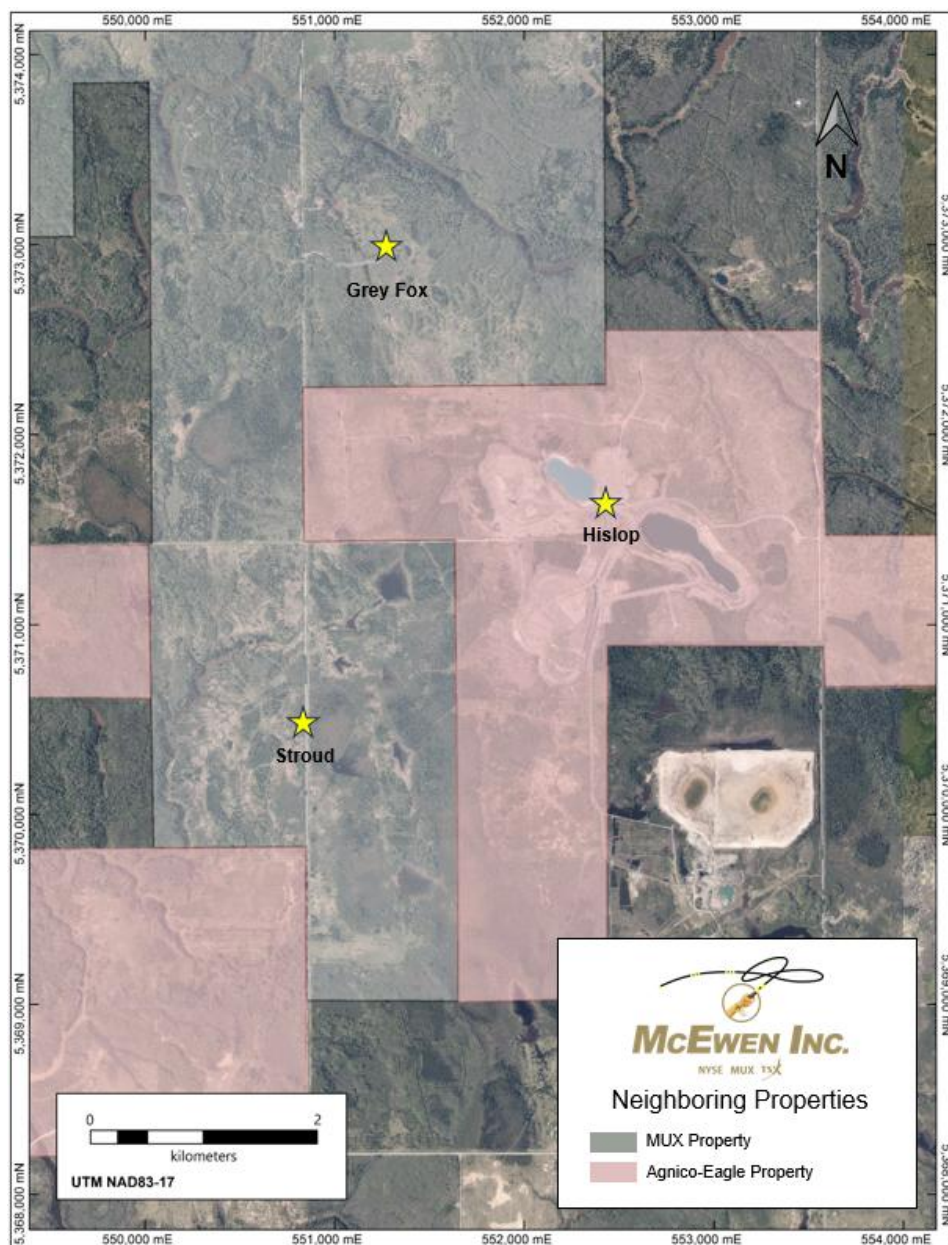


Figure 23-1: Plan View of Grey Fox and Stroud Projects in relation to Hislop Property

24 Other Relevant Data and Information

There is no other relevant data or information.



25 Interpretation and Conclusions

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the reviews and interpretations of data available for this Report.

25.1 Ownership

In the opinion of the Qualified Person, the ownership of the Grey Fox Project is well established, with McEwen holding a 100% interest in the mineral tenure covering the deposit area. The mineralized zones are situated on a contiguous group of patented and leased mining claims that collectively provide full control over the resource area. This ownership structure is sufficient to support the development and operation of the Project as contemplated in this Pre-Feasibility Study, and no material ownership constraints have been identified.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

In the opinion of the Qualified Person, the mineral tenure, surface rights, and associated agreements for the Grey Fox Project are well defined and provide a solid foundation for Project development. The deposit is located on a combination of patented claims and mining leases, with surface rights largely held or accessible for planned activities. The Project is subject to royalty agreements that are typical for the region and have been incorporated into the economic analysis. Existing agreements and ongoing engagement with Indigenous communities and other stakeholders are in place and support Project advancement. Overall, the tenure position, rights, and encumbrances are considered appropriate for the stage of study and do not present any material limitations to the Project as evaluated.

25.3 Geology and Mineralization

The Grey Fox deposit is interpreted, based on the geological, structural, alteration, and mineralization relationships described above, to represent a structurally controlled gold system with characteristics consistent with an epizonal orogenic gold environment. Mineralization is hosted within a geologically complex package of mafic metavolcanic rocks, metasedimentary units, and the Gibson Syenite. The interpretation presented herein is considered appropriate for the current level of study and is intended to support geological modelling, Mineral Resource estimation, and PFS-level mine planning under NI 43-101 and S-K 1300 reporting frameworks.

The principal controls on gold mineralization are interpreted to be the interaction of favourable host lithologies, rheological contrasts, and multiple generations of veining and deformation. The dominant mineralized system comprises steeply dipping, northeast-southwest striking C1-vein arrays that are best developed in the 147, 147NE, Grey Fox South, and Gibson zones. Additional mineralization is localized along the Contact Fault and within the Whiskey Jack shear system. These domains are genetically and spatially related within the broader Grey Fox system, but they display sufficient geological, structural, and



grade-continuity differences to warrant separate consideration during geological interpretation, domain modelling, estimation, and mine-design work.

The strongest and most laterally persistent mineralization is interpreted to occur where the C-vein system intersects chemically favourable iron-rich variolitic mafic metavolcanic flows and, in the Gibson area, where veins crosscut metasedimentary rocks proximal to the Gibson Syenite. These relationships indicate that both structure and host-rock composition exert first-order controls on gold deposition. In the volcanic-hosted zones, quartz-carbonate-albite veining, iron-rich host rocks, hematite-bearing alteration, and later sulphide-bearing alteration are interpreted to reflect fluid-rock reaction and changes in redox conditions that may have contributed to gold precipitation. In the Gibson zone, some mineralization is preferentially associated with quartz-sericite-sulphide alteration in metasedimentary rocks, localized brecciation, and stronger alteration near and below the syenite intrusion.

The structural architecture of the deposit is a material consideration for resource interpretation and mine planning. The Contact Fault, A-1 thrust fault, Gibson-Kelore Shear Zone, and Whiskey Jack shear define important geological boundaries and mineralized corridors. Variations in C-vein orientation across the Contact Fault are interpreted to reflect changes in host-rock competency between mafic metavolcanic rocks, metasedimentary rocks, and syenite. These variations should be reflected in geological modelling, domain definition, estimation strategy, and geotechnical assessment. The Whiskey Jack Zone is distinguished from the C-vein dominated zones by more continuous shear-hosted mineralization, higher average grades, and a deeper brittle-ductile style of deformation.

Overall, the geological setting and mineralization characteristics support the interpretation that Grey Fox comprises multiple related mineralized domains within a broader structurally controlled gold system. The primary opportunities for resource growth and refinement are expected along down-plunge and along-strike extensions of mineralized C-vein intersections with favourable metavolcanic and metasedimentary units, within structurally prepared zones adjacent to major faults and shears, and at depth in the Gibson area where drilling has indicated potential for higher-grade breccia-hosted mineralization below approximately 500 m. Further work should focus on refining structural controls, validating interpreted plunge continuity, assessing the extent and continuity of deeper Gibson mineralization, and integrating department results into metallurgical, sampling, reconciliation, and resource-modelling assumptions. These recommendations are intended to reduce geological uncertainty and support future updates to the Mineral Resource estimate and mine plan in accordance with applicable NI 43-101 and S-K 1300 disclosure requirements.

25.4 Exploration

The Grey Fox deposit appears to be a combination of an epizonal and orogenic gold deposit made up of a network of sheeted vein sets present in multiple orientations. The drilling densities at the various zones that make up the Grey Fox deposit are good and in the QP's opinion, the current geological and mineralization models are satisfactory.

Additional stripping for potential outcrops containing structures and mineralization is recommended to test against the validity of the current geological and mineralization models. It will be important to tighten the drill spacing for select areas at Grey Fox to improve the classification of mineral resources. Finally, the



Exploration team should continue to update the lithological-structural models for local accuracy, control of density and resource domaining.

25.5 Metallurgical Testwork

The principal conclusions arising from the metallurgical testwork completed on Grey Fox mineralized material are summarized below:

- **Comminution Characteristics**

Comminution testwork conducted on representative samples of Grey Fox mineralized material indicates that the material is relatively hard and competent compared with ores typically processed in conventional gold treatment facilities. The tested samples returned an average Bond Ball Mill Work Index (BWi) of 23.4 kWh/t, with the hardest sample, representing Domain 31, exhibiting a BWi of 27.1 kWh/t. These results indicate that the Grey Fox material will require relatively high grinding energy for comminution, and that installed grinding power is expected to be a key design consideration for process plant throughput and overall circuit performance.

- **Gravity Recoverable Gold Potential**

Gravity concentration testwork returned gold recoveries to gravity concentrate ranging from 4% to 32%, with average gravity recovery below 20%. These results indicate that the gravity recoverable gold component is generally modest, although certain samples exhibited recoveries that may justify consideration of gravity concentration within the process flowsheet. On the basis of the testwork completed to date, inclusion of a gravity recovery stage may provide incremental recovery benefit but is not considered fundamental to the overall metallurgical recovery strategy.

- **Whole-Ore Cyanidation Performance**

Whole-ore cyanidation testwork returned gold recoveries ranging from approximately 71% to 94%. The broad range of recoveries demonstrates significant metallurgical variability within the Grey Fox mineralized material. This variability is expected to have implications for recovery forecasting, geometallurgical domaining, and potential process optimization strategies, and should be considered in future metallurgical evaluation and mine planning.

25.6 Mineral Resource Estimates

The Mineral Resources estimates are considered reliable and thorough, and based on quality data that has been evaluated and verified. In cases where historical data was unable to be fully validated, resources were restricted to the inferred classification. Geological interpretations and parameters used in the MRE's were assessed and considered reasonable.

- The Grey Fox deposit is estimated to contain (inclusive of mineral reserves) 18.8 Mt at 3.28 g/t for 2.0 Moz of gold at the indicated classification and 5.0 Mt at 2.69 g/t for 438 koz of gold at the inferred classification.
- The Grey Fox deposit is estimated to contain (exclusive of mineral reserves) 9.7 Mt at 2.25 g/t for 701 koz of gold at the indicated classification and 4.7 Mt at 2.57 g/t for 388 koz of gold at the inferred classification.
- The Stroud deposit is estimated to contain, in inferred classification, 1.9 Mt at 2.80g/t for 168 koz of gold.



- The Black Fox deposit is estimated to contain, in the combined measured and indicated classification, 385 kt at 3.57 g/t for 44 koz of gold. At the inferred classification it is estimated to contain 385 kt at 2.90g/t for 36 koz of gold.
- The Froome deposit is estimated to contain, in the combined measured and indicated classification, 651 kt at 2.78 g/t for 58 koz of gold. At the inferred classification it is estimated to contain 302 kt at 2.93 g/t for 28 koz of gold.
- The Stock West and Main deposits are estimated to contain 3.2 Mt at 2.63 g/t for 268 koz of gold at the indicated classification and 2.5 Mt at 2.31g/t for 184 koz of gold at the inferred classification.
- The Stock East deposit is estimated to contain 1.8 Mt at 2.12 g/t for 122 koz of gold at the indicated classification and 1.4 Mt at 2.02g/t for 87 koz of gold at the inferred classification.
- The Fuller deposit is estimated to contain 2.2 Mt at 3.14 g/t for 221 koz of gold at the indicated classification and 1.6 Mt at 2.19 g/t for 114 koz of gold at the inferred classification.
- The Davidson-Tisdale deposit is estimated to contain, in the combined measured and indicated classification, 294 kt at 6.23 g/t for 59 koz of gold. At the inferred classification it is estimated to contain 200 kt at 3.44 g/t for 22 koz of gold.

25.7 Mineral Reserve Estimate

The mineral reserves estimate for the Grey Fox Project has been completed in accordance with accepted practice and meets the CIM Definition Standards for Mineral Resources and Reserves (2014).

Many factors may affect the mineral reserves estimate such as dilution, metal prices, metallurgical recoveries, geotechnical characteristics of the deposit and host rock, and the capital and operating cost assumptions. These factors have been taken into account subject to the information and means available to QPs at the time of this study, as described in this report.

The metal price used to estimate Mineral Reserves was \$2,200 per ounce of gold. The QP considers this price to be reasonable based on historical three-year trailing averages.

The reserves were estimated by designing mineable stope shapes in Deswik using DSO. The dimensions of the shapes were determined using geotechnical modelling and modifying factors such as dilution and mining recovery were applied. The shapes were then interrogated against the resource block model and included in the mine plan where the Gold value exceeded the cut-off value. The result was the reserve estimate on which the mine plan was based.

The mineral reserve estimate is shown in Table 25-1.

Table 25-1: Mineral Reserve Estimate, March 27, 2026

Classification	Ore Tonnes (t)	Au (g/t)	Gold Ounces (Oz)
Total Reserve	9,405,829	3.24	980,303



25.8 Mining

McEwen has other existing mining properties (operating and shutdown) in the same area as the Grey Fox Project, and thus has significant experience in mining operations in the region. The planned production rates, modifying factors, and costs are benchmarked against other operations to ensure they are suitable. The project is planning to employ an underground mining method consisting primarily of longitudinal longhole open stoping, which is deemed suitable considering the geometry and quality of the orebody. A small portion of the tonnes will be recovered via cut-and-fill. Longhole open stoping provides operational flexibility and scalability. Given the steeply dipping orebody geometry and the competent hanging wall and footwall conditions, the project is proposed to use predominantly longhole stoping on 20 m to 30 m sublevels in ground conditions expected to range from fair to good. The stopes will be backfilled with a combination of unconsolidated rock fill plus cemented rock fill to prevent backfill dilution in subsequent stopes while also minimizing the use of cement.

All mining activities will be conducted using rubber-tired equipment. Ore will be hauled to surface via the ramp using underground haul trucks. An existing portal near the Whiskey Jack and Gibson deposits will provide initial access to the underground operations. The operation is targeting a steady-state production rate of approximately 1,500 t/d until 2035 then ramping up to 2,700 t/d until 2040, ramping down in 2041. The equipment fleet has been sized to deliver the scheduled mining volumes with fleet expansion and the rebuild and replacement of gear included in the capital spending schedule. Push-pull ventilation system using intake fans at raises/portals and exhaust fans at return raises, with heaters to prevent freezing delivering up to 250 m³/s of fresh air.

25.9 Infrastructure

The Grey Fox Project will consist of a compact, centralized mining infrastructure supporting underground operations via North and South portals. Core site infrastructure includes ore and waste stockpiles, a primary crushing facility, water management systems, ventilation plants, and key support buildings such as administration offices, mine dry, maintenance shop, warehouses, and security gatehouse. Site access is provided via Tamarack Road connecting to Highway 101, with ore hauled ~34 km to the Stock Mill. Electrical power will be supplied through a 27.6 kV grid connection along Tamarack Road.

25.10 Tailings Storage

Primary conclusions related to tailings storage are outlined below:

- The planned western expansion of the Stock Mill TMA is anticipated to provide approximately 4.2 Mt of storage capacity within an outer footprint of approximately 41 ha. Existing leased lands at the Stock Mill are anticipated to provide at least an additional 193 ha of future expansion potential, for an estimated total expansion capacity of approximately 24 Mt.
- Due to the geotechnical challenges associated with the underlying, soft, varved clays at the Stock Mill site, foundation-dewatering and seepage-control measures, in the form of prefabricated vertical drains (PVDs; wick drains) and appurtenant dewatering measures, have been included in the costs for the Stock Mill western expansion.



- Other tailings management and storage methods, including using the Black Fox open pit, are currently being evaluated that may provide more economical and safer alternatives to slurry placement.

25.11 Water/Solute Mass Balance and Water Treatment

A water and solute mass balance model was developed in GoldSim to quantify mine contact water flows and water quality that may require treatment prior to discharge. Details on methods and results of this water/solute mass balance model are provided in Section 20. These results were used as the basis for the concept-level design of a water treatment plant (described in Section 18), as well as treatment plant capital and annual operation cost estimates (provided in Section 21). The water balance was constructed based on the proposed mine water management plan involving pumping of contact water from Grey Fox (including underground workings dewatering water and runoff from mine facilities) to the Black Fox Mine holding pond.

Key conclusions of the water/solute mass balance evaluation are summarized below.

- Average flows to the Holding Pond are anticipated to vary seasonally between 2,200 and 4,700 m³/day, with an annual average of approximately 3,300 m³/day. This includes water pumped from Grey Fox, as well as contact runoff at Black Fox from the Waste Rock Stockpile.
- Although several analytes may be elevated in concentration in mine contact water, arsenic is the only analyte predicted to exceed Canadian MDMER water quality standards, and the exceedance is predicted to be minimal and potentially seasonal.
- Arsenic and other constituents (cobalt, copper, iron, nickel, molybdenum, and zinc) are predicted to exceed Provincial PWQO values. There is also a smaller chance that vanadium may exceed PWQO values.
- Importantly, model results are highly sensitive to uncertainties in development rock geochemical source terms.

Based on these simulation results, the anticipated water treatment system will consist of iron oxidation (potentially using a chemical oxidant such as oxygen or sodium hypochlorite) and greensand filtration, followed by ion exchange for low level metals removal, including a SBA resin for oxyanion removal (arsenic, molybdenum, and vanadium) and a SAC resin for removal of divalent cation metals (copper, cobalt, nickel, and zinc).

25.12 Market Studies and Contracts

Gold is traded freely at widely known prices, consequently, the prospects for the sale of any production of the metal are virtually assured. A market study for the saleability of gold products or for the expected price of gold was not undertaken.

25.13 Capital and Operating Cost Estimates

The estimate meets the classification standard for a Class 3 estimate as defined by AACE International and has an intended accuracy of +20% / -15%. The estimate is reported in Q1 2026 US dollars.



An all in project capital cost is estimated at \$181 million. SUSEX are estimated at \$174 million, and a closure cost of \$7.4 million.

The total operating cost is estimated at \$1,398 million or \$149 per tonne of ore milled.

25.14 Economic Analysis

The Project indicates an after-tax cash flow of \$493.2 million, after-tax NPV (5%) of \$282.1 million, and after-tax IRR of 24.8%. The Project is most sensitive to the head grade and gold price.

25.15 Risks

The QPs have not identified any significant risks or uncertainties that would reasonably be expected to impact the reliability or confidence of the exploration data, mineral resource estimates, or projected results. They are also not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other factors that could materially influence the Mineral Resource estimates. However, it should be noted that substantial fluctuations in metal prices could potentially affect the project's economic viability.



26 Recommendations

The work completed as part of this Technical Report indicates that the Grey Fox Project is economically feasible based on the stated inputs and assumptions. The authors recommend that McEwen advance development of the Grey Fox Project.

The following work programs are recommended by the QPs to further support project advancement and enhance project value. Estimated costs for the recommended work are provided in Table 26-1.

26.1 Recommended Work Programs

26.1.1 Early Works

An early works program is recommended to confirm assumptions related to surface geotechnical conditions, including surface overburden characteristics, the condition of the existing portal, and surface conditions at the planned locations of the ventilation raise collars and the south portal.

26.1.2 Further Exploration

Future exploration drilling at Grey Fox should focus on:

- The northwestern extent of the Gibson Zone, where there is evidence of repeating vein sets associated with the contact between the Gibson syenite and adjacent metasediments to the east.
- Areas below the 300 m elevation at the Whiskey Jack Zone, where mineralization remains open.
- Deeper drilling (>300 m elevation) at Grey Fox South and 147, where drill density is lower relative to other areas of the property.

In addition to continued diamond drilling, recommended exploration activities include geochemical modelling, downhole televiewer surveys to refine vein orientation, and additional LIBS scanning to better characterize lithological and mineralized domains.

26.1.3 Mining

26.1.3.1 Geomechanical

The following geochemical work is recommended to support advancement beyond PFS-level studies:

- Conduct cemented rock fill mix trials to identify designs that meet required strength criteria, including for conditions involving mining beneath cemented rock fill sills.
- Complete additional geotechnical core logging and sampling to support IFC design, including:
 - Laboratory testing (e.g., UCS testing by lithology).
 - Enhanced characterization of joint conditions (e.g., roughness and alteration).

Further evaluation of crown pillar stability over the LOM is recommended, including:



- Drilling a minimum of two boreholes within the planned crown pillar to obtain oriented geotechnical core logging data, with packer testing in zones of low RQD.
- Collection of laboratory samples (e.g., UCS and Brazilian strength tests) from representative lithologies.
- Numerical modelling of crown pillar and upper stope stability, as required based on changes to design assumptions or geological conditions.

26.1.3.2 Mining Optimization

Opportunities have been identified to further improve project economics, including:

- Optimization of mine plan and stope sequencing, evaluation of alternative mining methods, and assessment of overall mine development sequence (e.g., earlier development of the South Portal).
- Continued optimization of cut-off grades to improve resource conversion, extend mine life, and balance economic and sustainability objectives.
- Ongoing evaluation of methodologies for excavation and support of ventilation raise collars particularly in areas characterized by poor-quality overburden.
- Review stope excavation dimensions versus geomechanical recommendations. Confirm smaller stopes with less high cost per tonne vs. large tonnes, lower cost per tonne but dilution and recovery factors.
- Evaluation mining equipment for multiple purpose, i.e. slot raise machines can be use for escape raises, etc.
- Evaluation of emerging technologies to improve operational efficiency and safety, including increased use of automation and semi-autonomous production equipment.

26.1.4 Mineral Processing and Metallurgical Testwork

The following metallurgical testwork is recommended to further characterize processing response of mineralized material from the Grey Fox deposit and to evaluate opportunities for process optimization. As of the publication date of this Technical Report, a metallurgical program is underway that is expected to address items outlined below:

- **Additional Flotation Testwork** to evaluate recovery potential and assess concentrate upgrading through cleaner flotation stages.
- **Preg-Robbing Assessment** using cyanidation testwork with and without activated carbon to confirm potential impact on gold recovery.
- **Sedimentation and Settling Testwork** to establish settling rates, underflow density potential, and flocculant requirements, and to provide design criteria for thickening and tailings management infrastructure.
- **Oxidation and Fine Grinding Evaluation** to assess potential improvement in liberation of gold encapsulated within the pyrite matrix and downstream gold recovery. Future metallurgical programs should evaluate this treatment option to determine its technical effectiveness and economic viability relative to the base case processing flowsheet.



26.1.5 Tailings Storage

Additional evaluation of tailings management alternatives is recommended to align with GISTM Best Available Practices and Best Available Technologies and to reduce risk. In particular, consideration should be given to avoiding tailings deposition at the Stock Mill by evaluating the use of the Black Fox open pit as a tailings management facility.

26.1.6 Surface Infrastructure

Based on the current level of engineering, the following work is recommended:

- **Access and Transportation**
 - Inspection of the Tamarack Road Bridge prior to construction.
 - Completion of hydrotechnical and geotechnical investigations to confirm bridge substructure capacity.
- **Infrastructure and Site Development**
 - Detailed site investigations to refine topsoil volume estimates and support reclamation planning.
 - Investigate bringing increased line power to site for future expansion and electrification options.
 - Confirmation of availability and integration of Black Fox infrastructure to support early works.

26.1.7 Hydrogeology

The current hydrogeological model was developed based on an open pit mining scenario at Grey Fox. It is recommended that the model be updated to reflect the current underground-only mining approach in order to confirm the assumptions regarding groundwater inflows to the underground workings. Based on available data, additional drilling is not considered necessary to complete the model update.

26.2 Estimated Costs

Estimated costs for the recommended work programs are summarized in Table 26-1.

Table 26-1: Estimated Costs for Recommended Work for the Grey Fox Project

Description	Amount (\$ millions)
Early Works	\$6.0
Further Exploration	\$5.0
Geotechnical and mining studies	\$0.9
Mineral Processing and Metallurgical Testwork	\$0.2
Tailings Storage	\$0.1
Surface Infrastructure	\$0.7
Hydrogeology	\$0.6
Total	\$13.5



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28 QP Certificates



CERTIFICATE OF QUALIFICATIONS

I, Mark Hatton, P.Eng, do hereby certify that:

1. I am currently employed as a Mining Engineer by Stantec Consulting Ltd. (Stantec) of 1-1760 Regent St, Sudbury, ON, Canada P3E 3Z8.
2. I graduated from Queen's University in Kingston, Ontario, with a Bachelor of Science Degree in Mining Engineering, in 1994.
3. I am a member in good standing of the Association of Professional Engineers of Ontario (Reg.#90433939).
4. I have practiced as a Mining Engineer since 1994, with 27 years at Stantec and 4 years of on-site mine operations experience at nickel, gold, potash and zinc mines in Canada, Tanzania, and Turkey. My experience includes underground mine design, surface and underground infrastructure, capital and operating cost estimates, and cash flow modelling and financial analysis for gold, nickel, zinc, copper, and uranium mines in Canada and Mexico.
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 meet the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1.1, 1.13, 1.14, 1.16, 1.17, 1.19, 1.20, 1.22, 1.23, 2, 3, 15, 16, 18.1 to 18.7, 19, 21.1, 21.2, 21.2.1, 21.2.4 to 21.3.1, 22, 24, 25.7 to 25.9, 25.12 to 25.15, 26.1.1, 26.1.3, 26.1.6, 26.2, 27 and 28 of the report titled "Grey Fox Project Prefeasibility Study NI 43-101 Technical Report" dated July 17, 2026, with an effective date of June 8, 2026.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I personally inspected the property during a site visit October 27, 2025.
9. I have had no prior involvement with the Grey Fox Project that is subject to this Report.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
12. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

"Original Signed and Sealed by Author"

Mark Hatton, P. Eng.

CERTIFICATE OF QUALIFICATIONS

I, Carson Cybolsky P.Geol, do hereby certify that:

1. I am currently employed as Senior Resource Geologist by McEwen Ontario, a subsidiary of McEwen Inc. (McEwen) 2839 Highway 101 East, Matheson, Ontario, Canada, P0K 1N0.
2. I graduated with a Bachelor of Science (Geology) degree from Laurentian University in 2013.
3. I am a member in-good-standing of the Association of Professional Geoscientists of Ontario (membership #: 3351).
4. I have worked as an Geoscientist since graduating from my undergraduate degree in Geology in 2013. I have 13 years of experience in production geology including 7 years of experience in resource and block model estimation.
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 meet the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1.8 to 1.10, 1.12, 6.4.1, 10.11, 10.12, 11, 12.1 to 12.2, 14.1 to 14.8 and 25.6, of the report titled “Grey Fox Project Prefeasibility Study NI 43-101 Technical Report” dated July 17, 2026, with an effective date of June 8, 2026.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I am a full-time employee of McEwen Ontario and work at the Black Fox Complex Mine Site. I have been with McEwen Ontario since 2019.
9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
11. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

“Original Signed and Sealed by Author”

Carson Cybolsky, P.Geol.

CERTIFICATE OF QUALIFICATIONS

I, Sean Bradley Farrell, P.Ge., do hereby certify that:

1. I am currently employed as Exploration Manager by McEwen Ontario, a subsidiary of McEwen Inc. (McEwen), 2839 Hwy 101 East, P.O. Box 605, Matheson, Ontario, Canada, P0K 1N0.
2. I graduated with a B.Sc. (hons) degree from Carleton University in 2001 and a M.Sc. degree from the University of Ottawa in 2005.
3. I am a member in-good-standing of the Association of Professional Geoscientists of Ontario (membership #: 2161).
4. I have worked as a Geoscientist since graduating from post graduate degree in 2005. I have 21 years of experience in the evaluation and analysis of underground mineable projects.
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 meet the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections: 1.2 to 1.7, 1.21, 4, 5, 6.1 to 6.4, 6.5, 7 to 9, 10.1 to 1.10, 23, 25.1 to 25.4 and 26.1.2, of the report titled “Grey Fox Project Prefeasibility Study NI 43-101 Technical Report” dated July 17, 2026, with an effective date of June 8, 2026.

I am a full-time employee of McEwen Ontario and work at the Black Fox Complex Mine Site. I have been with McEwen Ontario since 2019.
7. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
9. I am not independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

“Original Signed and Sealed by Author”

Sean Bradley Farrell, P.Ge.

CERTIFICATE OF QUALIFICATIONS

I, William Luke Willis, P.Geol., do hereby certify that:

1. I am currently employed as Director, Resource Modelling and Senior Geologist by McEwen Inc., 150 King Street West, Toronto, Ontario, Canada M5H 1J9.
2. I am a graduate of the University of Wales, College of Cardiff, UK with a BSc. degree in Exploration and Mining Geology (1995).
3. I am registered as a Professional Geoscientist with the Professional Geoscientists Ontario (Reg.# 2146).
4. I have practiced my profession as a Geologist in Exploration, Mining and Resource Geology for over 25 years.
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 meet the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 14.9 and 14.10 of the report titled “Grey Fox Project Prefeasibility Study NI 43-101 Technical Report” dated July 17, 2026, with an effective date of June 8, 2026.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I personally inspected the property during a site visit January 2025 and make site visits on a regular basis to the greater Fox Complex .
9. As an employee of McEwen Inc., I have had continual involvement in a corporate role with the Fox Complex since 2017, when McEwen acquired the property.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
12. I am not independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

“Original Signed and Sealed by Author”

William Luke Willis, P.Geol.

CERTIFICATE OF QUALIFICATIONS

I, Gerd Michael Wiatzka P.Eng, do hereby certify that:

1. I am currently employed as a National Expert, Vice President and Director Mining of Arcadis Professional Services (Canada) Inc. (Arcadis), 55 St Clair West, Toronto, Ontario, M4V 2Y7.
2. I graduated with a Bachelor Applied (Hons) Science degree from the University of Waterloo, Ontario, in 1974.
3. I am a member in-good-standing of the Association of Professional Engineers of Ontario (Reg.#49882012), the Northwest Territories and Nunavut Association of Professional Engineers (Reg.#L1701), and the Engineers Geoscientists Manitoba (Reg.#47832).
4. I have worked as a Civil/Environmental Engineer since graduation with more than 45 professional years of experience primarily in the resource sector including more than 35 years' experience as an environmental professional and worldwide project experience in the mining sector spanning engineering design, construction management, environmental assessments, closure planning, due diligence and liability assessments, and NI 43-101 reviews of major mining operations.
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 meet the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 12.3 to 12.5, 18.10 to 18.12, 21.2.3, 21.3.3, 21.3.4, 25.10, 25.11, and 26.1.5 of the report titled "Grey Fox Project Prefeasibility Study NI 43-101 Technical Report" dated July 17, 2026, with an effective date of June 8, 2026.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I personally inspected the property during a site visit October 27, 2025.
9. I have had no prior involvement with the Project that is subject to this Report.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
12. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

"Original Signed and Sealed by Author"

Gerd Michael Wiatzka, P. Eng.

CERTIFICATE OF QUALIFICATIONS

I, Daniel Weich Morrison P Eng, do hereby certify that:

1. I am currently employed as Principal Process Engineer by DRA Global, 20 Queen Street West, Toronto, ON, M5H 3R3.
2. I graduated with a Bachelor of Engineering degree from the University of Stellenbosch, South Africa in 1990.
3. I am a member in-good-standing of the Professional Engineers of Ontario (Membership Number 100134360)
4. I have worked as an Engineer since graduating from my undergraduate degree in Chemical Engineering. I have 34 years of experience in the design and operation of metallurgical process plants.
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 meet the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1.11, 1.15, 12.6, 13, 17, 18.8, 21.2.2, 21.3.2, 25.5 and 26.1.4, of the report titled “Grey Fox Project Prefeasibility Study NI 43-101 Technical Report” dated July 17, 2026, with an effective date of June 8, 2026.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I personally inspected the property during two site visits dated 15 to 18 Jul 2025 and 23 to 26 Feb 2026.
9. I have had no prior involvement with the Project that is subject to this Report. OR explain prior involvement
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
12. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

“Original Signed and Sealed by Author”

Daniel Weich Morrison, P. Eng.

CERTIFICATE OF QUALIFICATIONS

I, Sheldon Smith, MES, P.Geo. do hereby certify that:

1. I am currently employed as a Senior Hydrologist with Stantec Consulting Ltd., ("Stantec"), 300-125 Commerce Valley Drive West, Markham, Ontario, Canada L3T 7W4.
2. I graduated from the Memorial University of Newfoundland in St. John's, NL in 1994 with a B.Sc.(H) in Physical Geography and the University of Waterloo, Waterloo, Ontario in 1998 with a Master of Environmental Studies.
3. I am a member in-good-standing of the Professional Geoscientists of Ontario (Member #1418).
4. I have practiced my profession continuously for over 30 years with experience in mine environmental and water management from over 40 similar studies or projects including Vale at more than 25 locations in Canada and South America, Glencore, Newmont, Barrick, Newfoundland Gold, Firefly Metals, Alderon Iron Ore, Century Iron Mines, Altius Resources, Palladin/Aurora Energy, Atlantic Gold, Trevali, Thomas Resources, Marathon Gold (Equinox), Canada Nickel, Baffinland, Premier Gold, Greenstone Gold (Equinox), Alamos, Wesdome, Norcliff Resources, DeBeers, Richmond, Ontario Graphite, Northern Graphite, Ferromin Inc., KGHM, Pan American Silver, Signal Gold, Generation PGM, Treasury Metals, Clean Air Metals, Matador Mining, Wallbridge, NorthX, Magna Mining, McEwen Mining, Kinross and others.
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 meet the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1.18, 18.9, 20, and 26.1.7 of the report titled "Grey Fox Project Prefeasibility Study NI 43-101 Technical Report" dated July 17, 2026, with an effective date of June 8, 2026.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8. I have not personally inspected the property.
9. I have had no prior involvement with the Grey Fox Project that is subject to this Report.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.
12. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated this 17th day of July 2026.

"Original Signed and Sealed by Author"

Sheldon Smith, MES, P.Geo.

With every community, we redefine what's possible.

Stantec is a global leader in sustainable engineering, architecture, and environmental consulting. The diverse perspectives of our partners and interested parties drive us to think beyond what's previously been done on critical issues like climate change, digital transformation, and future-proofing our cities and infrastructure. We innovate at the intersection of community, creativity, and client relationships to advance communities everywhere, so that together we can redefine what's possible.



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