



NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada

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1.0 SUMMARY

1.1 Introduction

WSP Canada Inc. (WSP) was commissioned by Fuerte Metals Corporation (Fuerte), in coordination with other consultants (Micon, SRK, Ensero, Forte Dynamics, Lorax & Open Contour), to complete a Preliminary Economic Assessment (PEA) and prepare a technical report in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for the Coffee Gold Project (or Project).

This Report was prepared to support the results of the PEA presented in news release “Fuerte Announces a Positive Preliminary Economic Assessment for the Coffee Gold Project; Positioning the Company as one of Canada’s Next Gold Producers” on February 22, 2026, followed by a second Press Release on April 8, 2026, “Fuerte Files Technical Report for the Coffee Gold Project”. The purpose of this technical report is to present the PEA on the Coffee Gold Project and to make recommendations on the programs of work required to move the Project to the next stages of study and development.

Metric units are used throughout this report, and all dollar amounts are reported in CA and U.S. Dollars unless otherwise stated.

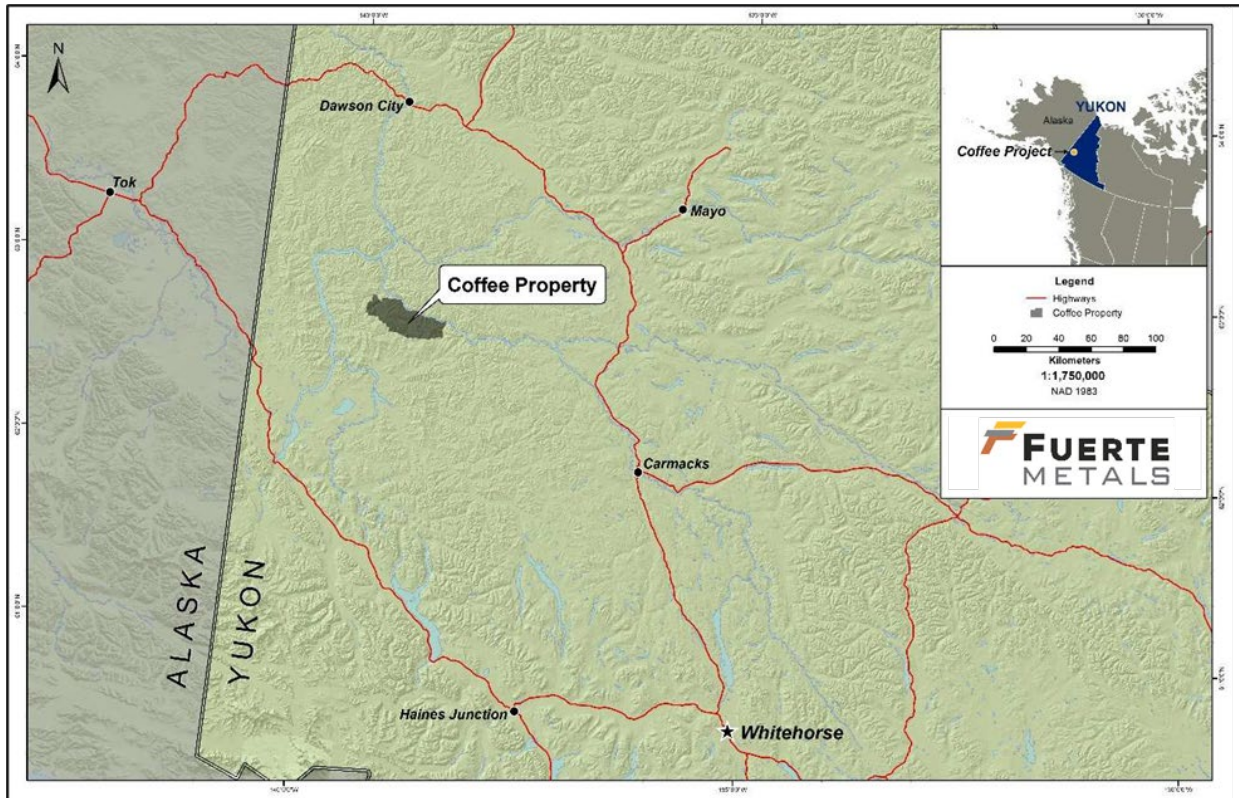
1.2 Property Description, Location and Access

The Coffee Gold Project comprises 3,542 contiguous claims covering an aggregate area of approximately 70,256 ha and lies within the Traditional Territories of the Tr’ondëk Hwëch’in and Selkirk First Nation, and the asserted traditional territory of the White River First Nation.

It is in the Whitehorse Mining District of west-central Yukon Territory, Canada, approximately 130 km south of Dawson and about 160 km northwest of Carmacks (Figure 1.1). The Coffee site is currently accessible by air or by barge along the Yukon River. A proposed 214 km all-season road (the Northern Access Route or NAR) from Dawson is currently in the permitting process, with permits expected to be received in the first half of 2026. Construction of the NAR is anticipated to commence in 2026, with the contract awarded and equipment mobilized. A 1,000 m all-weather airstrip has been permitted under the exploration permit (MLU LQ00552) and will supplement the existing airstrip, allowing for larger aircraft and more efficient use.



Figure 1.1
Location of the Coffee Gold Project



Modified after the JDS in-house engineering report 2024

Agreements

The Coffee Gold Project was previously owned by Goldcorp Kaminak Ltd. (Kaminak), an indirect subsidiary of Newmont Corporation (Newmont). Fuerte entered into an agreement dated September 12, 2025 to acquire all of the shares of Kaminak from a wholly owned subsidiary of Newmont in consideration for: (i) payment in cash in the amount of US\$10M; (ii) the issuance of common shares and preferred shares of Fuerte having an aggregate value of US\$40M; and (iii) the assumption of a 3% net smelter return (NSR) on the Coffee Gold Project which is payable to the Newmont and may be repurchased by Fuerte for US\$100 million within 12 months of the commencement of commercial production. The transaction closed on October 17, 2025, and the Project is now 100% owned by Fuerte under the indirect subsidiary Kaminak Gold Ltd.

1.3 Geology And Mineralization

1.3.1 Regional Setting

The Coffee Gold Project is in an allochthonous peri-cratonic terrane known the Yukon-Tanana Terrane (YTT). This terrane has a protracted history of rifting, magmatism, and metamorphism spanning from the Late Devonian into the Mesozoic. Numerous orogenic and magmatic-hydrothermal mineral deposits occur throughout the YTT as a result of orogenesis and widespread magmatism.

The Dawson Range in which the Coffee Gold Project is housed is a west-northwest-trending topographic plateau (approximately 250 km) in the YTT that remained unglaciated during the last ice age. In the Dawson



Range the metamorphic rocks of the YTT consist of psammitic to semi-pelitic schists of the Snowcap assemblage, orthogneiss of the Sulphur Creek and Simpson Range suites, felsic to mafic metavolcanic and metasedimentary rocks of the Finlayson assemblage, micaceous schist of the Klondike assemblage, and undifferentiated ultramafic rocks (Ryan et al., 2013a, 2013b; MacWilliam, 2018). The metamorphic units are intruded by granite of the mid-Cretaceous Dawson Range batholith (DRB) and the Coffee Creek pluton (CCP), both of which are part of the Whitehorse plutonic suite (Ryan et al., 2013a, 2013b).

1.3.2 Property Geology

The host rocks at Coffee were deformed by a series of YTT-wide Paleozoic tectonic events, as well as Cretaceous deformation (Nelson et al., 2013; MacWilliam, 2018).

Based on recent work completed under the auspices of the Yukon Geological Survey (YGS), 22 map units occur on the Coffee Gold Project. The pre-Late Devonian meta-sedimentary rocks of the Snowcap assemblage are structurally overlying Late Permian meta-granitoid rocks of the Klondike assemblage, with the contact presumed to be a transposed intrusive contact (MacWilliam, 2018). Amphibolite and metabasalt occur at the base of the Snowcap assemblage, and undifferentiated ultramafic units occur as lenses throughout the assemblage (MacWilliam, 2018). Finlayson assemblage rocks are observed near the southern end of the property and stratigraphically overlie Snowcap assemblage psammite. The undeformed CCP intrudes the Paleozoic metamorphic assemblage, as do a series of Cretaceous mafic to felsic dikes. These Cretaceous dikes are spatially associated with subvertical fault-fracture networks that are the primary controls of mineralization at Coffee. These faults are interpreted as high-order structures related to the Coffee Creek fault (CCF) (MacWilliam, 2018).

1.3.3 Mineralization/Deposit Type

The mineral endowment at Coffee is hosted within several zones each with varying host rocks and structural orientations. The bulk of the mineral resource occurs within the Supremo zone which is predominantly hosted within augen-bearing orthogneiss of the Sulphur Creek suite. The Supremo zone is further broken down into different 'T'-structures named after exploration trenches (T1 through T9, excluding T6); the most significant of which is T3 which trends north-northeast and extends for approximately 5 km of strike length. The Supremo T-structures are mostly north-northwest to northeast-trending and form an interconnected array of fault-fracture networks. The Latte and Double-Double zones are located south of Supremo, and each consist of nearly east-west trending structures that crosscut psammitic to semi-pelitic schists of the Snowcap assemblage. The T-structures intersect Latte and Double-Double south of the Snowcap-Sulphur Creek contact. The Kona and Kona North zones are the main bodies of mineralization hosted within the CCP. These zones consist of east-northeast trending fault-fracture networks. Other mineralized zones at Coffee are situated in east-west trending fault fracture networks like Kazaar, Forte and Americano; north-northwest to north-northeast trending structures like Arabica, Decaf and French Press; and northeast trending structures like Supremo Extension and Sumatragur. Other mineral occurrences within the Coffee Gold Project that are more distal to the Coffee deposit, such as the Sugar occurrence, are not discussed in detail in this report.

Gold at Coffee occurs as fine-grained auriferous pyrite, arsenopyrite and arsenian pyrite. The mineralization is controlled by subvertical fault-fracture networks, characterized by variably mature, polyphase tectonic fault gouge and fault-fill breccias and cataclasites.

1.3.4 Status Of Exploration

Exploration work conducted includes: Mineral Project and deposit-scale mapping, geochemical soil and silt sampling, Bulk Leach-Extractable Gold (BLEG) and heavy mineral stream sediment sampling, biochemical sampling, ground and airborne geophysical surveys (VLF-EM, VTEM, ELF-EM, HLEM, Ohm Mapper, GPR, H-V and MASW seismic, magnetics, IP resistivity, gravity, and borehole EM), airborne



radiometrics, trenching, reverse circulation drilling (365,112 m), rotary air blast drilling (2,197 m), sonic drilling (156 m), and diamond drilling (242,242 m) which includes drilling for geotechnical purposes (3,010 m).

The exploration/drilling completed to date has led to the discovery of gold mineralization in 15 separate areas of the Coffee Gold Project: Supremo, Sumatra, Latte, Double-Double, Arabica, Americano West, Americano, Espresso, Kona, Kona North, Supremo Extension, Cappuccino, Dolce, French Press and Sugar. Gold mineralization occurs in narrow to broad gold-bearing locally brecciated structures with quartz, albite, ankerite, dolomite, sericite, and pyrite alteration.

1.4 Deposit Types

The Micon QP has reviewed the descriptions of the deposit types presented in the JDS in-house engineering report for Newmont and concurs with the following statement: “Coffee is a mid-Cretaceous, gold-only deposit that has been classified as a variety of deposit types over the Project’s history including epithermal, reduced intrusion related, Carlin-type, and orogenic-gold, as well as combinations of the four proposed deposit types (i.e., over-printing or combination of fluid sources).” This statement implies that currently, the Coffee Gold Project deposit type(s)/genetic model is yet to be resolved.

1.5 Exploration

Exploration work conducted on the Coffee Gold Project includes the following: property and deposit-scale mapping, geochemical soil and silt sampling, bulk leach-extractable gold (BLEG) and heavy mineral stream sediment sampling, biochemical sampling, ground and airborne geophysical surveys (VLF-EM, VTEM, ELF-EM, HLEM, Ohm Mapper, GPR, H-V and MASW seismic, magnetics, IP resistivity, gravity, and borehole EM), and airborne radiometric surveys.

1.6 Drilling

Micon’s QP has not identified any drilling, sampling or recovery factors that could result in sampling bias or otherwise materially impact the accuracy and reliability of the assays and, hence, the resource database. Core recoveries of >90% were confirmed during the site visit.

In summary, the drilling campaigns have successfully outlined a large mineral resource of over 3 million ounces of gold, as detailed in this Technical Report.

1.7 Sample Preparation

Following a thorough review the QA/QC results plus the accompanying Quality Control Reports from Issuers, the Micon QP concludes that the data provided by the Issuers and laboratories is adequately reliable for the purposes of mineral resource estimation.

1.8 Data Verification

The MRE data used in this Technical Report were generated in a credible manner with proper procedures and have been accurately transcribed from the assay certificates and are of sufficient analytical quality to support a MRE for the Coffee Gold Project.

1.9 Mineral Processing and Metallurgical Testing

Metallurgical testing for the Coffee Gold deposits commenced in 2011 and has continued since its inception. The initial testwork, pre-2013, was carried out by Inspectorate Exploration & Mining Services Ltd. (Inspectorate) of Richmond, British Columbia. Since 2013, metallurgical testing has been completed by



several independent laboratories, including Kappes, Cassiday and Associates (KCA) in Reno, Nevada; SGS Canada in Burnaby, British Columbia; Phillips Enterprises, LLC in Golden in Colorado, ALS Metallurgy Kamloops in Kamloops, British Columbia; and Forte Dynamics (Forte) in Fort Collins, Colorado..

The metallurgical testing utilized samples derived from both bulk surface samples and drill core composites. Testing included column leaching, bottle roll leaching, vat leaching, pressure oxidation and roasting, flotation, column percolation and drain down, multi-element head assay analyses, column leach head and tailings assay screen analyses, ball mill work indices, crushing impact, and abrasion indices.

The samples used for the various testwork programs completed to date were not only representative of all the main types of lithologies and mineralized material types identified at the Project, but also included mineralization from all of the identified individual deposits.

The mineralized material at Coffee demonstrated amenability to the heap leaching process. Generally, oxide material had high recoveries, transition material showed moderate results, while the sulphide samples had poor recovery, indicating refractory behavior. Temperature impacts on leaching recovery has been tested and the results indicated that minimal variation in leaching kinetics is to be expected between a cold temperature of 4°C and warm ambient temperature of 22°C. Based on the agglomeration and compacted permeability tests, cement agglomeration is not considered necessary.

Commercial heap leach operations typically achieve lower gold recoveries than laboratory tests suggest. This difference is due to various factors specific to the site and operation, such as local climate, crushing circuit design, heap leach pad design, stacking, and overall operating approach. A recovery discount of approximately 3-5% was recommended by KCA in their testwork reports, which is a general accepted practice. Actual recoveries in an operating heap leach pad and associated process plant may vary due to operating conditions such as weather, crushing plant design criteria, heap leach pad design and operating philosophy.

Table 1.1 provides an overview of the adjusted final average gold recoveries and the amount of reagents used, based on bottle roll and column leach tests carried out by KCA and SGS. It includes gold recovery data for Latte, Supremo, Double-Double, and Kona, broken down by material type (oxide or transition).

**Table 1.1
Estimated Gold Recovery and Reagent Consumption**

Description	Gold Recovery (%)*	Reagent Consumption	
		NaCN (kg/t)	Lime Addition (kg/t)
Latte			
Oxide	88.6	0.2	1.2
Upper Transition	77.4	0.2	1.4
Middle Transition	60.3	0.3	1.5
Lower Transition	30.0	0.3	1.6
Supremo			
Oxide	87.2	0.3	1.4
Upper Transition	79.3	0.2	1.3
Middle Transition	52.6	0.2	1.5
Lower Transition	34.2	0.2	1.5
Double-Double			
Oxide	89.1	0.2	1.5
Upper Transition	77.0	0.3	1.5
Middle Transition	42.6	0.3	1.6



Description	Gold Recovery (%)*	Reagent Consumption	
		NaCN (kg/t)	Lime Addition (kg/t)
Lower Transition	30.2	0.3	1.5
Kona			
Oxide	83.0	0.2	1.6
Upper Transition	71.3	0.2	1.5
Middle Transition	57.3	0.3	1.5
Lower Transition	28.6	0.3	1.5

Source: KCA (2024)

* The recoveries presented in the table reflect laboratory test results adjusted by a 3-5% discount to account for anticipated field heap leaching conditions. Actual recoveries may differ due to various influencing factors.

1.10 Mineral Resource Estimate

The estimation work was completed under the supervision of Micon's Qualified Person (QP), Charley Murahwi, P. Geo. FAusIMM, in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines adopted on November 29, 2019 (2019 CIM Best Practice Guidelines). The MRE has an effective date of August 21, 2025. The modelling work was prepared internally by Fuerte technical staff and reviewed, edited and accepted by the Micon QP.

After applying the technical and economic assumptions for open pit mining to the block models with a constraining pit shell, the total mineral resource, summarized across all areas (Supremo, Latte, Double-Double, Supremo Extension, Arabica, Cappuccino, French Press, and Kona), is reported in Table 1.2.

Table 1.2
Coffee Gold Project Mineral Resource Statement, Effective Date: August 21, 2025

Area	Category	Tonnage (kt)	Gold Grade (g/t)	Gold Ounces (koz)	Strip Ratio
All Areas	Measured	1,200	1.80	69	5.1
	Indicated	78,846	1.14	2,888	
	M+I	80,046	1.15	2,957	
	Inferred	21,200	1.17	800	
Notes:					
<ol style="list-style-type: none"> Economic parameters used in the resource are a gold price of US\$2,500/oz; heap leach average recoveries for the individual metallurgical domains of 86.3% for Oxide, 76.0% for Upper Transition, 54.5% for Middle Transition and 31.4% for Lower Transition; a mining cost of C\$3.27-\$3.50/t, processing costs of C\$6.64/t, and general and administrative costs of C\$6.0/t. A CA\$:US\$ exchange rate of 1.35 was also assumed. The calculated cut-off grades vary between 0.13 g/t Au and 0.48 g/t Au, depending on the metallurgical domain. The global weighted average cut-off grade is 0.18 g/t Au, with domain tonnage contributions comprising 64% Oxide, 18% Upper Transition, 5% Middle Transition, and 13% Lower Transition. Pit slope angles vary between 45.0 and 48.8 degrees depending on the pit area. Pit optimization was done on 12x12x10 m re-block model with a minimum of 4x4x5 m regularized SMU. Numbers have been rounded to the nearest thousand for tonnes and ounces. Differences may occur in totals due to rounding. The mineral resources described above have been prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards and Practices. Charley Murahwi, P. Geo., FAusIMM, from Micon International Limited is the Qualified Person (QP) for this Mineral Resource Estimate. Mineral resources unlike mineral reserves have no demonstrated economic viability. The tonnages and grade of the reported Indicated and Inferred mineral resources in this Technical Report are uncertain in nature; however, it is reasonably expected that significant portions of the Indicated and Inferred Mineral Resources can be upgraded into Measured and Indicated Mineral Resources, respectively, with further exploration field work. 					



Area	Category	Tonnage (kt)	Gold Grade (g/t)	Gold Ounces (koz)	Strip Ratio
9. Micon's QP has not identified any legal, political, environmental, or other factors that could materially affect the potential development of this Mineral Resource.					

The MRE is supported by a geological database compiled from drillholes completed by Kaminak, Goldcorp, and Newmont between 2010 and 2023. The full database contained 4,638 drillholes, of which 4,029 drillholes (379,130 m of sampling) were used in the mineral resource estimation. The mineral resource database includes 27.1% diamond drillholes (core) and 72.9% reverse circulation (RC) drillholes.

Key steps in the estimation included:

1. Geological Modelling: Gold mineralization is structurally controlled and was modelled using the Vein System tool in Leapfrog Geo, based on a 0.2 g/t Au cut-off grade for domain construction. This resulted in 478 individual vein solids grouped into 38 mineralized estimation domains.
2. Oxidation: Oxidation was modelled using the ratio of cyanide-soluble gold assay (Au_AA) to fire assay (Au_FA), classifying material into Oxide (OX), Upper Transitional (UT), Middle Transitional (MT), Lower Transitional (LT), and Sulphide (SU) types.
3. Compositing and Capping: A 1.0 m composite interval was selected for estimation. Grade capping analysis was performed, resulting in metal loss of less than 5% for most mineralized domains.
4. Grade Interpolation: Ordinary Kriging (OK) was selected as the final interpolation method. Other methods such as Inverse distance cubed (ID3), Nearest Neighbour (NN), were also used for comparison purposes.
5. Density: Bulk density values were assigned based on a combination of host lithology, oxidation type, and mineral domain.

Mineral resource classification criteria were determined based on the range of variogram models:

- Measured Resource: Categorized based on a three-hole drill spacing of 15 m or less.
- Indicated Resource: Categorized based on a three-hole drill spacing of 30 m or less.
- Inferred Resource: Categorized based on a three-hole drill spacing of 60 m or less.

Model validation checks, including statistical comparison between OK, NN, and ID3 estimates and visual validation using swath plots and sectional validation, demonstrated that the OK estimates reproduce composite grades well.

The MRE satisfies the Reasonable Prospects for Eventual Economic Extraction (RPEEE) criteria, which was determined using pit optimization (Lerchs-Grossmann algorithm in Datamine NPVS) based on specific economic and technical assumptions summarized in Table 1.3.



**Table 1.3
Coffee Gold Project Economic and Technical Pit Optimization Assumptions and Gold Cut-off
Grades for Resource**

Item	Unit	Coffee Gold Project Areas				
		Supremo	Latte	Double-Double	Kona	Other Areas
Exchange Rate	US\$/CA\$	1.35				
Gold Price	US\$/oz	2,500.00				
Processing Cost	CA\$/t	6.64				
G&A Cost	CA\$/t	6.00				
Gold Transportation & Refining	CA\$/oz	1.30				
Gold Royalty	%	2.38				
Gold Payability	%	99.80				
Mining Cost	CA\$/t	3.27	3.30	3.32	3.50	3.50
Gold Metallurgical Recoveries						
Oxide (OX)	%	87.20	88.60	89.10	83.00	80.00
Upper Transition (UT)	%	79.30	77.40	77.00	71.30	70.00
Middle Transition (MT)	%	52.60	60.30	42.60	57.30	50.00
Lower Transition (LT)	%	34.20	30.00	30.20	28.60	25.00
Open Pit Optimization Parameters						
Overall Slope Angle	degrees	48.8	46.5	47.8	47.8	45.0
Block Model re-blocked (X, Y, Z)	m	12 x 12 x 10				
Regularized SMU (X, Y, Z)	m	4 x 4 x 5				
Mining Recovery	%	100				
Mining Dilution	%	0				
Minimum mining width (bottom)	m	12				
Gold Cut-off Grades						
Oxide (OX)	g/t	0.14	0.14	0.13	0.14	0.15
Upper Transition (UT)	g/t	0.15	0.15	0.16	0.17	0.17
Middle Transition (MT)	g/t	0.23	0.20	0.28	0.21	0.24
Lower Transition (LT)	g/t	0.35	0.40	0.40	0.42	0.48

1.11 Mining Methods

The Coffee Gold Project is an advanced project that can be mined as an open pit using conventional mining methods. The main mining risks involve climatic influences and dilution if not mitigated through effective operational practices.

The Coffee Gold Project is a shallow high-grade deposit that is amenable to development as an open pit mine. This study outlines up to eight separate pits totaling 778 Mt with 90.5 Mt as plant feed at a 7.5:1 strip ratio using the parameters outlined in this Report. Open pit mining is assumed to be carried out by the owner using conventional drill, blast, load, and haul methods. The schedule mines between 30 to 85 Mt of material within 12 years. The plan schedule mines 15 separate phases and optimizes value by focusing on mining the highest value phases early in the schedule. Scheduling used truck hours as the basis rather than tonnage given the variation in haulage distances. The schedule focused on maintaining consistent crusher feed while minimizing stockpiling on an annual basis.

Open pit mining will use a fleet of owner-owned 16 m³ shovels, 13 m³ front-end loaders (FEL) and 150 t haul trucks. Mining will use a 10 m bench configuration. Double-benching will optimize catch-bench width.



1.11.1 Geotechnical Analysis and Recommendations

Open pit geotechnical design criteria were initially developed for the pits by SRK (2016) and then updated by JDS in 2018 based on additional geotechnical core drilling, strength testing and downhole televiwer surveying. A total of 13 HQ3 diameter core holes were logged and tested between the two programs totalling 2,105 m in length. Both geotechnical drilling campaigns consisted of detailed geotechnical core logging and discontinuity orientation, field point load testing, and laboratory strength testing of core samples.

Review of the geotechnical conditions indicated the design criteria will need to address the effects of structural controls rather than rock mass strength. Results of the rock mass characterization suggest that (other than the oxide materials which will be mostly mined and processed) the rock mass at Coffee is generally of 'Fair' to 'Good' geomechanical quality according to the Bieniawski (1989) criteria.

Review of the rock mass fabric and structural characterization broadly indicates dominant, shallow to moderately south-southwest dipping foliation that has a high potential to create planar instabilities and will likely control achievable bench face angles in the pit north walls. Due mostly to the north-south orientation of the pit, the east and west pit wall benches are not anticipated to be significantly impacted by structurally controlled instabilities based on the current understanding of the structural model.

Bench design and kinematic analyses are based solely on orientations of geologic structure and do not directly consider effects of weathering, alteration, blasting or excavation techniques. Recommendations for south facing pit walls assume that the achievable bench face angles are controlled by an assumed amount of partial backbreak along foliation sets. This assumption will require follow-up assessment in the next stage of geotechnical study.

1.12 Recovery Methods

1.12.1 Crushing and ADR

The selection of the flowsheet, operating parameters, and design criteria are based on metallurgical test results and are consistent with the updated mine plan. The gold recovery process is designed to treat 7.4 Mt/a with an average gold head grade of 1.25 g/t and an overall gold recovery rate of 77.5%.

The main processing steps include crushing, heap leaching, and adsorption-desorption-recovery (ADR). A two-stage crushing plant will operate at an average throughput of 1,495 t/h, running 18 h/d across 275 d/a. The crushed material will have a target size of 80% passing (P_{80}) at 50 mm before being sent to the heap leach pad. Primary crushing uses a gyratory crusher, followed by secondary crushing with a cone crusher. Following crushing, material will be transported to the heap leach pad via overland and grasshopper conveyors. Crushing and heap leach pad loading will pause during the coldest period each year, which is assumed to be January through March.

A barren leach solution containing cyanide will be pumped to the top of the heap, where it percolates through the heap and dissolves gold. The resulting Pregnant Leach Solution (PLS) will be collected and fed into the Carbon-In-Column (CIC) circuit in the ADR process plant. The CIC circuit consists of six sequential, casCAing carbon columns. The solution will pass through a trash removal screen and then flow counter-currently to the movement of carbon, advancing from column 1 to column 6. Overflow from the final column is discharged onto a recovery screen to capture any carbon particles carried forward, after which the solution flows to the barren solution tank for preparation and recirculation to the heap leach pad.

Loaded carbon from the CIC circuits will be processed by acid wash and elution. Finally, the gold-rich solution will be processed through electrowinning and refining to produce gold doré as the finished product.



1.12.2 Heap Leach Facility

A conventional heap leach facility (HLF) will be developed in stages and is designed with a capacity to process 91 Mt of crushed leached material following the mine schedule. As material is placed on the leach pad, irrigation with barren solution containing cyanide will percolate the heap to dissolve gold. The gold-bearing process solution will be collected at the base of the lined heap leach pad via a network of collection piping, and the process solution will be routed to the process plant for gold recovery and reagent reconditioning. Once an area has been leached for the target time or metal recovery, the next lift will be placed on top of the already leached material and the process is repeated. Several cold weather engineering controls will ensure continuous recovery during extreme winter conditions, including heating of barren solution, insulated and heat-traced piping, and buried drip irrigation lines. While leaching and gold recovery is continuous, it is expected that stacking operations would cease during periods of extreme low temperatures.

The HLF includes perimeter channels and event ponds to collect meteoric water that enters the process system. Water collected in the event ponds will be used as make up water to meet the nominal process flow rate, with excess meteoric water being removed for other mine uses or water treatment.

1.13 Project Infrastructure

1.13.1 General Infrastructure

The key facilities will be installed near the process plant to improve operational efficiency and reliability in challenging winter conditions. The infrastructure ensures year-round access where possible, with plans for seasonal intermittent access, and emphasizes self-sufficient utilities, water, and waste management.

The major infrastructure at the Coffee site includes:

- Access and site roads
- Heap leach facilities and event ponds
- Airstrip
- Power plant and power distribution
- Fuel storage
- Process-related facilities
- Truck shop, warehouse, and storage areas
- Mine dry and office complex
- Camps
- Waste management facilities
- Water treatment plants
- Explosives storage



- Water supply, and
- Site water management facilities.

The Northern Access Route (NAR), preferred for its lower cost and reduced environmental impact, connects the site to the Dawson City via a 214 km all-weather road. Logistics account for seasonal river crossings using barges or ice bridges, with consumable storage to cover up to 15 weeks of potential isolation. Onsite roads are built from local materials and the airstrip supports personnel and freight transfers. Centralized buildings support mining operations, while explosives and fuel storage are designed for extended disruptions.

Power is generated centrally with a mix of LNG and diesel generators and distributed throughout the site. Raw water is supplied from the Yukon River via dedicated pumping systems.

Water management strategy is to ensure adequate flexibility in water management to meet the operational needs of the Coffee Gold Project while remaining protective of the receiving environments downstream of the Coffee Gold Project.

1.13.2 Water Management

The water management program for the Coffee Gold Project is designed to provide flexible and reliable control of contact and non-contact water while protecting downstream receiving environments and maintaining the physical stability of site infrastructure. It integrates local climate and hydrologic data with engineered infrastructure, including collection ponds (with the largest being the Alpha Pond), diversion channels (with the largest being the Halfway Creek Diversion), and pit dewatering systems, to manage runoff, sediment, and potential water quality effects. Key facilities such as the Alpha Pond and Facility Pond are sized to meet regulatory requirements and to handle design flood events, while diversion channels and rock drains are configured to minimize disturbance to natural drainage patterns and maintain long-term system performance during operations and into closure.

1.13.3 Waste Rock Management

Waste rock management at the Coffee Gold Project will be centered on the Alpha Waste Rock Storage Facility (WRSF) and the strategic backfilling of mined-out pits to optimize storage, minimize disturbance, and maintain long-term geotechnical stability. Over the 13-year active mining period, approximately 688 Mt of waste rock are projected to be generated, with annual volumes varying according to the mine schedule and number of active pits. The Alpha WRSF will be developed in stages, including engineered rock drains and careful placement over variable permafrost and ice-rich foundations to maintain stability and control drainage, while pit backfilling will account for swell and haul distances to efficiently utilize void space and reduce the overall surface footprint of waste storage.

1.13.4 Heap Leach Facility Water Treatment

The proposed water treatment plant (WTP) is designed to treat for cyanide, ammonia, several metals, nitrate, and dissolved solids. The treatment stages include chemical, biochemical and membrane treatment processes. The treated effluent from the WTP flows by gravity to Alpha WRSF rock drain to the Alpha Pond and ultimately to Halfway Creek. The treatment process will produce sludge streams high in solids and as a concentrated brine stream rich in dissolved solids. The sludge streams will be dewatered, and the solids will be disposed, while the brine stream will be treated and recirculated back into the HLF. The treatment is expected to commence in year 2 of the mine operation, and to continue through year 13. From year 14 onwards the WTP is expected to operate at a lower flowrate during closure until it is no longer required.



1.14 Market Studies and Contracts

The gold market is mature globally with numerous reputable refiners readily available. As such, a market study for the saleability of gold products or for the expected price of gold was not undertaken. 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.

The QP (William Richard McBride, P. Eng.) has adopted a price projection for the PEA economic analysis model representative of the average value of the consensus opinions of several financial institutions during the December 2025 to January 2026 period.

The consensus price used was US\$4,110/oz Au for Y1 of production and US\$3,620/oz Au long term for the balance of the life-of mine (LOM). The gold price averages US\$3,656/oz Au over the 13-year processing period and was viewed as conservative compared to the February 18, 2026 end of business (EOB) spot price of near US\$5,000/oz.

Contract terms and refining charges are yet to be established with refining companies. It is expected that any future contract terms will be typical for the industry.

1.15 Environmental Studies, Permitting and Social or Community Impact

1.15.1 Environmental Baseline Studies

Baseline studies across disciplines have been undertaken since 2010 and continue to be ongoing to support the Water Licence and Quartz Mining Licence applications and to meet the Decision Document for the Coffee Gold Mine Project (YESAB File No. 2017-0211).

Engagement on the Project occurred as part of the environmental and socio-economic process (YESAB file number: 2017-0211) with the Tr'ondëk Hwëch'in (TH), Selkirk First Nation, White River First Nation, and the First Nation of Na-cho Nyäk Dun (collectively the "Affected First Nations"). Engagement with several First Nation communities is ongoing.

1.15.2 Closure, Decommissioning and Reclamation

A Reclamation and Closure Plan (RCP) and associated closure liability estimate has previously been prepared for the then defined project. The liability estimate was updated to reflect project amendments as a result of this PEA. Primary uncertainties and pre-existing closure planning and cost assumptions that have been carried forward and expanded into the PEA design are listed in Section 20.4. As a result of these uncertainties, the closure cost estimate could materially change as the closure strategy is developed, uncertainties addressed and assumptions updated.

1.15.3 Environmental Assessment and Environmental Permitting

The Coffee Gold Project was subject to an assessment under the *Yukon Environmental and Socio-Economic Assessment Act*, administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB), which was completed in March 2022. The assessment documents the potential environmental and socio-economic effects of the Coffee Gold Project by evaluating baseline information, the proposed mine plan, mitigations, and the assessment of project effects. The Decision Document for the Coffee Gold Mine Project was issued on behalf of the Decision Bodies (Natural Resources Canada, Transport Canada, and Government of Yukon) that accept the recommendation that the Project be allowed to proceed subject to terms and conditions specified in the Referral Report (Yukon Environmental and Socio-Economic Assessment Board 2022). The Coffee Gold Project is in the process of obtaining a Type



A Water Use Licence and a Quartz Mining Licence. Changes to the assessed mine plan resulting from this PEA may require additional permitting to carry out activities.

1.16 Capital and Operating Costs

1.16.1 General

The project capital cost estimate (CAPEX), sustaining capital cost estimate (SUSEX), and operating cost estimate (OPEX) were compiled by WSP; and are based on the Project and ongoing scope of work presented in this Report. Table 1.4 outlines all of the contributors to the CAPEX, SUSEX, and OPEX for the Coffee Project.

**Table 1.4:
List of Cost Estimation by Contributor Firm**

Description	
Open Pit Mining	WSP, Open Contour
On-site development (airstrip)	WSP
Crushing and Handling	WSP
Heap Leach	Forte Dynamics
Process Plant	WSP
On-Site Infrastructure	WSP
Off-Site Infrastructure	WSP
Water Management	SRK, WSP
Water Treatment	Ensero
Indirect	All
Contingency	All
Reclamation and Closure	WSP

Unless indicated otherwise, all CAPEX and OPEX costs are expressed in Canadian Dollars (CA\$) and are based on Q1 2026 pricing. The exchange rate used for items quoted in US\$ was CA\$1.39.

1.16.2 Capital Costs

The Project CAPEX consists of direct and indirect capital costs as well as contingency. Provisions for SUSEX are also included, mainly for mining, water management, and electrics related to the mining. The CAPEX was developed to deliver an overall accuracy range of -30% to +50%.

Table 1.5 presents the CAPEX, SUSEX, and total capital by area. Shown separately in the Table are Indirects, representative of Owner's costs and EPCM, and the Overall Contingency cost. A contingency ranging from 10% to 25% was applied by WSP to the total cost of direct and indirects for all the capital items except for Reclamation and Closure.



The project CAPEX totals \$998.2 million and the SUSEX totals \$692.7 million, leading to an overall capital expenditure of \$1,690.9 million.

**Table 1.5:
Capital Cost Estimate by Area**

Description	CAPEX (CA\$ million)	SUSEX (CA\$ million)	Total Capital (CA\$ million)
Open Pit Mining	85.3	252.3	337.6
On-site development	15.9	24.1	40.0
Crushing and Handling	65.2	7.6	72.7
Heap Leach	87.1	159.7	246.8
Process Plant	76.7		76.7
On-Site Infrastructure	174.8		174.8
Off-Site Infrastructure	71.3		71.3
Water Management	31.9	25.4	57.3
Water Treatment Plant	26.2	26.2	52.3
Indirects	188.0	52.0	240.0
Contingency*	175.9	145.4	321.3
Total Major Area Capital**	998.2	692.7	1,690.9

*Contingencies applied vary from 10% if quoted to 25% if from prior study or benchmarked.

**Note: Numbers may not sum precisely due to rounding.

The exchange rate used for items quoted in US\$ was 1.39 CAN\$: 1.00 US\$.

1.16.3 Operating Costs

The estimated life-of-mine operating cost per tonne of material processed along with the US\$ cost per ounce gold produced for sale are summarized in Table 1.6



**Table 1.6:
Site Operating Unit Costs**

Description	CA\$/t processed	US\$/oz
Open Pit Mining	30.30	699
Site Services	2.81	65
Processing	6.94	160
General & Administrative	5.41	125
Water Treatment Plant	1.95	45
Total Site OPEX*	47.41	1,094

*Numbers may not sum precisely due to rounding.

The unit costs as shown in the above table are averages over the LOM plan and there can be fluctuations in either annual costs or annual gold ounces sold that can affect a particular year's \$US/oz sold result for the overall operation.

1.17 Economic Analysis

The economic analysis presented in this report contains forward-looking information under Canadian securities law. The results of the analysis rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ materially from those presented here.

The economic analysis of the Coffee Gold Project is preliminary in nature and per allowances for PEA level of study, includes Inferred mineral resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. As a result, there is no certainty that this PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The economic analysis is based on the discounted cash flow (DCF) method on a pre-tax and after-tax basis. Current Yukon tax regulations were used to assess corporate tax liabilities. The key metrics determined in the analysis are the Net Present Value (NPV) at a discount rate of 5%, the Internal Rate of Return (IRR) and the payback period in years. A sensitivity analysis was carried out to assess the impact of variations in gold grade, recoveries, gold price, CAPEX, and OPEX on the financial metrics. For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Coffee Gold Project has been evaluated on an unlevered, all-equity basis.

The cash flow model uses inputs from all elements of the Coffee Gold Project to provide a comprehensive financial projection for the entire Coffee Gold Project, on an annual basis, supporting, as shown in Figure 1.2 depicting the annualized cashflows, a 16-year life of Coffee Gold Project with a 13-year processing period. All prices and costs are in Canadian dollars and accurate as of Q1 2026. No provisions have been made for the effects of inflation.

Table 1.7 provides a summary of the key technical assumptions and inputs.



Table 1.7:
Coffee Gold Project PEA Summary: Key Technical Assumptions and Cost Inputs

Description	Unit	Value
Gold Price (Average LOM Base Case)	US\$/oz Au	3,656
Exchange Rate	US\$/CA\$	1.39
Discount Rate	%	5
Project Parameters		
LOM	years	13
Total Mill Feed Tonnage	Mt	90.5
Average Mined Grade	g/t Au	1.25
Average Gold Recovery	%	77.5
Contained Gold	koz	3,644
Recovered Gold	koz	2,824
Average Annual Gold Production	koz/a	217
Average Mining Rate – All material per calendar day	t/d	177,687
Strip Ratio	-	7.6:1
Revenue Related Costs		
Selling Costs	CA\$ million	78.5
Royalty NSR	CA\$ million	394.9
LOM Operating Costs		
Open Pit Mining	CA\$ million	2,743.4
Site Services	CA\$ million	254.6
Processing	CA\$ million	628.5
General and Administrative (G&A)	CA\$ million	489.4
Water Treatment Plant	CA\$ million	176.5
Total Operating Cost¹	CA\$ million	4,292.4



Description	Unit	Value
Sustaining Cost Estimate		
Direct Cost	CA\$ million	495.2
Indirect Cost	CA\$ million	52.0
Contingency	CA\$ million	145.4
Total Sustaining Capital Cost	CA\$ million	692.7
Project Capital Estimate		
Direct Cost	CA\$ million	634.3
Indirect Cost	CA\$ million	188.0
Contingency	CA\$ million	175.9
Total Project Capital Cost	CA\$ million	998.2
Other Costs		
Sustaining Working Capital	CA\$ million	-16.0
Project Capital Working Capital	CA\$ million	16.0
Closure	CA\$ million	177.6
Royalty Buybacks	CA\$ million	141.0
Salvage Value	CA\$ million	-22.4
LOM Operating Unit Costs		
Open Pit Mining	CA\$/t processed	30.30
Site Services	CA\$/t processed	2.81
Processing	CA\$/t processed	6.94
General and Administrative (G&A)	CA\$/t processed	5.41
Water Treatment Plant	CA\$/t processed	1.95
Total Operating Unit Cost	CA\$/t processed	47.41
Total Cash Cost ²	US\$/oz Au	1,214



Description	Unit	Value
AISC ³ (LOM)	US\$/oz Au	1,386

¹ Total operating cost is after transfer of Year -1 to capital and refers to onsite charges that cover open pit mining, processing and material handling, and onsite general and administrative costs during the production period.

² Total cash costs are a non-GAAP financial measure and are calculated as the sum of doré transportation, treatment and refining charges, royalties, and total operating costs, divided by the quantity of ounces sold.

³ AISC is a non-GAAP financial measure and calculated as the sum of doré transportation, treatment and refining charges, royalties, total operating costs, and sustaining capital costs, divided by the quantity of ounces sold.

*Numbers may not add up due to rounding

1.17.1 Non-GAAP Measures

Generally accepted accounting principles (GAAP) is a set of principals widely used in the United States (USA) for financial reporting by corporations and government agencies. The generally accepted accounting principles (GAAP) are a set of accounting rules, standards, and procedures issued and frequently revised by the Financial Accounting Standards Board (FASB) and the Governmental Accounting Standards Board (GASB). These principles ensure consistency, accuracy, and transparency in financial reporting across various industries in the United States. Public companies must follow GAAP when preparing their financial statements, which is also widely used in governmental accounting.

The Report has included herein certain performance measures (non-GAAP measures) which are not specified, defined, or determined under generally accepted accounting principles (GAAP). These non-GAAP measures are common performance measures in the gold mining industry, but because they do not have any mandated standardized definitions, they may not be comparable to similar measures presented by other issuers. Accordingly, such measures are used to provide additional information, and readers should not consider these non-GAAP measures in isolation or as a substitute for measures of performance prepared in accordance with GAAP. As the Coffee Project is not in production, it does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.

Cash Costs - The Report calculated total cash costs as the sum of mining, processing, refining & transport, G&A and royalty costs. Cash costs per ounce is calculated by taking total cash costs and dividing such amount by payable gold ounces. While there is no standardized meaning of the measure across the industry, this measure is useful to external users in assessing operating performance.

All-In Sustaining Cost - All-in sustaining costs are comprised of total cash costs, sustaining capital expenditures to support ongoing operations and closure costs. All-in sustaining costs per ounce is calculated as all-in sustaining costs divided by payable gold ounces. All-in sustaining costs capture the important components of Coffee's production and related costs and are used by the Company and investors to understand projected cost performance at the Coffee Project. Sustaining capital is a supplementary financial measure which reflects cash-basis expenditures which are expected to maintain operations and sustain production levels at the Coffee Project.

At an assumed weighted average gold price of US\$ 3,656/oz based on gold ounces recovered over the Project life, the financial results shown in Table 1.8 indicate a positive pre-tax NPV of CA\$ million 4,916.3 and an after-tax NPV of CA\$ million 3,004.5, both at a discount rate of 5%. The after-tax IRR shown is 43.5% and the after-tax payback is 1.97 years.

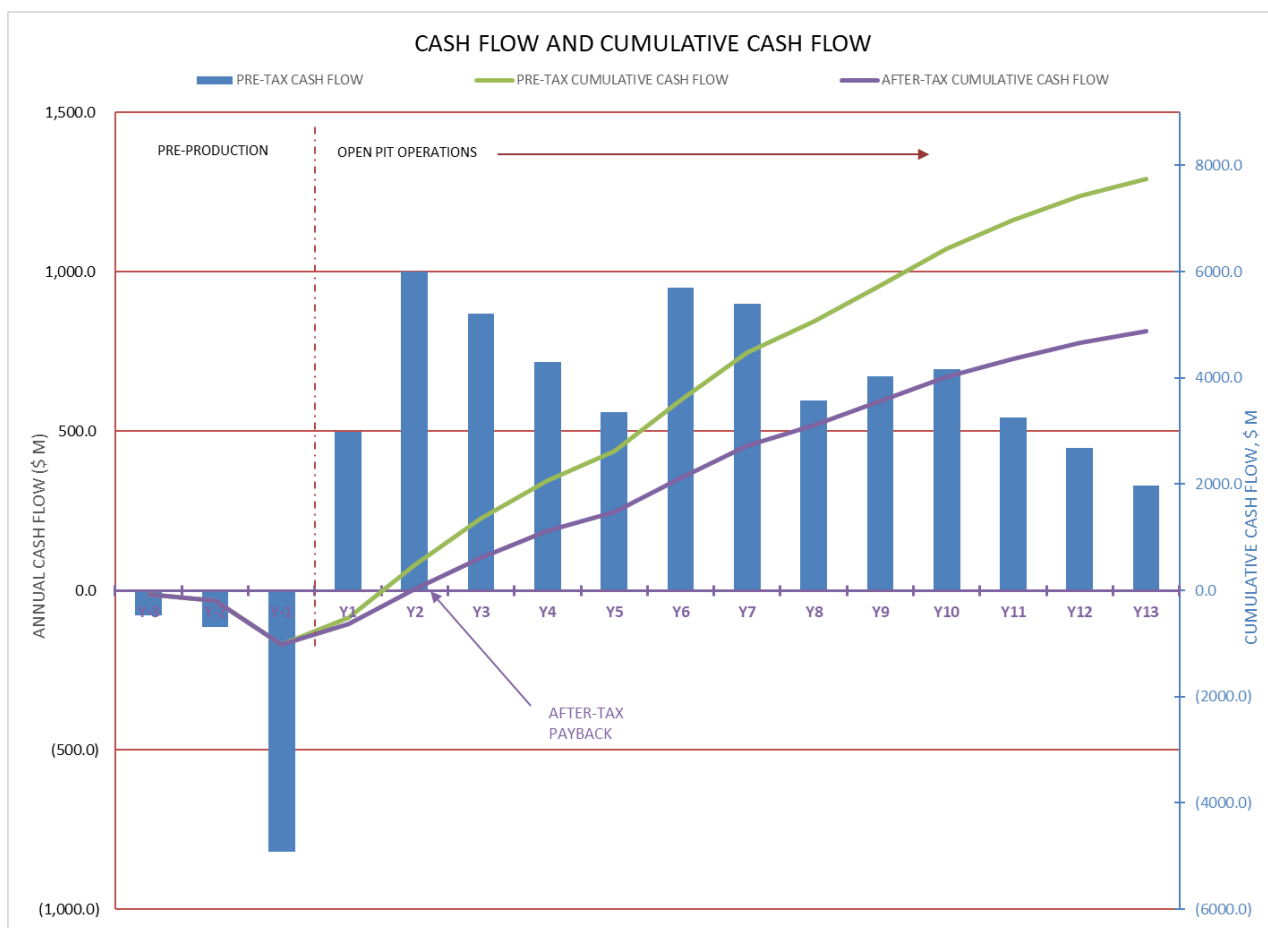


**Table 1.8
Economic Results Summary**

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	CA\$ million	4,916.3	3,004.5
IRR	%	59.8	43.5
Payback Period	Years	1.52	1.97

Figure 1.2 graphically depicts the pre-tax cash flows along with the cumulative pre-tax and after-tax cash flows.

**Figure 1.2:
Annual and Cumulative Cash Flows**



Source: WSP, 2026

Table 1.9 depicts the results of a sensitivity analysis that was conducted on the base case to assess the impact of changes in the price of gold on the Project's NPV at a 5% discount rate, payback period and IRR.



**Table 1.9
Economic Metrics Sensitivity to Variations in the Gold Price**

Gold Price (US\$/oz)	2,500	3,300	PEA 3,656 ¹	4,500	Near Spot 5,000 ²
Pre-tax NPV (5%) (CA\$ million)	2,008	3,980	4,916	7,201	8,340
After-tax NPV (5%) (CA\$ million)	1,138	2,403	3,004	4,470	5,200
After-tax IRR (%)	22.3%	36.0%	43.5%	57.7%	62.1%
Payback (years)	3.51	2.39	1.97	1.51	1.46

1. Analyst consensus prices as at February 11, 2026: US\$4,110/oz Au in 2029 and US\$3,620/oz Au in 2030 and beyond provide weighted average of US\$3,656/oz Au over total gold recovery period
2. Near Spot Price of US\$5,000/oz on February 18, 2026, reflecting LBMA PM closing price of US\$5,003/oz Au

The sensitivity analysis also assessed the impact of changes in gold grade, process recovery, gold price, CAPEX, SUSEX and in OPEX on the Project's NPV at 5%, the IRR, the payback in years, and the Percent variation in the NPV at 5%. The impact of each variable is examined individually with an interval of $\pm 20\%$, applied at increments of 10%. It is to be noted that the margin of error for cost estimates at the PEA study level is typically -30% to +50%.

The after-tax results of the sensitivity analysis are shown in Figure 1.3 and Figure 1.4. The NPV of the Project is most sensitive to variations in the gold grade, gold price and gold recovery projections, followed by OPEX and CAPEX. Although the intervals and increments assessed are different, the spider graphs show the NPV appears to be equally sensitive to variations in grade, recovery and price. Overall, within the limits of accuracy of the cost estimates in this study, the Project's potential after-tax viability does not seem significantly vulnerable to the under-estimation of either CAPEX or OPEX, when taken individually.



Figure 1.3
After-Tax NPV 5%: Sensitivity to Price, Grade/Recovery, Project Capital Cost, Sustaining Capital Cost and Total Operating Cost

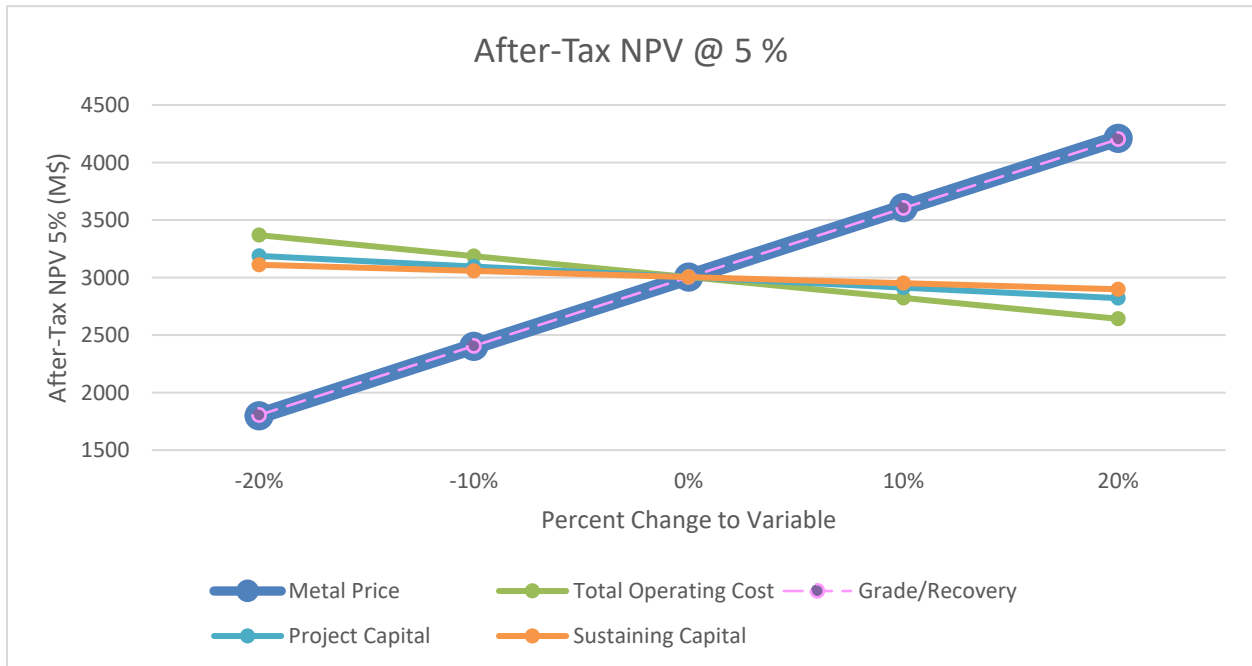


Figure 1.4
After-Tax Percent Variation in NPV 5%: Sensitivity to Price, Grade/Recovery, Project Capital Cost, Sustaining Capital Cost and Total Operating Cost

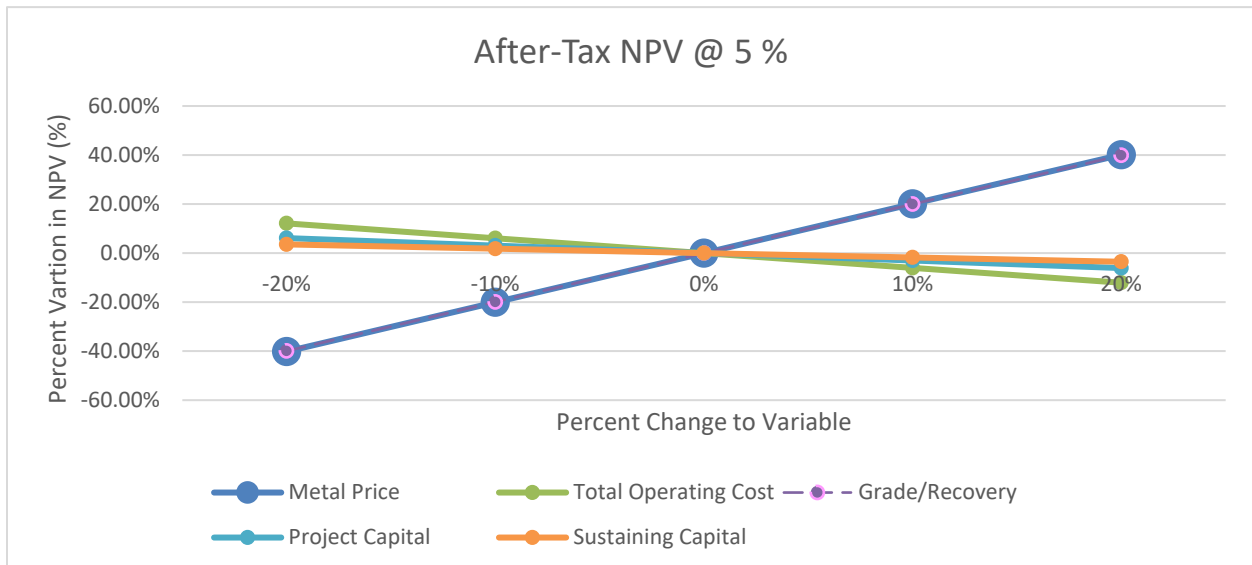


Table 1.10 shows the NPV sensitivity of the Project to alternative discount rates of 0%, 5%, 8%, 10%, and 12%. The economic indicators, both pre-tax and after-tax, for the various discount rates are provided.



**Table 1.10
Economic Indicators with Variations on Discount Rate**

Discounting rate 5%	Units	Pre-Tax	After-Tax
Payback period	years	1.52	1.97
IRR	%	59.8%	43.5%
Discount Cash Flow	CA\$ million	4,916	3,004
Discount Rate Sensitivity			
NPV @ 0%	CA\$ million	7,596	4,743
NPV @ 5%	CA\$ million	4,916	3,004
NPV @ 8%	CA\$ million	3,837	2,301
NPV @ 10%	CA\$ million	3,268	1,931
NPV @ 12%	CA\$ million	2,794	1,622

1.18 Adjacent Properties

The Micon QPs note that there are three known Mineral Projects adjacent to the Coffee Gold Project. The Casino Copper-Gold Porphyry Deposit owned by Western Copper and Gold Corporation is located approximately 30 km southeast of the main drilled area, and the White Gold Corporation mineral claims are located immediately to the northwest of the Coffee Gold Project. The Boulevard project owned by Independence Gold Corp. is located immediately to the southwest of the Coffee Gold Project.

1.19 Other Relevant Data and Information

All relevant data and information pertaining to the Coffee Gold Project have been disclosed under the relevant sections of this report. No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

1.20 Interpretation And Conclusions

1.20.1 Geology and MRE

Based on the exploration and drilling completed to date, the following facts are pertinent to the progression of the Coffee Gold Project.

1.20.1.1 Exploration/Drilling

The gold mineralization discovered at the Coffee Gold Project to date is hydrothermal in origin and both structurally and lithologically controlled. The current interpretation leans more towards linear shear hosting structures; however, a closer look at the “epicentre” of the mineralization suggests that the existence of a mega (intrusion related gold) IRG system could be possible. This requires a systematic investigation.



Reconnaissance exploration/drilling elsewhere on the Coffee Gold Project has been successful in discovering the presence of silver mineralization of potentially economic grades associated with gold in some areas of the deposits across the 70,256 ha Project area.

1.20.1.2 Mineral Resources Status

The growth potential for the mineral resource is favourable as the main Supremo deposit(s) remain open for expansion along strike in the north-northeast direction and down dip. In addition, there is potential for growing the resource via new discoveries, as several known mineral occurrences within the greater Project area remain to be test drilled for resource evaluation. In the Micon QP's opinion, the deposit/mineral resource is poised for growth on three fronts as follows:

- Additions from the already discovered deposits not included in the current resource (i.e., sparsely drilled parts of Americano, Espresso, Cappuccino, Dolce, French Press and Sugar).
- Additional exploration in the greater Project area.
- Deeper drilling to explore the likely potential for sulphides at depth.

Furthermore, the addition of silver into the MRE will increase the value of the resource; hence, the need to assess the silver enriched areas. This can be achieved initially by mineralogical work to determine whether the silver enrichment is due to localised phenomena.

1.20.1.3 Metallurgy

There have been a number of metallurgical test programs conducted by reputable laboratories. The preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. Good metal accountability across a range of testwork results were observed and added confidence to the data review.

The mineralized material demonstrated amenability to heap leaching process. Generally, oxide material had high recoveries, transition material showed moderate results, while the sulphide sample had poor recovery, indicating refractory behaviour. Temperature impacts on leaching recovery has been tested and the results indicated that minimal variation in leaching kinetics is to be expected between a cold temperature of 4°C and warm ambient temperature of 22°C. Based on the agglomeration and compacted permeability tests, cement agglomeration is considered not necessary.

In the WSP QP's opinion, the samples used and testing overall are adequate for a PEA study and align with processing plans

1.20.1.4 Uncertainties & Risks

All mineral resource estimates have a degree of uncertainty or risk associated with them, due to technical, environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors, among others. All mineral resource estimates also present their own opportunities.

Factors that may affect the MRE include fluctuations in the price of metals, in particular gold and changes in the metallurgical recoveries and bulk density assignments. In addition, it is the QP's opinion that the factors set out below could affect the mineral resource estimate.

- The geological interpretations and assumptions used to generate the estimation domain.
- The confidence assumptions and methods used in the mineral resource classification.



- Economic assumptions used in the cut-off grade determination.
- Input and design parameter assumptions that pertain to the open pit mining constraints.
- Assumptions as to the continued ability to access the Project site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

To mitigate risks related to metallurgy/bulk density, additional detailed investigations involving pilot plant tests are recommended prior to undertaking Feasibility Studies. Risks associated with fluctuations in the price of metals are uncontrollable; however, a modest long-term gold metal price has already been considered in determining the economic factors for the Mineral Resource Estimate.

1.20.1.5 Overall Geology Conclusions

The exploration work completed, and the results obtained to date, are satisfactory to justify further work to move the Coffee Gold Project to the next level towards building a mining venture.

The systematic progression of the Coffee Gold Project towards becoming a mining Project will be guided by clearly defined short-term and medium-/long-term objectives whilst not losing sight of the fact that the overall resource size within the Mineral Project area remains the topmost factor for a sustainable future and the associated long term investment decisions.

1.20.2 Mining Methods

Considerable previous work including mine iterations have been completed for this project, and most of the inputs for the current study have been taken from this previous work. The mine has increased in total size due to higher gold price inputs. Some of the previous work will require additional engineering given the increase in size. Haul roads and in pit dumping configuration will have to be updated.

Work for this report was done to further optimize the sequencing and timing of these phases. It is believed that this work has improved the economics and should guide further technical work.

Grade control remains an important consideration not just for fleet sizing, but also for bench configuration and mining strategy.

As mentioned in Section 26, some of the newest autonomy technologies could be implemented in this project given its size.

1.20.3 Metallurgy

There have been a number of test programs conducted on Coffee material by reputable laboratories. The preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. Mineralized material demonstrated amenability to the heap leaching process. Generally oxide material had high recoveries, transition material showed moderate results, while the sulphide sample had poor recovery, indicating refractory behaviour. In the QP's opinion, the existing samples and testing overall are adequate for a PEA study and align with processing plans.



1.20.4 Recovery Methods

1.20.4.1 *Crushing and ADR*

The processing design for the Project has undergone several studies in the past. The current process configuration is now sufficiently developed to meet and exceed the PEA requirements, with production capacity aligned to the mine plan. The selected combination of gyratory and cone crushers is expected to provide a reliable solution as compared to mineral sizers for the required crushing operations.

1.20.4.2 *Heap Leach*

The heap leach pad has been designed to meet the leach capacity requirements per this PEA. Ore properties have been assumed at this time, with additional site investigations and material characterization work including ore testing required, as included in the recommendations. Current analyses indicate the heap leach pad will contain the design tonnage. Cost estimates made for the CAPEX & SUSEX associated with the heap leach pad were developed using pricing based on project-specific experience and available site investigation information.

1.20.5 Project Infrastructure

1.20.5.1 *General Infrastructure*

The infrastructure key consideration will focus on ensuring year-round access where possible, with contingency plans for seasonal river-crossing limitations, with emphasis on self-sufficient utilities, water, and waste management.

Several comprehensive studies have been conducted for the Coffee Gold Project, resulting in well-planned infrastructure for a PEA level of study that enables the project to advance to its next phase of development.

1.20.5.2 *Water And Waste Management*

The water and waste management components have been developed following regional best practice and incorporating available site-specific information.

The water management program for the Coffee Gold Project is designed to provide flexible and reliable control of contact and non-contact water while protecting downstream receiving environments and maintaining the physical stability of site infrastructure. It integrates local climate and hydrologic data with engineered infrastructure, including collection ponds (with the largest being the Alpha Pond), diversion channels (with the largest being the Halfway Creek Diversion), and pit dewatering systems, to manage runoff, sediment, and potential water quality effects. Key facilities such as the Alpha Pond and Facility Pond are sized to meet regulatory requirements and to handle design flood events, while diversion channels and rock drains are configured to minimize disturbance to natural drainage patterns and maintain long-term system performance during operations and into closure.

Waste rock management at the Coffee Gold Project will be centered on the Alpha Waste Rock Storage Facility (WRSF) and the strategic backfilling of mined-out pits to optimize storage, minimize disturbance, and maintain long-term geotechnical stability. Over the 13-year active mining period, approximately 688 Mt of waste rock are projected to be generated, with annual volumes varying according to the mine schedule and number of active pits. The Alpha WRSF will be developed in stages, including engineered rock drains and careful placement over variable permafrost and ice-rich foundations to maintain stability and control drainage, while pit backfilling will account for swell and haul distances to efficiently utilize void space and reduce the overall surface footprint of waste storage.



1.20.6 Environmental Studies, Permitting, Social or Community Impact

1.20.6.1 Closure

The closure cost estimate has been updated from prior estimates to reflect the updated PEA design and is considered sufficient for a PEA level study. Closure uncertainties are identified in Section 20.4.3, with recommendations for the production of a more robust closure liability estimate suitable for subsequent study phases presented in Section 26.6.1.

1.20.7 Costs

Costs as presented are representative of a PEA with a level of accuracy of -30% to +50%.

1.20.8 Economics

Economic Analysis provides After-Tax financial results of a NPV@5% of \$3,004.5 million, an IRR of 43.5%, and a Payback in Years of 1.97. The Project results are most sensitive to gold price and metal recovery assumptions. Sensitivity analysis shows that were the realized gold selling price to be US\$2,500/oz Au the Project NPV@5% would still be \$1,138 million.

1.21 Recommendations

1.21.1 Geology & MRE

Coffee is a Mineral Project with significant potential that warrants additional work. A considerable effort has been made to delineate the deposit and test metallurgical methods and recoveries as well as mining and associated infrastructure to develop the Project. All activities completed to date have yielded positive results encouraging Fuerte to continue further studies. Accordingly, infill and step-out drilling are recommended to upgrade and expand the resource. The approximate budget for this work is CA\$ 20 M.



Micon's QP believes that the proposed programs of infill and step-out drilling and the budget are appropriate and recommend that Fuerte undertake the programs subject to funding or other matters which may cause the proposed programs to be altered in the normal course of its business activities, or alterations which may affect the program as a result of the activities themselves.

1.21.2 Mining Methods

The following recommendations are related to mining:

- A technology 'roadmap' should be written in collaboration with other vendors in the space.
- Ore control practices with respect to mining the type of vertical vein orebodies should be detailed before fleets are chosen.
- Permit constraints should be understood and implemented within the mineplan detail such as permafrost mining and dump building.
- Haul road plans should be detailed given the newest pit designs.
- Inpit backfilling plans should be detailed given the newest pit designs.
- Leach pad construction (earthworks fill material) should be included in the mine planning detail.



1.21.3 Metallurgy

In the next phase of project development, it is recommended to evaluate the impact of ore blending on extraction performance and flowsheet selection. Preliminary flotation testing on sulphide samples has yielded promising results. It is recommended to conduct trade-off studies to assess the feasibility of a flotation processing route for sulphide ore.

1.21.4 Recovery Methods

1.21.4.1 General

The following are recommended for the process plant design in the next phase of the Project:

- Evaluate secondary crushing in an open-circuit versus a closed-circuit configuration;
- Optimize final crushed ore product size and crushing circuit general arrangement;
- Perform detailed thermal analysis for solution management under cold weather conditions;
- Develop a detailed operations and equipment maintenance plan during the coldest period each year, which is assumed to be from January to March.

1.21.4.2 Heap Leach Pad

A detailed geotechnical field investigation will be required to further define subsurface conditions beneath and surrounding the HLP, process plant areas, and ponds, to include detailing permafrost extents. Ore strength testing is required and should be performed for ore delivered in the first 6 months, years 1-2, years 3-4 and beyond and include any other ore type changes that may occur periodically during mining that may contain varying strength parameters. Ore testing shall include shear strength testing, decrepitation testing, and compacted permeability at a minimum. A comprehensive site investigation plan for material collection, testing, and characterization will be developed for these activities.

Detailed thermal modelling should be performed once additional ore strength data, updated climate data, and other design parameters are obtained or refined.

1.21.5 Project Infrastructure

1.21.5.1 Infrastructure General

Infrastructure development for remote heap leach operations should be guided by a self-sufficient design approach tailored to cold climates, with an emphasis on constructability, reliability, and long-term operational performance. It is essential to address construction sequencing and logistics during the early planning stages, given the limited summer construction period and reliance on seasonal transportation.

Fuel storage and handling facilities should be appropriately sized to accommodate extended resupply intervals. These facilities will include secondary containment, winterized transfer systems, and spill prevention measures.

1.21.5.2 Water And Waste Management

Additional geotechnical investigations will be required to support the next stage of engineering, specifically along the footprint of the proposed Alpha Waste Rock Dump, Alpha Pond, and new diversion alignments



around the pits. This additional investigation (test pitting and drilling) would be done to improve foundation characterization and construction material availability, and to give additional information and parameters for stability and deformation calculations. Similarly, the next stage of engineering will include more advanced hydrologic modelling to evaluate ranges of long-term snowmelt volumes to inform pond sizing.

For the Water and Waste Management Infrastructure additional site investigation data is required to advanced designs to PFS or high level. This would be expected to be comprised of additional test pitting and drilling within the waste rock and water management (diversions and pond) footprints.

1.21.6 Heap Leach Facility Water Treatment

The following are recommended to confirm and optimize the HLF Water Treatment Plant (WTP) design:

- Reassess the WTP design basis to reflect changes associated with the HLF design advancement and the potential treatment of the pit dewatering stream.
- Conduct additional treatability testing using ore-leached water samples to reassess treatment requirements and operating conditions.
- Update and optimize the WTP design based on the revised design basis and the treatability test results.
- Complete process waste classification for dewatered solids and clean-in-place (CIP) wastes and evaluate appropriate management and disposal strategies.
- Update the cost estimate to reflect the optimization.

1.21.7 Environmental Studies, Permitting, Social Impact or Community

1.21.7.1 Closure Planning Recommendations

To progress closure planning to the next stage of project development, with the objective of producing an integrated life of mine plan and reducing uncertainty in the closure cost estimate, the following actions are recommended. It is noted that closure planning, hence associated closure costs, always carry a level of uncertainty into operations that can only be addressed once the mine is operating. As such, the next steps below are focussed on those activities that can address the key uncertainties relevant to this stage of the Project.

- Improve progressive closure inclusion in operational planning to increase confidence that progressive closure commitments, currently unaccounted for in the closure cost estimate, are included in the project design and operational plan.
- Improve closure cost estimation for the project through collation of a project-specific closure cost model built up from first principles, to provide for a more logical calculation of closure liability and to include a more detailed consideration of closure logistics and schedule.
- Progress closure design to reflect the updated PEA mine design.
- Stakeholder engagement - progressing early and meaningful stakeholder and rightsholder engagement for improved understanding of closure requirements and socio-economic transition requirements for more robust closure planning and liability estimates.



2.0 INTRODUCTION

2.1 Authorization, Terms of Reference and Purpose

Fuerte has retained WSP, in coordination with other consultants (Micon, SRK, Ensero, Forte Dynamics, Lorax & Open Contour), to complete a PEA of the Coffee Gold Project located in the Whitehorse Mining District of west-central Yukon (YT Canada), approximately 130 km south of Dawson, and to prepare an NI 43-101 technical report (Technical Report or Report) in compliance with Form 43-101F1, to support public disclosure. The Coffee Gold Project houses several gold deposits within an exploration concession covering an area of 70,256 ha.

This Report was prepared to support the results of the PEA presented in news release “Fuerte Announces a Positive Preliminary Economic Assessment for the Coffee Gold Project; Positioning the Company as one of Canada’s Next Gold Producers” on February 22, 2026, followed by a second Press Release on April 8, 2026, “Fuerte Files Technical Report for the Coffee Gold Project”. The purpose of this Technical Report is to present the PEA on the Coffee Gold Project and to make recommendations on the programs of work required to move the Project to the next stages of study and development. Fuerte requires an independent Technical Report in order to support regulatory disclosures.

2.1.1 Cautionary Statement

The PEA is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. No Mineral Reserves are defined for the Coffee Gold Project.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the consultants’ services. The information, conclusions, and estimates contained herein are based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Fuerte, subject to the terms and conditions of its contract with Fuerte various consultants and relevant securities legislation. The contract permits Fuerte to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party’s sole risk.

2.2 Qualifications of Consultants

The QPs who prepared this Report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, civil, mechanical, electrical, capital and operating cost estimation, and mineral economics.

The individuals listed in Table 2.1 are by virtue of their education, experience and professional association considered QPs, as defined by NI 43-101.



**Table 2.1
Qualified Person Responsibility**

QP	Designation	Company	QP Responsibility/Role	Report Sections/Sub-sections
Charley Murahwi	P.Geo., FAusIMM.	Micon	Property Description and Location, Accessibility, Climate, Local Resources, Infrastructure, and Physiography, History, Geological Setting and Mineralization, Deposit Types Exploration, Drilling, Sample Preparation, Analyses, and Security (Data Verification, Mineral Resource Estimate, Adjacent Properties	1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.18, 1.19, 1.20.1, 1.21.1, part of 3, 4, 5, 6, 7, 8, 9, 10 11, Part of 12.1, 12.2, 12.5, 14, 23, 24, 25.1, 26.1 & 27.2
Hannah Chiew	P.Eng.	Ensero	HLF Water Treatment	1.13.4, 1.21.6, 18.12, 21.3.9, 21.4.9, 21.6.5, 25.5.3, 26.5.3
Samantha Barnes	P.Eng.	SRK	Infrastructure – Water Management Sub-Sections Including Climate Considerations	1.13.2, 1.20.5.2, 1.21.5.2, 18.11, 21.3.8, 21.4.8, 25.5.2, 26.5.2
John Kurylo	P.Eng.	SRK	Infrastructure and Waste Rock - Waste and Water Management Sub-Sections. Including Waste Rock Storage external to pits	1.13.2, 1.13.3, 1.20.5.2 1.21.5.2, 18.13, 21.3.8, 21.4.8, 25.5.2, 26.5.2
Barry Carlson	P.Eng., SME-RM	Forte Dynamics	Heap Leach CAPEX & SUSEX	1.12.2, 1.20.4.2, 17.2.4, 21.3.4, 21.4.4, 25.4.2, 26.4.2
Russell Downer	P.Eng.	Opencontour	Pit Optimization, Mine Design, Mine Schedule, Mine Methods, Haul Roads & Stockpiles	1.11, 1.20.2, 1.21.2, 16, 25.2, 26.2
Rachel Wyles	P.Eng.	WSP Canada Inc.	Environmental Studies, Permitting & Social or Community Impact	1.15.1, 1.15.3, 20.1, 20.2, 20.4, 20.5, 25.6, 27.4
Marc Rougier	P.Eng.	WSP Canada Inc.	Mining Methods – Geotechnical Sub-Section	1.11.1, 16.1, 26.2.2
Kim Ferguson	FAusIMM	WSP Canada Inc.	Closure, Decommissioning and Reclamation sub-sections	1.15.2, 1.20.6.1, 1.21.7.1, 20.3, 25.6.2, 26.6.1
Dachun (David) Jin	P.Eng.	WSP Canada Inc.	Mineral Processing and Metallurgy, Recovery Methods, Project Infrastructure, Electrical and Capital Costs, Mill Operating Costs, G&A costs.	1.9, 1.12 1.13.1, 1.20.1.3, 1.20.3, 1.20.4.1, 1.20.5.1, 1.21.3, 1.21.4, 1.21.5.1, 13, 17.1, 17.2.2.1, 17.2.3, 17.2.5- 17.2.8, 17.3, 18.1-18.10, 21.3.2, 21.3.3, 21.3.5-21.3.7, 21.4.2, 21.4.3, 21.4.5-21.4.7, 21.6.2-21.6.4, 25.3, 25.4.1, 25.5.1, 26.3, 26.4.1, 26.5.1, 27.1, 27.3
William Richard McBride	P.Eng.	WSP Canada Inc.	Introduction, Market Studies and Contracts, compiling of Capital and Operating Costs, Economic Analysis	1.1, 1.14, 1.16, 1.17, 1.20.7, 1.20.8, 2, parts of 3, 19, 21.1, 21.2.1, 21.3.1, 21.4.1, 21.6.1, 22, 25.7, 25.8
Laura-Lee Findlater	P.Geo.	Lorax Environmental Services Ltd.	Hydrogeology	20.6



2.3 Sources of Information

1. The principal sources of information for this report are: JDS Energy & Mining Inc. (JDS), 2024. Internal Prefeasibility Study on the Coffee Gold Project, Yukon Territory, Canada, dated October 9, 2024; prepared for Newmont Corporation (Internal Engineering Report; JDS, 2024)
2. Micon International Ltd. (Micon), 2025 “NI 43-101 Technical Report for the 2025 Mineral Resource Estimate Update on the Coffee Gold Project, Yukon, Canada”
3. Yukon Geological Survey (YGS), 2025. Geology and bedrock mapping updates at the Coffee Gold Project gold deposit: implication for deposit classification. (YGS Open File 2025-6).
4. Drillhole databases supplied by Fuerte.
5. Observations made during the site visits by the QP’s.
6. Discussions with Fuerte management and staff familiar with the Mineral Project.

The reference sources for published material researched by the QPs for this report are identified in Section 27.0. Additional information in reference to expert documentation is referenced in Section 3.0.

2.4 Scope of Personal Inspection

Micon QP (Charley Murahwi, P. Geo., FAusIMM) conducted a site visit to the Project from 28 to 29 July 2025. During his visit, QP Murahwi verified the channel chip sampling previously completed by Newmont at surface in trench 3 (T3), examined the geology of key outcrops/exposures, discussed the geologic models, verified some of the drillhole collar positions, reviewed in-house density measurements, examined drill cores, reviewed drillhole logs and core sampling/data collection methods, reviewed mineralization types, and discussed the Quality Assurance/Quality Control (QA/QC) protocols previously used by Newmont.

Open Contour QP (Russell Downer, P.Eng.) conducted a site visit to the Project on August 28, 2018. During his visit, QP Downer toured the project area, including active and historical drill workings, and inspected the core shed and associated facilities. He reviewed drill core, observed site infrastructure and operational conditions, and gained an understanding of the exploration activities and geological setting. Discussions were held with site personnel regarding drilling procedures, core handling practices, and general project operations.

A WSP employee (Mitch Trottier, Senior Geotechnical Engineer, P.Eng.) conducted a site visit from March 31 to April 1, 2026 on behalf of WSP QP’s Marc Rougier, P.Eng and David Jin, P.Eng. The site visit included completion of site orientation, meetings with the senior site geologist and site superintendent, review of available project layout mapping for the proposed pits, heap leach facilities, waste rock storage areas, and associated infrastructure, and a helicopter reconnaissance of the proposed mine area. Observations were limited to surface conditions visible at the time of the visit. Georeferenced photographs of the proposed pit and infrastructure areas were collected during the helicopter flight for review by WSP Qualified Person personnel.

Richard Gowans, a Micon QP responsible for the metallurgical testing section of this report, did not conduct a site visit as there was no relevant work being undertaken on the site pertaining to his area of responsibility.

Mr. Barry Carlson, the QP responsible for heap leach capex and susex, performed a site visit on August 24th-28th, 2018. During the visit, Mr. Carlson inspected the site area layout for the heap leach facility,



reviewed waste rock designated areas, visited each planned pit location and other area drainages, visited the admin and camp areas, and conducted general observations of the project site.

Kim Ferguson, a WSP QP responsible for the Closure, Decommissioning and Reclamation section of this report, did not conduct a site visit as there was no relevant operational disturbance or landforms existing on the site pertaining to her area of responsibility.

William Richard McBride, a WSP QP responsible for mining costs, market studies, and economic analysis did not make a site visit as there is no pertinent information to examine on the site pertaining to his area of responsibility.

John Kurylo and Samantha Barnes, are the SRK QPs responsible for the waste and water management infrastructure. John Kurylo completed a one day site visit on March 24th, 2026. Samantha Barnes did not conduct a site visit as John was able to complete a site visit on SRK's behalf and as the portion of the scope she assisted with would not be further advanced, and there was no pertinent information to examine on the site; as the ground surface was covered with snow for the period the PEA study was completed and thus no surface water flow was able to be observed. Prior to this PEA SRK has been involved in past water management infrastructure designs, past hydrotechnical analysis, past dam break analysis, and permitting support (from approximately 2018 to 2025) and assisted with some site investigations (linked to water management infrastructure) in 2018.

Laura-Lee Findlater, a Lorax QP responsible for hydrogeology, did not conduct a site visit in 2026 as Ms. Findlater had spent numerous weeks at the site in 2014 and 2015 to supervise hydrogeological drilling programs. Her most recent site presence extended from May 13th to June 10th, 2015.

Hannah Chiew, an Ensero QP responsible for the water treatment components of this Technical Report, did not conduct a personal site inspection in 2026. Her scope of work was limited to the review and interpretation of existing water quality data, flow information, site water balances, and conceptual and preliminary engineering information provided by the Issuer, which were sufficient to characterize influent water chemistry, treatment objectives, and process constraints relevant to the water treatment assessment. No aspects of the evaluation relied on site-specific visual observations or field measurements; accordingly, a personal inspection was not required to support the conclusions presented herein. A site visit may be warranted at later stages of Project development.

2.5 List of Previous Relevant Technical Reports

The previous Technical Report, "NI 43-101 Technical Report for the 2025 Mineral Resource Estimate Update on the Coffee Gold Project, Yukon, Canada"; prepared for Fuerte Metals Corp. by Micon.

2.6 Units Of Measurement and Abbreviations

Metric units are used throughout this report, and all dollar amounts are reported in CA and U.S. Dollars unless otherwise stated. Coordinates within this report use NAD83 UTM Zone 7N (EPSG 26909) unless otherwise stated. The list of abbreviations which may be used in this report is presented in Table 2.2.



**Table 2.2
Measurement Units and Abbreviations**

Name	Abbreviation
Adsorption desorption recovery	ADR
Year or annum	a
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian National Instrument 43-101	NI 43-101
Carbon-in-column	CIC
Centimetre(s)	cm
Carbon replacement deposits	CRD
Days	d
Degree(s)	°
Degrees Celsius	°C
Digital elevation model	DEM
Digital Terrain Model	DTM
Dollar(s), Canadian and US	\$, CA\$ and US\$
Generally Accepted Accounting Principals	GAAP
Geological Survey of Canada	GSC
Hours	h
Gram(s)	g
Grams per metric tonne	g/t
Greater than	>
Hectare(s)	ha
Induced polarization	IP
Kilogram(s)	kg
Kilometre(s)	km
Less than	<
Litre(s)	l
Life-of-mine	LOM
Metre(s)	m
Metres above sea level	masl
Micon International Limited	Micon
Million tonnes	Mt
Million ounces	Moz
Million years	Ma
Newmont Corporation	Newmont
North American Datum	NAD
Not available/applicable	n.a.
Parts per billion	ppb
Parts per million	ppm
Percent(age)	%
Qualified Person	QP
Quality Assurance/Quality Control	QA/QC
Run-of-mine	ROM
Specific gravity	SG
System for Electronic Document Analysis and Retrieval	SEDAR
Système International d'Unités	SI
Three-dimension	3D
Tonne (metric)	t



Name	Abbreviation
Troy ounce(s)	oz
Universal Transverse Mercator	UTM
Very-low-frequency electromagnetic	VLF-EM
Yukon Geological Survey	YGS

2.7 Effective Date

The issue date of this report is April 8, 2026. The effective date of the PEA is March 27, 2026.



3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied upon expert reports that provided information regarding mineral tenure, surface rights, obligations, royalties, agreements, taxation and commodity price information contained within this Report.

3.1 Mineral Tenure, Surface Rights, Obligations, Royalties and Agreements

The Micon, Opencontour and WSP QPs have not independently reviewed the mineral tenure, surface rights, obligations, royalties, and agreements. The QPs have fully relied upon and disclaim responsibility for, information supplied by Fuerte through the following document:

- Email from Fuerte entitled “Coffee Gold Project Technical Report Section 4” with attachment of the content for inclusion in Section 4.0 in this Report, dated April 7, 2026

This expert information is used in Section 4.0 of the Report and was also used to support considerations of RPEEE and the declaration of mineral resources in Section 14.0, in Section 16.0 for the mine plan, and for royalties and agreements considered for the economic analysis in Section 22.0.

3.2 Taxation

The WSP QP has not independently reviewed the taxation information. The WSP QP has fully relied upon and disclaims responsibility for, information supplied by Fuerte for this information through the following document:

- Email from Fuerte entitled “Consent as Taxation Expert” with attachment of the economic model with the tax inputs, dated April 7, 2026.

This expert information is used to summarize the taxation information, and to support the after-tax economic analysis in Section 22.0.

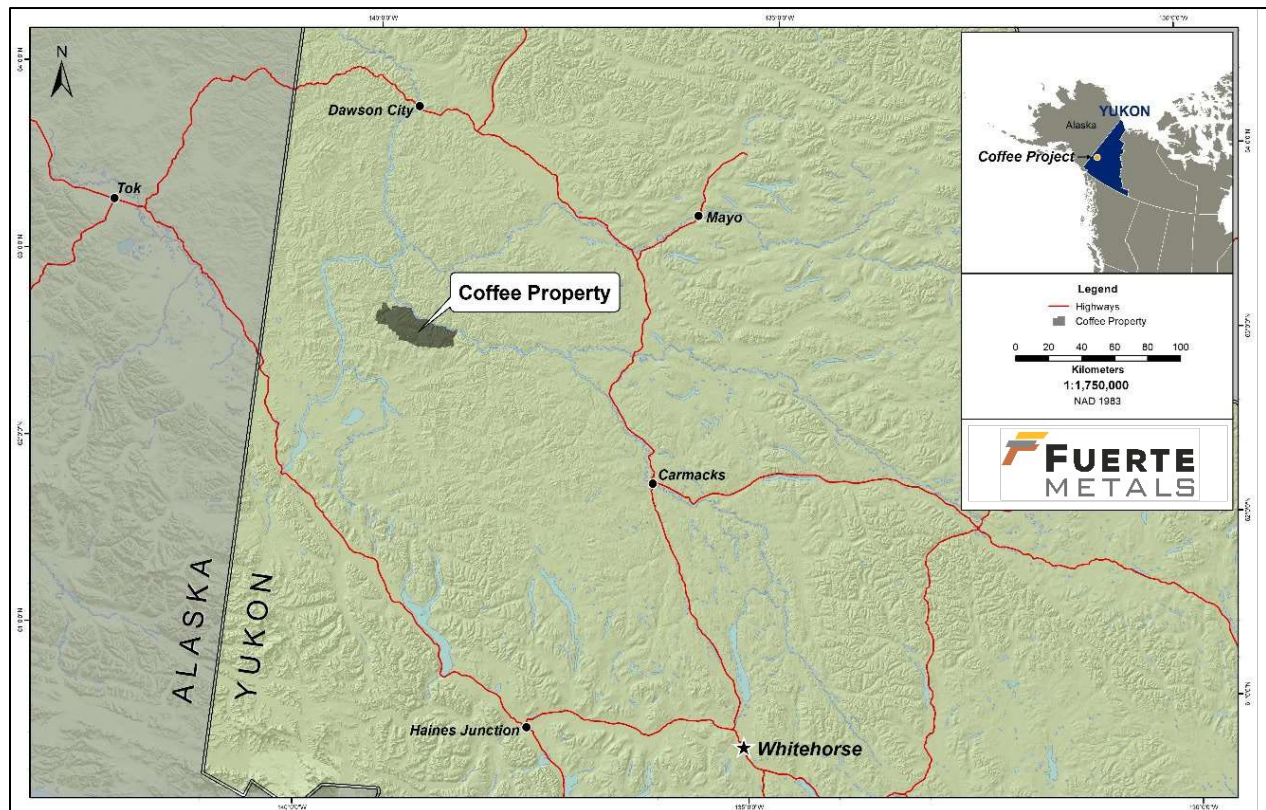


4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Coffee Gold Project is located in the Whitehorse Mining District of west-central Yukon Territory, Canada. The Mineral Project is approximately 130 km south of Dawson and about 160 km northwest of Carmacks (Figure 4.1).

Figure 4.1
Location of the Coffee Gold Project



Source: Modified after the JDS report, 2024

The Coffee Gold Project covers parts of 1:50,000 scale national topographic system (NTS) map sheets 115J-13, 115J-14, and 115J-15. The core of the mineralized zones are located roughly at the UTM NAD83 coordinates of 6,974,000 mN and 584,000 mE.

The Mineral Project lies within the Traditional Territories of the Tr'ondëk Hwëch'in and Selkirk First Nation, and the asserted traditional territory of the White River First Nation.

4.2 Mineral Project Description and Land Tenure

4.2.1 Mineral Project Details

The Project comprises 3,542 contiguous claims covering an aggregate area of approximately 70,256 ha. The boundaries of the individual claims have not been legally surveyed. The list of claims is presented in Table 4.1.



There are two groups of placer claims on Coffee-Latte and Halfway Creeks.

**Table 4.1
Coffee Project Summary of Mineral Tenure Information**

Claim Name and Number	Grant No.	Expiry Date	NTS #s	Grouping
COFFEE 1 - 6	YC46734 - YC46739	12/15/2032	115J14	HW07440
COFFEE 7	YC46740	12/15/2036	115J14	HW07442
COFFEE 8	YC46741	12/15/2036	115J14	HW07629
COFFEE 9 - 12	YC46742 - YC46745	12/15/2036	115J14	HW07444
COFFEE 13 - 14	YC46746 - YC46747	12/15/2036	115J14	HW07442
COFFEE 15	YC46748	12/15/2036	115J14	HW07443
COFFEE 16	YC46749	12/15/2036	115J14	HW07442
COFFEE 17 - 18	YC53949 - YC53950	12/15/2032	115J14	HW07444
COFFEE 19 - 24	YC53951 - YC53956	12/15/2032	115J14	HW07440
COFFEE 25 - 36	YC53957 - YC53968	12/15/2032	115J14	HW07629
COFFEE 37 - 39	YC54445 - YC54447	12/15/2032	115J14	HW07443
COFFEE 40	YC54448	12/15/2033	115J14	HW07443
COFFEE 41 - 56	YC54449 - YC54464	12/15/2032	115J14	HW07629
COFFEE 57 - 62	YC54465 - YC54470	12/15/2032	115J14	HW07630
COFFEE 63 - 70	YC54471 - YC54478	12/15/2032	115J14	HW07629
COFFEE 71 - 92	YC54479 - YC54500	12/15/2032	115J14	HW07630
COFFEE 105 - 112	YC60176 - YC60183	12/15/2032	115J14	HW07443
COFFEE 113 - 122	YC83190 - YC83199	12/15/2032	115J14	HW07442
COFFEE 123 - 128	YC83200 - YC83205	12/15/2032	115J14	HW07444
COFFEE 129 - 132	YC83206 - YC83209	12/15/2032	115J14	HW07440
COFFEE 133 - 136	YC83210 - YC83213	12/15/2032	115J14	HW07444
COFFEE 137 - 140	YC83214 - YC83217	12/15/2032	115J14	HW07440
COFFEE 141 - 144	YC83218 - YC83221	12/15/2032	115J14	HW07444
COFFEE 145 - 148	YC83222 - YC83225	12/15/2032	115J14	HW07440
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COFFEE 189 - 226	YC83266 - YC83303	12/15/2032	115J14	HW07630
COFFEE 227 - 276	YC83652 - YC83701	12/15/2032	115J14	HW07629
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COFFEE 283 - 288	YC89411 - YC89416	12/15/2032	115J14	HW07444
COFFEE 289 - 292	YC89417 - YC89420	12/15/2032	115J14	HW07440
COFFEE 293 - 298	YC89421 - YC89426	12/15/2032	115J14	HW07442
COFFEE 299 - 304	YC89427 - YC89432	12/15/2032	115J14	HW07444
COFFEE 305 - 308	YC89433 - YC89436	12/15/2032	115J14	HW07440



Claim Name and Number	Grant No.	Expiry Date	NTS #s	Grouping
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COFFEE 317 - 328	YC89445 - YC89456	12/15/2032	115J14	HW07443
COFFEE 329 - 344	YC89457 - YC89472	12/15/2032	115J14	HW07442
COFFEE 345 - 366	YC93441 - YC93462	12/15/2032	115J13,115J14	HW07630
COFFEE 367 - 382	YC93463 - YC93478	12/15/2032	115J14	HW07629
COFFEE 383 - 394	YC93479 - YC93490	12/15/2032	115J14	HW07443
COFFEE 395 - 404	YC93491 - YC93500	12/15/2032	115J14	HW07442
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COFFEE 411 - 514	YC92601 - YC92704	12/15/2032	115J13,115J14	HW07630
COFFEE 515 - 522	YC92705 - YC92712	12/15/2032	115J14	HW07629
COFFEE 523 - 542	YC92713 - YC92732	12/15/2032	115J14	HW07630
COFFEE 543 - 550	YC92733 - YC92740	12/15/2032	115J14	HW07629
COFFEE 551 - 570	YC92741 - YC92760	12/15/2032	115J14	HW07630
COFFEE 571 - 578	YC92761 - YC92768	12/15/2032	115J14	HW07443
COFFEE 587 - 590	YC92777 - YC92780	12/15/2032	115J14	HW07630
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COFFEE 617 - 624	YC93357 - YC93364	12/15/2032	115J14	HW07443
COFFEE 625	YC93365	12/15/2032	115J13	HW07630
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COFFEE 659 - 676	YC96833 - YC96850	12/15/2032	115J14	HW07443
COFFEE 677 - 684	YC96851 - YC96858	12/15/2032	115J14	HW07629
COFFEE 685 - 718	YC96859 - YC96892	12/15/2032	115J14	HW07443
COFFEE 719 - 726	YC96893 - YC96900	12/15/2032	115J14	HW07442
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COFFEE 783	YC92591	12/15/2032	115J14	HW07444
COFFEE 784	YC92592	12/15/2032	115J14	HW07442
COFFEE 785	YC92593	12/15/2032	115J14	HW07444
COFFEE 786	YC92594	12/15/2032	115J14	HW07442
COFFEE 787	YC92595	12/15/2032	115J14	HW07444
COFFEE 788	YC92596	12/15/2032	115J14	HW07442
COFFEE 789	YC92597	12/15/2032	115J14	HW07444
COFFEE 790	YC92598	12/15/2032	115J14	HW07442
COFFEE 791	YC92599	12/15/2032	115J14	HW07444
COFFEE 792	YC92600	12/15/2032	115J14	HW07442
COFFEE 793	YC92818	12/15/2032	115J14	HW07444



Claim Name and Number	Grant No.	Expiry Date	NTS #s	Grouping
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COFFEE 797	YC92822	12/15/2032	115J14	HW07444
COFFEE 798 - 865	YC92823 - YC92890	12/15/2032	115J14	HW07442
COFFEE 866 - 894	YC93271 - YC93299	12/15/2032	115J14	HW07442
COFFEE 895 - 910	YC92801 - YC92816	12/15/2032	115J14	HW07442
COFFEE 911 - 928	YD12701 - YD12718	12/15/2032	115J14	HW07443
COFFEE 929 - 960	YD12719 - YD12750	12/15/2032	115J14	HW07442
COFFEE 961 - 969	YD13231 - YD13239	12/15/2032	115J14	HW07443
COFFEE 970 - 1040	YD13241 - YD13311	12/15/2032	115J14	HW07443
COFFEE 1041 - 1168	YD13312 - YD13439	12/15/2032	115J14	HW07442
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COFFEE 1173 - 1180	YD13444 - YD13451	12/15/2032	115J14	HW07440
COFFEE 1181 - 1184	YD13452 - YD13455	12/15/2032	115J14	HW07444
COFFEE 1185 - 1192	YD13456 - YD13463	12/15/2032	115J14	HW07440
COFFEE 1193 - 1196	YD13464 - YD13467	12/15/2032	115J14	HW07444
COFFEE 1197 - 1204	YD13468 - YD13475	12/15/2032	115J14	HW07440
COFFEE 1205 - 1208	YD13476 - YD13479	12/15/2032	115J14	HW07444
COFFEE 1209 - 1216	YD13480 - YD13487	12/15/2032	115J14	HW07440
COFFEE 1217 - 1224	YD13488 - YD13495	12/15/2032	115J14	HW07444
COFFEE 1225 - 1244	YD13496 - YD13515	12/15/2032	115J14	HW07442
COFFEE 1245 - 1252	YD13516 - YD13523	12/15/2032	115J14	HW07444
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COFFEE 1273 - 1280	YD13544 - YD13551	12/15/2032	115J14	HW07444
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COFFEE 1301 - 1308	YD13572 - YD13579	12/15/2032	115J14	HW07444
COFFEE 1309 - 1328	YD13580 - YD13599	12/15/2032	115J14	HW07442
COFFEE 1329 - 1332	YD13600 - YD13603	12/15/2032	115J14	HW07444
COFFEE 1333 - 1352	YD13604 - YD13623	12/15/2032	115J14	HW07442
COFFEE 1353 - 1356	YD13624 - YD13627	12/15/2032	115J14	HW07444
COFFEE 1357 - 1376	YD13628 - YD13647	12/15/2032	115J14	HW07442
COFFEE 1377 - 1380	YD13648 - YD13651	12/15/2032	115J14	HW07444
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COFFEE 1413 - 1416	YD13684 - YD13687	12/15/2032	115J14	HW07442
COFFEE 1421 - 1429	YD13692 - YD13700	12/15/2032	115J14	HW07442
COFFEE 1430	YD42501	12/15/2032	115J14	HW07442
COFFEE 1435 - 1472	YD42506 - YD42543	12/15/2032	115J14	HW07442



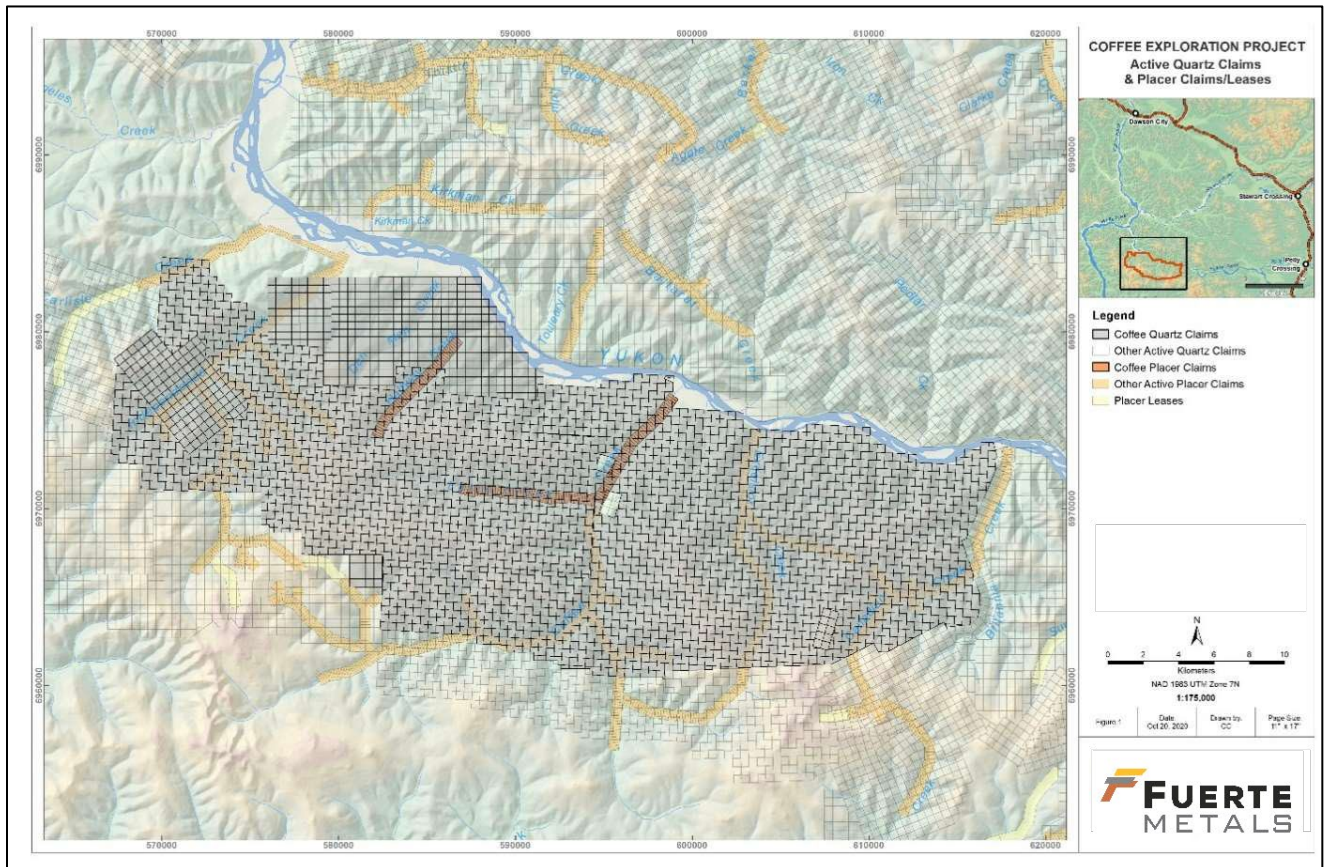
Claim Name and Number	Grant No.	Expiry Date	NTS #s	Grouping
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COFFEE 1497 - 1714	YD42701 - YD42918	12/15/2032	115J14, 115J15	HW07444
COFFEE 1715 - 1718	YD43085 - YD43088	12/15/2032	115J14	HW07442
COFFEE 1719 - 1798	YD43929 - YD44008	12/15/2032	115J14, 115J13	HW07629
COFFEE 1799 - 1954	YD44009 - YD44164	12/15/2032	115J13	HW07630
COFFEE 1955 - 1986	YD16283 - YD16314	12/15/2032	115J14	HW07440
COFFEE 1987 - 1992	YD16315 - YD16320	12/15/2032	115J14	HW07444
COFFEE 1993 - 1996	YD16321 - YD16324	12/15/2032	115J14	HW07440
COFFEE 1997 - 2006	YD16325 - YD16334	12/15/2032	115J14	HW07442
COFFEE 2007 - 2012	YD16335 - YD16340	12/15/2032	115J14	HW07444
COFFEE 2013 - 2016	YD16341 - YD16344	12/15/2032	115J14	HW07440
COFFEE 2017 - 2022	YD16345 - YD16350	12/15/2032	115J14	HW07442
COFFEE 2023 - 2028	YD16351 - YD16356	12/15/2032	115J14	HW07444
COFFEE 2029 - 2032	YD16357 - YD16360	12/15/2032	115J14	HW07440
COFFEE 2033 - 2038	YD16361 - YD16366	12/15/2032	115J14	HW07442
COFFEE 2039 - 2044	YD16367 - YD16372	12/15/2032	115J14	HW07444
COFFEE 2045 - 2048	YD16373 - YD16376	12/15/2032	115J14	HW07440
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COFFEE 2055 - 2060	YD16383 - YD16388	12/15/2032	115J14	HW07444
COFFEE 2061 - 2064	YD16389 - YD16392	12/15/2032	115J14	HW07440
COFFEE 2065 - 2076	YD16393 - YD16404	12/15/2032	115J14	HW07444
COFFEE 2077 - 2080	YD16405 - YD16408	12/15/2032	115J14	HW07440
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COFFEE 2093 - 2096	YD16421 - YD16424	12/15/2032	115J14	HW07440
COFFEE 2097 - 2116	YD16425 - YD16444	12/15/2032	115J14	HW07442
COFFEE 2117 - 2124	YD16445 - YD16452	12/15/2032	115J14	HW07444
COFFEE 2125 - 2264	YD89255 - YD89394	12/15/2032	115J15	HW07444
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COFFEE 2309 - 2316	YD89439 - YD89446	12/15/2032	115J14	HW07444
COFFEE 2317 - 2346	YD89447 - YD89476	12/15/2032	115J14	HW07442
COFFEE 2347-2846	YD91501-YD92000	12/15/2032	115J15, 115J14	HW07440
COFFEE 2847 - 2936	YD90101 -YD90190	12/15/2032	115J15	HW07440
COFFEE 93	YC60164	12/15/2032	115J14	HW07444
COFFEE 94	YC60165	12/15/2032	115J14	HW07442
COFFEE 95	YC60166	12/15/2032	115J14	HW07444
COFFEE 96	YC60167	12/15/2032	115J14	HW07442
COFFEE 97	YC60168	12/15/2032	115J14	HW07444



Claim Name and Number	Grant No.	Expiry Date	NTS #s	Grouping
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COFFEE 99	YC60170	12/15/2032	115J14	HW07444
COFFEE 100	YC60171	12/15/2032	115J14	HW07442
COFFEE 101	YC60172	12/15/2032	115J14	HW07444
COFFEE 102	YC60173	12/15/2032	115J14	HW07442
COFFEE 103	YC60174	12/15/2032	115J14	HW07444
COFFEE 104	YC60175	12/15/2032	115J14	HW07442
COFFEE 579 - 586	YC92769 - YC92776	12/15/2032	115J14	HW07630
COFFEE NW 1 - 108	YF01901 - YF02008	12/15/2029	115J13	HW07629
CREAM 1 - 22	YC60088 - YC60109	12/15/2032	115J13	HW07630
CREAM 23 - 68	YC83144 - YC83189	12/15/2032	115J13	HW07630
DM 1 - 6	YC83307 - YC83312	4/29/2030	115J14	HW07760
DM 21 - 114	YC83327 - YC83420	4/29/2030	115J14	HW07760
DM 115 - 124	YD10495 - YD10504	4/29/2027	115J14	HW07760
DM 125 - 427	YD60615 - YD60917	4/29/2027	115J14	HW07760
LION 1 - 16	YC83761 - YC83776	12/15/2032	115J14	HW07443
SUGAR 1 - 10	YC95568 - YC95577	12/15/2032	115J15	HW07444

Figure 4.2 shows the current NGC claims as well as overlapping placer claims held by other entities on the Coffee Gold Project.

Figure 4.2
Layout of the Coffee Gold Project Claims



Source: Modified after the JDS Internal Engineering Report, 2024

The Mineral Project is primarily located on Crown land, except for one fee simple parcel owned by the Proponent in the vicinity of the existing exploration camp near the Yukon River. Fuerte currently holds quartz mineral claims for the Coffee Gold Project. In addition, Fuerte acquired placer claims and leases on Halfway, Latte and Lower Coffee Creek, to ensure there is not simultaneous or competing mineral development.

4.2.2 Mineral Title Obligations

All mineral claims in Yukon are valid for one year after recording. To maintain a claim, the recorded holder must, on or before the expiry date (or "good to date") of the claim, either perform, or have performed, exploration and development work on that claim and register such work online or register an online payment in lieu of exploration and development work. Previous operators/owners (Newmont) accrued enough work on the Coffee Gold Project claims so that their expiry dates are in 2027 and 2032 as specified in Table 4.1. On an annual basis, a mineral project owner must expend CA\$100 per claim to hold the claims in good standing. Claims can also be held in lieu of work by submitting a cash payment equivalent to the work requirement of CA\$100 per claim. The Project is in good standing order, and all the mining claims are deemed current.

Only work prescribed in the Yukon Quartz Mining Act is acceptable for registration as assessment credit on a claim. The necessary approvals and permits under the Quartz Mining Act must be obtained before



any mechanical disturbance of the surface of the ground is performed by, or on behalf of, the recorded holder.

4.2.3 Surface Rights

Fuerte has inherited from Newmont all permits and authorizations required from governmental agencies to allow surface drilling and exploration activities on the Coffee Gold Project.

The Project holds a Class 4 Quartz Mining Land Use Approval (LQ00552) issued by Yukon Government, Department of Energy, Mines and Resources. The Class 4 Permit expires May 27, 2031, and includes provisions for year round operation of the Java camp, seasonal camp, 120 km of existing and new access roads, a winter road, construction of the Java airstrip and, substantial surface drilling and exploration activities on the property. The authorization also contains a number of operating and environmental protection restrictions and requires CA\$220,400.91 of security held for the Project.

The Project holds a Type "B" Water Licence (MN16-034-2) for the use of water and deposition of waste associated with the Coffee camp, and an occupancy of 100 persons. The permit expires July 11, 2026.

4.3 Royalties and Agreements

The Coffee Gold Project was previously owned by Kaminak, an indirect subsidiary of Newmont. Fuerte entered into an agreement dated September 12, 2025, to acquire all of the shares of Kaminak from a wholly owned subsidiary of Newmont in consideration for: (i) payment in cash in the amount of US\$10 million; (ii) the issuance of common shares and preferred shares of Fuerte having an aggregate value of US\$40 million; and (iii) the assumption of a 3% NSR on the Coffee Gold Project which is payable to Newmont and may be repurchased by Fuerte for US\$100 million within 12 months of the commencement of commercial production. The transaction closed on October 17, 2025, and the Project is now 100% owned by Fuerte under the indirect subsidiary Kaminak Gold Ltd.

Additional royalty agreements pertaining to the property include NSRs with a total sliding scale effective rate of 2.2%-2.8%, net of a one-time buyback for CA\$2 million, which can be exercised ahead of the commencement of production. The effective rate increases over time, reaching a maximum in year ten.

To the QP's knowledge there are no outstanding encumbrances or challenges to title of the mining claims, as shown in records held by the Yukon Department of Energy, Mines and Resources (EMR).

4.4 Permitting And Environmental Liabilities

While progressive reclamation of exploration disturbance has occurred over years of exploration, access trails, drill pads, roads, infrastructure and equipment (e.g., camp trailers, heavy equipment) anticipated to be utilized or disturbed as part of mine development remain in place. It is the Micon QP's (Charley Murahwi) understanding that further permitting and environmental studies will be required, if further exploration, test mining and economic studies demonstrate that the mineralization is sufficient to host a mining operation.

Applications for Water Use Licence and Quartz Mining Licence permits required for mine construction and operation were submitted in November 2023 and are currently under review. Authorizations for all-season access road construction are also under review. Permitting required to support the advancement of the Project is discussed in Section 20.0.

4.5 Other Significant Factors/Risks

Micon QP is not aware of any other significant factors or risks, other than those discussed in this Report, that may affect access, title or the right or ability to perform work on the Project by Fuerte.



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following descriptions have been excerpted from the Internal Engineering Report (JDS, 2024) with minor edits by the Micon QP.

5.1 Accessibility

The Coffee Gold Project is located approximately 130 km south of Dawson and 160 km northwest of Carmacks, within the Dawson Range, Yukon Territory, Canada. Access to the Project is primarily by airplane or helicopter from Whitehorse and/or Dawson, or by barge via the Yukon River.

An airstrip is located at the Coffee Gold Project camp, approximately 10 km from the gold mineralization areas. Currently, river transport along the Yukon River is available for five months during the summer ice-free period.

Previously (in 2011), Kaminak constructed a 23 km road from the barge landing at the Coffee Gold Project camp to the Supremo and Latte drilling areas, which served as the main exploration access road from 2012 to 2015.

5.2 Climate And Vegetation

5.2.1 Climate

The Project area and the Yukon as a whole, has a subarctic, continental climate with a summer mean temperature of 10°C and a winter mean temperature of -23°C. Summer and winter temperatures can reach up to 38°C and -55°C, respectively. Dawson, the nearest access point, has a daily temperature average above freezing for 180 days of the year.

The climatic conditions restrict exploration activities to the summer months. However, mining activities could be structured for year-round operations.

5.2.2 Vegetation

The Coffee Gold Project claims encompass an area of partially tree-covered hills. The Project area has local mature pine forests with thick moss cover on the ground. However, the majority of the Project is above tree line and contains short shrubby vegetation.

5.3 Physiography

The Coffee Gold Project is located in the northern Dawson Range, forming a moderate plateau that escaped Pleistocene glaciation. As such, the topography of the area is defined by stream erosion resulting in gently rounded hills with tightly incised valleys. Across the Project, elevations range from 400 to 1,500 m above sea level.

Bedrock exposures are scarce. A typical landscape/topographic view in the Mineral Project area is depicted in Figure 5.1.

Figure 5.1
Aerial View of the Kona, Latte, Supremo and Double-Double Mineralized Zones (Looking Northwest)



Source: Kaminak, 2016.

5.4 Infrastructure And Local Resources

There are no all-weather or winter roads connecting the Coffee Gold Project to any of the major communities in the Yukon. However, there is proposed construction of a 214 km all-weather gravel road between Dawson and the Coffee Gold Project (Figure 5.2). Crossing of the Stewart and Yukon Rivers will be by barge in summer and ice road in winter. An airstrip is located at the Coffee camp approximately 10 km from the areas of gold mineralization.

Currently, river transport along the Yukon River, with multiple barge access points to the Coffee camp, is available for five months during the summer period when the river is free of ice.

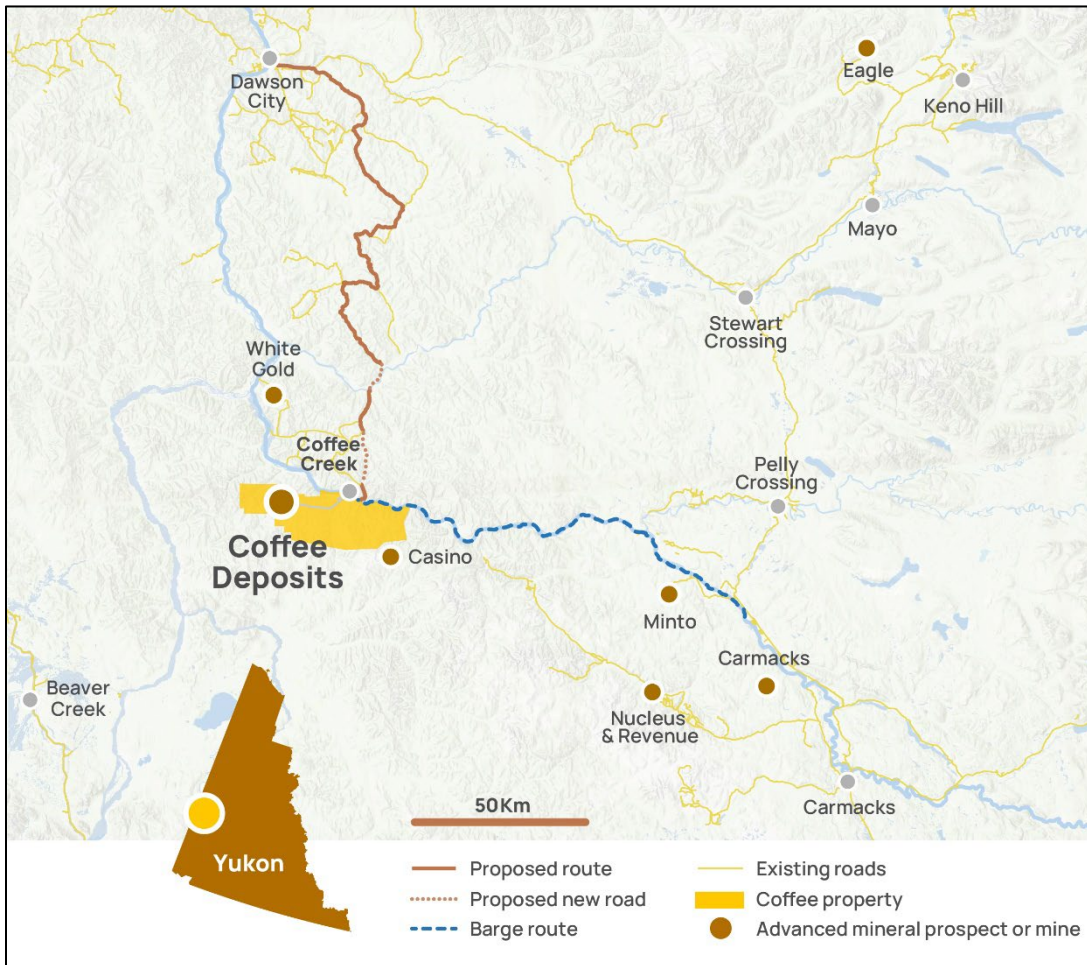
For the immediate/foreseeable future, power could be from generators (LNG, diesel, or solar powered). Water is available from boreholes and the nearby Yukon River.

The Coffee Gold Project is isolated from population centres. Mining personnel would have to be transported from other regions in the Yukon/Canada, depending on the specific tasks.

5.5 Sufficiency of Surface Rights for Mining Operations

The Project is of sufficient size to accommodate facilities including tailings storage areas, waste disposal areas, heap leach pad and processing plant required to allow mining activities to proceed, if economic mineralization in sufficient quantities is discovered on the Project.

**Figure 5.2
Proposed All-weather Access Road Alignment**



Source: Fuerte, 2025

5.6 Local Resources

5.6.1 (i) Availability of Mining Personnel

The Coffee Gold Project is isolated from population centres. Thus, mining personnel would have to be transported from other regions in the Yukon/Canada, depending on the specific tasks.

5.6.2 (ii) Sufficiency of Surface Rights for Mining Operations

The Mineral Project is of sufficient size to accommodate all facilities required to allow mining activities to proceed, if economic mineralization in sufficient quantities is discovered on the Mineral Project.



6.0 HISTORY

6.1 General Statement

In this section, the discovery history of the Coffee Gold Project has been excerpted from the YGS Open File 2025-6 while the prior ownership and exploration history is edited from the Internal Engineering Report prepared for Newmont. The Coffee Gold Project is a recent discovery without known artisanal workings.

6.2 Discovery History

Early exploration work in the vicinity of the Project commenced in the 1960s and 1970s following the discovery of the Casino copper-gold porphyry and continued into 1986 when the Geological Survey of Canada conducted regional-scale stream sediment sampling. Results from this survey did not produce any anomalous gold values from samples collected in the tributaries draining the main deposit, but elevated arsenic and antimony were noted. In 1999, Deltango Gold Ltd. conducted silt and soil geochemical sampling programs. Prospector International Resources Inc. also conducted silt and soil geochemical sampling programs in 1999 and 2000 and successfully delineated a gold-in-soil anomaly approximately 400 x 900 m on the eastern margin of Supremo. No follow-up work was conducted by Deltango Gold Ltd. or Prospector International Resources Inc. In 2006, Shawn Ryan staked the recently lapsed claims and conducted grid and ridge-top soil sampling utilizing Yukon Mineral Exploration Program (YMEP) grants issued by the Government of Yukon outlining an extensive gold-in-soil anomaly.

In 2009, Kaminak Gold Corporation entered into an option agreement with Shawn Ryan to acquire the Coffee Gold Project. After the acquisition, the claim package was expanded and exploration continued with surface trenching and soil sampling over the initial gold-in-soil anomaly in the main deposit area (Doerksen et al., 2016).

6.3 Prior Ownership and Exploration History

6.3.1 Kaminak (2009 – 2016)

The initial diamond drilling in 2010 (76 holes with a total metreage of 16,105 m) led to the discovery of the Supremo, Latte, Double-Double, Kona and Americano zones (Doerksen et al., 2016). Most significantly, the first drillhole CFD0001 directed at the Supremo deposit intersected 17.1 g/t gold over 15.5 m.

Following the discovery, Kaminak pursued an assortment of drilling programs (ranging from infill to step-out drillholes) from 2010 through 2015 on Supremo, Latte, Double-Double, Kona, Espresso, Americano, and Sugar. In addition to drilling and expanding the resources, Kaminak also conducted metallurgical tests to support the estimation of mineral resources.

6.3.2 Goldcorp (2016 – 2018)

In 2016, Kaminak Gold Corporation was acquired by Goldcorp Inc. (Goldcorp), who took ownership of the Coffee Gold Project and conducted additional metallurgical tests and limited drilling. Details of the work conducted by Goldcorp are provided in Sections 9 and 10.

6.3.3 Newmont (2019 – 2024)

Goldcorp subsequently merged with Newmont in 2019, with Newmont taking full ownership of the Coffee Gold Project.



The exploration and drilling activities completed by Goldcorp, and subsequently Newmont, from 2017 through 2023 are discussed in Sections 9 and 10.

6.4 Production History

There are no mineral production figures declared from the Coffee Gold Project.

6.5 Summary of Exploration Programs (2009 – 2023)

An annual summary of exploration programs from 2012 to 2017 highlights the extensive work conducted, including drilling, soil sampling, trenching, and geophysical surveys. Cumulatively, from 2009 to 2023, a total of 609,708 m of drilling and 54,404 soil samples have been collected. Several targets arising from these exploration programs remain to be drill tested as described under section 9.



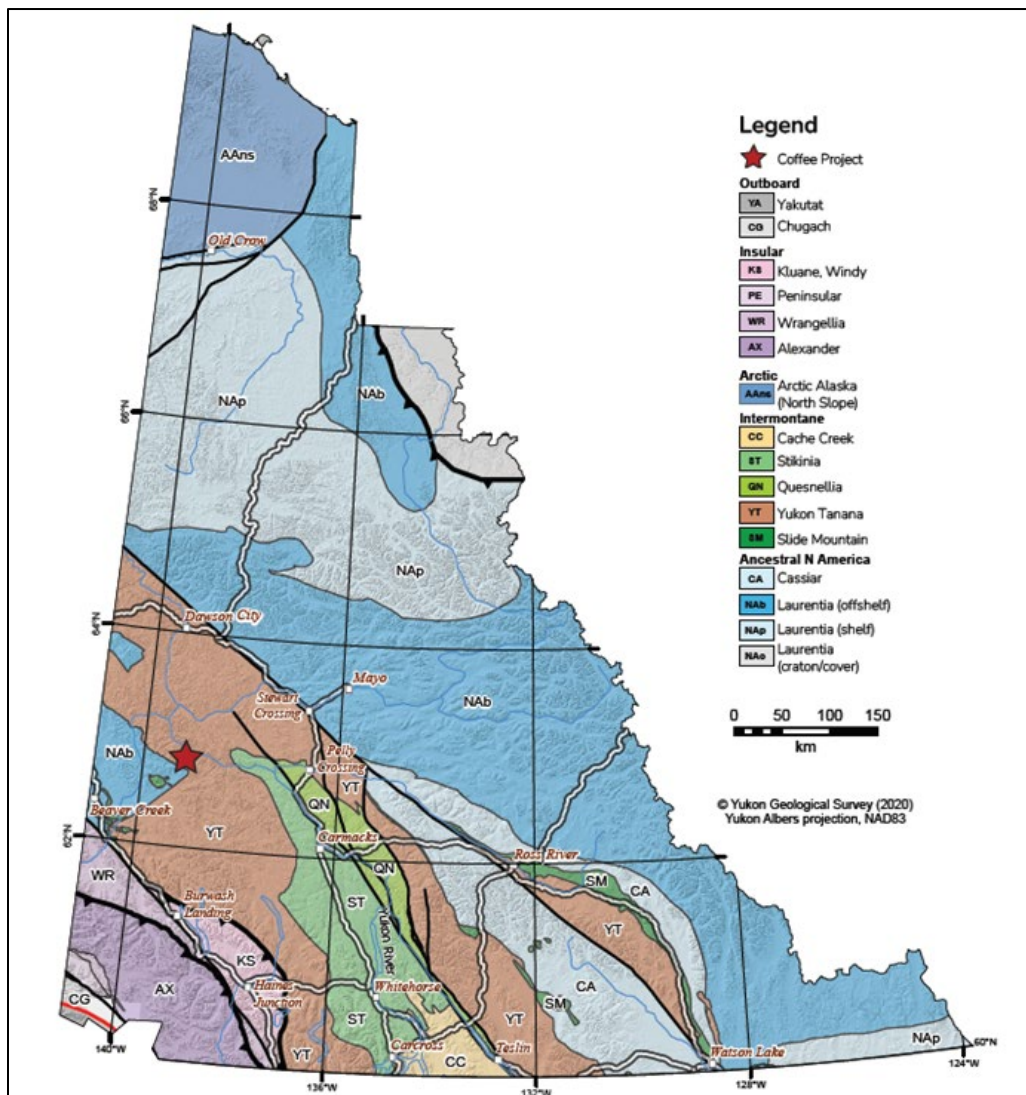
7.0 GEOLOGICAL SETTING AND MINERALIZATION

The Micon QP has reviewed the geology content of the YGS Open File 2025-6 and the Internal Engineering Report (JDS, 2024) and considers it current. The following discussion summarizes content from these reports.

7.1 Regional Geology

The Coffee Gold Project is located in the Yukon-Tanana Terrane (YTT). The YTT (Figure 7.1) is an allochthonous peri-cratonic terrane which has a protracted history of rifting, magmatism, and metamorphism spanning from the Late Devonian into the Mesozoic (Colpron et al., 2006).

Figure 7.1
Terrane Map of Yukon

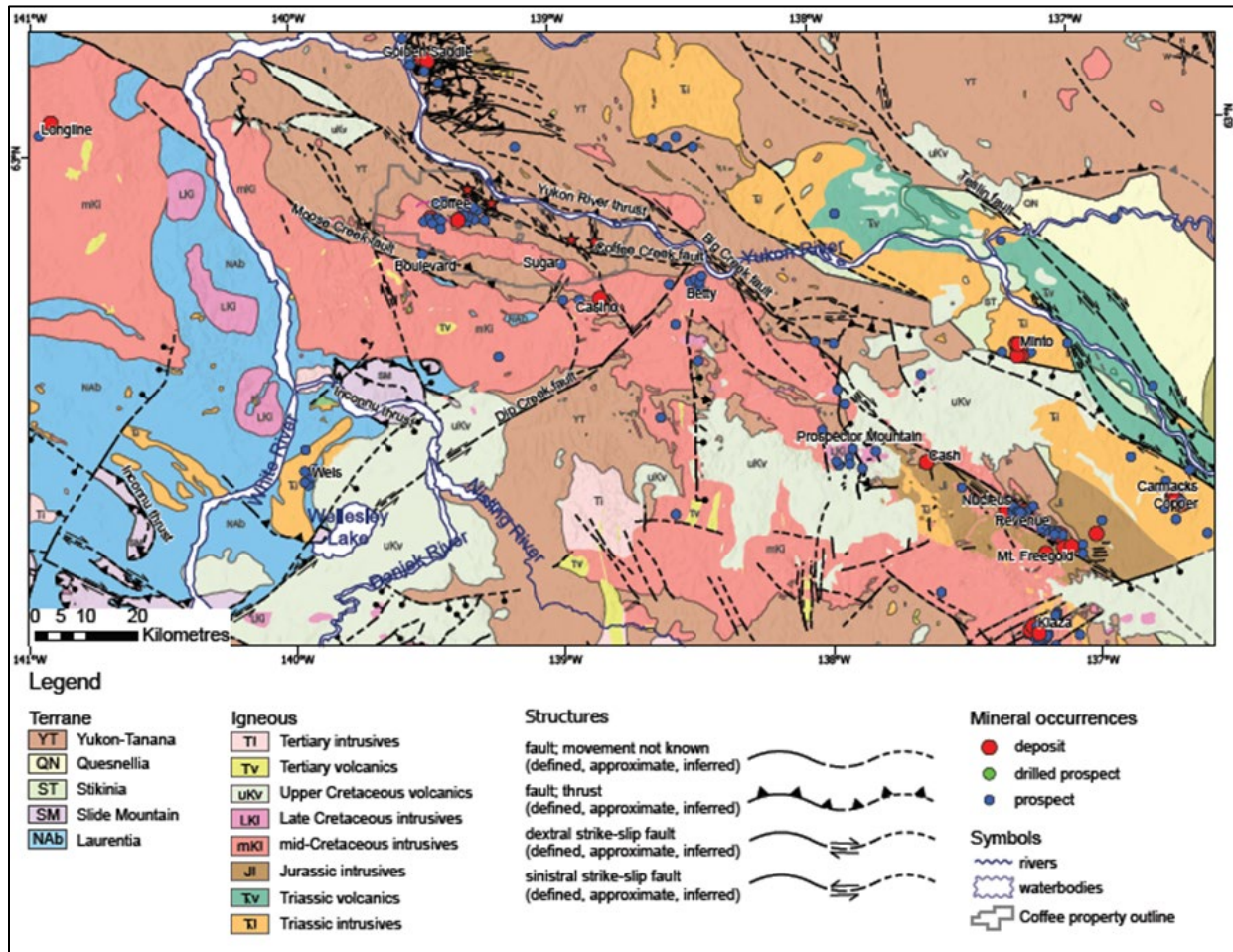


Source: YGS Open File 2025-6 [The red star indicates the location of the Coffee Gold Project.]



Numerous orogenic and magmatic-hydrothermal mineral deposits occur throughout the YTT as a result of orogenesis and widespread magmatism in the Mesozoic. These events range from Late Triassic magmatism that produced Cu-Au mineralization, to Late Cretaceous porphyry systems. Orogenic deposits also occur over two distinct intervals: between 163 and 155 Ma (e.g., Golden Saddle; Bailey, 2013); and the mid-Cretaceous at 96 to 92 Ma, such as the Boulevard and Longline occurrences (Figure 7.2; Joyce, 2002; Allan et al., 2013; McKenzie et al., 2013).

Figure 7.2
Overview Geology Map of the Dawson Range



Source: YGS Open File 2025-6

Figure 7.2 shows that metamorphic and sedimentary rocks have been simplified into their respective terranes, with intrusive and extrusive igneous rocks overlain. Faults displayed are from the YGS study (2025) and Yukon Geological Survey (2023a). Mineral occurrences are from MINFILE (Yukon Geological Survey, 2023a).

The Dawson Range in which the Coffee Gold Project is housed is a west-northwest-trending topographic plateau (approximately 250 km) in the YTT that remained unglaciated during the last ice age. In the Dawson Range the metamorphic rocks of the YTT consist of psammitic to semi-pelitic schists of the Snowcap assemblage, orthogneiss of the Sulphur Creek and Simpson Range suites, felsic to mafic metavolcanic and metasedimentary rocks of the Finlayson assemblage, micaceous schist of the Klondike assemblage, and undifferentiated ultramafic rocks (Ryan et al., 2013a, 2013b). The metamorphic units are intruded by granite



of the mid-Cretaceous Dawson Range batholith (DRB) and the Coffee Creek pluton (CCP), both of which are part of the Whitehorse plutonic suite (Ryan et al., 2013a, 2013b).

7.2 Regional Structures

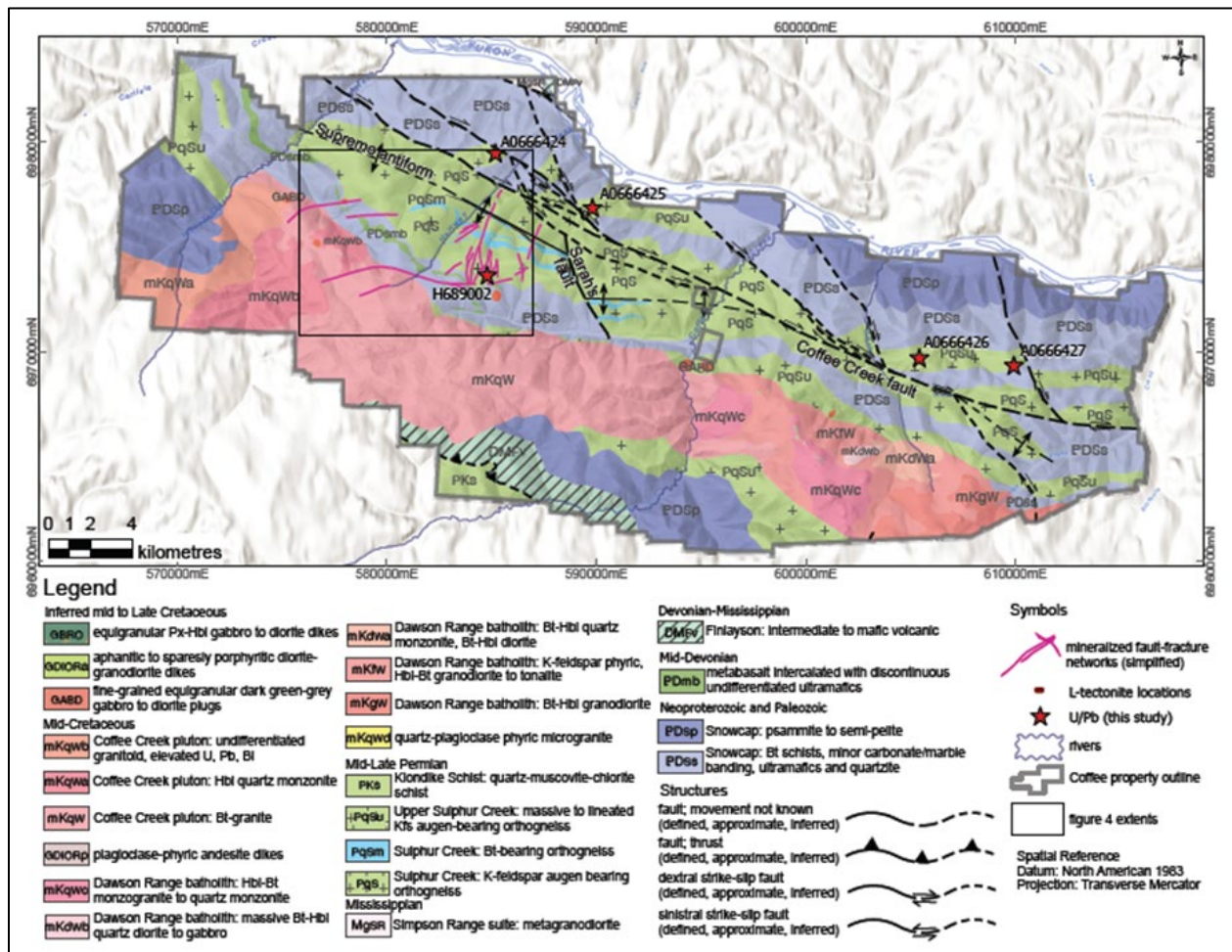
Early-stage deformation in the Dawson Range is recorded by Late Triassic to Early Jurassic metamorphic foliation fabrics and thrust imbrication in the Snowcap and Klondike assemblages, as well as northeast-verging folds and associated crenulation fabrics (Allan et al., 2013; Colpron et al., 2022). This deformation is also recorded in the Permian Sulphur Creek suite, which is locally intercalated with the Snowcap assemblage.

The earliest stages of deformation are cut by faults of various orientations. North of the Coffee Gold Project the Yukon River thrust fault juxtaposes Late Devonian to Early Mississippian meta-igneous rocks of the Simpson Range suite on top of the pre-Late Devonian Snowcap assemblage (Ryan et al., 2013a, 2013b). The Yukon River thrust is broadly east-west oriented and occurs roughly parallel to the Moose Creek fault, located south of the CCP (Figure 7.2). The Moose Creek fault is interpreted as a major bounding structure between YTT rocks to the northeast and North American basement rocks to the southeast (Figure 7.2). Large northeast-trending faults are identified in regional-scale maps, such as the Dip Creek fault (Ryan et al., 2013a) and are interpreted as oblique-sinistral faults broadly coeval with Late Cretaceous magmatism (Allan et al., 2013). The Dawson Range is also affected by a network of broadly northwest-trending faults (Figure 7.2), the most significant of which is the Big Creek fault (BCF) which runs for more than 150 km from the Mount Freegold area in the southeast past the Yukon River thrust in the northwest (Figure 7.2). The Coffee Creek fault (CCF) intersects the BCF system to the east of Coffee Gold Project and has been interpreted to be cut by and perhaps dextrally offset by the BCF based on lineation interpretation of aeromagnetic data (MacWilliam, 2018; Yukon Geological Survey, 2023a; M. Colpron, pers. comm., 2024). The CCF has been interpreted as the primary controlling structure for the Coffee deposit (MacWilliam, 2018) with recent mapping updating the location and nature of the deposit scale faults (see Discussion). Lastly, multiple north to northwest-trending faults are observed (Figure 7.2); the dextral slip fault crosscutting the Moose Creek fault, the normal fault separating the DRB from Paleozoic metamorphic rocks southeast of Casino, and the normal fault dissecting the Prospector Mountain suite (Figure 7.2).

7.3 Local Geology

During the period 2020 to 2023, several studies were undertaken to improve the geological understanding of the Coffee deposits. These studies included field and desktop mapping, relogging of drill core from the Sugar prospect, U-Pb geochronology, geochemical analysis of late intrusive phases and atypical gold mineralization, and structural analysis and reinterpretation of the main mineralization-controlling faults. Together these studies have further elucidated the geological and structural setting of the Coffee deposits during formation. The current understanding of the Project geology is summarized in Figure 7.3. The pre-Late Devonian meta-sedimentary rocks of the Snowcap assemblage are structurally overlying Late Permian meta-granitoid rocks of the Klondike assemblage, with the contact presumed to be a transposed intrusive contact (MacWilliam, 2018). Amphibolite and meta-basalt occur at the base of the Snowcap assemblage, and undifferentiated ultramafic units occur as lenses throughout the assemblage (MacWilliam, 2018). Finlayson assemblage rocks are observed near the southern end of the Project and stratigraphically overlie Snowcap assemblage psammite. The undeformed CCP intrudes the Paleozoic metamorphic assemblage, as do a series of Cretaceous mafic to felsic dikes (Figure 7.4).

Figure 7.3
Simplified Project Geology Map



Source: YGS Open File 2025-6.

Note: Red stars = Zircon U-Pb geochronology samples. Black box delineates the extents shown in Figure 7.4.

7.3.1 Pre-Late Devonian Snowcap assemblage

7.3.1.1 Semi-pelitic schist, marble and psammite (PDSs and PDSp)

The Snowcap assemblage is divided into two property-scale units that are readily identified in outcrop mapping, soil litho-geochemistry and radiometrics. The dominant unit is a dark blue grey to black, fine to medium-grained, strongly foliated muscovite-biotite schist with varying abundances of millimetre-scale quartz and feldspar augen.

The second subunit of the Snowcap assemblage consists of dark blue grey to black fine-grained, weakly foliated, compositionally banded psammite and minor pelite with local hornfels proximal to the CCP.



7.3.1.2 *Amphibolite and meta-basalt (PDmb)*

Towards the structural base of the Snowcap schist package there are discrete and discontinuous lenses of dark blue grey to green-grey fine to medium-grained, variably foliated amphibolite. These lenses are dominated by hornblende, biotite, and lesser quartz and feldspar.

Meta-basalt is also observed within the Snowcap schist as massive to banded lenses of striped to mottled dark green-grey to white fine to coarse-grained augite, biotite, chlorite, epidote and albite. Penetrative foliation is observed throughout and is defined by distinct compositional layering comprising strained albite, augite-biotite and chlorite bands (ibid.). Three-dimensional modelling of drillhole data and analysis of soil geochemistry data indicates that there are two major bands of this unit within the Snowcap package, with other minor lenses occurring near the Latte and Double-Double zones.

7.3.1.3 *Paleozoic ultramafic rocks*

Throughout the Snowcap assemblage are discontinuous bodies up to 5 m wide of light grey and light to dark green-grey fine-grained talc schist, listwanite, serpentinite, and variably carbonate, magnesite, fuchsite, and epidote-altered rock (MacWilliam, 2018). Observed contacts between ultramafic rocks and surrounding schists are commonly sharp and typically crosscut the regional foliation at low angles. Lenses commonly display a strong to intense ductile shear fabric with locally observed recumbent folding. Intense and pervasive carbonatization and serpentinization completely destroy the primary mineralogy and texture of these rocks (ibid.).

7.3.2 Devonian to Mississippian Finlayson Assemblage (DmFv)

The Finlayson assemblage consists of moderate to strongly foliated dark green-grey amphibolite with fine-grained hornblende-biotite and trace garnet. Foliation within this unit is subvertical and foliation displays parasitic folding. These rocks are not observed in drill core and are not present in the main deposit area.

7.3.3 Late Permian Klondike assemblage

7.3.3.1 *Sulphur Creek suite – Supremo pluton (PqS)*

The K-feldspar augen-bearing orthogneiss of the Sulphur Creek suite (PqS), is pink-grey, medium to coarse-grained, and interlayered on a centimetre to metre-scale with medium blue-grey fine to medium-grained biotite-muscovite-feldspar-quartz schist (PqSm; Plate 1; MacWilliam, 2018). There is a main central panel of the augen orthogneiss, herein called the Supremo pluton, and a structurally higher intrusion of the orthogneiss referred to here as 'Upper Sulphur Creek' (PqSu).

7.3.3.2 *Upper Sulphur Creek (PqSu)*

The previously unknown texturally distinct upper subunit of the Sulphur Creek suite was first mapped in 2022 and was not immediately recognized as being part of the Sulphur Creek suite. Local textural similarities and U-Pb dating in this study led to this conclusion (see Geochronology results). This unit is separated from the main Supremo pluton orthogneiss unit by a panel of Snowcap assemblage and is observed on the northern limb of the Supremo antiform as well as truncating against the CCP on the southern limb of the antiform. The subunit is mineralogically identical to the orthogneiss; however, in addition to the typical S-tectonite textures it locally exhibits both an undeformed phaneritic texture and distinctive lineation fabric (L-tectonite) defined by elongated and plastically deformed feldspars. All three textures are observed intermittently along strike of the unit.



7.3.3.3 Klondike Schist (PKf and PKs)

Regionally, the Klondike Schist is mapped as a quartz-muscovite-chlorite schist (Yukon Geological Survey, 2023a). A single package of Klondike Schist overlies the mapped Finlayson assemblage in the southern portion of the Coffee Gold Project.

7.3.4 Mid to Late Cretaceous plutonic rocks

7.3.4.1 Coffee Creek pluton (mKfw, mKdwa, mKdwb, mKdwc, mKqw, mKqwa, mKqwb, mKqwc)

The mid-Cretaceous CCP is a composite pluton that intrudes the metamorphic rock package south of the Supremo antiform and forms a west-trending ridge south of the deposit where resistive outcrops and tors of granite are exposed. This unit connects to the DRB southeast of the Coffee Gold Project near the Casino deposit and extends to the western edge of the Coffee Gold Project boundary.

East of the Coffee Creek drainage there are five phases of mid-Cretaceous intrusions ranging from gabbro to monzogranite. Dated units in this area pre-date the western portion of the CCP slightly with U-Pb zircon crystallization ages ranging from 110.9 to 104 Ma (Yukon Geological Survey, 2023b) which are closer in age to the DRB.

West of the Coffee Creek drainage the CCP yielded dates ranging from ~103 to ~97 Ma (McKenzie et al., 2013; Buitenhuis, 2014; MacWilliam, 2018). Three phases are recognized within the CCP west of the Coffee Creek drainage: biotite granite (mKqw), hornblende quartz monzonite (mKqwa), and an undifferentiated granite phase mapped primarily by remote sensing techniques (mKqwb). These three younger western phases of the CCP are collectively referred to as the Coffee Creek granite.

7.3.4.2 Cretaceous intrusions

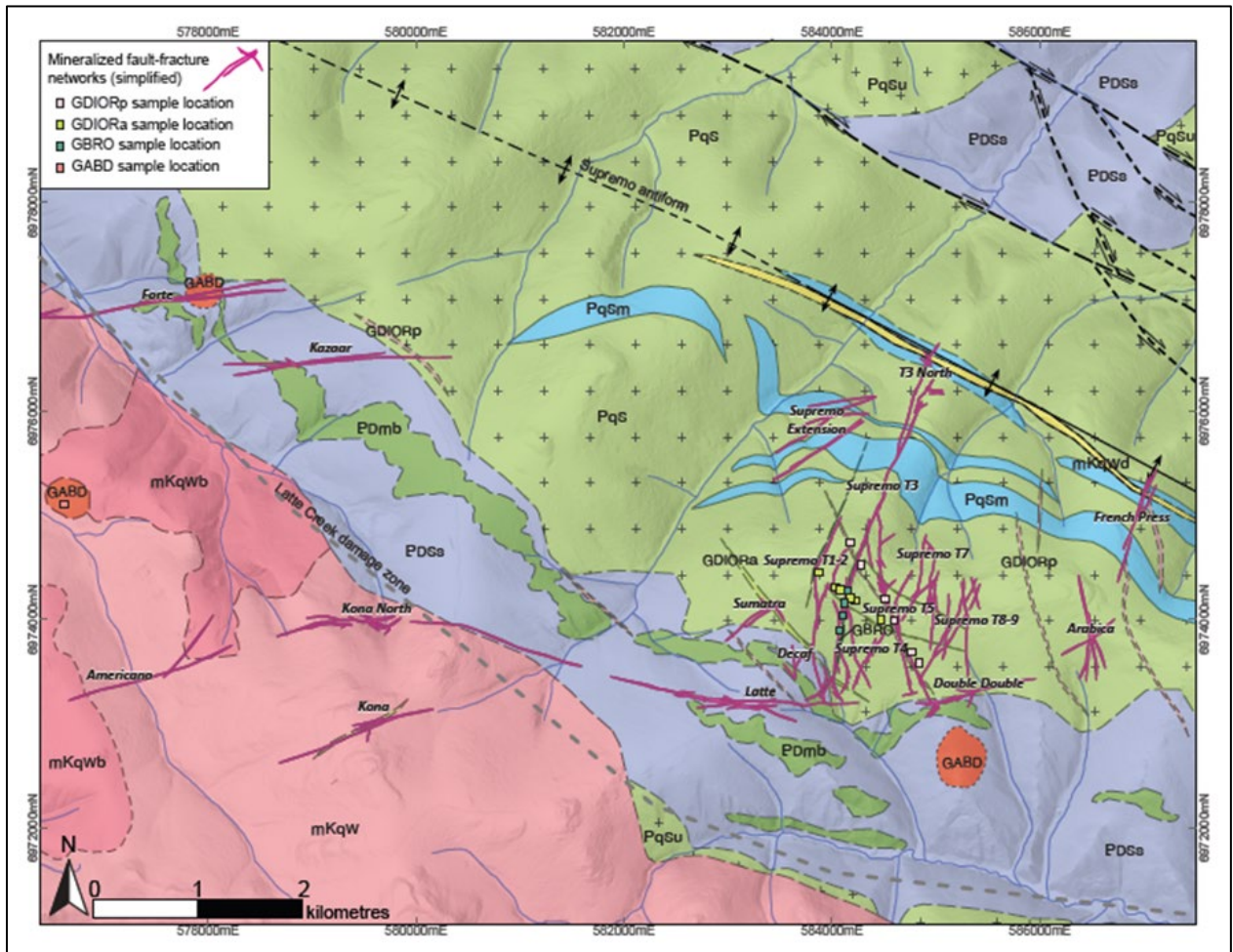
Five distinct phases of Cretaceous dikes and plugs intrude the metamorphic units; three of which also intrude the CCP. Geochemically they are grouped as microgranite, diorite to granodiorite, and monzogabbro to gabbroic diorite (MacWilliam, 2018). Two phases of diorite dikes are observed: a porphyritic phase and an aphanitic phase. All dikes are steeply dipping and crosscut the regional metamorphic foliation, striking east, north to northeast, and northwest.

7.4 Mineralization

7.4.1 Overview of Coffee Deposits

An overview of the main Coffee deposits is presented in Figure 7.4. The mineralized fault-fracture networks are represented in pink, with deposit and inventory zones/subzones labelled. Dike geochemistry sampling locations are shown as coloured squares. See Figure 7.3 for lithology legend.

Figure 7.4
Overview the Coffee Deposits



Source: YGS Open File 2025-06.
Note: See Figure 7.3 for lithology legend.

7.4.2 Deposit Structural Setting

The host rocks at the Coffee Gold Project were deformed by a series of YTT-wide Paleozoic tectonic events, as well as Cretaceous deformation (e.g., MacWilliam, 2018). Shallow south-southwest-dipping compositional layering (S0) is observed locally within schist of the Snowcap assemblage, and as layers of marble and quartzite (MacWilliam, 2018). Tectonic foliation (S1) formed parallel to S0 during a flattening episode (D1) likely in the Early Jurassic (Colpron et al., 2022). The S1 fabric is axial planar to the earliest phase of folding (F1) observed, which consists of tight to isoclinal folds (MacWilliam, 2018). This schistosity is defined by alignment of biotite and muscovite in mica-bearing units and by hornblende in metagabbro (ibid.). Quartz veins interpreted as metamorphic segregations or transposed early quartz veins also help define this schistosity (ibid.).

Following the development of the early D1 fabrics, the regional S1 metamorphic fabric was broadly folded into the Supremo antiform (Figure 7.3). Foliation measurements in drill core from the southern limb indicate S1 dips shallowly (~30°) to the southwest, steepening slightly towards the south in the Latte area. To the north of the Coffee deposits the fabric in the northern limb dips moderately to the north-northeast and



similarly becomes steeper in proximity to the Yukon River. The axial trace of this F2b antiform has largely been inferred from S1 measurements in core but can be observed in exploration trenches north of the main deposit and in outcrop in Coffee Creek.

The undeformed mid-Cretaceous CCP intrudes the metamorphic units. The Cretaceous dikes are also largely undeformed, apart from minor shearing along the contacts of the monzogabbro and aphanitic diorite dike phases. These Cretaceous dikes are spatially associated with subvertical fault-fracture networks that are the primary controls of mineralization at Coffee. These faults are interpreted as high-order structures related to the CCF (MacWilliam, 2018).

7.4.3 Mineralization and Alteration

The mineral endowment at the Coffee Gold Project is hosted within several zones each with varying host rocks and structural orientations. Dimensions are variable in length (100 m to 5000 m), widths (1 to > 10 m) and depths not fully explored to date. The bulk of the mineral resource occurs within the Supremo zone which is predominantly hosted within augen-bearing orthogneiss of the Sulphur Creek suite (Figure 7.4). The Supremo zone is further broken down into different 'T'-structures named after exploration trenches (T1 through T9, excluding T6); the most significant of which is T3 which trends north-northeast and extends for approximately 5 km of strike length (Figure 7.4). The Supremo T-structures are mostly north-northwest to northeast-trending and form an interconnected array of fault-fracture networks (Figure 7.4). The Latte and Double-Double zones are located south of Supremo, and each consist of nearly east-west trending structures that crosscut psammitic to semi-pelitic schists of the Snowcap assemblage (Figure 7.4; MacWilliam, 2018). The T-structures intersect Latte and Double-Double south of the Snowcap-Sulphur Creek contact. The Kona and Kona North zones are the main bodies of mineralization hosted within the CCP. These zones consist of east-northeast trending fault-fracture networks (Figure 7.4). Other mineralized zones at the Coffee Gold Project are situated in east-west trending fault fracture networks like Kazaar, Forte and Americano; north-northwest to north-northeast trending structures like Arabica, Decaf and French Press; and northeast trending structures like Supremo Extension and Sumatra (Figure 7.4).

Gold occurs as fine-grained auriferous pyrite, arsenopyrite and arsenian pyrite (MacWilliam, 2018). Mineralization is controlled by subvertical fault-fracture networks, characterized by variably mature, polyphase tectonic fault gouge and fault-fill breccias and cataclasites (MacWilliam, 2018). These brittle structures range in deformation intensity from angular clast-supported crackle breccias to intensely milled fault gouge. The mature, gouge-rich breccias represent the centre of these fault-fracture networks, or 'fault core', whereas the less mature crackle breccias are typical of the edges of these fault-fracture networks. Quartz-carbonate cement is present in some degree within the matrix of some tectonic breccias; however, textures indicated that the quartz-carbonate fluid is infilling the breccia and did not play a major role in hydrothermally fracturing the rock. There is a notable lack of quartz or carbonate veining associated with the mineralizing fault corridors; however, rarely sulphide minerals can be associated with quartz-dolomite gangue as part of the quartz-rich breccias (Buitenhuis, 2014; Buitenhuis et al., 2015; MacWilliam, 2018). Competent Fe-carbonate and quartz-carbonate cemented breccias occur sporadically in the fault zones but are volumetrically minor and exhibit no clear spatial correlation. Fe-carbonate-rich breccias are post-mineralization (e.g., Buitenhuis, 2014). In addition to fault-controlled gold mineralization, minerals of interest also occur disseminated in the wallrock immediately surrounding these fault arrays. Supergene oxide mineralization has been the focus of exploration since Coffee's discovery and occurs as limonite-hematite-goethite at depths of up to ~400 m and affects all styles of hypogene mineralization described above (Buitenhuis, 2014; Buitenhuis et al., 2015; MacWilliam, 2018).

Lithology also acted as a primary control on mineralization, as mineralization favours Fe-rich units containing pyrite and biotite (MacWilliam, 2018). In Latte and Double-Double, the dominant lithology is the Snowcap assemblage biotite schist. Since there is more iron available in biotite-rich units for mineralizing reactions, disseminated mineralization tends to extend farther into the wallrock from the controlling fault zones (up to 20 m in Latte zone; MacWilliam, 2018). In these zones sulphidation reactions consume Fe-bearing micas and produce foliation-parallel white mica + illite + arsenian pyrite pseudomorphs (Buitenhuis,



2014; MacWilliam, 2018). In contrast, disseminated mineralization adjacent to controlling structures in the relatively biotite-poor Sulphur Creek orthogneiss (Supremo zone) and CCP (i.e., Kona zone) tends to reach 5 to 10 m into the wallrock (MacWilliam, 2018). Additionally, high-grade gold mineralization occurs along the intersection lineation between biotite-rich bands in the Sulphur Creek orthogneiss and mineralized fault-fracture networks. The fault-fracture networks are also exploited by gabbroic and granodioritic dikes which themselves are commonly brecciated and mineralized (MacWilliam, 2018).

Alteration typically consists of white mica + illite + kaolinite + quartz + ankerite (Buitenhuis, 2014). Arsenopyrite, stibnite, realgar and native arsenic are also observed locally within mineralized zones in minor abundances (MacWilliam, 2018). White micas, as interpreted from available short-wave hyperspectral analyses, consist of NH₄ white mica, paragonitic muscovite and muscovite in mineralized zones (MacWilliam, 2018). Available data suggests some concentric zonation of NH₄ white mica and paragonitic muscovite may exist in mineralized zones. No other vertical or horizontal alteration zonation is documented at Coffee.

Arsenic is intimately associated with gold at the Coffee Gold Project as the gold is hosted within arsenian pyrite (e.g., Wainwright et al., 2011; Buitenhuis et al., 2015; MacWilliam, 2018). Antimony is also used as an indicator of gold as it occurs as stibnite and within arsenian pyrite (MacKenzie et al., 2013). Sodium and potassium are observed to decrease with increasing gold mineralization as a result of the alteration of biotite, phengite and feldspar to the alteration assemblage of white mica and kaolinite (MacKenzie et al., 2014).



8.0 DEPOSIT TYPES

8.1 General Statement

Coffee has been described as a mid-Cretaceous, gold-only deposit that has been classified as a variety of deposit types over the Project's history including epithermal, reduced intrusion related, Carlin-type, and orogenic-gold, as well as combinations of the four proposed deposit types (i.e., over-printing or combination of fluid sources) (JDS, 2024). This implies that the Coffee Gold Project deposit type(s)/genetic model is yet to be resolved.

8.2 Current Ideas on the Deposit Type

The difficulty in elucidating the deposit type has been reinforced by the most recent work as recorded in the YT review statement excerpted below:

“Over the Project's history, it has been stated that the Coffee deposit shares similarities with low-sulphidation epithermal, reduced intrusion-related, Carlin-type (and 'Carlin-like'), orogenic gold deposits, and a combination of these (Cruikshank, 2011;; Buitenhuis et al., 2015; MacWilliam, 2018). 'Carlin-like' is not a formally defined term but is often used to describe gold deposits that share some, but not all, characteristics of Carlin-type deposits.

Allan et al. (2013), Buitenhuis (2014), and MacKenzie et al. (2014) suggested that Coffee resembles an epizonal orogenic gold system based on similarities to nearby prospects (i.e., Boulevard). Other studies, as early as 2013, interpreted Coffee deposits to be at least partially an epithermal gold deposits despite lacking typical characteristics such as a $\text{Ag} \pm \text{Pb-Zn}$ association and quartz + adularia veins (Cruikshank, 2011; Chartier et al., 2013; Buitenhuis et al., 2015). Buitenhuis et al. (2015) described 'epithermal-like textures' and interpreted them as an overprinting epithermal stage of mineralization, or the latest stages of a single evolving epizonal orogenic system. MacKenzie et al. (2013) also described sooty-sulphide-bearing veins with prismatic quartz and banded carbonate veins that crosscut the main mineralizing phase in Latte and Double-Double but did not suggest that these represent an epithermal environment.

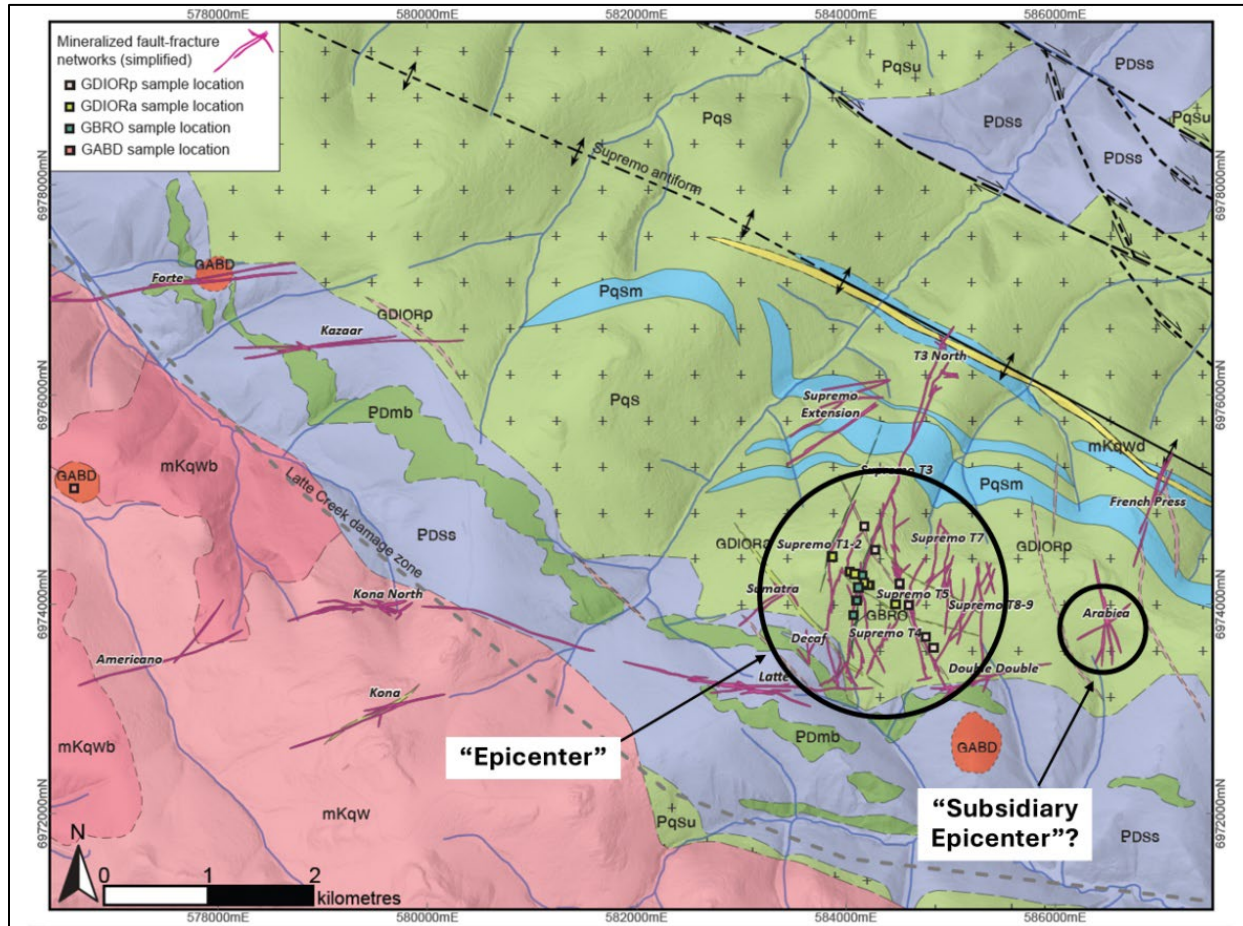
In more recent work, MacWilliam (2018) states that the Coffee deposits are epizonal orogenic gold deposits, with the CCP acting as the main fluid conduit. The Coffee gold mineralization was interpreted to be deposited at a maximum depth of ~5 km following the emplacement of the mid-Cretaceous CCP, with a mineralization age of 97–93 Ma interpreted from $^{40}\text{Ar}/^{39}\text{Ar}$ dating of hydrothermal sericite and fuchsite (MacWilliam, 2018). A maximum age of gold deposition was demonstrated by the CCP crystallization age of 103-99 Ma, which also hosts mineralized fault-fracture networks in Kona and Kona North (ibid.). MacWilliam (2018) also acknowledged that mineralization could be as young as ~57 Ma based on the inferred post-mineral deposition of the Rhyolite Creek volcanics near the Coffee Gold Project; however, 97–93 Ma has become the most cited age range of the deposit. Rapid exhumation rates of 2.0-3.5 km/Myr were inferred based on the depth and timing of mineralization, and an emplacement depth of ~12 km for the CCP (ibid.). MacWilliam (2018) also included mineralization fluid characterization that used alteration and mineralization mineral assemblages to bracket the fluids to ~250°C and a pH of ~5. Lead and sulphur isotope analysis completed by MacWilliam (2018) was inconclusive at determining the source of fluids and metals that formed the Coffee deposits. The lack of vertical and lateral zonation was used as evidence against a magmatic source (ibid.). Instead, the fluids were interpreted to be sourced more distally from the devolatilization of sedimentary rocks at depth (ibid.). However, MacWilliam (2018) also recognized that Coffee's characteristics are somewhat unique compared to typical orogenic deposits, and acknowledged overlap between orogenic, (reduced) intrusion-related, and Carlin-like deposit models, highlighting a need for reassessment of epizonal gold-only deposit models.”



8.3 Basis For Future Exploration And QP Comments

Based on the Micon QP’s observations during the site visit and analysis of the data maps available, the deposit combines an intrusion related gold IRG type and orogenic zonal type. The surficial “epicentre” of the Coffee Gold Project deposits is shown in Figure 8.1.

Figure 8.1
Major Coffee Gold Project Deposits Map and the Surficial Location of a Hypothetical Mega-Breccia Pipe



Source: Modified from the YGS Open File 2025-6

This deposit type can be further explored using downhole geophysical techniques combined with reverse circulation (RC) drilling across the envisaged surficial epicentre of the intrusion/deposit. The surficial epicentre of the Coffee Gold Project deposits could represent a mega-breccia zone with subsidiaries scattered across the map as exemplified by the one to the east of Double-Double, as shown in Figure 8.1.



9.0 EXPLORATION

9.1 Overview

The Micon QP has reviewed the exploration work including geochemical and geophysical surveys, and geological mapping from the Internal Engineering Report (JDS, 2024) prepared for Newmont and considers it relevant and current. The following discussion summarizes content from that report.

Exploration work conducted on the Coffee Gold Project includes: property and deposit-scale mapping, geochemical soil and silt sampling, bulk leach-extractable gold (BLEG) and heavy mineral stream sediment sampling, biochemical sampling, ground and airborne geophysical surveys (VLF-EM, VTEM, ELF-EM, HLEM, Ohm Mapper, GPR, H-V and MASW seismic, magnetics, IP resistivity, gravity, and borehole EM), and airborne radiometric surveys.

Systematic exploration commenced in 2009 after acquisition of the Coffee Gold Project by Kaminak. The exploration works thereafter are listed in Table 9.1.

Table 9.1
Exploration Work Completed by Kaminak, Goldcorp and Newmont

Operator	Year	Drill (m)	Soil Samples	Stream Sediment Samples	Trenching (m)	Trench Samples	Mapping and Sampling (days)	Geophysics	Geomorphology
Kaminak	2009	-	3,876	-	4,164	828	10	261 line-km ground magnetic survey	-
	2010	16,105	8,851	-	4,470	826	10	579 line-km ground magnetic survey	-
	2011	47,990	10,689	-	3,926	799	15	4,842 line-km airborne magnetic and gamma-ray spectrometric; 15.9 line-km HLEM and Ohm mapper surveys	Mapping
	2012	65,548	4,438	-	-	N/A	40	=1.48 line-km, 51 points IP/DC lines	-
	2013	55,477	5,027	-	153	147	2	18 days of IP	-
	2014	52,760	2,955	-	6,252	2,025	30	5,300 line-km airborne magnetic infill survey	-
	2015	41,895	N/A	-	-	-	30	=	-
Goldcorp	2016	27,546	1,072	-	1,277	472	14	=	-
	2017	66,617	2,875	12	99	36	2	221 line-km ground geophysical surveys, 189 line-km airborne geomagnetic surveys	-
	2018	98,583	9,745	-	-	-	-	513 line-km ground VLF- EM surveys, 34.5 km ground magnetic survey	-
Newmont	2019	90,696	3,747	-	-	-	13	387 line-km ground VLF- EM surveys	-
	2020	-	1,623	118	878	300	50	-	Mapping
	2021	8,424	1,552	15	-	-	-	265.5 line-km VLF-EM surveys	Mapping
	2022	10,276	97 (DSG)	-	-	-	23	128 line-km VLF-EM surveys	Mapping
	2023	18,830	-	304	-	-	40	3.3 line-km MT and DC surveys	Mapping
Totals	609,708	54,404	449	21,219	5,433	279	-	-	

Notes: DGS = deep sensing geochemistry; HLEM = horizontal loop electromagnetic; IP = induced polarization; VLF-EM = very low frequency electromagnetics; MT = magnetotelluric; DC = direct current Source: Newmont 2024



9.2 Geochemistry Survey

9.2.1 Soil Sampling and Deep Sensing Geochemistry

Soil geochemistry has been an efficient tool for mapping the distribution of gold and associated elements, as well as providing data input to large scale lithological mapping. However, the presence of permafrost on northerly aspects, downslope creep of unmineralized colluvium, and thick vegetation coverage along valley bottoms has provided challenges to acquiring quality samples in some parts of the Project. Most gold discoveries at the Coffee Gold Project to date have emanated from soil geochemistry targeting, in conjunction with other vectoring tools.

GroundTruth Exploration Ltd. has been the sole contractor for all soil sampling programs since 2009. This has resulted in a high level of consistency with regards to sampling methodology and sample data quality. The approach for soil sampling field programs has been systematic:

- i) initial ridge-and-spur sampling to identify areas of interest
- ii) 200 m x 50 m to 200 m x 100 m north-south trending soil lines targeting identified anomalies
- iii) 100 m x 50 m spaced soil lines for filling in established targets
- iv) further infill sampling as required (down to 25 m x 25 m).

To date, 54,404 soil samples have been collected within the Coffee Gold Project.

Samples were collected using a hand auger to various depths depending on the soil profile. The organic A-horizon material was discarded, and augering continued until the C-horizon rock chips were encountered, checking for false bottoms on the A-horizon profile. Soil samples were collected over intervals varying from 60 to 70 cm, with maximum depth not exceeding the 1.25 m length of the auger. Samples were placed directly in pre-marked bags. A field duplicate sample was collected at a rate of one every 25 samples. Sample number, location, depth, and geological parameters were recorded directly into a handheld computer with a global positioning system (GPS) reading of sample location also stored separately as a backup. The sample location was marked with flagging tape and a metal tag on a nearby tree. In more recent soil sampling programs, tags with sample specific barcodes were left at each station. Pre-2016 soil samples were analyzed at Acme Analytical Laboratories in Vancouver, British Columbia (Acme). Post-2016 soil samples were analyzed at Bureau Veritas Minerals Laboratory also in Vancouver, BC.

In 2022, a Newmont proprietary deep sensing geochemistry (DSG) survey test grid was conducted during the months of June and July in the Kazaar South (KZS) prospect to determine whether this technique can detect anomalies in permafrost-covered areas. KZS was chosen as a test area because the gold-in-soil signature showed little to no anomalous gold, but rock sampling along the road cut and later trenching revealed that the very low frequency electromagnetic (VLF-EM) lineaments in the area were in fact mineralized, leading to the belief that permafrost is masking some of the mineralization normally detected by soils. A total of 96 wet bulk samples were taken over the KZS east-west trending VLF-lineament, then dried and sieved before shipping to Newmont's Malozemoff Technical Facility (MTF) for analysis. Survey spacing was 50 m x 25 m over known unmineralized ground, and 50 m x 10 m over the mineralized structure. The results suggest that the DSG method was marginally successful in penetrating through permafrost cover. Further tests with spacing equal to that of previous soils samples may be required to further test its efficacy before employing the method at a more substantial scale at the Coffee Gold Project.



9.2.2 Biogeochemistry

Geochemistry of C-horizon soils has proven to be an extremely useful exploration technique at the Coffee Gold Project; however, in certain areas soil C-horizons have either not developed or cannot be accessed for sampling due to thick organic cover or permafrost, typically on North facing slopes or near valley bottoms. With the objective of evaluating biogeochemistry as an alternative method for discerning geochemical anomalism, a total of 516 samples of Black Spruce (needles, twigs, bark), *Betula Nana* (Dwarf Birch; leaves and twigs), and Mountain Alder (leaves, twigs), were collected in 2017. The samples were collected in a series of north-south transects across the western portion of Latte (permafrost-rich ground), as well as across the Kazaar prospect (north-facing slope, thick organic cover). The sampling transects coincided with previous soil sampling lines to facilitate comparison between the two methods.

Results from this biogeochemical orientation survey revealed that samples of Black Spruce needles showed the strongest correlation to Au/As-in-soil geochemistry, presenting biogeochemistry as a potential alternative method for gaining targeting data in areas of poor soil-sample suitability. However, the small dataset obtained showed variability in correlation between the two test areas, and larger scale orientation studies were deemed necessary to further evaluate the method. No further biogeochemistry surveys have been conducted since 2017.

9.2.3 Stream Sediments

With the objective to build on previous government stream sediment work, as well as evaluating a helicopter-based stream sediment sampling method, Aurora Geosciences was contracted in 2017 for a planned 100 sample program. Due to sample mechanism malfunctioning, only 12 samples were collected before the program was aborted.

9.2.4 Bulk Leach Extractable Gold

The Micon QP has reviewed and excerpted the following statements in their original form from the Internal Engineering Report (JDS, 2024) prepared for Newmont.

To gain further knowledge of Project-wide catchment gold distribution and to explore potential new geochemical signatures of mineralization at the Coffee Gold Project, 118 bulk leach extractable gold (BLEG) samples were collected from a multitude of streams and creeks during the 2020 field season, achieving coverage of catchments across the Project. This program was completed using Newmont's proprietary BLEG sampling and sample preparation techniques, with final analytical tests completed at ALS Limited based in Australia.

Results from this program revealed a series of distinct geochemical signatures related to gold mineralization across the Coffee Gold Project, both in areas of known mineralization (i.e. main Coffee deposit) as well as in under-explored areas. BLEG sampling was successful in highlighting unique Coffee-style signatures that were used for Project-wide analysis.

9.3 Geological Mapping/Prospecting

9.3.1 Overview

The limited bedrock exposure within the Coffee Project has hampered detailed bedrock mapping efforts. Recurring mapping and prospecting programs from 2010 through 2023 have largely consisted of: i) reconnaissance traverses mapping exposed bedrock or talus material in field or along exploration road cuts to increase the understanding of the district geology; ii) follow-up prospecting in areas newly discovered of gold-in-soil anomalies. Mapping and prospecting have dominantly been conducted by Project geologists, although an external contractor performed a substantial mapping traverse program in 2014. Mapping and



prospecting programs have provided control points for the development of Project-scale lithological maps, as well contributing to exploration targeting. A mapping program conducted during the 2022 and 2023 field seasons led to significant improvements in knowledge of the bedrock geology across the Project. Samples collected for age-dating and/or whole-rock geochemistry throughout this period have helped determine the sequence of events that have occurred throughout the history of the Coffee deposit.

9.3.2 Surface Rock Sampling

Surface sampling during mapping and prospecting programs were undertaken by the Project geologists. The sampling procedure consisted of compositing several (where applicable) representative rock samples from outcrop, colluvial float, or hand-excavated test pits to gather enough material for assay analysis. Sampling site information including handheld GPS coordinates, terrain, and vegetation data were recorded in field notebooks and later recorded into an access database. Sample lithology descriptions were entered into the database, and samples were analyzed by desktop x-ray fluorescence (XRF) for detection of pathfinder elements.

9.3.3 Trenching

Heavy equipment-supported excavation of trenches has occurred throughout the Coffee Gold Project's history, with the initial discovery trenches at Supremo, Latte, Double-Double and Kona completed in 2009 and 2010. Following the success of these early trenching programs, trenching of exploration targets identified by soil-geochemistry and prospecting programs has been conducted as a precursor to execution of drill programs. Trenching has also been utilized on a larger scale for gaining materials for metallurgical testwork for all the main mineralized structures comprising the main Coffee deposit.

9.3.3.1 Trench Sampling

Early exploration trenches (2009-2012) were completed by light Heli-portable Can-Dig machinery which limited the depth of excavations, often not reaching bedrock. For trenching operations after 2013, heavy equipment was used to excavate to a minimum of 0.5 m below the bedrock colluvium interface. Some excavations failed to reach bedrock due to permafrost or excessive colluvium cover. The start and end coordinates of each trench were recorded with a handheld GPS unit, and metre markers, delineating positions along the trench, were laid out with measuring tape for referencing lithological logging and sampling intervals. Lithological logs of trenches were recorded in field notebooks and later transferred to a Microsoft Access database. Composite rock samples were taken at 5 m horizontal intervals, with detailed sampling on 2 m or 1 m intervals within probable zones of mineralization. Representative samples were collected by chipping rock on the wall or base of the trench, and each sample was assigned to prelabelled sample bags and sampling information was recorded in sample booklets for subsequent transfer to the Microsoft Access database. For exploration trenches, samples were analyzed using desktop XRF equipment in the field prior to submittal to ALS Geochemistry for multi-element analysis.

9.4 Geophysical Surveys

9.4.1 Magnetic

The purpose of collecting magnetic data has been to map magnetic field contrasts between lithological units and to discern structural patterns related to gold mineralization across the Coffee Gold Project. The magnetic destructive nature of syn-mineralization alteration has allowed magnetic field data (TMI and Tilt derivatives) to be important tools for delineating mineralized structures. Drill testing of magnetic-low anomalies with coincident soil geochemical signatures played an important role in the initial discovery and follow-up delineation of many of the main resource targets.



9.4.1.1 Ground

Initial ground magnetic datasets were collected by Ryanwood Exploration in 2009 and 2010. A total of 840 line-km of ground magnetic data were collected over the main Supremo, Latte and Double-Double zones. The surveys were completed along north-trending lines spaced 100 m apart and at reading intervals of 0.5 seconds. The bulk of the existing ground magnetics dataset was collected by Aurora Geoscience between 2017 and 2022 in conjunction with VLF-EM surveys. This survey dataset consists of 1,514.5 line-km covering much of the area around Supremo, Supremo Extension, Latte, Double-Double and Arabica over to Kona/Kona North and Kazaar as well as surrounding early-stage targets.

The ground magnetic dataset was reprocessed by Newmont internal geophysicists in 2020. Ground magnetics collected after 2020 have not been reprocessed. Ground magnetics imagery played an important role during the 2022 and 2023 Project geology updates.

During the 2023 Coffee field season Quantec Geoscience performed a Magnetotelluric (MT) and Direct-Current (DC) 3.3 line-km geophysical survey over the main Supremo deposit. Preliminary data includes an elevation map of the survey area and a graphical representation of the MT and DC surveys. Complications with permafrost in the area affected the quality of the data collected. No significant findings were reported from this study.

9.4.1.2 Airborne

The technical objective of airborne magnetic surveys was to provide high-resolution total field magnetic and radiometric data suitable for a structural geology interpretation and identification of lithological trends.

In 2011, New-Sense Geophysics Ltd. of Markham, Ontario was contracted to run a high sensitivity helicopter magnetic and gamma-ray spectrometric airborne survey. The survey consisted of 4,842 line-km and was flown at two resolutions: 200 m spacing over a Project wide block; and 100 m spacing over two separate blocks covering a ~ 20 km x 8 km block (east-west x north-south) centred at Supremo, as well as a ~ 10 km x 10 km block centred over the Sugar prospect (1,304 line-km for the two infill blocks). Fully processed and corrected magnetic and radiometric maps were prepared by New-Sense Geophysics Ltd.

Additional infill of the aeromagnetic dataset was conducted in 2014 (contracted to CGG Canada Services Ltd., no additional spectrometry data was collected), where a 50 m line spacing was achieved over the Supremo and Sugar areas. Project wide infill to 100 m line spacing was also executed, for a total of 5,300 line-km infill surveys.

The airborne magnetic and radiometric datasets were reprocessed by Mira Geoscience Ltd. in 2014, and later by Newmont geophysicists in 2020. Both datasets were used heavily during the 2022-2023 Project geology map update.

9.4.2 Radiometrics

Airborne gamma-ray spectrometry data was collected during the 2011 New-Sense Geophysics aeromagnetic survey (see 2011 airborne magnetic survey above for details on coverage). The original dataset was reprocessed in 2019 and 2020 by Newmont internal geophysicists. The radiometric dataset has primarily been used for Project scale lithology mapping (K-Th-U ternary plots), in conjunction with other datasets.

9.4.3 Light Detection and Ranging

An airborne Light Detection and Range (LiDAR) survey was flown in 2011 by Eagle Mapping Services Ltd, Vancouver B.C. The survey was flown within a semi-rectangular ~ 20 km east-west x 8 km north-south



block, roughly centred at the main Supremo area. The resulting high resolution LiDAR dataset has primarily been utilized for slope and lineament analysis.

In 2021 a LiDAR survey was conducted over the entire Coffee Gold Project and over the proposed NAR. Data was collected over two flight missions on August 2nd and August 31st for the Project area and from July 27th to August 29th for the NAR. Flight lines were flown in an east-west direction. Final products included 1 m cell bare-earth grid points, georeferenced raster hill shade images with a 1 m pixel size, and ortho-image mosaics with a 20 cm pixel size. Following delivery of these products, lineament analysis was conducted and an initial update to the Coffee Road network was completed using the imagery.

No further LiDAR surveys have been conducted since 2021.

9.4.4 Aerial Imagery

9.4.4.1 Orthophotograph

A colour aerial photograph survey was flown in August of 2011 by Geographic Air Surveys of Edmonton, Alberta. The survey was completed at a scale of 1:20,000 covering the entirety of the property and the surrounding area. The aerial photography was triangulated using a combination of airborne GPS, surveyed ground control points and LiDAR data. An expanded digital elevation model (DEM) was generated for the Project area using the triangulated data and existing LiDAR dataset. A mosaic orthophoto was created from the merging of digitized imagery and generated DEM. The aerial imagery dataset has been used in a variety of exploration related applications, including soil sample suitability and geomorphology analysis.

In 2021 an orthophoto survey was conducted over the entire Project area and over the NAR in conjunction with the LiDAR surveys (mentioned above). Georeferenced, colour, digital orthomosaics with 20 cm pixel size were delivered in compressed enhanced compressed wavelet (ECW) and GeoTIFF formats. The mosaics were divided into tiles using the same tile structure as the LiDAR tiles and trimmed to the Project boundary. The compressed ECW tiles were created using a 5:1 compression ratio.

9.4.4.2 Drone Imagery

During the 2022 and 2023 field seasons, a DJI eight rotor drone was used to capture aerial photos of prospect areas as well as the Coffee camp area. Flight paths were planned using the drone's internal software which created grid patterns for the drone to fly. The flight paths allow the drone to take photos of the complete area from multiple angles. Using Pix4D photogrammetry software and known ground control points, orthomosaic images and DEMs were created.

This process was used during the 2022 and 2023 field season to accurately track ground disturbance and plan drillholes throughout the Supremo Extension prospect.

9.4.5 Versatile Time Domain Electromagnetics

A helicopter versatile time domain electromagnetics (VTEM) survey was flown in 2017 by Geotech Ltd. as a free method trial, covering a ~ 5 km x 0.85 km block trending northeast from Latte to Cappuccino (100 m spaced survey lines). While the Latte structure correlated well with signals in the resulting VTEM magnetic, resistivity, and chargeability datasets, no comparable correlation was present in the survey data along the surveyed Supremo structures. This ambiguity may be attributed to the narrower brittle structural domains in Supremo versus the wider structural domains at Latte.



9.4.6 Very Low Frequency Electromagnetics

The purpose of collecting VLF-EM data has been to map shallow conductive structures across the Project. The VLF-EM method has proven a vital method for exploration target vectoring by identifying brittle/clay-rich/water-bearing structures. The application of the VLF-EM dataset to exploration program design has strongly contributed to the discoveries of high value prospects, such as Supremo Extension, Kazaar, French Press, and extensions and refinement of existing targets such as Decaf and Espresso.

With the addition of an airborne magnetic dataset in 2011 and 2014, no further ground magnetic surveys were conducted until 2017. As part of the VLF-EM surveys conducted between 2017-2022, ground magnetic data was collected simultaneously as VLF-EM in all areas covered by these survey programs.

9.4.6.1 *Ground*

While initial ground-based VLF-EM orientation survey was conducted over a limited area of Supremo in 2010, no further surveys were conducted until 2017 when Aurora Geosciences Ltd. of Whitehorse was contracted to conduct a multi-method geophysical orientation survey program. The initial VLF-EM orientation survey covered an east-west ~ 6.5 km x 0.4 km area over Supremo. Results from the orientation grid revealed spatial correlation between conductive lineaments in the VLF-EM dataset and established structural corridors of drill-delineated mineralization. Given the success of the orientation grid, the ground VLF-EM dataset coverage was expanded from 2017-2022. A total of 1,514.5 line-km of VLF-EM have been completed. VLF-EM surveys utilize electromagnetic signals from different naval bases. Surveys at Coffee were completed using signals from Maine, Hawaii, and Washington.

The ground surveys were completed by contractor field personnel walking predefined survey lines while taking total magnetic field readings at a frequency of one reading per second, and VLF-EM readings every 10 m (nominally). Additional ground magnetic data were collected across the survey lines to assist with data levelling.

The final processed dataset delivered from Aurora Geosciences Ltd. (also later re-processed by Newmont geophysicists) was split into three data stitched mosaic layers depicting each transmitter station separately.

9.4.6.2 *Airborne*

Following the success of the ground-based VLF-EM program in 2017, Geophysics GPR International was contracted the same year to conduct a helicopter-based VLF-EM orientation survey. The objective of this program was to produce similar results to the ground VLF-EM survey in a more time-efficient manner. Two test blocks, overlapping the existing ground VLF-EM dataset, were flown at 30 m above ground level. Preliminary results were ambiguous as the airborne dataset did not replicate the conductive trend depicted by the ground VLF-EM in either of the two test blocks. The ambiguity in the airborne dataset was attributed to the 30 m instrumentation flying height, resulting in coarser resolution. Given the limitation in reducing the survey height due to varying terrain and vegetation conditions in Coffee survey areas, no further airborne VLF-EM were conducted.

9.4.7 Extremely Low Frequency Electromagnetics

Aurora Geoscience Ltd. was contracted to perform an extremely low frequency electromagnetics (ELF-EM) orientation survey in 2017, with the objective of mapping conductive structures at depth to explore potential down-dip extensions of known mineralized structures on the property. The survey consisted of a 143 survey point at Latte and Supremo areas, as well as 18 points collected from Kona area for a total of 4.8 line-km at 100 m station spacing. No strong responses or conductive features were present in the processed dataset. The lack of response may be attributed to the resolution of the ELF survey being insufficient to register the relatively narrow structures of known mineralization. No further ELF-EM was conducted.



9.4.8 Gravity

9.4.8.1 Ground

Aurora Geoscience Ltd. was contracted to perform a ground gravity orientation survey to aid in the direct detection of mineralized bedrock. The 2017 survey consisted of two separate northeast-southwest and northwest-southeast survey lines for a total of 17.4 line-km over the Kona and Supremo areas. Due to the limited surveyed area (isolated lines rather than surveyed blocks), the program was unsuccessful in locating Coffee-style mineralization.

9.4.8.2 Airborne

CGG Canada was contracted to conduct a helicopter-supported gravity and magnetic survey of the Coffee Gold Project in 2020, with the objective of obtaining a Project-wide gravimetric dataset to aid in exploration targeting and bedrock mapping. The survey was completed at 200 m line spacing along east-west oriented lines for a total of ~ 4,000 line-km surveyed. Flight lines to tie lines were conducted at a ratio of 10:1. Findings from the processed dataset confirmed updated Project-wide lithological interpretation, as well as first order structural geometries. The resulting dataset was not successful in delineating known mineralization. This was due to topography leveling effects and the inability of the survey to delineate small-scale structures and lithological changes.

9.4.9 Horizontal-Loop Electromagnetic/Ohm Mapper

With the objective to explore subsurface electromagnetic conductivity of mineralized structures as a tool for target vectoring, an orientation study of ground HLEM and Ohm mapper surveys were conducted over sections of Supremo, Latte and Kona areas in 2011. A total of 15.9 line-km of HLEM survey lines were collected, using a 100 m coil separation and readings at 25 m separation. Additionally, 4.9 line-km of Ohm mapper survey lines were collected at various rope lengths and dipole lengths to determine best suited survey parameters, with 60 m rope length and 20 m dipoles as the chosen survey configuration. Results from the EM work was generally ambiguous; however, a weak electromagnetic anomaly was present at Kona and corresponds to the known gold mineralization. No further HLEM surveys have been conducted.

9.4.10 Induced Polarization and Direct-Current resistivity

The purpose of this survey was to test the high resolution DC resistivity and IP survey method over known mineralization zones to determine the accuracy of the method in varying surface materials and conditions. GroundTruth Exploration Ltd. was contracted to perform IP and DC resistivity surveys in 2012 and 2013. Four IP/DC lines were completed across the Supremo and Latte structures in 2012 (1.48 line-km, 51 points), and 18 additional survey lines were completed in 2013 (5.2 line-km, 173 points). The 2013 survey lines targeted both known mineralization (Supremo, Latte, Double-Double), as well as prospective targets (Latte West, Cappuccino). Survey lines were orientated perpendicular to known/Projected mineralized trends at 5 m electrode spacing, achieving an optimal 2.5 m horizontal resolution and a maximum reading depth of 90 m. Each line was surveyed using both Dipole-Dipole and Inverse Schlumberger arrays to maximize qualitative data and complete the most robust and fault tolerant dataset possible. Resistivity and induced polarization measurements were taken at every reading to give complementing data.

Ground conditions (talus, permafrost/ice) presented a challenge by prohibiting proper electrode ground connection and limiting the depth of the survey. Results from the processed IP-Resistivity dataset delivered by GroundTruth Exploration were overall ambiguous, with variable degrees of positive correlation between interpreted sub-vertical structures from survey data and drill-defined gold-bearing structures.



9.4.11 Ground Penetrating Radar

A ground penetrating radar (GPR) survey was completed by Aurora Geosciences Ltd. of Whitehorse during the 2017 field season. A total of 4.8 line-km was surveyed across the proposed waste rock facility and parts of the proposed Supremo pit, with the objective to explore the ability of the GPR method for mapping overburden thickness for geotechnical purposes. Results from this survey did not warrant additional GPR programs.

9.4.12 Seismic Surveys

Aurora Geoscience Ltd. was contracted to perform Horizontal to Vertical (H-V) and Multichannel Analysis of Surface Waves (MASW) surveys to evaluate the effectiveness of these techniques to determine the nature of overburden for geotechnical/exploration purposes and to correct gravity data in areas of low-density cover. The survey consisted of 41 H-V and 3 MASW sites in the Supremo and Latte areas coincident with detailed gravity surveyed stations. Results of this program did not warrant any follow-up.

9.4.13 Self-Potential

Aurora Geoscience Ltd. was contracted to perform a Spontaneous /Self Potential (SP) orientation survey with the objective to evaluate capabilities of the SP method for target vectoring. The survey consisted of measurements collected along two separate NE-SW and NW-SE survey lines for a total of 6.6 line-km across and along drill-delineated mineralized structures at Latte and Supremo. Results were ambiguous; no clear correlation between depicted self-potential anomalies in the SP dataset and known mineralized structures could be made.

9.4.14 Borehole

9.4.14.1 BHEM-Volterra Surveys

Aurora Geoscience Ltd. was contracted to complete electromagnetic (BHEM-Volterra) surveys of two boreholes located at Latte; CFD0694 (870 m down-hole surveyed) and CFD0696 (636 m down-hole surveyed). Survey points were recorded every 6 m at 45 s reading time in CFD0696 and were adjusted to 120 s and variable stations intervals between 9 m and 12 m for CFD0694. Results from these down-hole EM surveys revealed weak responses indicating that this methodology does not yield direct detection for nearby off-hole conductors within the mineral system.

9.4.14.2 Downhole Imaging

With the objective to explore downhole imaging of RC drillholes as a cost saving supplement to diamond drilling, as well as exploring the method's capabilities of recording structural measurements, DGI Geoscience Inc. (DGI) was contracted to conduct a series of Optical Televiewer (OTV) surveys in both diamond and RC drillholes in 2018. Additional surveys of 22 RC holes were completed during the 2019 Supremo infill drill program and 2021 exploration drilling at Supremo Extension. Both dry and wet hole conditions were tested for RC drillholes, although the majority of surveys were done after RC holes were flushed with water to improve image quality. All surveys consisted of continuous downhole high-resolution image profiles and magnetic directional surveys, from which a digital image of the length of hole was stitched into an oriented core image. From the resulting image, DGI personnel extracted structural orientations for identified feature such as lithological contacts, veins, foliation, and fractures.

Results from the OTV surveys were deemed to moderately aid in lithological logging of RC drillholes; however, it was determined that it may be of more use for internal Project personnel to conduct the structural pictures from the resulting imagery, as more intimate knowledge of the various structure types is important to integrate the OTV structural data with the greater diamond drillhole structural dataset.



Additional OTV and ATV (acoustic televiewer) surveys were conducted by DGI in the 2023 field season alongside SRK consulting during the drilling of geotechnical diamond drillholes. These surveys were moderately successful, as only segments of the surveys were completed on the geotechnical holes due to a combination of hole stability and a lack of the required equipment.



10.0 DRILLING

10.1 Overview

The drilling completed on the Coffee Gold Project as of the effective date of this Technical Report is summarized in Table 10.1.



**Table 10.1
Summary of Drilling Completed at the Coffee Gold Project**

Operator	Year	Core		Core-GT		RAB		RC		RC-Spacing		Sonic		Total	
		Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)
Kaminak	2010	76	16,105											76	16,105
	2011	111	29,963					145	19,518					256	49,481
	2012	124	29,609					223	39,451					347	69,060
	2013	62	12,273					240	43,204					302	55,477
	2014	145	26,794	2	100			206	25,867					353	52,760
	2015	103	15,840			35	2,198	197	23,702			35	156	370	41,895
Goldcorp	2016	26	6,703	35	370			261	20,473					322	27,546
	2017	74	17,697	43	723			329	48,197	219	5,535			665	72,152
	2018	195	36,691	7	1,148			590	60,655	90	5,823			882	104,317
Newmont	2019	194	30,203					550	60,403					744	90,606
	2021	1	161					49	8,263					50	8,424
	2022	16	3,456					39	6,820					55	10,276
	2023	43	9,780	4	669			169	8,381					216	18,830
	Total	1,170	235,274	91	3,010	35	2,198	2,998	364,933	309	11,358	35	156	4,638	616,930

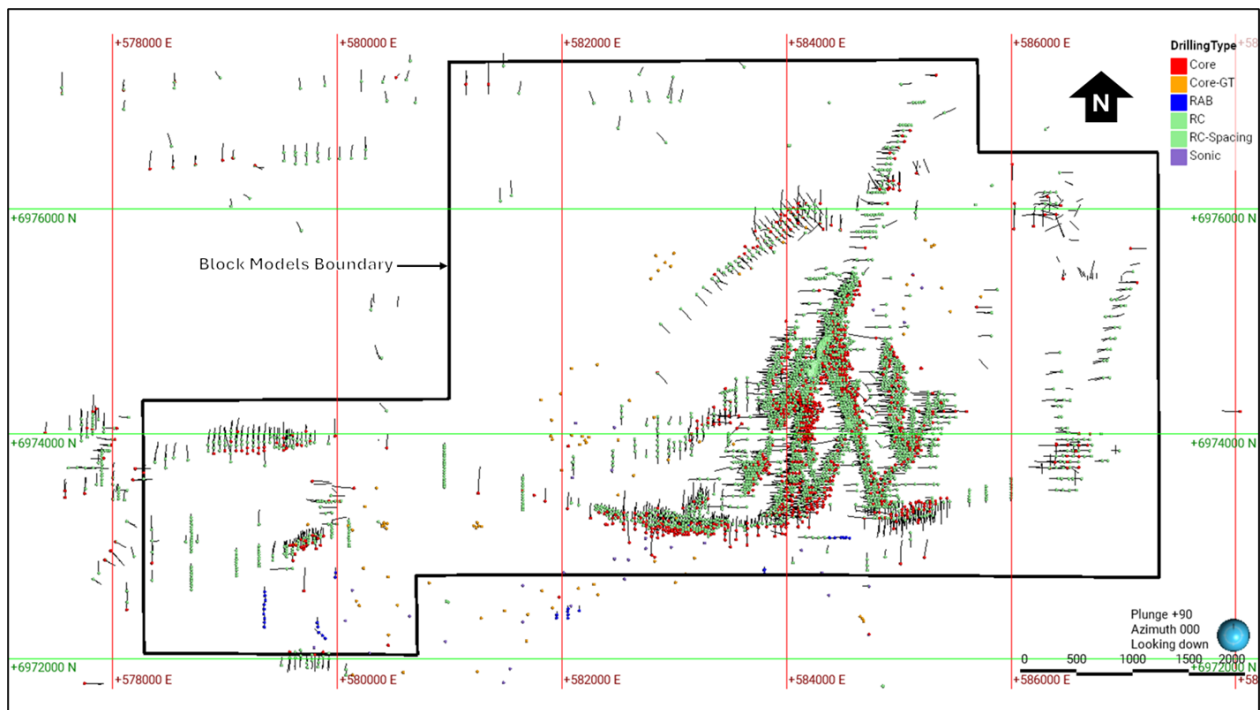
Notes: GT = geotechnical holes; RAB = rotary air blast. Figures may not sum due to rounding.
Source: Newmont 2024.



To date a total of 4,638 holes, totalling 616,930 m have been drilled on the Project. No drilling was conducted in 2020 due to the Covid-19 pandemic, otherwise drilling activities have been conducted consecutively since 2010. The layout of the drillholes completed is shown in Figure 10.1. Drill plans per campaign and representative examples of drill sections through the deposits are provided in subsection 10.6.

Initial diamond core drilling was contracted to Kluane Drilling Ltd. (Kluane Drilling) of Whitehorse, Yukon. In 2012, CYR Drilling International of Winnipeg, Manitoba (CYR Drilling) replaced Kluane Drilling as the sole diamond drilling contractor. CYR Drilling remained at the Project until 2021, after which Boart Longyear took over in 2022. RC drilling has been utilized at the Project since 2011 with Northspan Exploration Ltd. as the sole contractor.

Figure 10.1
Coffee Gold Project Drillhole Layout Plan



Source: Micon 2025 (Generated from the resource database)

10.2 Drilling Methods

The main drilling methods conducted within the Project area include diamond core (DD) and reverse circulation (RC) rotary air blast (RAB) and sonic. Diamond core is primarily composed of NQ2 (50.7 mm) and NQ (starting in 2022; 47.6 mm) diameter core. Some metallurgical (not included in Table 10.1) and geotechnical drill programs used HQ2 and HQ3 or PQ (with split tubes), respectively. RC drill rods were predominantly 3.5" diameter; however, in 2019, some 6" diameter rods were utilized to help maintain production and drill deeper than the standard 3.5" rods would allow. In 2015, a suite of RAB and sonic drillholes were completed. The RAB was used for condemnation testing, and the sonic drill was used for geotechnical testing for permafrost.



10.3 Drilling Procedures

10.3.1 Drillhole Positioning

Drillholes are planned to target specific geochemical and/or structural trends with fences of one or more holes drilled roughly perpendicular to the strike of the interpreted mineralized structure. Drillholes are collared on variably spaced sections typically ranging from 25 m to 50 m, to average maximum depths of 200 m below surface with some holes extending past 350 m. Drilling is planned to intercept targets at angles where sampling can represent the true thickness of mineralization; however, the structurally complex nature of the deposit does not always allow this.

Typically, the target ratio of RC to DDH of 3:1 was maintained during infill drilling campaigns. This allows for rapid drill production while maintaining structural and lithological understanding.

10.3.1.1 Diamond Drilling

For DD, when drilling is underway cores are taken from the core tube and placed into wooden core boxes. No core logging or sampling is completed at the drill rigs.

Oriented core has been used over the course of the Project. Orientation marks (ORI Marks) are drawn on the down-hole side of a 3 m run with a red grease pencil by the contracted drilling company. The mark is produced from the centre of the core (perpendicular to the core axis) to the edge, signifying the bottom side of the core. ORI marks are used to collect subsurface structural data and are important in the development of structural models and Au grade shell.

10.3.1.2 RC Drilling

The RC drilling process breaks rock using an air compressed hammer which produces a sample of fine rock cuttings as opposed to drill core. Compressed air drives a pneumatic hammer attached to a rotating face sampling bit with tungsten carbide nodes. Chips and rock dust generated by the hammer are forced through openings in the face of the bit and up into the sample return tube inside the rod string. The 5-foot rods are attached to an air and sample hose that continues into a cyclone module. The sample is separated from the air in the cyclone and drops out the bottom into a 5-gallon bucket. Each sample comprises one 1.52 m run, with the drillhole and rods being blown out (cleaned) between each run. Once the complete sample has been collected in the five-gallon bucket the total weight of the sample is recorded in kilograms. The weight of the empty bucket is recorded at the start of each drillhole on the first sample tag. The sample is then reduced through a 1:8 riffle splitter into a sample typically weighing 2 kg that is retained for analysis. The weight of the split sample is also recorded. A sub-sample taken from the reject material is wet sieved and put into chip trays for logging by onsite Project geologist.

10.4 Drillhole Collar and Downhole Surveys

The Micon QP has reviewed and excerpted the following statements in their original form from (and occasionally with minor edits in some cases) from the JDS inhouse PFS report prepared for Newmont.

10.4.1 Drill Collar Surveys

Drillhole collars are surveyed three times for skid drill pads and twice for fly drill pads using a Leica GS16 Real Time Kinematic (RTK) GPS setup. All coordinates are recorded in NAD83 UTM Zone 7N. The base station for the Leica GS16 RTK GPS is set up over known control points laid out by Challenger Geomatics (third party surveying contractor) or by trained Newmont geologists/drill supervisors. The RTK rover unit communicates to the base station via a radio link to deliver highly accurate satellite location data.



Planned collar locations are initially laid out using the RTK GPS or a handheld Garmin GPS unit. Once the drill pad has been constructed the coordinates are checked using the RTK GPS to ensure the pad has been adequately constructed. This step is not conducted on helicopter supported drill pads as once the pad is built, there is no feasible way to adjust the collar location. Once drilling has been completed the drilled collar location is measured (on both the fly and skid pads). Only the final collar coordinates and planned coordinates are recorded in the Project acQuire database.

The accuracy of the tool is verified by regular 'check' shots taken from other known control points laid out by Challenger Geomatics. At each collar location two separate 15 second readings (averaged measurements) are taken and stored in the GPS control unit then averaged to give a reliable final collar location. Measurements and staking are not performed if the 2D and 1D accuracy of the device is greater than 0.09 m.

10.4.2 Drill Collar Alignment Surveys

Prior to 2017, drill collar alignment was performed using a Suunto or Brunton handheld compass. In 2017, a north seeking Minnovare AziAligner gyrocompass tool was used for the Latte infill program. In 2018, the Devico DeviAligner north seeking gyrocompass tool was introduced as part of the regular drill alignment procedure. Current survey procedures ensure that each drill line-up uses a the DeviAligner gyrocompass for maximum accuracy. Line-ups are performed to get the drill rig as close to the planned dip and azimuth; dip/azimuth/roll must be within 0.5° of the proposed drilling orientation. A calibration check station with a known orientation was constructed in 2020 to be used for QA/QC checks in subsequent drill programs. Data from the alignment survey is captured digitally and entered directly into the Coffee Gold Project database.

10.4.3 Downhole Diamond Drill Surveys

Downhole surveys for diamond drillholes were originally conducted using the Reflex EzShot® gyro tool but the switch was made to Reflex Gyro Sprint-IQ tool in 2022. This unit is capable of both North Seeking gyroscopic measurements and reference measurements. North Seeking measurements are recorded from surface at 30 m intervals to the bottom of the hole using a wireline (IN survey). The tool is then retracted by the wireline, and the same process is repeated in the up-hole direction (OUT survey). Digital copies of the IN, OUT and averaged IN-OUT surveys are collected by the geologist at the drill site and transferred to the Project Geologist. The Project Geologist assesses the survey data and enters it into the Project acQuire database, where it is subsequently employed in 3D modelling practices. Additionally, single shot surveys are taken every 30 m downhole as drilling is in progress. This acts as a safeguard in case the hole collapses and a final continuous survey is no longer possible. To ensure that downhole survey data is accurate and precise, a minimum of 5% of drillholes receive a secondary survey via the Ez-trac or Optical Televiwer survey (which also acts as a gyro).

10.4.4 Downhole RC Drill Surveys

Downhole surveys for RC drillholes are conducted using the Inertial Sensing Slim Gyro reference gyro tool. The DeviAligner tool is secured to the drill string upon completion of the hole, recording both azimuth and inclination data which are utilized as reference information during down-hole surveying with the Slim Gyro tool. The Slim Gyro tool is affixed to a wireline set up with a footage counter to measure depth stations from surface as the tool is lowered downhole. Survey stations are recorded for a minimum of 10 seconds at 25 feet intervals from surface. Once target depth is reached, a new unique survey is initiated, and the survey procedure described above is repeated from target depth to surface. Once complete, the survey data is downloaded from the tool to the tablet and exported to IS Analysis software, which allows for visual and quantitative comparison of the survey data to determine its integrity. If the IN and OUT surveys do not match within a $<5^\circ$ azimuth at most stations or a significant dog leg is detected in the drillhole, the survey is repeated until consistent results are achieved. Inclination readings are also compared for similarity to



ensure a similar level of consistency. Once verified, the survey data is exported to USB storage and then imported into the Project acquire database. A calibration check station with a known orientation was constructed in 2020 to be used for QA/QC inspections of both the Slim Gyro and DeviAligner survey tools. These inspections are done regularly and collected quantitatively in a database.

10.4.5 Dog Leg Severity QA/QC

In 2020, an audit of downhole surveys collected between 2010-2019 was completed using the Newmont dog leg severity (DLS) excel tool. The audit was divided into three scenarios to reflect the major drill types: NQ-diamond, 3.5" RC, and 6" RC. Parameters were adjusted within the tool to reflect the variability between these drill methods, allowing higher DLS thresholds for RC drilling given the higher downhole deviation inherent for this drilling method due to the shorter rod length and more flexible rod joints. Diamond drillhole surveys were audited against original paper records of downhole EzShot measurements where data irregularities (such as survey depths) were discovered or where high DLS values were calculated. Downhole deviation within surveyed RC drillholes was also further separated into drift and lift to identify the largest contributor to downhole DLS, and IN and OUT surveys were compared when available. If RC survey data were inconsistent compared to original survey data records, the original survey files were reprocessed using downhole tool manufacturer supplied software (Inertial Sensing Analysis for SlimGyro, Icefield Tools Process Gyro for Gyroshot). If the inconsistencies could not be rectified, the erroneous survey stations and collars were flagged in the database. Survey stations that exceeded established DLS thresholds were also flagged for both RC and diamond drillholes. Beyond the 2020 downhole survey audit all surveys collected during subsequent drilling campaigns were also analyzed for DLS following the same methodology as applied in the 2020 audit. The same approach will be used in the subsequent programs.

10.5 Drillhole Data Collection

The Micon QP has reviewed and excerpted the following statements in their original form from (and occasionally with minor edits in some cases) from the JDS inhouse PFS report prepared for Newmont.

10.5.1 Drillhole Data Collection

10.5.1.1 Drill Core Logging

Once the drill core arrives at the core logging facility the geologist and geo-technician inspect every core box to ensure that 3 m interval blocks and box numbering are in the correct order. First the core is oriented based on the core orientation mark at the end of each 3 m run. The geo-technician then measures each run of core and records the total recovery, rock quality designation (RQD), fracture frequency, number of fractures along foliation, joint roughness, orientation mark quality, and run-on-run consistency (RORC). If the orientation mark quality is poor or not useable (i.e., angular uncertainty of $>10^\circ$) the geo-technician informs the core logging geologist who discusses the poor-quality marks with the drill foreman. Using the recovered core length, metre marks are put on the core at 1 m intervals and meterage for each core box are labelled. XRF analysis is performed at 1 m increments across the entire drillhole. When the XRF measures >200 ppm As, the core is then analyzed at every 50 cm interval up hole and downhole until it measures <200 ppm As. All XRF and geotechnical data is entered into the Project database.

Once the geotechnical logging is completed the core logging geologist begins the geological logging process. The core logging geologist breaks out intervals and records data for changes in lithology, mineralogy, alteration, oxidation, mineralized zones, and structures. The structural measurements are taken using a kenometer; however, goniometers and wrap-around protractors have also been used. Once the geological information has been recorded the geologist identifies specific gravity (SG) samples and core samples. Sampling is typically completed on 1 m intervals with a minimum acceptable sample of 0.5 m. Starting in 2018, sample breaks were placed at all lithological and mineralization contacts, prior to this most sampling was completed on 1 m intervals. Specific gravity sampling is completed at the logging



geologist's discretion but aims to include/capture representative data for all observed lithologies and mineralization styles (see Section 10.3.1.1 and 11.4.3). The intervals, sampling and geological data are recorded into the database. Sample intervals are recorded within the sample tag books as a back-up.

Once geological logging has been completed the geo-technician measures the SG of the samples chosen by the logging geologist and then writes out aluminum box tags. The core boxes are then photographed dry and wet and then off-loaded back onto pallets. Once the core has been palletized, the box tags are stapled onto the box ends, and the core is moved to the core cutting facility.

10.5.1.2 Drill Core Losses

Drill core losses are assessed and recorded during the logging stage. Records indicate that drill core recovery in all the campaigns averaged above 90%. Thus, sample quality would not have been negatively impacted by core losses.

10.5.1.3 RC Chip Logging

The RC chip trays are logged by the logging geologist and intervals are determined by changes in lithology, mineralogy, alteration, mineralized zones, and oxidation. The corresponding data is entered into the acQuire database. Once logging is completed the chip trays are brought to the geo-technician who photographs the chips and prepares them for storage.

10.5.2 Sampling Procedures

10.5.2.1 Drill Core Sampling

Once geotechnical and geologic logging have been completed sample intervals are marked and tagged by the logging geologist. Core is then transported to the core cutting facility where it is cut and sampled as per the logging geologists' instructions and procedures. Standards or blanks are alternated every tenth sample. The standard or blank tags are placed beside the preceding sample tag in the core box. Standard type is randomly generated in the database upon entry and are inserted into the sample sequence once cutting is complete. Duplicate samples are required at least once every 50 samples with priority going to mineralized intervals. Duplicate samples are cut in sequence along with the regular samples. Sample interval data, standards, blanks, and duplicates are all entered into the database in conjunction with the drillhole log. Samples are then assembled into shipments and flown off site to Whitehorse. Sample shipment information including hole ID, first and last sample of the drillhole, flagging tape colour used to tie sample bags, number of bags in the shipment, the purchase order (PO) number, the number of shipments if the drillhole included multiple PO's, the standards used in the shipment, the weight of each sample bag in the shipment, and the security tag numbers, are all recorded in to a shipment notebook and within the acQuire database. A shipment dispatch form is then filled out in and is included with the shipment. The dispatch form includes hole ID, PO number, security tag numbers, bag weights, and sample numbers. Once all data has been verified by the Project geologist the sample shipment is cleared for dispatch from site. Samples are picked up in Whitehorse by either Fuerte personnel, ALS laboratory personnel, or Smalls Expediting Services and are transported to the appropriate laboratory for preparation and assay analysis. Chain of custody documents and additional sampling information are discussed further in.

The sampling process steps are: geotechnical and geological logging; sample interval selection, marking, and tagging; core cutting, including duplicates, and bagging; insertion of blanks and standards; shipment preparation, weight recording, and security tagging; and shipping to external laboratories.

Sampling is typically done at 1-m intervals, with forced breaks at lithological/mineralized contacts. Minimum sample interval is 0.5 m, maximum is 1 m. Entire holes may be sampled (exploration) or sections (infill), with selective sampling extending 10 m up and down from mineralized zones. Poor recovery (<80% or <2.4



m in mineralized zone) triggers an automatic re-drill or flagging for future re-drill. Composite samples may be made for poor recovery, if necessary, within 0.5 m to 3 m limits, and flagged in the database. Intervals are marked with yellow grease pencil and recorded to two decimal places. Hard copies of sampling information are archived, and digital copies stored in the acQuire database.

Specific gravity (SG) measurements are taken using the water immersion method before core cutting. From 2010-2011, measurements were nominally every 10 m in non-mineralized rock and 5 m in structural/mineralized zones. From 2012-2023, one sample per mineralized zone and per major lithology in non-mineralized rock, or every ~50 m. For closely spaced mineralized zones, SG was measured only for those zones. Dry and wax-coated weights, then immersed weights, were recorded. A standard was measured every ten samples to measure instrumental drift.

10.5.2.2 RC Sampling

RC samples are collected at the drill rig by a qualified subcontractor in conjunction with rock chips for logging. Samples are collected every ~1.52 m for the entire hole, with the exception of the first sample of each hole where intervals may be longer depending on recovery while casing is set. Large poly mineralization retention bags (36" x 24") are marked with the depth of the interval and small poly mineralization sample bags (12" x 20") are marked with a sample ID corresponding to a sample tag book where depth interval and hole information are recorded.

The sample is collected in a clean bucket from under the cyclone. The full sample and bucket are weighed and then the sample is split using a riffle splitter (1/8 split) into a retention bag and a sample bag. The retention and sample bags are weighed on a scale, and the total weight, retention bag weight, and sample bag weight are recorded in the corresponding sample tag book. The weighing of samples was implemented in 2022. Historically, the retention bags were kept on site in case any errors occurred with sampling. Since 2020, the retention bags have either been flown back to the drill lay-down for burial there.

A representative spear sample is taken from the waste bucket and is dry and wet sieved on site. Chips from the sieved sample are placed in a chip tray marked with the corresponding depth interval. Retention bags are folded over and are stacked neatly at the drill site, sample bags are closed and arranged into sample shipments at the drill site. A geologist regularly checks in on each drill to ensure sample integrity, quality, and that the site is being maintained in a clean and orderly manner to prevent sampling errors and contamination. When groundwater is encountered, the drillers will attempt to 'dry' the interval before it reaches the cyclone. If a sample is wet, the sample will be spear sampled rather than split, and 'wet sample' is noted in the tag book. A geologist will supervise sampling during wet drilling conditions to ensure the sample is representative.

The cyclone is checked regularly for contamination and cleaned as necessary, regardless of drilling conditions. The collected samples are brought to the main core processing area at the end of every shift where they are laid out and the sample count and sequence numbers are checked to verify against the completed tag books. Standards and blanks are inserted every 10 samples and the standard type is generated randomly in the acQuire database upon entry. Standards and blanks are inserted by the geotechnicians during shipment preparation. Standards and blanks procedures for RC sampling are the same as diamond drill core sampling.

10.6 Significant Drill Results/Interpretation

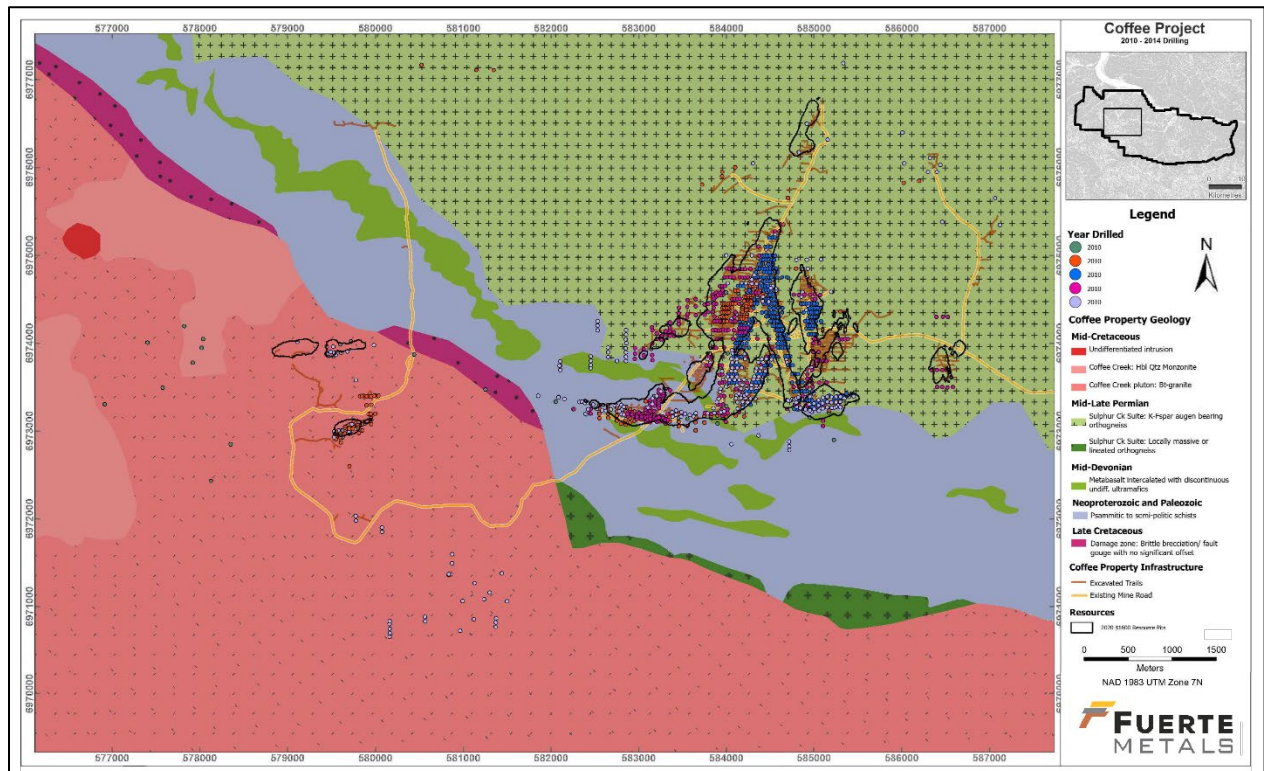
In addition to the cross sections shown in Figure 10.5, Figure 10.6 and Figure 10.8, interpretation of geology and mineralization from drill results can also be seen in Figure 14.25 to Figure 14.30.



10.6.1 Drilling Campaign 2010–2014 (Kaminak)

Initial drilling efforts targeted gold-in-soil anomalies identified by soil sampling programs and trends delineated in shallow trenches excavated with a CanDig excavator. This culminated in discoveries at Supremo, Latte, Double-Double, Kona, Kona North, Americano, and Espresso. Most significantly, the first drillhole CFD0001 directed at what is now known as the Supremo deposit, intersected 17.1 g/t gold over 15.5 m oblique intersection width. Subsequent step-out and infill drilling was conducted for the delineation of a mineral resource. The drilling completed is summarized in Figure 10.2.

Figure 10.2
Coffee Gold Project Initial Discovery and Mineral Resource Estimation Delineation Drilling



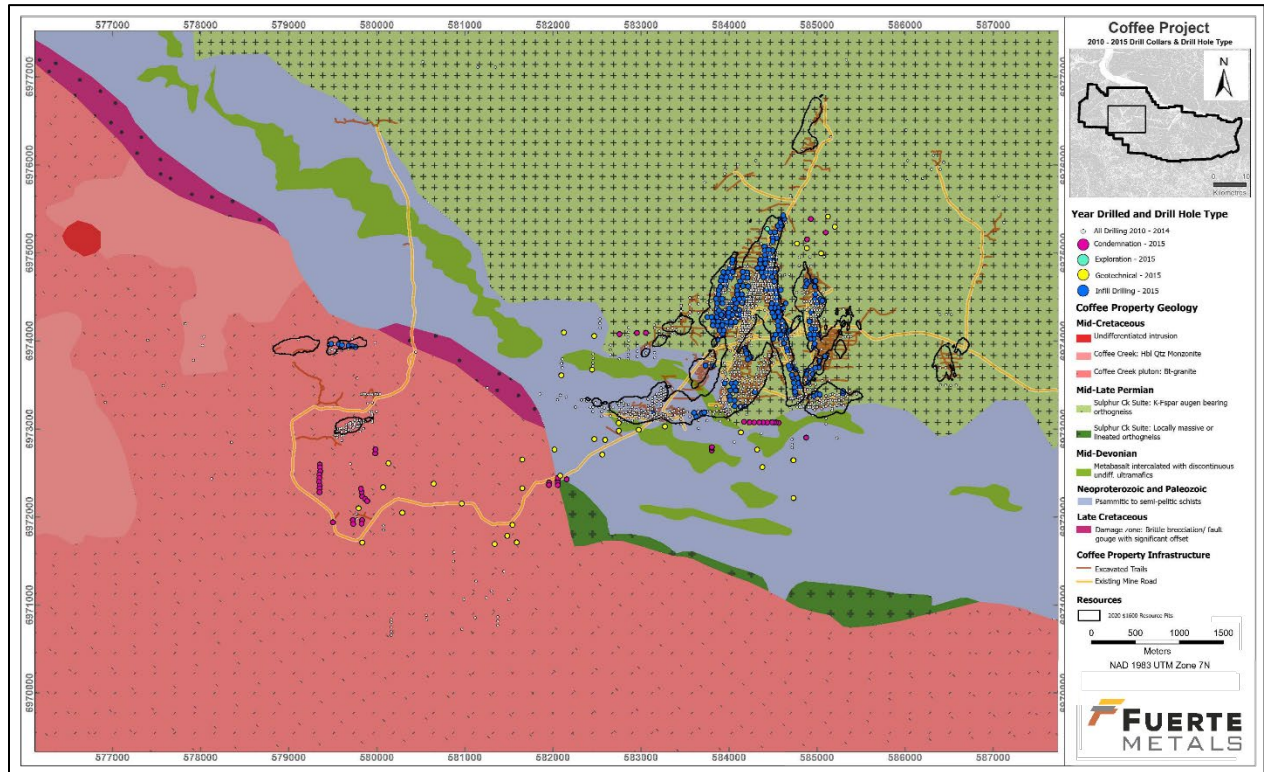
Source: Kaminak, 2014

The 2010 discovery drilling was immediately followed by the 2011–2013 delineation of mineralized structures during which the Cappuccino and Supremo Extension were discovered. Infill and metallurgical drilling followed suit in 2014.

10.6.2 Drilling Campaign 2015 (Kaminak)

Drilling efforts focused on continued infill drilling at Supremo with continued condemnation, metallurgical and geotechnical drilling also completed. A RAB condemnation campaign and sonic geotechnical drilling campaign were undertaken to determine permafrost conditions and assist with future engineering studies. The drilling completed is summarized in Figure 10.3.

Figure 10.3
Infill, Condemnation and Geotechnical Drilling Completed in 2015



Source: Kaminak, 2024

10.6.3 Drilling Campaign 2016 (Kaminak)

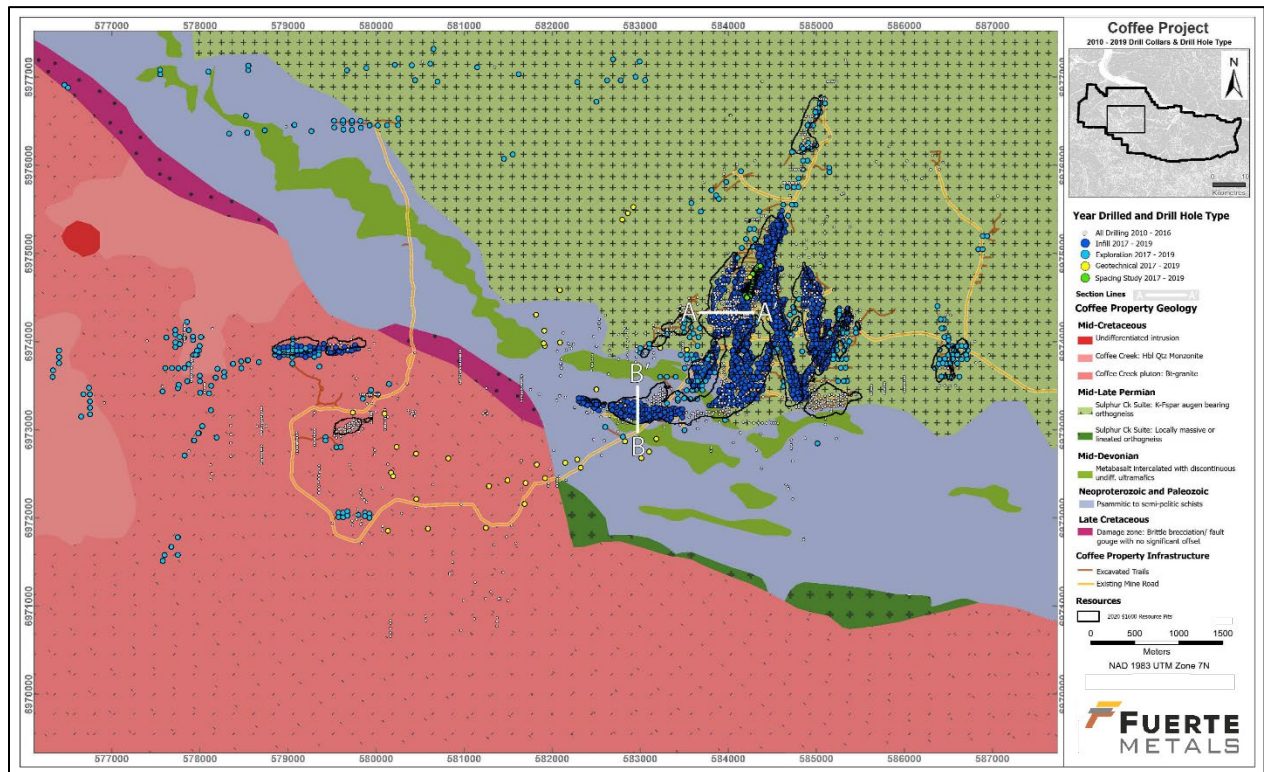
Drilling focused on extension of known mineralized structures including Americano, Kona, Latte, Double-Double, Supremo T3, Supremo T8-9, and Arabica.

10.6.4 Drilling Campaign 2017–2019 (Goldcorp)

The initial effort was to infill Latte to 25 m x 25 m spacing and was driven by the need to have a measured resource classification. In addition to the dense drilling campaign at Latte, infill drilling was also conducted at Kona North and step out drilling identified a westward extension of mineralization at Kona North leading to the discovery of the AmeriKona prospect. Other infill programs included exploration infill at Supremo T8-9 and Arabica. Exploration drilling at Americano, Kazaar and Decaf successfully intersected and delineated mineralization.

The drilling for this campaign is summarized in Figure 10.4, Figure 10.5, and Figure 10.6.

Figure 10.4
Drilling Campaign 2017-2019

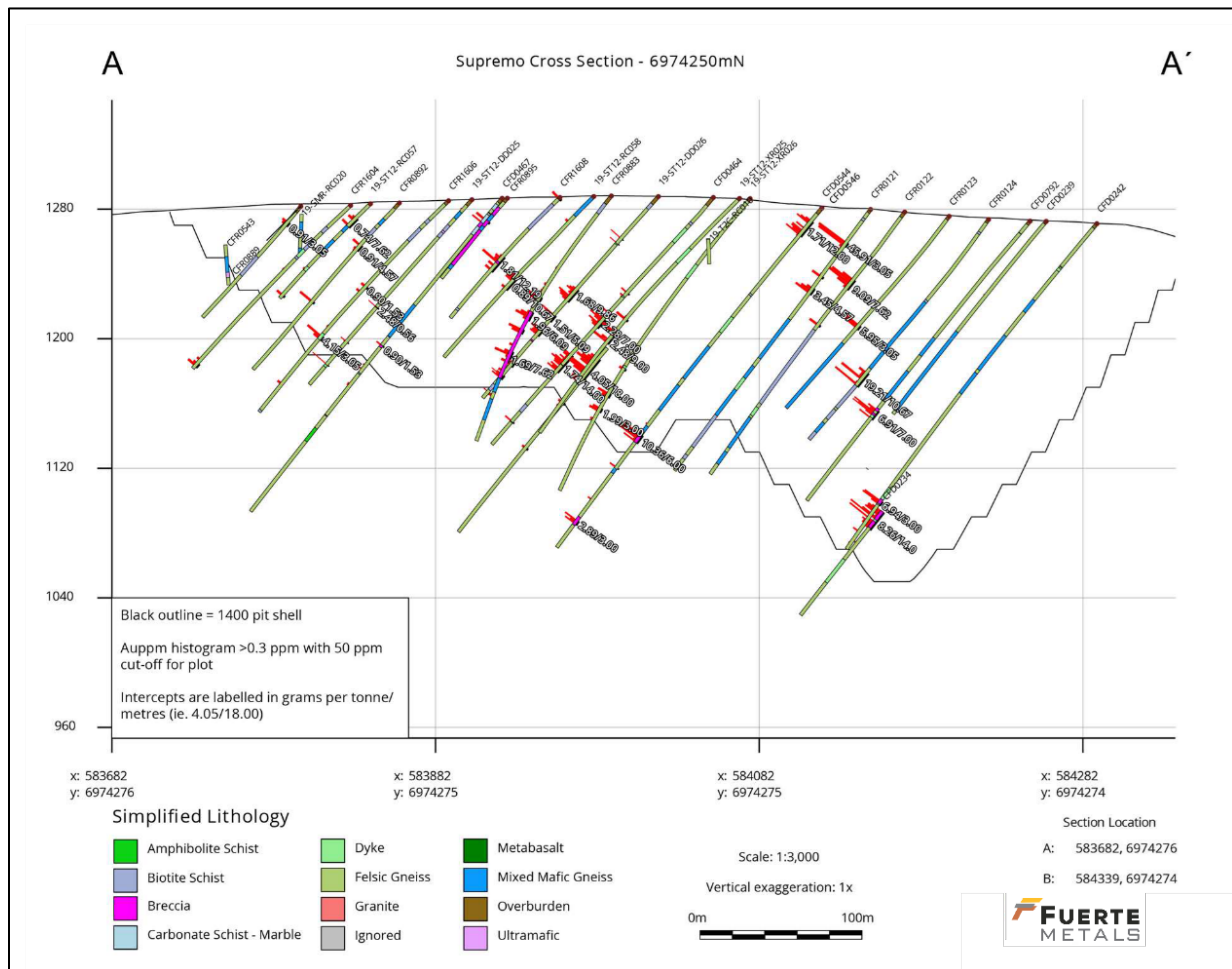


Source: Goldcorp, Newmont, 2024

The year 2018 commenced with the completion of the spacing study along Supremo T3. Exploration drilling was limited to sporadic testing of coincident VLF-EM and soil anomalies at Espresso, Americano, Double-Double, and Mocha. The majority of drilling completed focused on infilling Supremo, AmeriKona, Dolce, and Decaf to bring the drill spacing to 25 m x 25 m.



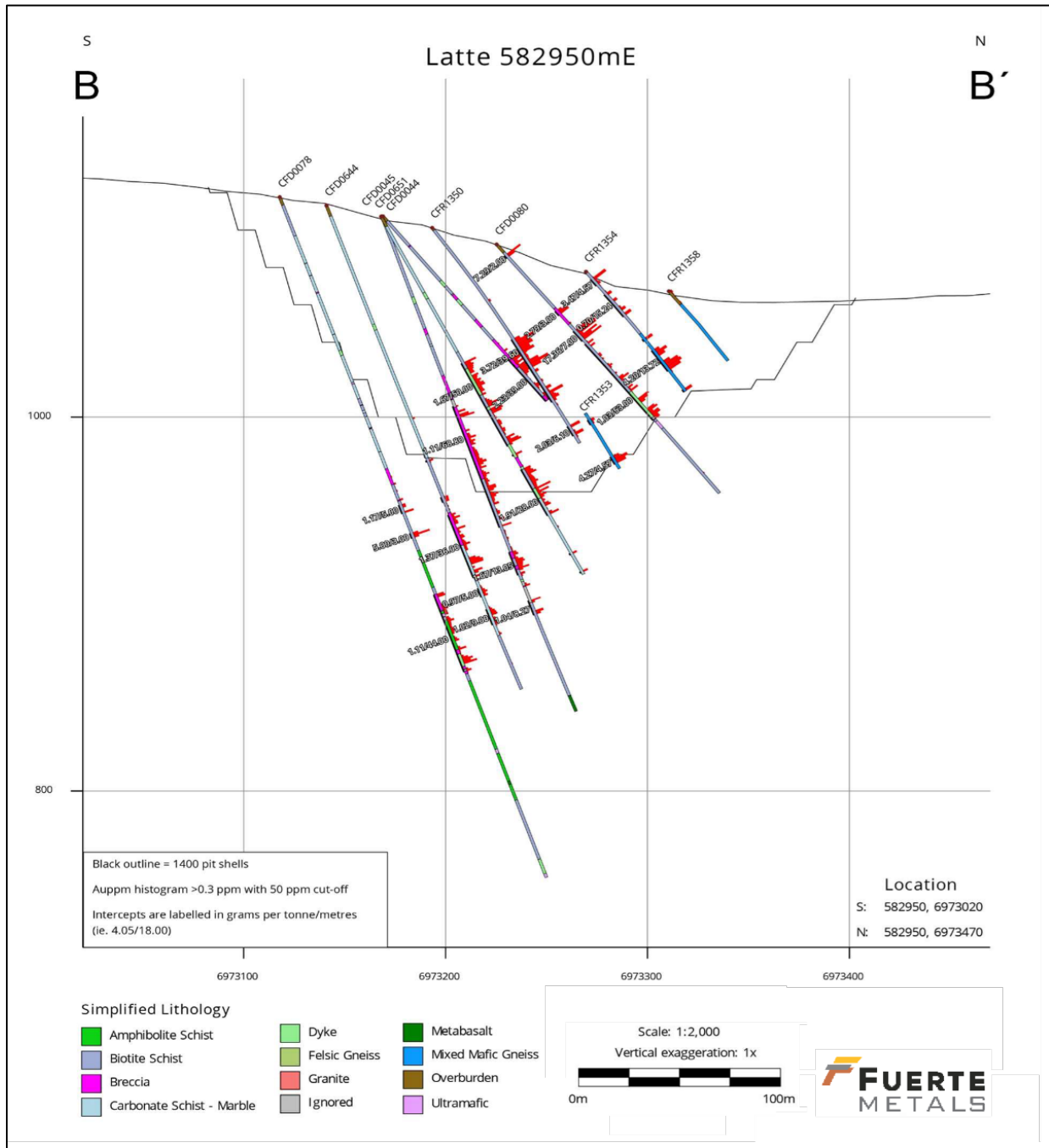
Figure 10.5
Cross Section Across Supremo T3 and Supremo T5



Source: Goldcorp, Newmont, 2024



Figure 10.6
Cross Section Across Latte



Source: Goldcorp, Newmont, 2024



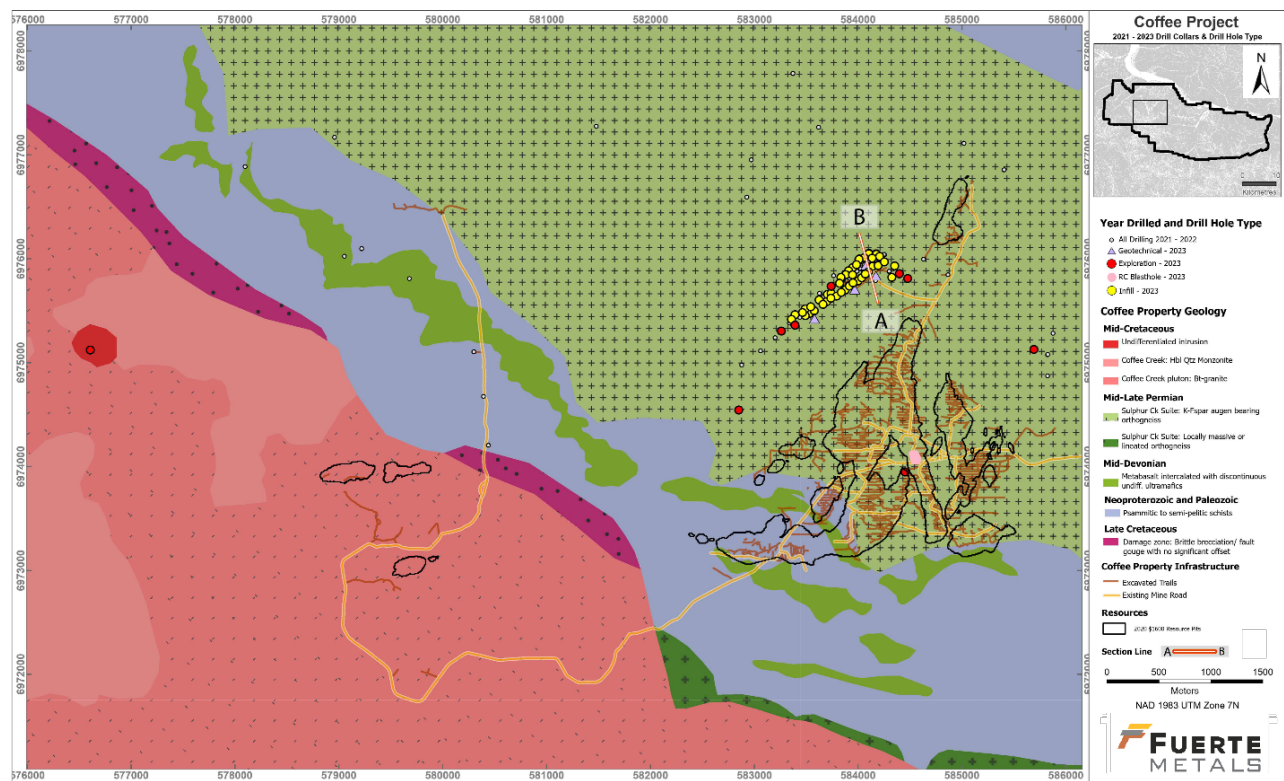
10.6.5 Drilling Campaign 2020–2023 (Newmont)

Drilling in Supremo Extension during the 2021 field season was able to successfully improve spacing within the prospect from 200 m to 100 m spacing along the Northern limb resulting in an additional 24 significant intercepts up to 137.45 g/t gold. The lack of diamond drilling during this period; however, made it difficult to improve the geologic model for the prospect.

Most of the diamond drilling completed in 2022 focused on the Supremo Extension target to add confidence to previous delineated mineralization and collect structural data that was missing from the previous season. Additional diamond and RC drilling were conducted at Supremo Extension to test along strike and down-dip of known mineralized structures. Drilling consistently intersected the three main structures that host the majority of the gold in Supremo Extension, as well as numerous secondary structures.

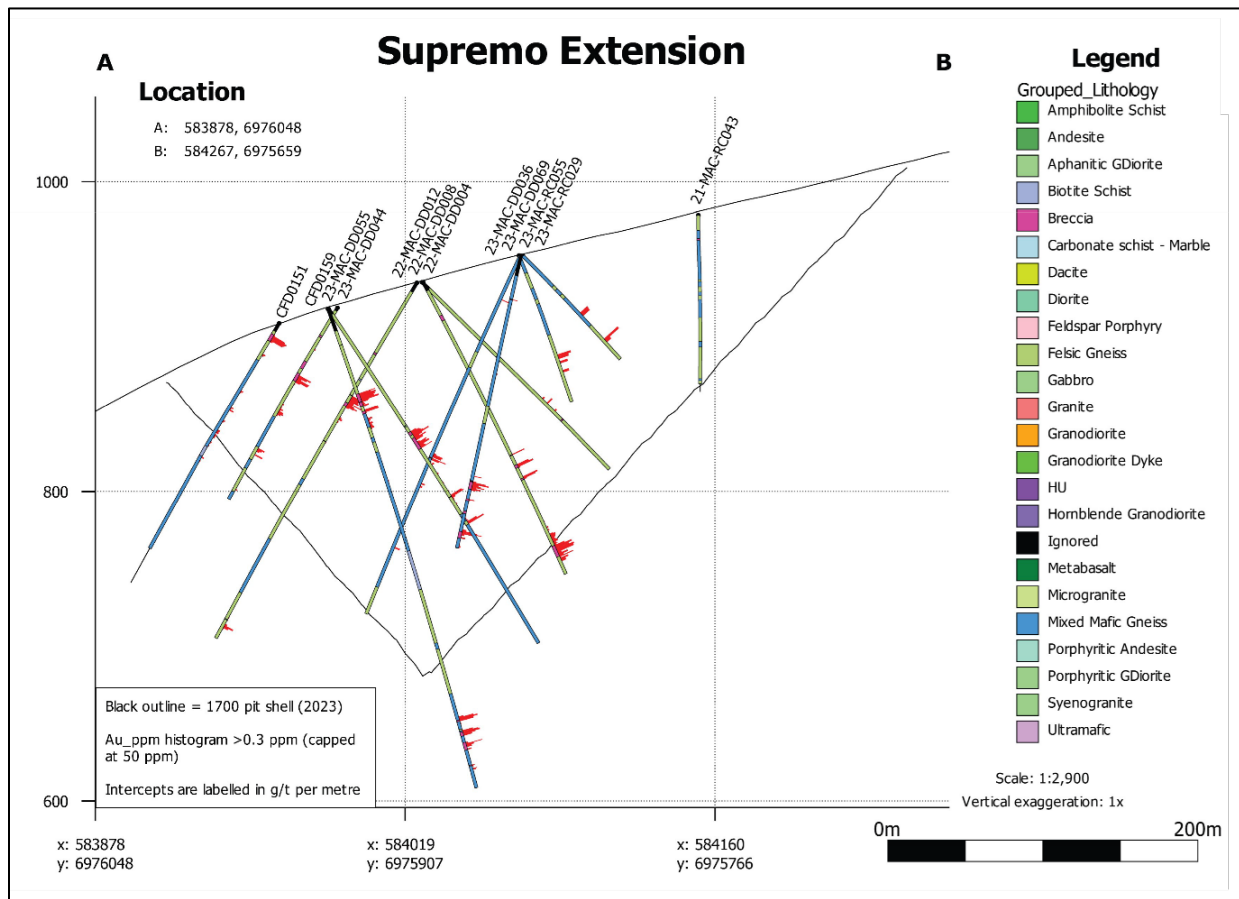
During the 2023 field season the bulk of the diamond and RC drilling was focused on advancing the Supremo Extension prospect to inferred mineral resource classification by decreasing drill spacing to 50 m x 50 m. The drilling achieved is summarized in Figure 10.7 and Figure 10.8.

Figure 10.7
Supremo Extension Drilling Campaign 2023



Source: Newmont, 2024

Figure 10.8
Cross Section of the Supremo Extension



Source: Newmont, 2024

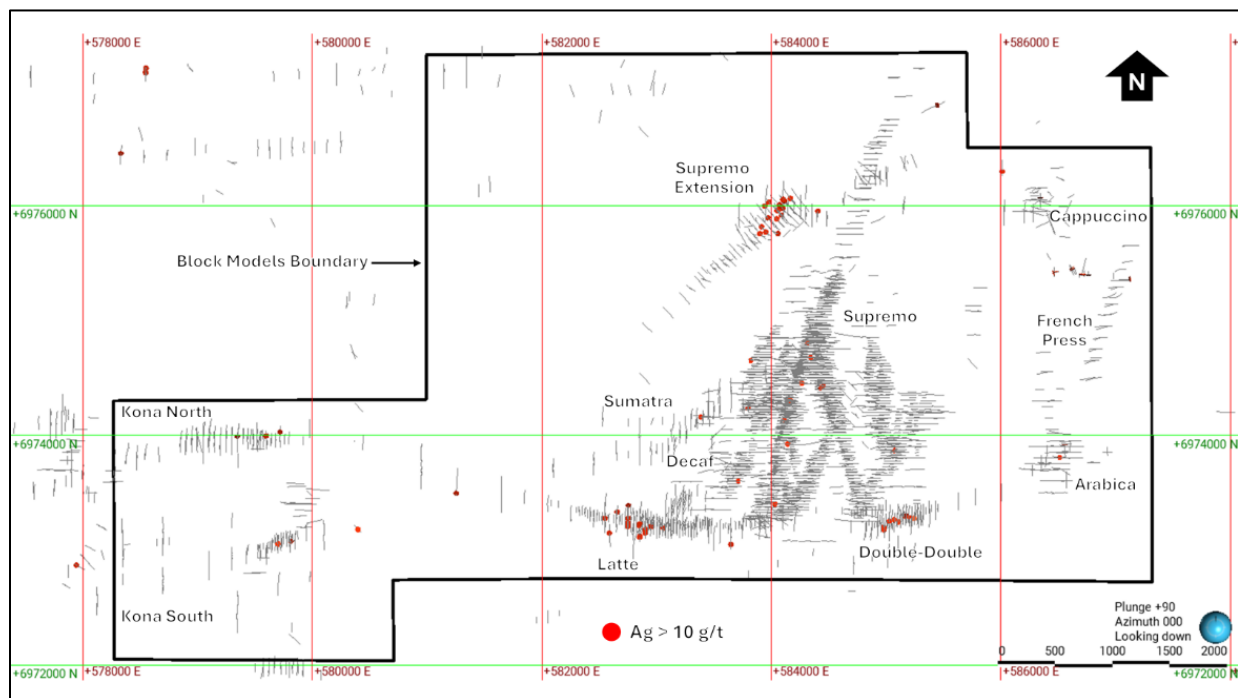
10.6.6 Soil Samples Preparation and Analyses

Soil samples collected by Shawn Ryan in 2007 were analyzed by Acme. Soil samples were prepared using a conventional preparation procedure and analyzed for a suite of 36 elements using aqua regia digestion followed by inductively coupled plasma-atomic emission spectrometry (ICP-AES) on 15 g sub-samples (method code 1DX2). Soil samples collected between 2009 and 2021 were also submitted to BV. The samples were prepared and assayed using the same methodology used to assay samples submitted in 2007. Soil samples were prepared using standard preparation procedures and analyzed for a suite of 36 elements using aqua regia digestion followed by ICP-AES on 15 g sub-samples (method codes 1DX2/AQ201). Soil samples were not submitted for umpire testing at a secondary lab.

10.6.7 Silver Rich Areas

The association of gold and silver remains to be fully investigated. The areas so far established to have silver of significance in addition to gold are Supremo Extension, Latte and Double-Double as shown in Figure 10.9.

Figure 10.9
Coffee Gold Project Silver Rich Areas



Source: Micon, 2025 (Generated from the MRE database)

10.7 Summary of the Best Drillhole Intercepts

A summary of drillholes with greater than 10 m intersection length (about 7 m true thickness) and greater than 10 g/t Au is shown in Table 10.2 below.

Table 10.2
Table Showing the Best Drill Intersections Encountered at the Coffee Gold Project

Hole ID	From	To	Au g/t	Length	Easting	Northing	Elevation
19-ST3-DD011	148.00	161.00	15.37	13.00	584,138	6,974,212	1,123
19-ST4-DD003	137.40	156.63	11.40	19.23	584,287	6,973,581	987
CFD0001	17.00	29.20	21.07	12.20	584,186	6,974,454	1,273
CFD0199	158.00	169.00	14.10	11.00	584,071	6,973,999	1,074
CFD0210	31.00	42.00	15.52	11.00	584,027	6,973,751	1,092
CFD0618	44.00	66.00	12.94	22.00	582,852	6,973,308	1,007
CFD0802	189.43	200.00	15.23	10.57	584,146	6,974,230	1,100
CFR0035	33.53	44.20	18.48	10.67	584,180	6,974,422	1,256



Hole ID	From	To	Au g/t	Length	Easting	Northing	Elevation
CFR0252	184.40	198.12	19.07	13.72	584,278	6,974,649	1,086
CFR0433	57.91	68.58	10.18	10.67	583,077	6,973,250	1,062
CFR1452	51.82	67.06	10.72	15.24	577,927	6,973,850	1,046
CFR1894	18.29	33.53	10.36	15.24	584,066	6,974,049	1,199
CGC00242	36.58	50.29	10.20	13.71	584,280	6,974,735	1,214
CGC00304	0.00	15.24	22.83	15.24	584,204	6,974,496	1,268

10.8 MICON QP Comments

Micon's QP has not identified any drilling, sampling or recovery factors that could result in sampling bias that would otherwise materially impact the accuracy and reliability of the assays and, hence, the mineral resource database. Core recoveries of >90% were confirmed during the site visit.

In summary, the drilling campaigns have successfully outlined a large mineral resource of over 3 million ounces of gold, as detailed in this Technical Report.



11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Two main types of sampling have been collected at the Coffee Gold Project.

- (i) soil samples - are primarily used to direct exploration efforts across the Mineral Project
- (ii) rock samples - mainly derived from diamond and RC drilling which are primarily used for the estimation of mineral resources.

There are also a relatively small number of additional “grab” samples and samples collected from trenches, as well as stream sediment samples which were used for exploration and evaluation purposes but not used directly in the estimation of mineral resources. Detailed sample preparation, assaying and QA/QC protocols for soil and rock samples are included in the sections below.

11.1 Sample Preparation/Quality Control Before Dispatch to Analytical Laboratory

11.1.1 Sample Preparation

Sample preparation for the different types of samples is as follows:

11.1.1.1 (i) Soil Samples

The soil samples are collected from the B-C horizon and put into sample bags at the time they are collected in the field; there is no further preparation required before dispatch to the laboratory.

11.1.1.2 (ii) Rock Samples

These are differentiated into drill core, RC chips and trench/grab samples.

11.1.1.3 Drill Core

Drill core is cut/split longitudinally into symmetrical halves prior to sampling using a diamond saw; one half is taken as the sample for laboratory analysis, and the other half is retained for future reference. The drill core samples are typically collected at 1 m intervals; however, where warranted, sample lengths are varied based on visible mineralization, lithological, alteration and mineralogical changes.

11.1.1.4 RC Chips

During drilling, the sample is separated from the air in the cyclone and drops out the bottom into a 5-gallon bucket. Each sample comprises one 1.52 m run, with the drillhole and rods being blown out (cleaned) between each run. Once the complete sample has been collected in the five-gallon bucket the total weight of the sample is recorded in kilograms. The weight of the empty bucket is recorded at the start of each drillhole on the first sample tag. The sample is then reduced through a 1:8 riffle splitter into a sample typically weighing 2 kg that is retained for analysis. The weight of the split sample is also recorded. A sub-sample taken from the reject material is wet sieved and put into chip trays for logging by onsite Project geologist.

11.1.1.5 Trench/Grab Samples

These samples are collected in sample bags in the field; there is no further preparation required before dispatch to the laboratory.



11.1.2 Quality Assurance/Quality Control

During 2009, Kaminak did not implement specific analytical quality control measures to monitor the assay results delivered by Acme. The 2009 exploration program involved primarily soil sampling and trenching. Kaminak relied on the laboratory internal analytical quality control measures to monitor the reliability of assay results delivered by Acme.

From 2010, QA/QC samples have been inserted at regular intervals in the sample batches. The QA/QC samples comprise certified reference materials (CRMs) representing low, medium, and high-grade mineralization that were sourced from various research/commercial laboratories including Ore Research & Exploration Pty Ltd (OREAS), and CDN Resource Laboratories (CDN). The QA/QC samples also include certified analytical blanks from one or more of these laboratories.

The rate of insertion of CRMs and blanks has varied over the years. For rock samples collected in 2010, CRMs were inserted approximately at a rate of one every 30 samples. For rock samples collected between 2011 and 2023, blanks and CRMs were alternated and inserted at a rate of one every ten samples.

11.2 Sample Packaging and Security

All activities pertaining to sampling and insertion of control samples are conducted under the supervision of the Project geologist. There is no other action taken at site; thus, no aspect of the sample preparation for analysis is conducted by an employee, officer, director or associate of the issuer.

On site, all soil and rock samples (including QA/QC samples) are individually sealed in polymineralization bags and consolidated into sample batches in rice sacks which are sealed and uniquely numbered with security tags to control sample tampering. The sample batches are then shipped offsite by commercial fixed wing charter aircraft (operated by Alkan Air Ltd. and Great River Air Ltd.) from the Coffee Gold Project Airstrip to either Whitehorse or Dawson and then transported via truck directly to the ALS Minerals preparation facility in Whitehorse (ALS) by an expeditor, Fuerte staff, or ALS laboratory personnel. Security tags were/are tracked throughout transportation until receipt by ALS. Upon receipt, laboratory personnel check to ensure that no seal has been tampered with and acknowledge receipt of samples in good order via telephone/email. No samples were reported tampered with from 2010 through to 2023.

11.3 Details of Laboratories Used

Two primary laboratories have been used for sample preparation and assaying for samples collected between 2009 and 2023: i) Acme Analytical Laboratories / Bureau Veritas Commodities Canada (BV) was used for analysis of soils samples; ii) ALS Group of Laboratories (ALS) was used for drill, trench and grab samples. BV was also used as a secondary laboratory for umpire or check assaying of drill samples. BLEG samples were prepared and analyzed at an internal Newmont laboratory facility in Denver, Colorado.

The management system of the ALS Group of laboratories is ISO 9001:2015 accredited, and the employed primary and auxiliary analytical laboratories are also ISO/IEC 17025:2017 accredited by the Standards Council of Canada for certain testing procedures, including the methods used to assay samples submitted from the Coffee Gold Project. ALS participates in international proficiency tests such as those managed by CANMET and Geostats Pty. Ltd. BV's Vancouver laboratory is ISO/IEC 17025:2017 certified by the Standards Council of Canada for the provision of assays and geochemical analyses.

All the laboratories involved in the analyses of the Coffee Gold Project samples are independent of Fuerte.



11.4 Laboratory Sample Preparation And Analyses

11.4.1 Rock Samples Preparation and Analyses

All drill core, RC, trench, and grab samples collected from 2010 through 2023 were submitted to ALS for preparation and assaying. Samples were typically received, crushed, and pulverized by the ALS Whitehorse preparation facility, and shipped to ALS North Vancouver for assaying. In years of high sample volumes, sample preparation and/or analysis was distributed to secondary ALS laboratory facilities in Terrace, Kamloops, Yellowknife, North Vancouver, and Reno, Nevada, to improve turn-around times.

11.4.1.1 Preparation

Rock and core samples were prepared for assaying at ALS preparation facilities using a conventional preparation procedure (dry at 60°C, crushed and sieved to 70% passing 10 mesh ASTM, pulverized to 85% passing 75 µm or better). Prepared samples were then transferred to ALS analytical laboratory in North Vancouver (or auxiliary facilities) for analytical testwork.

11.4.1.2 Analyses

Prepared samples were assayed for gold using a conventional lead-fire assay procedure (Au-ICP21, ICP-AES) on 30 g sub-samples (50 g samples were used in 2010). Fire-assayed samples with greater than 10 g/t gold were re-assayed from a second 30 g split (50 g split in 2010) using a fire assay procedure and a gravimetric finish (Au-GRA21). From 2012 to 2023, samples with grades greater than 20 g/t gold were also submitted for screened fire assay from a 1,000 g coarse reject split (Au-SCR21). The screened fire assay was passed through a 100 µm mesh, with the oversize fraction undergoing gravimetric analysis following fusion, whereas the undersize fraction was split into two 50 g samples and analyzed using atomic absorption. The average between the two minus fractions was then combined with the plus fraction to provide the total weighted average gold.

From 2013 to 2019 rock samples with grades greater than 0.3 g/t gold, were also submitted for cyanide soluble gold assay (Au-AA13), after which the triggering threshold was lowered to 0.2 g/t gold and applies to all drill samples produced from 2021 to 2023. For this analysis, a 30 g sub-sample was weighed in a closed 100 mL plastic vessel followed by the addition of 60 mL of sodium cyanide solution (0.25% NaCN, 0.05% NaOH) before shaking until homogenized.

Following homogenization, the solution was rolled for an hour before an aliquot was taken and centrifuged. Finally, the sample was analyzed by atomic absorption spectrometry. In 2013, 8,016 sample pulps from previous drilling programs (2010 through 2012 inclusive) were subjected to cyanide leach analyses. A summary of the number of samples analyzed by Au-AA13 can be found in Table 11-1.

In addition to gold fire assays, all drill samples produced in 2010 and 2011 were also analyzed for a 35-element suite using an aqua regia digestion and ICP-AES finish on 0.5 g sub-samples (ME-ICP41). The ME-ICP41 ALS multi-element method was also utilized in 2012 to 2019, although only on selected drillholes/samples. In 2018, a selection of samples from exploration drillholes were analyzed for 35-48 elements using a four-acid digestion and ICP-AES finish on 0.25 g sub samples (ME-ICP61). From 2021 through 2023, all regular drill samples were analyzed for a suite of 48 elements utilizing a four-acid digestion with a combination of ICP-AES and ICP-MS finishes (ME-MS61).

Additionally, a selection of drill samples with silver grades of higher than 100 g/t were re-assayed using either digestion followed by ICP-AES or by conventional fire assay with gravimetric finish on 30 g charges (2010: three samples; 2011: two samples; 2012: two samples; 2018: two samples; 2023: one sample). Two samples from 2011 and three samples from 2012 reported higher than 100 g/t silver but were not re-assayed.



Approximately 1% of all master pulps from core and RC samples submitted to ALS in 2010 through 2023 were submitted at the conclusion of each exploration season, to BV Laboratories for umpire gold check assay testing. BV employed comparable analytical gold methods to analyze the umpire samples as described above, including developing a customized cyanide leach assay method in 2014 to replicate the ALS Au-AA13 leach method described above. Additionally, between 0.2%-0.7% of coarse reject samples, were pulverized and assayed for gold at ALS (Au-ICP21 and Au-AA13), as laboratory preparation duplicates from 2013 to 2023.

All zones drilled in a given year were represented in the check-assay samples. Although samples covered a wide range of assay results (from detection limit to greater than 20 g/t gold), preference was given to individual samples that displayed greater than 0.2 g/t gold to provide an accurate test of laboratory performance and avoid analyzing a large number of near-detection level samples.

11.4.2 Rock Samples Specific Gravity Analysis

A total of 10,584 SG field measurements of primary core samples and 986 measurements of field standards were documented between 2010 and 2023 (see Section 10.21.2.1). To ensure accuracy, the measured weights were recorded into an MS Excel Spreadsheet from 2010-2019 in which the SG value was calculated. For SG values less than 2.40 or greater than 3.50, samples were re-weighed and entered data was checked by technicians prior to being entered into the MS Access drill database. From 2021 to 2023, measured weights were entered into an acquire database for live validation of the entered data prompted warnings and instructions to check data.

Independent specific gravity testing was also conducted on a selection of samples by ALS to verify the accuracy of the on-site methodology by utilizing ALS analytical method OA-GRA08a which is analogous to the field SG method employed at site. ALS OA-GRA08a results are in close agreement with field measurements and, therefore, indicate good reproducibility. Additionally, the 46 samples submitted for OA-GRA08a analysis in 2021 were re-analyzed utilizing ALS method OA-GRA08b which measured SG from a pulverized split from the same sample. This was done to test the viability to gain SG data from archive master pulps samples from previous RC drilling. However, the OA-GRA08b results did not reliably reproduce SG values from field measurements or ALS OA-GRA08a.

11.5 Quality Assurance And Quality Control Protocols And Results

11.5.1 Overview of Protocols

The exploration work conducted by Newmont and its predecessors was carried out using a QA/QC program meeting industry best practices for exploration properties. Standardized procedures were used in all aspects of the exploration data acquisition and management including mapping, surveying, drilling, sampling, sample security, assaying, and database management.

During 2009, Kaminak did not implement specific analytical quality control measures to monitor the assay results delivered by Acme. The 2009 exploration program involved primarily soil sampling and trenching. Kaminak relied on the laboratory internal analytical quality control measures to monitor the reliability of assay results delivered by Acme.

At the commencement of core drilling in 2010, Kaminak began implementing external analytical quality control measures, in addition to choosing an ISO accredited primary laboratory. The analytical quality control measures involved the use of control samples (CRMs, blanks, field duplicates) and independent check assaying at an umpire laboratory.

Certified reference materials were sourced from CDN from 2010 to 2018. In 2018 and 2019, the CDN CRMs were replaced by CRMs from OREAS of Melbourne Australia. Typically, five to six unique standards were



used in each sampling program completed, with certified gold values ranging from ~ 0.2 ppm to ~ 10 ppm (fire assay). Pulp blanks were used exclusively from 2010 through 2017, after which a coarse silica blank was also introduced to enable further checking of the preparation laboratory crushing stage. All blanks used beyond 2010 were certified to contain less than 0.01 g/t gold. For 2010 rock samples, certified reference materials were inserted approximately at a rate of one every 30 samples. For 2011 through 2023 rock samples, blanks and CRMs were alternated and inserted at a rate of one every ten samples. The standards/CRMs utilized at the Coffee Gold Project are shown in Table 11.1.

**Table 11.1
Summary of Key Standards/CRMs Utilized at the Coffee Gold Project**

CRM ID	CRM TYPE	CERTIFIED AU G/T	CRM SOURCE
BL-9	Pulp Blank	0.01	CDN
BL-10	Pulp Blank	0.01	CDN
GS-1U	Pulp Standard	0.968	CDN
GS-3S	Pulp Standard	3.580	CDN
GS-6E	Pulp Standard	6.060	CDN
GS-9B	Pulp Standard	9.020	CDN
GS-P1A	Pulp Standard	0.143	CDN
GS-P7L	Pulp Standard	0.709	CDN
CSB	Coarse Blank	0.005	ORE
OR-21e	Pulp Blank	0.001	OREAS
OR-209	Pulp Standard	1.580	OREAS
OR-217	Pulp Standard	0.338	OREAS
OR-219	Pulp Standard	0.760	OREAS
OR-226	Pulp Standard	5.450	OREAS
OR-228b	Pulp Standard	8.570	OREAS

Field and laboratory duplicates were also inserted within the samples submitted for assaying. Diamond core field duplicate samples were collected by collecting two ¼ core pieces while leaving ½ core in the core box. Prior to 2013 core duplicates were done by sending ½ core for the parent sample and ¼ for the duplicate sample (with only ¼ core remaining in the core box). Reverse circulation sample duplicates were collected by running the retention bag of the original sample through the riffle splitter, splitting a second sample from the original sample directly at the drill site. Laboratory duplicates are repeat assays on pulverized samples originally assayed by ALS. Additionally, laboratory duplicates of cyanide shake test samples were taken at a rate of 1:50 total analyzed samples in 2013.

11.5.2 Overview of QA/QC Results

11.5.2.1 Standards/CRMs

Generally, QA/QC sample results are considered as failures if they are outside 2.5 to 3 standard deviations of the certified values. Assessment of the QA/QC sample results is conducted immediately after the assay results are received and remedial action taken where deemed necessary.

During an external audit of the QA/QC program at the end of 2018, it was found that reference materials sourced from CDN had a consistently high failure rate. It was found that this was attributed to poor homogeneity of the reference material. Some standards from CDN were still in use at the start of the first



quarter of 2019; however, their use was terminated in late February and a new set of standards from OREAS was put into use for the remainder of the year.

Control charts prepared for various CRMs and blanks are presented in Figure 11.1 and Figure 11.2 from which it is evident that >90% of assay batches did not require re-assaying.

Figure 11.1
Performance Plots of OREAS Low Grade CRMs

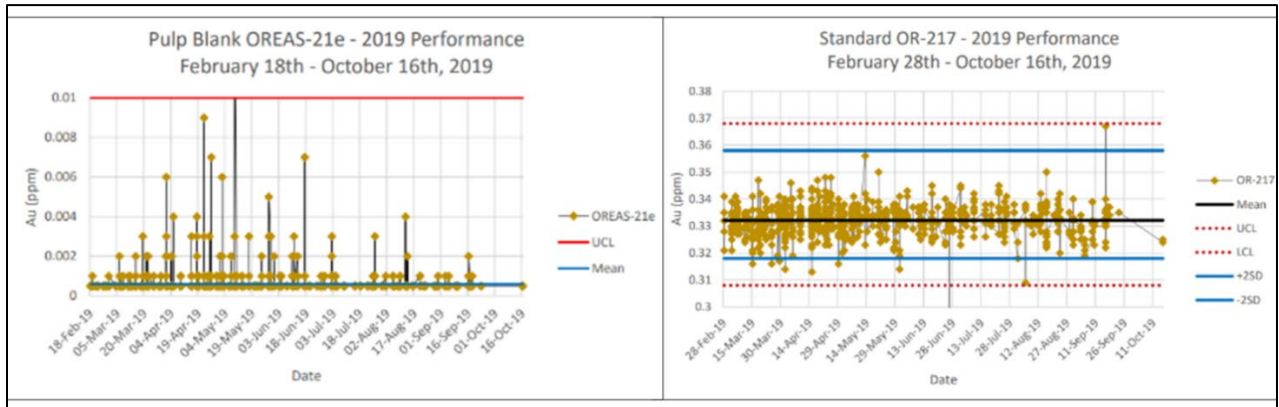
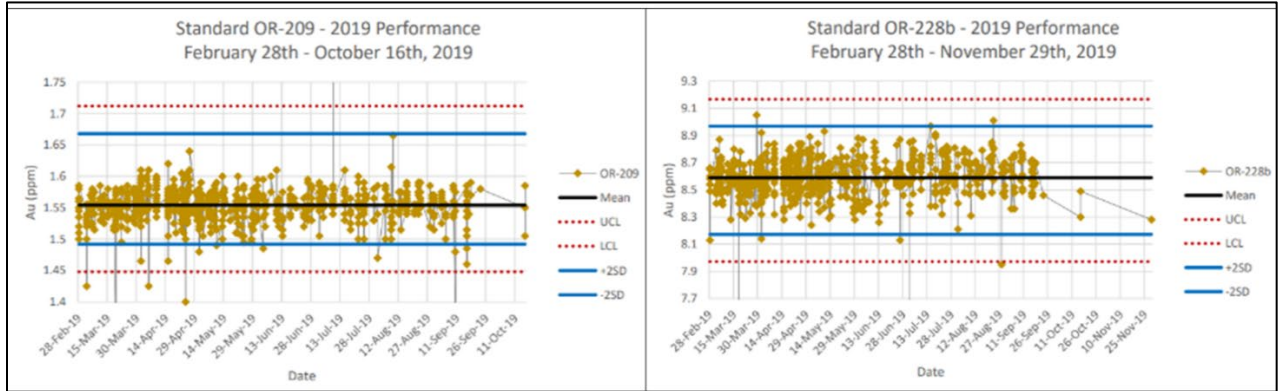


Figure 11.2
Performance Plots of High Grade CRMs



Wherever failures occurred, investigations were performed and remedial action taken as exemplified in the summary presented in Table 11.2.



**Table 11.2
Chart Summarizing Remedial Action Taken on CRM Failures**

Batch #	PO Number	Sample Start	Sample Finish	Hole ID	From	To	Status	Re-run	Reason
WH1217941 2	KGC-12-1639	KAM09114 8	KAM09128 7	CFD026 8	0	123	Fail	Pass	One standard failed high (KAM091270, GS-6A, 6.49ppm, max expected 6.17ppm). Fail associated with only grade in batch. Partial re-run requested 21-08-2012. All pass on re-run. No significant change in grade. 13-09-2012.
WH1217941 3	KGC-12-1640	KAM09632 2	KAM09642 1	CFD026 7	0	114	Pass		All QA/QC samples pass. 21-08-2012.
WH1217941 4	KGC-12-1643	KAM09642 2	KAM09651 7	CFD026 7	114	201	Pass		All QA/QC samples pass. 22-08-2012.
WH1217941 5	KGC-12-1644	KAM09651 8	KAM09661 7	CFD026 9	6	96	Pass		One standard failed slight high (KAM096590, GS-1J, 1.06ppm, max expected 1.048ppm). No significant grade in batch. All other QA/QC samples pass. No re-run. 04-09-2012.
WH1218175 4	KGC-12-1618	KAM10897 9	KAM10911 3	CFR026 1	0	181.6 6	Pass		All QA/QC samples pass. 04-09-2012.
WH1218175 5	KGC-12-1623	KAM10230 5	KAM10243 7	CFR026 2	0	182.8 8	Fail	Pass	One standard failed (KAM102430, GS-9A, 7.01ppm, min expected 8.62ppm). Only other standard in instrumental run also ran low, but within error (KAM102410, GS-1J, 0.866ppm, min 0.844ppm expected). This instrumental run carried all grade, and entire run was requested for re-run. 04-09-2012. All standards pass on re-run. Probable fluxing problem was culprit. Re-run accepted 13-09-2012.



The insertion of CRMs into sample batches was supported by several exhaustive duplicates/repeats/re-runs as summarized in Table 11.3. Around 2% of RC and core drilling samples were supported with field duplicates and 1% of all samples were checked with samples sent to an umpire laboratory.

**Table 11.3
Primary and Check Assay Drill Samples by Year**

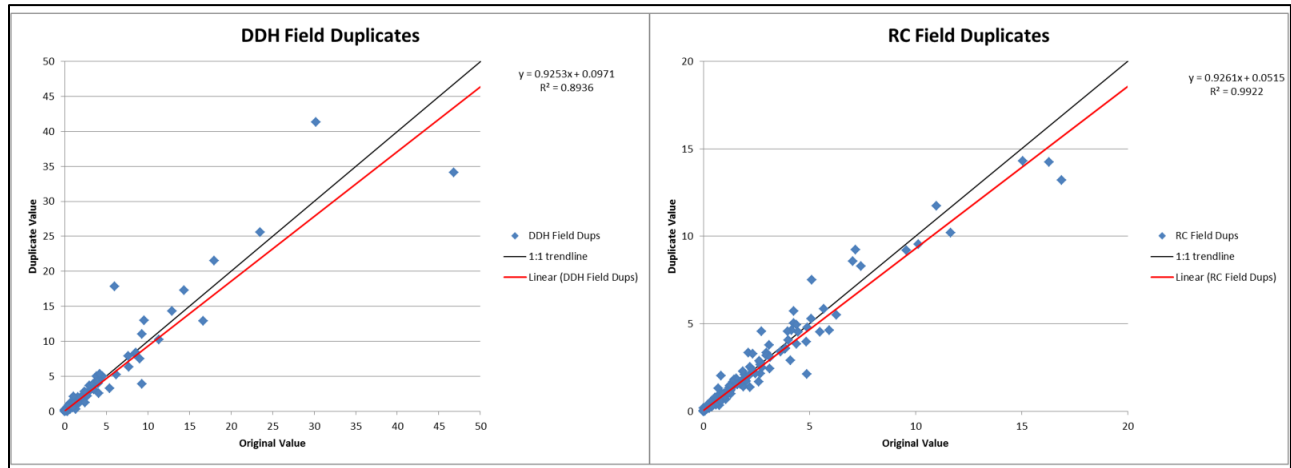
Year	RC Drill Samples	RC Field Duplicates	RC Field Dupl. (%)	DD Drill Samples	DD Field Duplicates	DD Field Dupl. (%)	Total Primary Samples	Umpire Pulp Checks	Umpire (%)	Coarse Reject Checks	Coarse Reject (%)
2010	0	0	0.0	15,955	411	2.6	15,955	178	1.1	0	0.0
2011	12,818	250	2.0	27,966	503	1.8	40,784	425	1.0	0	0.0
2012	25,691	485	1.9	26,581	546	2.1	52,272	672	1.3	0	0.0
2013	28,056	487	1.7	10,411	243	2.3	38,467	441	1.1	103	0.3
2014	16,850	321	1.9	19,187	373	1.9	35,837	448	1.3	125	0.3
2015	15,975	353	2.2	10,484	220	2.1	26,459	448	1.7	151	0.6
2016	12,989	518	4.0	6,597	125	1.9	19,586	450	2.3	141	70.0
2017	29,576	612	2.1	15,624	331	2.1	45,200	500	1.1	152	30.0
2018	39,519	1,173	3.0	27,736	606	2.2	67,255	500	0.7	150	0.2
2019	39,651	1,096	2.8	24,746	211	0.9	64,397	468	0.7	153	0.2
2020	0	0	0.0	0	0	0.0	0	0	0.0	0	0.0
2021	5,376	98	1.8	161	3	1.9	5,537	98	1.8	28	50.0
2022	4,448	205	4.6	3,464	72	2.1	7,912	97	1.2	31	0.4
2023	5,472	254	4.6	10,907	244	2.2	16,379	167	1.0	49	0.3
Total	236,221	5,852	2.5	199,819	3,888	1.9	436,040	4,892	1.1	1,083	0.2

Source: Newmont 2024.

An example of duplicates/repeat analyses results for diamond drillholes and RC duplicates is shown in Figure 11.3.



**Figure 11.3
Duplicate Analyses**



Source: Newmont 2024

11.6 QP Opinion and Comments

Micon QP considers the sample preparation, security, and analytical procedures to be adequate to ensure the credibility of the analytical results used for mineral resource estimation. The QA/QC protocols are comprehensive and in line with the 2019 CIMBest Practice Guidelines; they include the use of CRMs, duplicates, and repeat analyses at an umpire laboratory. All protocols are complimentary in ensuring the accuracy and integrity of data used in the MRE. The monitoring of the laboratory's performance on a real time basis ensures that corrective measures, if needed, are taken at the relevant time and gives confidence in the validity of the final certified assay data.

Following a thorough review the QA/QC results and quality control reports supporting the various drilling campaigns, the Micon QP concludes that the data provided by the Issuers and laboratories is adequately reliable for the purposes of mineral resource estimation.



12.0 DATA VERIFICATION

12.1 Site Visits

Mr. Barry Carlson, the QP responsible for heap leach capex and susex, performed a site visit on August 24th-28th, 2018. During the visit, Mr. Carlson inspected the site area layout for the heap leach facility, reviewed waste rock designated areas, visited each planned pit location, and visited the admin and camp areas, operated at the time by Goldcorp. The visual inspection included site surface soils, areas indicative of permafrost locations, and traversing the region which included drainages for general observation and review. Discussions were held with site personnel to discuss initial exploration results, previous drilling and testing for heap leach facility locations, and the proposed additional investigations and testing that were recommended.

A WSP employee (Mitch Trottier, Senior Geotechnical Engineer, P.Eng.) conducted a site visit from March 31 to April 1, 2026 on behalf of WSP QP's Marc Rougier, P.Eng and David Jin, P.Eng. The site visit included completion of site orientation, meetings with the senior site geologist and site superintendent, review of available project layout mapping for the proposed pits, heap leach facilities, waste rock storage areas, and associated infrastructure, and a helicopter reconnaissance of the proposed mine area. Georeferenced photographs of the proposed pit and infrastructure areas were collected during the helicopter flight for review by WSP Qualified Person personnel.

Russell Downer (Opencontour QP) conducted a site visit to the Project on August 28, 2018. During his visit, the QP toured the project area, including active and historical drill workings, and inspected the core shed and associated facilities. He reviewed drill core, observed site infrastructure and operational conditions, and gained an understanding of the exploration activities and geological setting. Discussions were held with site personnel regarding drilling procedures, core handling practices, and general project operations.

The Micon QP visited the Coffee Gold Project from 28 to 29 July 2025, during which time he undertook the verification exercises discussed below.

12.1.1 Ground Truthing

Ground truthing was conducted to confirm the geology, trends and magnitude of mineralization. This exercise was considered critical in the subsequent modelling of the deposits. The following was achieved:

- (i) An appreciation of the trenching completed as exemplified by Figure 12.1.

Figure 12.1
Trench 3 Site Showing Heavily Brecciated Oxidized Mineralized Zone



Source: Micon, 2025

(ii) Confirmation of structural controls of the mineralization as shown in Figure 12.2.

Figure 12.2
Micro-boudins + Micro-imbricate Structures in Mineralized Zone



Source: Micon, 2025

(iii) Verification of some drill collar positions as shown in Figure 12.3 and survey control points/procedures as depicted in Figure 12.4.

Figure 12.3
Verifying Drillhole Collars



Source: Micon, 2025

Figure 12.4
One of the Survey Control Points Observed During Ground Truthing



Source: Micon, 2025

12.1.2 Inspection of Logging and Sampling Facilities

The Micon QP conducted an inspection of the data gathering facilities utilized for logging and sampling drill cores and percussion chips. The key findings are summarized as follows:

- (i) The important drill cores are catalogued and kept in secure structures, as shown in in Figure 12.5.

Figure 12.5
Section of the Secure Core Shed Facilities at the Coffee Gold Project



Source: Micon, 2025

- (ii) Verification of the host lithologies for the Coffee Gold Project main deposits was achieved by reviewing sets of representative drillholes from each deposit. Examples are shown in Figure 12.6, Figure 12.7 and Figure 12.8. This review indirectly validates the lithology table of the mineral resource database.

Figure 12.6
Supremo Deposit Drill Core – Brecciated Granodiorite



Source: Micon, 2025

Figure 12.7
Latte Deposit Drill Core – Biotite Schist with Silicification



Source: Micon, 2025

Figure 12.8
Kona Deposit Drill Core – Subtle Brecciation



Source: Micon, 2025

A comparison of sample intervals in drill core trays with assay data sheets shows that in general the highest assays are recorded in areas of intense brecciation thereby emphasizing the strong structural control factor. Where selective sampling was conducted, the sampling encompassed extended zones of 10 m on either side of the core of mineralization.

Core recovery for the drillholes inspected was found to be >90% thereby confirming the average recovery as documented by Newmont.

12.2 Mineral Resource Database Check/Validation

12.2.1 Overview

The database was checked by:

- (i) examining and ensuring drillhole entries in the collar, survey, assay, and lithology tables, are matching
- (ii) ensuring that there were no duplicate entries in all the database tables. Details on procedures used for each component are as described below.

12.2.2 Drillhole Spacing

An assessment of the drilling pattern in 3D reveals a spacing of 25 m to 50 m up to 100 m between drillholes and the Micon QP considers this to be reasonable to demonstrate continuity for modelling purposes.



12.2.3 Assay Data

The Micon QP achieved assay data verification by:

- Comparing and ensuring the assays entered into the assay table correspond with the values in the original assay certificates issued by the laboratory.
- Reviewing the QA/QC protocols pertaining to procurement and utilization of control samples (i.e., standards/blanks) including duplicate analyses (see Section 11.21).

12.2.4 Survey Data

Procedures used for checking the survey involved examining downhole deviations and ensuring that the drillhole collar elevations matched the topography map surface.

12.2.5 Lithology Table Entries

The procedures used in the validation involved ensuring that the lithology table of the database matched the drillhole logs as recorded by the Project geologist (see Figure 12.6 to Figure 12.8). No issues were found.

12.3 Metallurgical Data Verification

The metallurgical QP has reviewed the metallurgical test work data and related reports, examined the analytical methodologies employed, evaluated the laboratory's credentials, and assessed the findings of previous analytical reports. All aspects were found to align with industry-standard practices. Additional statements on the adequacy of the metallurgical data are presented in Item 13.0.

12.4 Geotechnical Data Verification

The geotechnical QP has reviewed the slope design work conducted by others (JDS and SRK). The slope designs were found to align with industry standards for Preliminary Economic Assessments. Additional statements on the adequacy of the geotechnical data are presented in Item 16.1.

12.5 QP Opinion and Comment

12.5.1 Site Visit Due Diligence

The Micon QP site visit verification exercises demonstrated that the collection methods and the quality of the assayed samples in the database used to generate the MRE are satisfactory.

12.5.2 Database Check

The adequacy of data for MRE as summarized from Section 12.2.2 to subsection 12.2.5 are satisfactory for the Coffee Gold Project.

12.5.3 Overall Conclusion

The data used to estimate the mineral resources was generated in a credible manner with proper procedures and has been accurately transcribed from the assay certificates and is of sufficient analytical quality to support a MRE for the Coffee Gold Project.



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing for the Coffee Gold deposits commenced in 2011 and has continued since its inception. The initial testwork, pre-2013, was carried out by Inspectorate Exploration & Mining Services Ltd. (Inspectorate) of Richmond, British Columbia. Since 2013, metallurgical testing has been completed by several independent laboratories, including Kappes, Cassiday and Associates (KCA) in Reno, Nevada; SGS Canada Inc. (SGS) in Burnaby, British Columbia; Phillips Enterprises, LLC in Golden in Colorado, ALS Metallurgy Kamloops in Kamloops, British Columbia; and Forte Dynamics (Forte) in Fort Collins, Colorado.

The metallurgical testing utilized samples derived from both bulk surface samples and drill core composites. Testing included column leaching, bottle roll leaching, vat leaching, pressure oxidation and roasting, flotation, column percolation and drain down, multi-element head assay analyses, column leach head and tailings assay screen analyses, ball mill work indices, crushing impact, and abrasion indices.

The samples used for the various testwork programs completed to date were not only representative of all the main types of lithologies and mineralized material types identified at the Project, but also included mineralization from all of the identified individual deposits.

The summary of mineral processing and metallurgical testing presented in this Report is primarily based on a comprehensive analysis of the following existing documents:

- NI 43-101 Technical Report for the 2025 Mineral Resource Estimate Update on the Coffee Gold Project, Yukon, Canada authored by Micon International with a report date of October 6th, 2025 (the '2025 Micon Report')
- Internal Pre-feasibility Study Coffee Gold Project, Yukon Territory, Canada authored by JDS Energy & Mining Inc. with a report date of October 9th, 2024 (the 'Internal Engineering Report (JDS, 2024)')
- Various laboratory test results summaries in Excel and testwork reports, as further discussed in the following sub-sections.

13.1 Preliminary Testwork (Pre-2013)

According to the 2025 Micon Report (Micon, 2025), metallurgical testing conducted prior to 2013 has been identified; however, the associated laboratory reports were not made available for evaluation. Due to the same reason, the Micon QP for this Section of the Report has not been able to evaluate or make comments on testwork conducted prior to 2013.

13.2 Early Testwork (2013-2014)

During 2013 and 2014, metallurgical testing was carried out on both core and bulk samples from the Coffee Gold deposit at KCA for Kaminak as part of a comprehensive program. The following two metallurgical testwork reports were issued by KCA:

- Coffee Gold Project, Report of Metallurgical Testwork, November 2013, Prepared for: Kaminak Gold Corporation, Prepared by: Kappes, Cassiday & Associates, Reno, Nevada 89506, Report I.D.: KCA0130085_COF01_02; and
- Coffee Gold Project, Report of Metallurgical Testwork, April 2014, Prepared for: Kaminak Gold Corporation, Prepared by: Kappes, Cassiday & Associates, Reno, Nevada 89506, Report I.D.: KCA0130149_COF02_01.



The following sub-sections summarize the information contained in these two metallurgical reports.

13.2.1 Sampling

In June 2013, a total of 116 bags containing core representing 788 diamond drillhole interval material from the Latte and Supremo areas were delivered to KCA. The samples were utilized to generate seven drill core composite samples for the COF01_02 testwork program. Each composite sample was assigned a unique sample number (KCA Sample No. 68151 through 68157) and tested for head analyses, head screen analyses with assays by size fraction, bottle roll leaching, agglomeration, and column leaching. Additionally, portions of material from the Latte, Sulphide sample were utilized for comminution and flotation testwork.

In September 2013, 117 bags of bulk interval sample material were used to represent two sample groups from the following areas:

- A Supremo oxide sample, collected from a surface trench across the T3 mineralized structure at Location 6974250 mN, and
- A Latte oxide sample, collected from a surface trench across the Latte mineralized structure at Location 583250 mE.ard procedures.

These two composite samples were utilized by KCA for the COF02_01 testwork program. Each composite sample was assigned a unique sample number (KCA Sample No. 69580 and 69581) and tested for head analyses, head screen analyses with assays by size fraction, bottle roll leaching, agglomeration, and column leaching. From this work, the Supremo oxide sample comprised of 59 sub-samples, while the Latte oxide sample comprised of 58 sub-samples.

All preparation, assaying and metallurgical studies were performed utilizing industry-standard procedures.

13.2.2 Head Analyses

The composite samples prepared for COF01_02 and COF02_01 test programs are listed in Table 13.1 together with gold and silver head grades. Head analyses results via inductively coupled argon plasma-optical emission spectrophotometer (ICAP-OES) for mercury, copper, sulphur and arsenic are also listed in the table.



Table 13.1
Summary of Head Analyses – Preliminary Test Programs COF01_02 and COF02_01

KCA Sample No.	Sample Type	Description	Ave. Au (g/t)	Ave. Ag (g/t)	Total Hg (mg/kg)	Total Cu (mg/kg)	Total C (%)	Total S (%)	As (mg/kg)
68151	Core	Supremo, Oxide	1.46	0.41	0.31	13	0.22	0.10	1337
68152	Core	Supremo, Upper Transition	1.23	0.41	0.29	7	0.34	0.15	676
68153	Core	Supremo, Lower Transition	1.57	0.41	0.21	7	0.58	0.47	756
68154	Core	Latte, Oxide	1.49	0.62	0.23	23	1.75	0.16	1707
68155	Core	Latte, Upper Transition	1.48	0.89	0.36	35	2.03	1.03	2089
68156	Core	Latte, Lower Transition	1.66	0.62	0.29	31	2.97	1.27	2043
68157	Core	Latte, Sulphide	2.47	0.62	0.31	31	2.75	1.55	2011
69580	Bulk	583250 ME Latte Oxide	1.61	0.70	0.08	24	1.10	0.06	1817
69581	Bulk	6974250 MN Supremo	4.08	0.62	0.13	5	0.05	0.09	1310

Source: KCA (2013/2014)

13.2.3 Bottle Roll Leach Testwork

Amenability to cyanide leaching was assessed via bottle roll leach testing performed on portions of material from each composite sample. Gold and silver recoveries from the seven core samples depended on degree of oxidation. Typically, oxide samples had high recoveries, transition material showed moderate results, while the sulphide sample had poor recovery, indicating refractory behavior. Table 13.2 summarizes the bottle roll extraction results for the seven core samples.

Table 13.2
Summary of Bottle Roll Leach Tests – Drill Core Composites

KCA Sample No.	Description	Calculated Head Au (g/t)	Au Extracted (%)	Calculated Head Ag (g/t)	Ag Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
68151	Supremo, Oxide	1.44	94	0.41	50	1.29	1.50
68152	Supremo, Upper Transition	1.45	78	0.36	42	2.12	1.00
68153	Supremo, Lower Transition	1.64	53	0.64	68	1.45	1.00
68154	Latte, Oxide	1.57	92	0.94	78	1.27	1.50
68155	Latte, Upper Transition	1.37	51	0.85	40	1.15	1.50
68156	Latte, Lower Transition	1.46	38	0.61	33	1.57	1.50
68157	Latte, Sulphide	2.46	13	0.66	37	1.35	1.50

Source: KCA (2013)

Gold extractions of 92–94% from the two bulk samples show that this oxide mineralization responds well to cyanide leaching. The gold extraction results of the bottle roll leach tests are summarized in Table 13.3.



**Table 13.3
Summary of Bottle Roll Leach Tests – Bulk Samples**

KCA Sample No.	Description	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
				NaCN (kg/t)	Lime Addition (kg/t)
69580	Latte Oxide	1.47	94	0.06	2.5
69581	Supremo Oxide	3.59	96	1.62	2.5

Source: KCA (2014)

13.2.4 Agglomeration Testwork

Preliminary agglomeration testing was performed on crushed sub-samples from the two bulk samples as well as all core composites, except for the Latte sulphide composite. Multiple percolation tests were carried out on each agglomeration sample to assess the permeability of materials agglomerated with Portland cement. These tests conducted at a range of cement addition and without applying compressive load, using 2 kg portions of material crushed to target particle size of 100% passing 12.5 mm, 25 mm, or 31.5 mm, depending on the sample type.

The purpose of the percolation tests was to evaluate the permeability of the agglomerated material at various cement contents. The percolation tests were conducted in small (75 mm inside diameter) columns at a range of cement levels, with no compressive load applied. All agglomeration tests passed the criteria put forth by KCA, suggesting that agglomeration would not be required in heap leach operation. As a result, agglomeration was not applied in the subsequent column leach tests.

13.2.5 Column Leach Testwork

A total of ten column leach tests were conducted on the seven core composites, with material crushed to a target size of 80% passing 25 or 12.5 mm. During testing, the material was leached for 40 or 42 days with a sodium cyanide solution. Tests were conducted in an enclosed refrigeration unit at a target temperature of 4°C. A single test (KCA Test No. 69554) was conducted at ambient temperature (approximately 22°C).

The core composites column leach test results are presented in Table 13.4.



Table 13.4
Summary of Column Leach Tests – Drill Core Composites

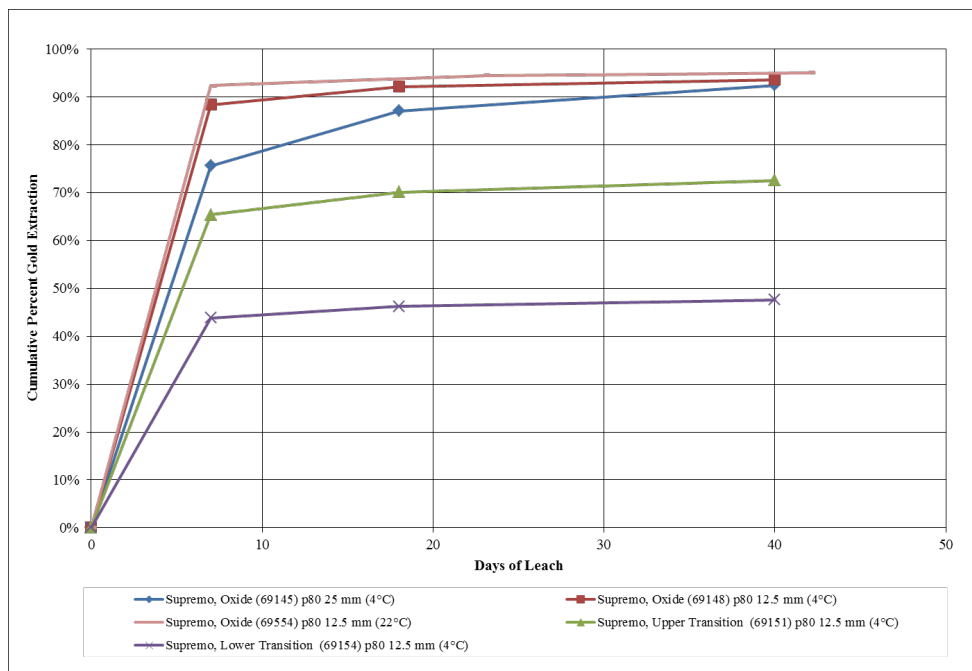
KCA Sample No.	Description	Crush P ₈₀ Size (mm)	Target Temp °C	Calculated Head Au (g/t)	Extraction Au (%)	Days of Leach	Reagent Consumption	
							NaCN (kg/t)	Lime Addition (kg/t)
68151 A	Supremo, Oxide	25	4	1.57	92	40	0.17	1.51
68151 B	Supremo, Oxide	12.5	4	1.44	94	40	0.28	1.50
68151 B	Supremo, Oxide	12.5	22	1.55	95	42	0.52	1.57
68152	Supremo, Upper Transition	12.5	4	1.49	73	40	0.31	1.00
68153	Supremo, Lower Transition	12.5	4	1.67	48	40	0.38	1.00
68154 A	Latte, Oxide	25	4	1.62	90	40	0.19	1.51
68154 B	Latte, Oxide	12.5	4	1.54	90	40	0.27	1.51
68155	Latte, Upper Transition	12.5	4	1.54	47	40	0.46	2.01
68156	Latte, Lower Transition	12.5	4	1.42	29	40	0.64	1.51
68157 B	Latte, Sulphide	12.5	4	2.37	5	42	0.46	1.51

Source: KCA (2013)

For each group of samples (Supremo and Latte), a graphical presentation of gold extraction over time is presented in Figure 13.1 and Figure 13.2.

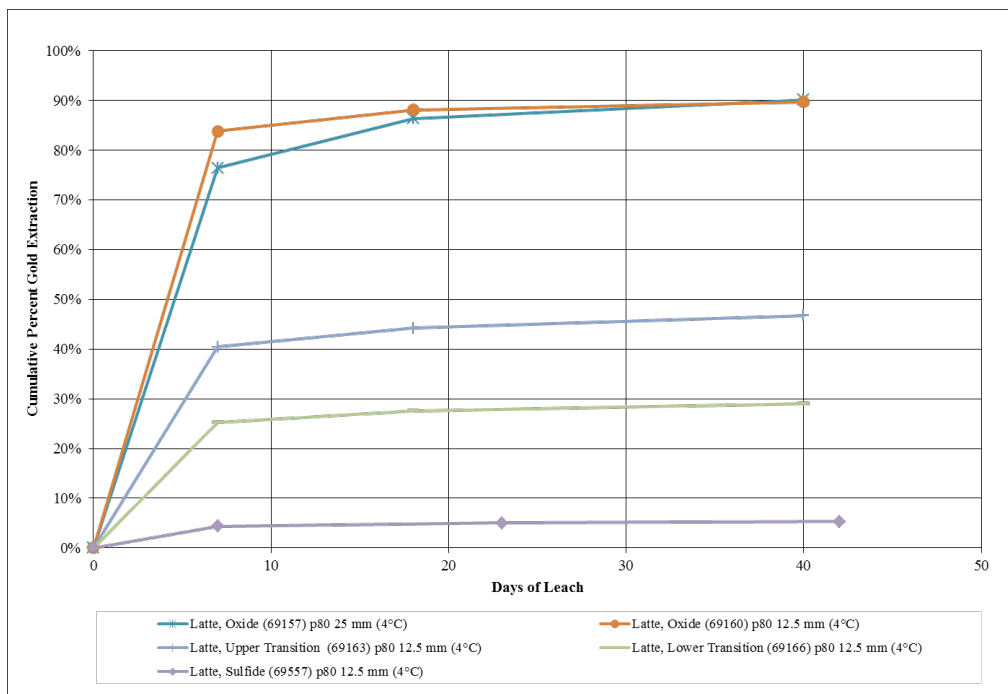


Figure 13.1
Supremo Core Composites - Column Leach Tests Gold Extraction



Source: KCA (2013)

Figure 13.2
Latte Core Composites - Column Leach Tests Gold Extraction



Source: KCA (2013)



13.2.6 Flotation Testwork

A sub-sample from the Latte sulphide composite (KCA Sample No. 68157) was submitted for kinetic flotation testing. Flotation tests were conducted in a laboratory-scale Denver flotation machine utilizing Reno municipal tap water. The products from each flotation test were dried, weighed, and individually assayed for gold, silver, copper, lead and total sulphur.

A total of four reagent scoping tests were conducted utilizing various reagent combinations and concentrations. Each test was conducted at a target grind size of 80% passing 75 μm . Applying various reagent schemes, the reagent scoping tests showed that around 62 to 69% of the gold can be concentrated into a rougher concentrate of approximately 21 g/t Au at a mass pull of about 8.0 to 8.9%.

Four grind size optimization tests were also conducted, utilizing standard reagent conditions and a range of grind sizes (80% passing 150, 75, 53 and 45 μm). The two coarser grind sizes gave the best results, with up to 72% Au recovery to a concentrate containing approximately 17 g/t Au with a mass pull of around 9.8 to 10.5%.

The results implied that a relatively coarse grind of between 75 μm and 150 μm was able to achieve reasonable flotation results for this material.

13.2.7 Comminution Testwork

A portion of the head material from the Latte, Sulphide composite (KCA Sample No. 68157) was submitted to Phillips Enterprises, LLC in Golden, Colorado for comminution testing to provide Bond Rod Mill and Ball Mill Work indices for the sample. The testwork illustrated a Bond Rod Mill work index of 12.7 kWh/t and a Ball Mill work index of 15.1 kWh/t.

13.2.8 Discussion

The gold head grades obtained throughout the test program were consistent, with strong agreement between the measured heads and the calculated values derived from the various parts of the test program.

Leach efficiency was evaluated by comparing gold extraction from head and tailings samples across size fractions at different temperatures. Results showed only 1–2% variation, confirming relatively consistent leaching kinetics at a range of operating temperatures (4°C and 22°C).

For the Latte, Oxide material, two column leach tests were completed to assess gold extraction at two particle sizes (80% passing 25 mm and 12.5 mm). The comparison showed no meaningful improvement in extraction with the finer crush, as both sizes achieved approximately 90% gold recovery.

For the column leach tests conducted on each group of samples (Supremo and Latte), a comparison of the gold extraction percentages with sulphide sulphur content showed generally a reduction in gold extraction with greater sulphide sulphur content.

A summary comparing gold extraction and sulphur speciation is presented in Table 13.5.



**Table 13.5
Column Leach Test Gold Extraction verse Sulphur Speciation**

KCA Sample	Description	Final Gold Extraction (%)	Sulphur Speciation Analyses (%)		
			Total	Sulphide	Sulphate
68151	Supremo, Oxide	94	0.10	0.06	0.04
68152	Supremo, Upper Transition	73	0.15	0.05	0.09
68153	Supremo, Lower Transition	48	0.47	0.30	0.17
68154	Latte, Oxide	90	0.16	0.02	0.14
68155	Latte, Upper Transition	47	1.03	0.70	0.33
68156	Latte, Lower Transition	29	1.27	0.93	0.34
68157	Latte, Sulphide	5	1.55	1.32	0.23

Source: KCA (2013)

13.3 Early Testwork (2014-2015)

Further metallurgical testing occurred in 2014 and 2015 by KCA, with the following report issued:

- Coffee Gold Project, Report of Metallurgical Testwork, June 2015, Prepared for: Kaminak Gold Corporation, Prepared by: Kappes, Cassidy & Associates, Reno, Nevada 89506, Report I.D.: KCA0140138_COF04_03.

The following sub-sections summarize the information contained in this metallurgical report.

13.3.1 Sampling

Testing at KCA continued in 2014 after receipt of approximately 8.7 tonnes of additional bulk/trench sample and half drill core material over a four-month period, as follows:

- September 2014: 148 individual bags of trench and surface bulk material from the Latte and Supremo areas, used to generate two Latte composites and one Supremo composite.
- October 2014: 119 bags of sample material from the Latte, Supremo and Kona, organized into seven individual samples.
- November 2014: 65 bags of sample material from the Latte, Kona, and Double-Double areas, organized into five individual samples.
- December 2014: 73 bags of material from the Supremo areas, organized into three individual samples.

These composites along with each individual remaining sample were utilized for head analyses, head screen analyses with assays by size fraction, bottle roll leach testwork and column leach testwork. All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures.

The selection of the cyanide soluble samples was designed to more adequately define the Transitional zones from oxide to sulphide. Some 80% cyanide (CN) soluble samples for certain mine areas was due to a lack of enough sub-samples available to generate a composite sample. For each head sample, developed



portions of the head material were ring and puck pulverized and analyzed for gold and silver by standard fire assay and wet chemistry methods. Head material was also assayed semi-quantitatively for an additional series of elements and for whole rock constituents. In addition to these semi-quantitative analyses, the head material was assayed by quantitative methods for carbon, sulphur and mercury. A cyanide shake test was also conducted on a portion of the pulverized head material. In addition to the analyses on pulverized head material, portions of material from each crush size utilized for column leach testwork were utilized for head screen analyses with assays by size fraction. The results of the head analyses for gold and silver are summarized in Table 13.6.

Table 13.6
Summary of Head Analyses – Gold and Silver ss

KCA Sample No.	Description	Average Assay Au (g/t)	Average Assay Ag (g/t)	Weighted Average Head Assay Au (g/t)*	Weighted Average Head Assay Ag (g/t)*
72101	Latte - 583150mE Trench	1.11	0.62	1.21	0.5
72102	Latte - 583350mE Trench	0.93	0.62	0.90	0.5
72133	Latte Mine Block	9.62	0.89	9.64	0.36
72135	Latte Oxide West	1.15	0.62	1.01	0.39
72136	Latte Oxide East	1.02	0.41	1.01	0.21
72137	Latte 80% CN Soluble	2.28	0.62	2.03	0.57
72138	Latte 60% CN Soluble	1.16	0.41	1.24	0.32
72134	Supremo Mine Block	5.25	0.62	5.50	0.33
72142	Supremo T2-T4 Composite	1.70	0.41	1.91	0.32
72143	Supremo T3 Composite	2.56	0.31	2.33	0.32
72144	Supremo Oxide West (T1, T2, T3)	1.31	0.41	1.49	0.25
72145	Supremo Oxide East (T4, T5, T7)	1.81	0.41	1.69	0.25
72146	Supremo 80% CN Soluble	1.02	0.41	1.05	0.28
72139	Kona Oxide	1.37	0.21	1.43	0.18
72140	Kona 80% CN Soluble	1.32	0.21	1.34	0.15
72141	Double Double Oxide	3.08	0.41	3.18	2.32

Source: KCA (2015)

Notes: *Average of all head assay screen analyses.

As shown in Table 13.6, silver grades in the head samples are low, and any silver production will not likely be economically significant.

Head samples were also analyzed for carbon and sulphur content. In addition to total carbon and sulphur, speciation analyses were completed for organic and inorganic carbon, as well as for sulphide and sulphate sulphur. Head analyses were also performed for mercury and copper, along with semi-quantitative analyses of selected individual elements and whole-rock constituents. Detailed analytical procedures and results are provided in the KCA 2015 report.

No problematic amounts of deleterious elements were discovered during the overall head analyses.

13.3.2 Bottle Roll Leach Testwork

Bottle roll leach testing was conducted on a portion of material from each sample. For each test, a 1,200 g portion of head material was ring and puck pulverized to a target size of 80% passing 0.075 mm. A 1,000 g



portion of the pulverized material was then utilized for a 96-hour leach test conducted, with the cyanide concentration maintained at a target concentration of 1.0 g/L throughout the test.

The results of the bottle roll leach testwork are summarized in Table 13.7. Except for the transition samples (60/80% CN soluble), which had gold extraction rates of 61% and 86%, column leach tests showed excellent gold recoveries of mid- to high-90%. Silver recovery ranged from 28% to 62%.

**Table 13.7
Summary of Bottle Roll Leach Tests**

KCA Sample No.	Description	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
				NaCN (kg/t)	Lime Addition (kg/t)
72101	Latte - 583150mE Trench	1.08	94	0.18	3.0
72102	Latte - 583350mE Trench	0.87	96	0.08	3.0
72133	Latte Mine Block	8.88	94	0.08	3.5
72135	Latte Oxide West	1.13	95	0.08	2.50
72136	Latte Oxide East	0.98	95	0.06	2.
72137	Latte 80% CN Soluble	2.00	86	0.15	2.5
72138	Latte 60% CN Soluble	1.18	61	0.16	2.5
72142	Supremo T2-T4 Composite	1.66	97	0.01	3.0
72143	Supremo T3 Composite	2.40	97	0.13	2.0
72134	Supremo Mine Block	5.02	98	0.03	2.8
72144	Supremo Oxide West	1.24	96	0.02	2.0
72145	Supremo Oxide East	1.78	97	0.10	1.8
72146	Supremo 80% CN Soluble	0.99	86	0.12	2.0
72139	Kona Oxide	1.23	90	0.31	2.5
72140	Kona 80% CN Soluble	1.31	81	1.33	2.8
72141	Double Double Oxide	2.64	96	0.15	2.5

Source: KCA (2015)

The carbon and sulphur assays for the Latte 80% CN soluble, Latte 60% CN soluble, Supremo 80% CN soluble and Kona 80% CN soluble samples were analyzed along with the gold extractions from the cyanide shake tests and the bottle roll leach tests. The organic carbon levels did not have an effect on the leachability of the material. However, the sulphide sulphur level trended more closely with the leachability of the material. An increase of the sulphide sulphur level led to a decrease in the gold extraction in both the cyanide shake tests and the bottle roll leach tests.

13.3.3 Column Leach Testwork

Column leach tests were performed on Latte and Supremo materials, which were crushed to sizes with 100% passing at 175 mm, 62.5 mm, or 16.0 mm. These samples were then leached in columns for 45 to 81 days using a sodium cyanide solution. For the remaining material types, column leach tests were conducted after crushing to 100% passing 62.5 mm or 16.0 mm, with all tests leaching 81 days.

All column leach tests utilizing material crushed to 100% passing 16 mm were run inside walk-in refrigerators in order to simulate heap leach conditions in the winter (4°C).



For the Latte material, gold extractions ranged from 54% (Latte 60% CN Soluble mineralization) to 96% for oxide mineralization based on calculated heads, which ranged from 0.89 to 9.54 g/t Au. The sodium cyanide consumptions ranged from 0.17 to 0.87 kg/t. The material utilized in leaching was blended with 0.98 to 1.53 kg/t hydrated lime.

For the Supremo material, gold extractions ranged from 81% (Supremo 80% CN Soluble) to 98% for oxide samples based on calculated heads, which ranged from 0.93 to 5.55 g/t Au. The sodium cyanide consumptions ranged from 0.29 to 0.96 kg/t. The material utilized in leaching was blended with 0.99 to 1.59 kg/t hydrated lime.

For the remaining material, gold extractions ranged from 72% (Kona 80% CN Soluble) to 95% for oxide samples based on calculated heads, which ranged from 1.35 to 4.33 g/t Au. The sodium cyanide consumptions ranged from 0.32 to 0.87 kg/t. The material utilized in leaching was blended with 1.53 to 2.06 kg/t hydrated lime.

A summary of the column leach test extractions is presented in Table 13.8.

Table 13.8
Summary of Column Leach Tests

KCA Sample No.	Description	Crush Size (mm)	Calculated Head Au (g/t)	Extraction Au (%)	Days of Leach	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
72101	Latte - 583150mE Trench	175	1.24	92	68	0.40	0.99
72101	Latte - 583150mE Trench	62.5	1.36	92	46	0.50	1.01
72101	Latte - 583150mE Trench	16	1.20	93	46	0.54	1.00
72102	Latte - 583350mE Trench	175	1.14	85	68	0.41	0.99
72102	Latte - 583350mE Trench	62.5	0.89	93	46	0.55	1.01
72102	Latte - 583350mE Trench	16	0.97	96	46	0.55	0.98
72133	Latte Mine Block	16	9.54	95	67	0.58	0.99
72135	Latte Oxide West	62.5	1.17	95	45	0.48	1.52
72135	Latte Oxide West	16	1.12	95	45	0.17	1.53
72136	Latte Oxide East	62.5	1.03	93	81	0.87	1.53
72136	Latte Oxide East	16	0.96	94	81	0.30	1.52
72137	Latte 80% CN Soluble	62.5	1.46	82	81	0.79	1.49
72137	Latte 80% CN Soluble	16	1.93	83	81	0.44	1.51
72138	Latte 60% CN Soluble	62.5	1.47	63	81	0.76	1.52
72138	Latte 60% CN Soluble	16	1.06	54	81	0.30	1.50



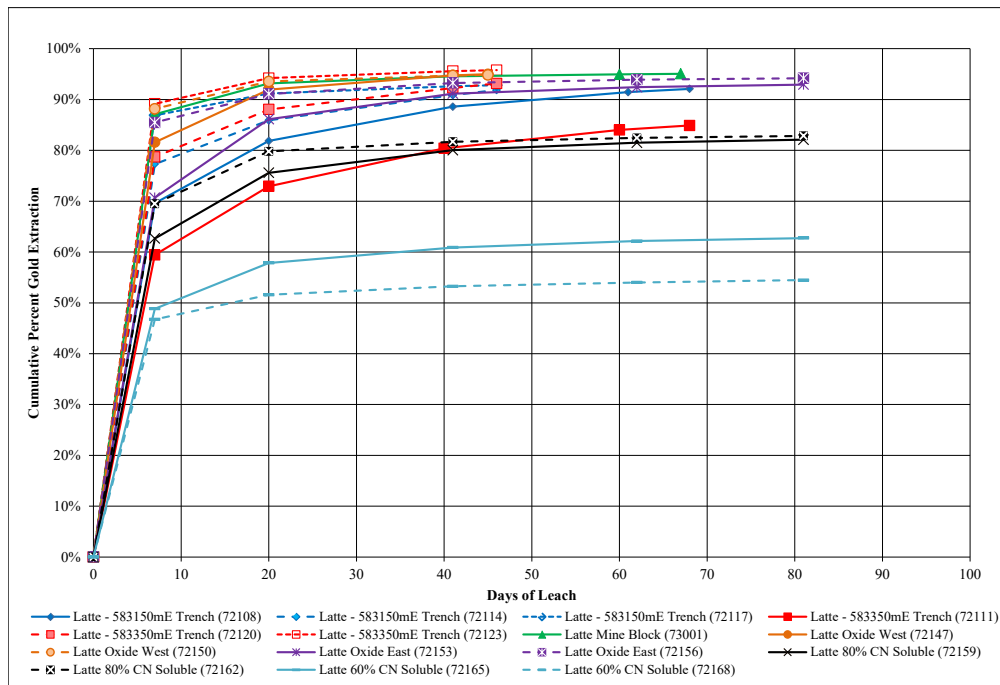
KCA Sample No.	Description	Crush Size (mm)	Calculated Head Au (g/t)	Extraction Au (%)	Days of Leach	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
72134	Supremo Mine Block	16	5.55	98	67	0.68	0.99
72142	Supremo T2-T4 Composite	175	2.10	92	67	0.53	1.49
72142	Supremo T2-T4 Composite	62.5	2.05	93	67	0.96	1.51
72142	Supremo T2-T4 Composite	16	1.90	97	67	0.59	1.49
72143	Supremo T3 Composite	175	2.86	82	67	0.46	1.51
72143	Supremo T3 Composite	62.5	2.28	93	67	0.72	1.59
72143	Supremo T3 Composite	16	2.26	95	67	0.29	1.51
72144	Supremo Oxide West (T1, T2, T3)	62.5	1.31	90	67	0.49	1.51
72144	Supremo Oxide West (T1, T2, T3)	16	1.28	95	67	0.84	1.50
72145	Supremo Oxide East (T4, T5, T7)	62.5	1.67	95	67	0.74	1.53
72145	Supremo Oxide East (T4, T5, T7)	16	1.78	96	67	0.32	1.49
72146	Supremo 80% CN Soluble	62.5	1.04	85	67	0.61	1.50
72146	Supremo 80% CN Soluble	16	0.93	81	67	0.43	1.20
72139	Kona Oxide	62.5	1.42	88	81	0.82	1.56
72139	Kona Oxide	16	1.36	91	81	0.37	2.06
72140	Kona 80% CN Soluble	62.5	1.36	72	81	0.76	1.53
72140	Kona 80% CN Soluble	16	1.35	76	81	0.50	2.01
72141	Double-Double Oxide	62.5	4.33	95	81	0.87	1.56
72141	Double-Double Oxide	16	2.99	95	81	0.32	2.02

Source: KCA (2015)

Kinetic gold extraction curves are shown in Figure 13.3, Figure 13.4 and Figure 13.5 for Latte samples, Supremo samples, and Kona and Double-double samples, respectively. Note that certain tests did not reach the full 81-day leaching period.



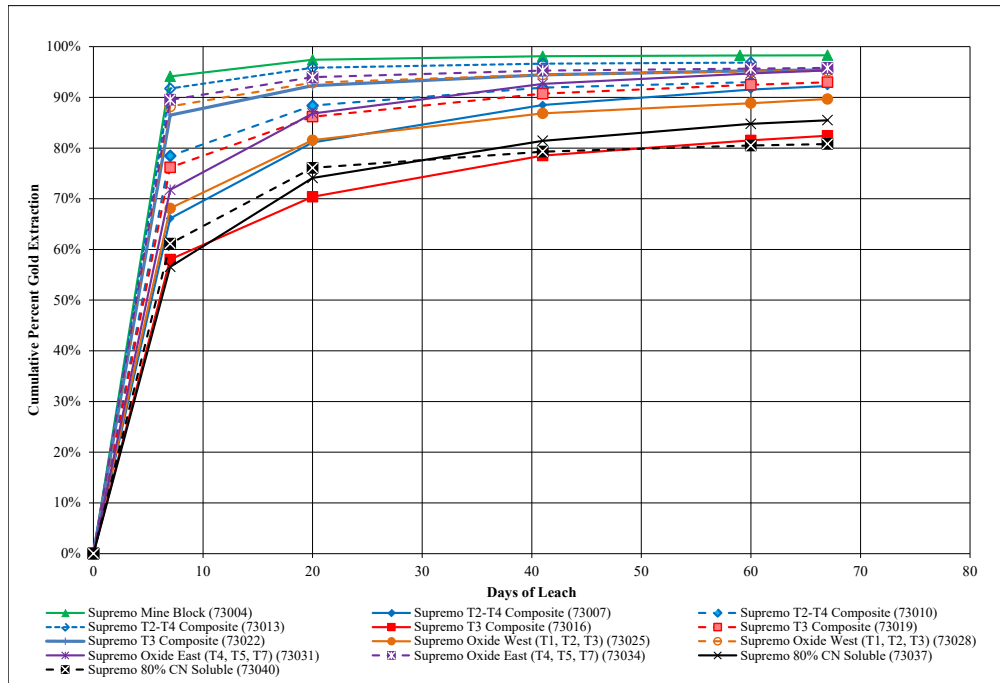
Figure 13.3
Latte Samples: Gold Extraction versus Days of Leach



Source: KCA (2015)

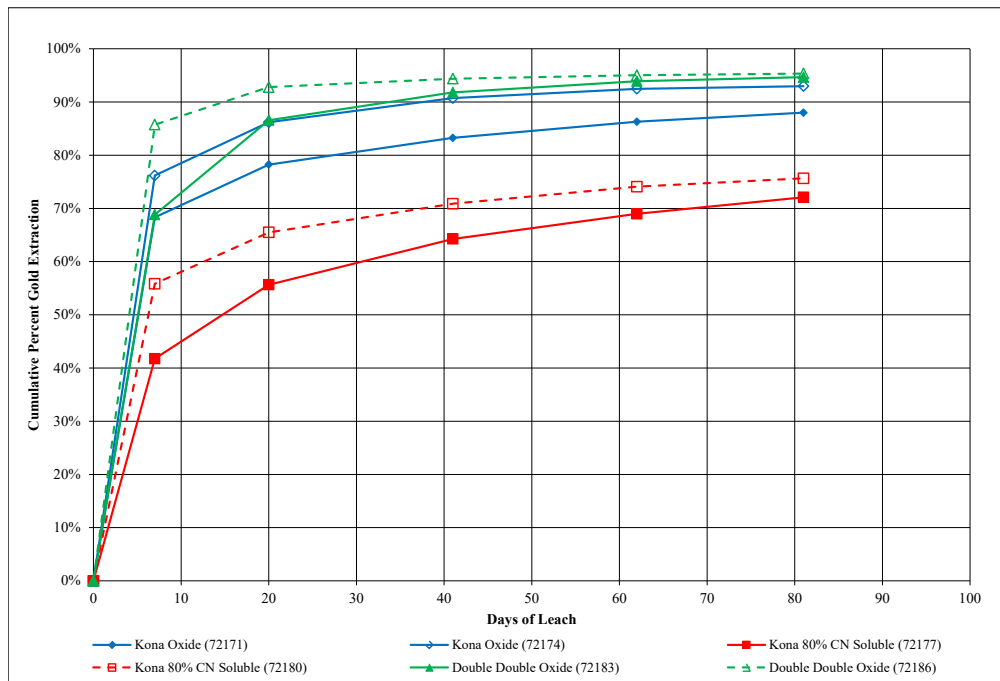


Figure 13.4
Supremo Samples: Gold Extraction versus Days of Leach



Source: KCA (2015)

Figure 13.5
Kona and Double-Double Samples: Gold Extraction versus Days of Leach



Source: KCA (2015)



Silver recoveries ranged from 13% to 63%. The height, slump and final apparent bulk density from the column leach tests were all favourable and it was determined that no agglomeration would be necessary.

The temperature of the test did not have any material effect on the rate of gold extraction or the overall extent of gold extraction. The cyanide consumptions also were not affected by the low temperature.

For the Coffee material, KCA recommended discounting laboratory gold extractions by three percentage points when estimating field extractions, assuming a well-managed heap leach operation. According to KCA's experience with mostly clean non-reactive gold bearing material, cyanide consumption for the Coffee material would be only 25% of the laboratory column test consumptions. KCA also noted that in practice, both heap leach recovery and cyanide consumption may be impacted by various operational factors, and actual performance could differ from laboratory predictions.

13.4 Recent Metallurgical Testwork (2018-2020, KCA)

Between November 2018 and June 2020, KCA carried out metallurgical testing for Goldcorp Kaminak Ltd on the Coffee Gold deposit, producing at least eight reports. Report I.Ds for these eight reports are: KCA0180004_COF09_01, KCA0180022_COF10_01, KCA0190003_COF11_01, CA0190019_COF12_01, KCA0190022_COF14_04, KCA0190023_COF15_01, KCA0190043_COF16_01, and KCA0190079_COF17_01.

The documented tests undertaken during this phase were wide-ranging and included size fraction analysis, mineralogy, comminution testing, preg-robbing assessments, bottle roll leach testing, compacted permeability testing, and column leach testing.

13.4.1 Sampling

In March 2018, KCA received 32 bags of ½ core material from the Coffee Gold Project, which was utilized for six composites from areas including Supremo T8-9, Supremiato T3N, and Arabica.

In January 2019, KCA received five wooden crates of ½ core material from the Coffee Gold Project. The material was compiled into composite samples representing the AmeriKona (Kona), Latte, and Supremo areas, and included oxide, transition, and waste (unmineralized) material. From this sample receipt, two Kona composites were prepared for the COF12_01 test program while nine composites representing the Supremo and Latte areas were generated for the COF11_01 testwork program.

Additional composite samples were prepared using reject material from previous KCA metallurgical test programs conducted on the Coffee Gold Project between 2013 and 2015.

13.4.2 Head Analyses

The results of the direct head assays for gold and silver and the average weighted head screen analyses for gold are presented in Table 13.9.



**Table 13.9
Summary of Head Analyses – Gold and Silver**

KCA Report ID	Description	Date Received	Average Assay Au (g/t)	Average Assay Ag (g/t)	Weighted Average Head Assay Au (g/t)	Weighted Average Head Assay Ag (g/t)
COF09_01	KGC-17-1703, Supremo T8-9 Composite #1	March 2018	1.03	0.55	0.84	0.42
COF09_01	KGC-17-1704, Supremo T8-9 Composite #2	March 2018	0.92	0.62	0.69	0.41
COF09_01	KGC-17-1705, Supremo T3N Composite #1	March 2018	0.63	0.48	0.79	0.30
COF09_01	KGC-17-1706, Supremo T3N Composite #2	March 2018	0.49	0.41	0.65	0.24
COF09_01	KGC-17-1707, Arabica Composite #1	March 2018	0.43	0.41	0.70	0.35
COF09_01	KGC-17-1708, Arabica Composite #2	March 2018	0.75	0.48	0.70	0.34
COF10_01	Latte 583250ME	2013, 2014	1.61	0.70	-	-
COF10_01	Supremo 6974250MN	2013, 2014	4.08	0.62	-	-
COF10_01	Latte 583150mE Trench	2013, 2014	1.11	0.62		
COF10_01	Supremo Oxide West (T1, T2, T3)	2013, 2014	0.93	0.62		
COF11_01	Latte Comp East Oxide Nov 2018	Jan 2019	-	-	1.41	0.61
COF11_01	Latte Comp West Oxide Nov 2018	Jan 2019	-	-	1.20	0.54
COF11_01	Latte Upper Trans Comp Nov 2018	Jan 2019	-	-	1.09	0.67
COF11_01	Latte Waste Comp Nov 2018	Jan 2019	-	-	0.02	0.47
COF11_01	Supremo T3 North Oxide Comp Nov 2018	Jan 2019	-	-	0.78	0.74
COF11_01	Supremo T3 Central Ox-Comp Nov 2018	Jan 2019	-	-	0.61	0.35
COF11_01	Supremo T3 South Oxide Comp Nov 2018	Jan 2019	-	-	0.96	0.39
COF11_01	Supremo T3 Upper Trans Nov 2018	Jan 2019	-	-	0.48	0.38
COF11_01	Supremo T3 Waste Comp Nov 2018	Jan 2019	-	-	0.01	0.15
COF12_01	Amerikona Composite 1	Jan 2019	1.04	0.21	-	-
COF12_01	Amerikona Composite 2	Jan 2019	1.19	0.27	-	-
COF14_01	PLS Composite - Lorax	Various	1.52	0.57	2.06	0.41
COF15_01	Supremo Oxide, 2014	Dec 2014	5.03	-	4.64	-
COF16_01	Supremo T5 Central Oxide 2019	June 2019	1.84	1.61	1.65	0.67
COF16_01	Supremo T5 North Oxide 2019	June 2019	2.61	0.86	1.98	0.44
COF16_01	Supremo T5 Upper Trans 2019	June 2019	1.19	0.24	1.37	0.32
COF16_01	Supremo T5 South Oxide 2019	June 2019	2.34	0.72	2.39	0.59
COF16_01	Supremo T5 Waste 2019	June 2019	0.009	0.14	-	-

Source: KCA (2018, 2020)



13.4.3 Bottle Roll Leach Testwork

As part of the 2018 CFO09_01 program, bottle roll leach testing was performed on twelve composite samples representing Supremo, Supremiatio, and Arabica mineralization, including oxide and transition material. Tests were conducted at varying crush sizes and leach times, with results presented in Table 13.10.

Table 13.10
Summary of Bottle Roll Leach Tests from KCA0180004_CO09_01

Description	Material Type	Feed Size P ₁₀₀ (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo T8-9 Composite #1	Oxide	25	240	0.52	77	0.02	0.75
Supremo T8-9 Composite #1	Oxide	0.15	96	1.04	98	0.25	1.76
Supremo T8-9 Composite #2	Upper Transition	25	240	0.47	84	0.03	0.50
Supremo T8-9 Composite #2	Upper Transition	0.15	96	0.91	96	0.17	1.52
Supremiatio T3N Composite #1	Upper Transition	25	240	0.78	73	0.06	0.50
Supremiatio T3N Composite #1	Upper Transition	0.15	96	0.62	95	0.19	1.76
Supremiatio T3N Composite #2	Oxide	25	240	0.53	79	0.02	0.50
Supremiatio T3N Composite #2	Oxide	0.15	96	0.44	94	0.22	1.50
Arabica Composite #1	Oxide	25	240	0.39	76	0.04	0.50
Arabica Composite #1	Oxide	0.15	96	0.33	94	0.21	1.26
Arabica Composite #2	Oxide	25	240	0.52	83	0.08	0.75
Arabica Composite #2	Oxide	0.15	96	0.69	96	0.21	1.52

Source: KCA (2018)

Bottle roll leach tests were conducted on Supremo and Latte sample material in the CFO10_01 program in 2018. The results are summarized in Table 13.11.



Table 13.11
Summary of Bottle Roll Leach Tests from KCA0180004_CO10_01

Description	Material Type	Feed Size P ₁₀₀ (mm)	Fine Screen Fraction (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
							NaCN (kg/t)	Lime Addition (kg/t)
Latte 583250 ME		31.5	+6.3	240	1.09	85	0.04	0.75
Latte 583250 ME		31.5	-6.3	240	1.45	92	0.13	1.50
Latte 583250 ME		16	+6.3	240	1.82	87	0.10	1.00
Latte 583250 ME		16	-6.3	240	1.71	93	0.22	1.25
Latte 583250 ME		4	-	240	1.26	92	0.15	1.25
Latte 583250 ME		1.7	-	240	1.01	96	0.31	1.26
Supremo 6974250 MN		31.5	+6.3	240	3.29	75	0.06	0.50
Supremo 6974250 MN		31.5	-6.3	240	3.54	92	0.18	1.75
Supremo 6974250 MN		16	+6.3	240	4.19	85	0.16	0.75
Supremo 6974250 MN		16	-6.3	240	3.70	93	0.16	1.75
Supremo 6974250 MN		4	-	240	3.30	92	0.15	1.51
Supremo 6974250 MN		1.7	-	240	2.54	94	0.33	1.50
Latte 583150 mE Trench	Oxide	62.5	+6.3	240	1.16	92	0.12	0.50
Latte 583150 mE Trench	Oxide	62.5	-6.3	240	1.62	94	0.17	1.50
Latte 583150 mE Trench	Oxide	16	+6.3	240	0.94	94	0.16	0.75
Latte 583150 mE Trench	Oxide	16	-6.3	240	1.39	91	0.20	1.25
Latte 583150 mE Trench	Oxide	1.7	-	240	1.27	92	0.43	1.26
Supremo Oxide West (T1, T2, T3)	Oxide	62.5	+6.3	240	0.98	81	0.07	0.50
Supremo Oxide West (T1, T2, T3)	Oxide	62.5	-6.3	240	3.25	95	0.18	1.50
Supremo Oxide West (T1, T2, T3)	Oxide	31.5	+6.3	240	0.91	79	0.01	0.50
Supremo Oxide West (T1, T2, T3)	Oxide	31.5	-6.3	240	2.85	96	0.17	1.50
Supremo Oxide West (T1, T2, T3)	Oxide	16	+6.3	240	0.76	92	0.08	0.75
Supremo Oxide West (T1, T2, T3)	Oxide	16	-6.3	240	1.62	95	0.20	1.00
Supremo Oxide West (T1, T2, T3)	Oxide	4	-	240	1.30	94	0.21	1.00
Supremo Oxide West (T1, T2, T3)	Oxide	1.7	-	240	1.39	96	0.37	1.25

Source: KCA (2018)



For the 2019/20 CFO11_01 program, bottle roll leach testing was conducted on samples from the Latte and Supremo areas. The testwork results are summarized in Table 13.12.

Table 13.12
Summary of Bottle Roll Leach Tests from KCA0190003_COF11_01

Description	Material Type	Feed Size P ₈₀ (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted %	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Latte Comp East Nov 2018	Oxide	19.0	240	1.28	88	0.29	0.75
Latte Comp East Nov 2018	Oxide	5.04	240	1.48	81	0.17	1.50
Latte Comp East Nov 2018	Oxide	0.06	96	1.45	87	0.16	2.00
Latte Comp West Nov 2018	Oxide	19.0	240	1.36	80	0.09	1.25
Latte Comp West Nov 2018	Oxide	5.04	240	1.30	91	0.14	1.25
Latte Comp West Nov 2018	Oxide	0.06	96	1.25	88	0.21	2.25
Latte Comp Nov 2018	Upper Transition	19.0	240	1.12	58	0.15	0.75
Latte Comp Nov 2018	Upper Transition	5.04	240	1.16	63	0.12	1.25
Latte Comp Nov 2018	Upper Transition	0.06	96	1.11	74	0.33	1.75
Supremo T3 North Comp Nov 2018	Oxide	19.0	240	0.88	79	0.06	0.75
Supremo T3 North Comp Nov 2018	Oxide	5.04	240	1.50	89	0.11	1.25
Supremo T3 North Comp Nov 2018	Oxide	5.04	240	1.29	89	0.11	1.25
Supremo T3 North Comp Nov 2018	Oxide	0.06	96	0.72	93	0.21	1.25
Supremo T3 Central Comp Nov 2018	Oxide	19.0	240	0.68	94	0.06	1.00
Supremo T3 Central Comp Nov 2018	Oxide	5.04	240	0.60	93	0.11	1.25
Supremo T3 Central Comp Nov 2018	Oxide	0.06	96	0.70	96	0.20	1.75

Source: KCA (2020)

In the CFO12_01 program, bottle roll leach testing was conducted on the two Amerikona samples. The results of the bottle roll leach testwork are summarized below in Table 13.13.



Table 13.13
Summary of Bottle Roll Leach Tests from KCA0190019_COF12_01

Description	Material Type	Feed Size P ₁₀₀ (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Amerikona Composite 1	Oxide	0.15	96	1.03	88	1.36	1.50
Amerikona Composite 2	Oxide	0.15	96	1.21	89	1.65	1.50

Source: KCA (2020)

The 2020 Bulk Pregnant Leach Solution generation program, CFO14_01, conducted a single bottle roll leach test on the sample consisting of Supremo, Latte, Double Double, and Kona Oxide and transition reject sample material from past test programs. The summary of the bottle roll test is shown in Table 13.14 below.

Table 13.14
Summary of Bottle Roll Leach Tests from KCA0190022_COF14_04

Description	Feed Size P ₈₀ (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
					NaCN (kg/t)	Lime Addition (kg/t)
PLS Generation Composite	0.075	96	1.53	90	0.21	1.75

Source: KCA (2020)

As part of the 2019/20 CFO15_01 program, a single bottle roll leach test was conducted on the Supremo Oxide material. The results of the test are summarized below in Table 13.15.

Table 13.15
Summary of Bottle Roll Leach Tests from KCA0190023_COF15_01

Description	Feed Size P ₈₀ (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
					NaCN (kg/t)	Lime Addition (kg/t)
Supremo Oxide, 2014	0.075	96	4.39	94	0.13	2.00

Source: KCA (2020)

A series of bottle roll leach tests was undertaken on five Supremo T5 composite samples representing oxide and upper transition during the 2019/2020 CFO16_01 metallurgical program. Test results are provided in Table 13.16.



Table 13.16
Summary of Bottle Roll Leach Tests from KCA0190043_COF16_01

Description	Material Type	Feed Size P ₈₀ (mm)	Leach Time (h)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo T5 Central Oxide 2019	Oxide	25	384	1.55	91	0.06	1.00
Supremo T5 Central Oxide 2019	Oxide	6.3	384	1.81	89	0.16	1.15
Supremo T5 Central Oxide 2019	Oxide	0.106	96	1.68	95	0.23	1.50
Supremo T5 North Oxide 2019	Oxide	25	384	2.56	94	0.09	1.05
Supremo T5 North Oxide 2019	Oxide	6.3	384	2.49	89	0.16	1.30
Supremo T5 North Oxide 2019	Oxide	0.106	96	2.56	96	0.09	1.75
Supremo T5 Upper Trans 2019	Upper Transition	25	384	1.27	79	0.13	1.05
Supremo T5 Upper Trans 2019	Upper Transition	6.3	384	1.34	80	0.30	1.20
Supremo T5 Upper Trans 2019	Upper Transition	0.106	96	1.19	82	0.20	2.00
Supremo T5 South Oxide 2019	Oxide	25	384	2.31	88	0.16	1.35
Supremo T5 South Oxide 2019	Oxide	6.3	384	2.42	89	0.24	1.55
Supremo T5 South Oxide 2019	Oxide	0.106	96	2.53	95	0.28	2.25

Source: KCA (2020)

13.4.4 Compacted Permeability Testwork

KCA conducted compacted permeability testing to simulate heap percolation rates at various heights under compressive loads. These test results can be found in the KCA reports KCA0190003_COF11_01 and KCA0190043_COF16_01.

The permeability tests were conducted on the material at a crush size of 100% passing 62.5 mm. Separate test samples were loaded into a column and subjected to loads equivalent to 10, 40 and 80 m of the overall heap height (assuming a heap density equivalent to 1.8 t/m³).

The overall results of the tests are then examined to determine pass or failure. The flow rate, slump, pellet breakdown and solution color and clarity are all monitored to provide meaningful indications and to help judge what represents a “Pass” or “Fail”.

One non-agglomerated sample (Supremo T3 Waste Comp Nov 2018) passed all KCA criteria. All remaining samples passed at the 10 and 40 m equivalent heights but failed at the 80 m equivalent height due to excessive slump.

13.4.5 Column Leach Testwork

Column testing was carried out during the most recent phase of KCA testwork, with the results summarized in Table 13.17 to Table 13.23 below.



Table 13.17
Summary of Column Leach Tests from KCA0180004_CO09_01

Description	Material Type	Size P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo T8-9 Composite #1	Oxide	50.1	135	0.98	89	2.32	0.75
Supremo T8-9 Composite #2	Transition	50.7	135	0.87	94	2.06	0.76
Supremiato T3N Composite #1	Transition	52.5	72	0.97	94	1.22	0.75
Supremiato T3N Composite #2	Oxide	49.6	72	0.82	97	1.23	0.74
Arabica Composite #1	Oxide	49.1	72	0.73	96	1.59	0.86
Arabica Composite #2	Oxide	50.4	72	0.84	96	1.21	0.75

Source: KCA (2018)

Table 13.18
Summary of Column Leach Tests from KCA0190003_COF11_01

Description	Material Type	Size P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Latte Comp East Nov 2018	Oxide	45.7	57	1.45	91	0.82	1.01
Latte Comp West Nov 2018	Oxide	37.6	57	1.40	86	0.70	1.01
Latte Comp Nov 2018	Upper Transition	43.7	57	1.08	76	0.60	1.04
Supremo T3 North Comp Nov 2018	Oxide	47.6	57	0.93	85	0.70	1.00
Supremo T3 Central Comp Nov 2018	Oxide	44.8	57	0.77	93	0.80	1.02

Source: KCA (2020)



Table 13.19
Summary of Column Leach Tests from KCA0190019_COF12_01

Description	Material Type	Crush Size (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Amerikona Composite 1	Oxide	62.5	60	1.23	88	1.35	1.53
Amerikona Composite 2	Oxide	62.5	60	1.18	90	1.33	1.55

Source: KCA (2020)

Table 13.20
Summary of Column Leach Tests from KCA0190022_COF14_04

Description	Material Type	Size P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
PLS Composite - Lorax	Various	32.2	78	1.69	92	1.69	1.75

Source: KCA (2020)

Table 13.21
Summary of Column Leach Tests from KCA0190023_COF15_01

Description	Material Type	Size P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo, Low NaCN	Oxide	23.9	129	4.32	93	2.37	1.62
Supremo, Med NaCN	Oxide	24.4	129	4.06	93	1.61	1.52
Supremo, High NaCN	Oxide	24.7	129	4.42	94	1.85	2.00

Source: KCA (2020)



Table 13.22
Summary of Column Leach Tests from KCA0190043_COF16_01

Description	Material Type	Column Diameter (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo T5 Central 2019	Oxide	200	70	1.81	92	1.07	1.26
Supremo T5 Central 2019	Oxide	150	70	1.95	95	1.55	1.28
Supremo T5 North 2019	Oxide	200	78	2.76	96	1.07	1.26
Supremo T5 North 2019	Oxide	150	78	2.80	93	1.67	1.27
Supremo T5 2019	Upper Transition	200	70	1.33	81	1.01	1.29
Supremo T5 2019	Upper Transition	150	70	1.37	80	1.83	1.26
Supremo T5 South 2019	Oxide	200	70	2.44	95	1.13	1.53
Supremo T5 South 2019	Oxide	150	70	2.64	97	1.55	1.56

Source: KCA (2020)

Table 13.23
Summary of Column Leach Tests from KCA0190079_COF17_01

Description	Material Type	P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo GM-01	Oxide	57.2	72	0.25	89	0.75	1.51
Supremo GM-02	Oxide	48.6	72	5.02	48	1.06	1.55
Supremo GM-03	Oxide	49	72	0.72	71	0.8	1.53
Supremo GM-04	Upper Trans	43.1	72	0.41	79	0.69	1.5
Supremo GM-05	Oxide	49.2	72	0.64	96	1.09	1.52
Supremo GM-06	Oxide	44.9	72	1.23	97	0.99	1.69
Supremo GM-07	Upper Trans	49	72	2.79	73	0.69	1.51
Supremo GM-08	Oxide	47.9	72	0.68	96	0.79	1.53
Supremo GM-09	Oxide	49.4	72	0.67	96	0.77	1.51
Supremo GM-10	Mid. Trans	44.5	72	0.37	53	0.85	1.53
Supremo GM-11	Oxide	47.5	72	1.42	98	0.79	1.55
Supremo GM-12	Oxide	49.9	72	1.22	95	1.03	1.54
Supremo GM-13	Oxide	49.8	72	0.19	91	0.54	1.5
Supremo GM-14	Oxide	42.6	72	0.51	96	0.96	1.57
Supremo GM-15	Lower Trans	36.3	72	0.64	23	0.67	1.5



Description	Material Type	P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
Supremo GM-16	Upper Trans	40.8	71	0.34	84	0.79	1.56
Supremo GM-17	Upper Trans	42.3	71	0.84	87	0.79	1.54
Supremo GM-18	Mid. Trans	45	71	0.48	48	0.61	1.51
Supremo GM-19	Oxide	42.6	71	0.34	89	0.72	1.52
Supremo GM-20	Upper Trans	44.7	71	0.89	71	0.79	1.52
Supremo GM-21	Mid. Trans	43.2	71	1.59	55	0.91	1.53
Supremo GM-22	Lower Trans	36.6	71	0.26	94	1.17	1.64
Supremo GM-23	Upper Trans	55.8	71	0.89	92	0.65	1.5
Supremo GM-24	Lower Trans	44.9	71	2.14	14	0.77	1.51
Supremo GM-25	Oxide	50.2	71	0.30	94	0.73	1.52
Supremo GM-26	Oxide	36	71	4.11	99	1.05	1.55
Latte GM-27	Upper Trans	46.6	71	0.35	96	0.85	1.57
Latte GM-28	Mid. Trans	41.1	71	0.82	53	1.06	1.53
Latte GM-29	Lower Trans	40.8	71	0.20	58	0.92	1.52
Latte GM-30	Lower Trans	48	71	1.35	20	0.84	1.52
Latte GM-31	Upper Trans	48	70	0.79	68	1.11	1.53
Latte GM-33	Lower Trans	45.2	70	1.01	40	1.44	1.65
Latte GM-34	Lower Trans	38.7	70	0.5	91	1.46	1.55
Latte GM-35	Lower Trans	44.4	70	0.6	44	1.36	1.53
Latte GM-36	Lower Trans	44.8	70	0.82	34	1.38	1.55
Latte GM-37	Mid. Trans	41.1	70	0.31	46	1.28	1.54
Latte GM-38	Lower Trans	35.6	70	1.21	50	1.37	1.55
Latte GM-39	Lower Trans	51.3	70	2.23	27	1.17	1.53
D/Double GM-46	Mid. Trans	51.8	70	0.3	70	1.33	1.61
D/Double GM-47	Oxide	50.6	70	0.21	93	1.11	1.52



Description	Material Type	P ₈₀ (mm)	Leach Time (days)	Calculated Head Au (g/t)	Au Extracted (%)	Reagent Consumption	
						NaCN (kg/t)	Lime Addition (kg/t)
D/Double GM-48	Lower Trans	38.8	70	1.85	84	1.36	1.52
D/Double GM-49	Mid. Trans	38.6	70	0.82	27	1.3	1.53
D/Double GM-50	Upper Trans	39.2	70	0.82	78	1.33	1.52
Latte GM-51	Mid. Trans	49.6	70	0.6	69	1.24	1.53
Latte GM-52	Lower Trans	40.9	70	1.04	58	1.39	1.54
Latte GM-53	Mid. Trans	48.1	70	0.76	85	1.32	1.53
Latte GM-55	Oxide	44.3	70	1.08	97	1.5	1.53

Source: KCA (2020)

13.5 Recent Testwork (2020-2021)

SGS was retained by Newmont to perform metallurgical testwork on material from the Coffee Gold deposit. The resulting SGS reports do not consistently identify the specific sample areas (Latte, Supremo, Double-Double, Kona) and oxidation levels are not explicitly reported; however, oxidation can be inferred from the detailed assay results for each sample.

SGS released five reports during 2020 and 2021:

- An Investigation into Characterization and Comminution Testing of 2018 Full NQ Core and Split Core Samples from the Coffee Gold Project, June 2020, Prepared for: Newmont Corporation, Prepared by: SGS Canada Inc., Burnaby, BC V5A 4W4, Report I.D.: 17176-001;
- An Investigation into Chemical Analysis and Characterization of HQ Core Samples from the Coffee Project, June 2020, Prepared for: Newmont Corporation, Prepared by: SGS Canada Inc., Burnaby, BC V5A 4W4, Report I.D.: 17176-002;
- An Investigation into Characterization, Comminution, and Metallurgical Testing on 2018 Composite Samples from the Coffee Project, July 2020, Prepared for: Newmont, Prepared by: SGS Canada Inc., Burnaby, BC V5A 4W4, Report I.D.: 17176-003;
- An Investigation into Characterization, Comminution, and Metallurgical Testing on Geometallurgical Variability Samples from the Coffee Project, July 2020, Prepared for: Newmont Corporation, Prepared by: SGS Canada Inc., Burnaby, BC V5A 4W4, Report I.D.: 17176-004; and
- An Investigation into Characterization, Comminution, and Metallurgical Testing on Geometallurgical Variability Samples from the Coffee Project, September 2027, Prepared for: Newmont Corporation, Prepared by: SGS Canada Inc., Burnaby, BC V5A 4W4, Report I.D.: 17176-005.

13.5.1 Characterization and Comminution Testing (Report No. 17176-001, Jun-2020)

SGS received two sample shipments in 2019 for this test program. The first shipment consisted of 59 continuous whole drill core intervals from 6 m to 65 m depth from the drillhole CFD0888 while the second



shipment included 10 bulk blocks and 33 broken core samples. These samples were subjected to chemical analysis, mineralogy by Quantitative X-Ray Diffraction (QXRD) and comminution testing.

13.5.1.1 Whole NQ Core Samples

Based on the chemical analysis, the gold grades were in the range of <0.02 to 15.94 g/t, with an average grade of 1.46 g/t. The total carbon grades ranged from 0.01% to 3.60%, with an average grade of 0.84%; the carbon was predominantly presented as inorganic carbon and total organic carbon was low. The total sulphur grade varied from <0.005% to 0.50%, with an average grade of 0.07%; sulphur was predominantly presented as sulphide sulphur, and the sulphate sulphur was low.

XRD analysis showed that quartz was the predominant mineral, present in amounts ranging from 8.7% to 63.4%. Albite concentrations varied between 0.3% and 32.7%, while microcline (K-feldspar) ranged from 0.4% to 31%. Sulphides and sulphates were detected at low or trace levels. Kaolinite (clay) content was observed within the range of 0% to 27.6%.

The comminution test showed wide-ranging results: Cerchar Abrasivity index (CAI) values ranged from 0.69 to 3.77 (average 2.15); UCS ranged from 7.0 to 179.2 MPa (average 56.6 MPa); point-load indices (Is50) ranged from 0.28 to 7.15 MPa (average 2.28 MPa); and abrasion indices (Ai) ranged from 0.007 to 0.662 g (average 0.189 g).

13.5.1.2 Bulk Blocks and Broken Core Samples

Block 1, Block 3 and Block 17 had higher gold grades of 2.78 g/t, 8.88 g/t, and 5.31 g/t, respectively, though the bulk blocks and broken NQ cores samples grades were low (<0.02 g/t). The total carbon grades were in the range of <0.005% to 4.16%, with an average value of 0.58%; the carbon was predominantly presented as inorganic carbon and total organic carbon was low. The total sulphur grades were in the range of 0.01% to 0.05%; generally, the sulphur was predominantly sulphide sulphur, and the sulphate content was low.

Based on XRD analysis, actinolite (a fibrous amphibole silicate mineral), was present as a major mineral, ranging from 4.4% to 92.0%, while quartz varied between 2.8% and 79.9%. Albite content reached up to 55.2%, microcline ranged from 1.2% to 41.3%, and calcite was found up to 30.4%. Neither arsenopyrite nor pyrite was detected. Kaolinite (a type of clay) was observed in concentrations from 0% to 23.2%.

Bond abrasion tests were conducted on the bulk block samples only. The Bond abrasion indices (Ai) ranged from 0.050 to 0.761 g, with an average value of 0.334 g. They fell in the mild to very abrasive range of abrasiveness.

13.5.2 Chemical Analysis and Characterization (Report No. 17176-002, Jun-2020)

The primary objective of this testwork program was to characterize 40 half HQ core intervals and compare these findings against surface scanning results. Each sample underwent crushing to -2 mm, homogenization, and subsequent riffle-splitting of a 300 g subsample, which was then pulverized to achieve a target size of 80% passing (P_{80}) 75 μm . Residual -2 mm material was retained as reject. Comprehensive chemical analyses were performed, including gold assay by Fire Assay, sulphur speciation via LECO and ICP Multi-Acid Digestion, as well as XRD analysis using Rietveld Refinement techniques to thoroughly characterize the samples.

The results indicated substantial variability in mineral composition across the 40 samples. Gold head grades varied significantly, ranging from less than 0.02 g/t in 12 samples to 41.9 g/t in sample V086154. Sulphur speciation revealed percentages for all three forms of sulphur, with sulphide sulphur content below 0.05% and total sulphur reaching up to 3.32%, predominantly as sulphides.



All samples were subjected to XRD analysis to identify mineral constituents. Sulphides and sulphates were generally present in low or trace amounts; pyrite concentrations ranged from 0.2% to 3.1%, while marcasite was identified in samples V086163 and V086164 at values of 1.1% and 1.4%, respectively. Gypsum constituted up to 5.7% in approximately half the samples. Jarosite was detected in samples V086143 and V086162 at 1.5% and 1.0%, respectively. Kaolinite (clay) content identified in Sample V086159, spanning from 0.2% to 21.6% and averaging 4.4%. Montmorillonite (swelling clay) ranged from 0.2% in sample V086133 to 7.7% in sample V086165). The principal minerals included quartz (15.9% to 72.7%), albite (plagioclase) (0.2% to 24.4%), and microcline (K-feldspar) (1.5% to 24.7%).

13.5.3 Characterization and Comminution Testing on Composite Sample (Report No. 17176-003, June-2020)

The primary objective of this campaign was to conduct head characterization and comminution testing on 2018 composite samples. Nineteen composite samples (oxide, transition, and waste material types) were assessed, including six high-grade composites and three deep sulphide composites. The program included column leach testing on a subset of samples to evaluate heap leach gold recovery. Additionally, another subset comprising sulphide composites underwent gravity separation, flotation, and bottle roll cyanidation to determine mill amenability.

Bond abrasion, point-load, and Cerchar abrasivity tests were administered to all samples except the three deep sulphide composites. SMC (semi-autogenous mill comminution) and Bond ball mill grindability assessments were performed exclusively on the waste and Supremo T5 composites. Bond abrasion indices ranged from 0.005 to 0.330 g, classifying the samples as mildly to moderately abrasive. Cerchar abrasivity indices spanned 0.88 to 4.50, while point-load indices (Is50) varied from 0.47 to 7.00 MPa. A x b values ranged from 40.8 to 63.5, indicating the samples within a moderately soft to medium hardness category in terms of resistance to impact breakage. The Bond ball mill work indices were observed between 14.6 and 16.6 kWh/t, generally categorizing the samples as moderately hard.

Chemical analyses were carried out for the six high-grade composites and three deep sulphide composites. Gold head grades measured between 5.35 g/t and 22.0 g/t, and sulphide head grades ranged from below 0.05% to 2.03%.

XRD analysis by Rietveld Refinement identified quartz, muscovite, and kaolinite as the predominant minerals present in the six high-grade samples. Pyrite was detected at 1.4% in sample SUP-T3-UT (HG), while gypsum was identified at 2.4% in sample LAT-West Oxide (HG) and at 1% in SUP-T3-UT (HG); these constituted the only sulphide and sulphate minerals characterized in the tested samples.

Five column cyanidation tests conducted on NO-25, SO-25, CO-25, and UT-25 demonstrated high calculated gold extractions ranging from 87.1% to 94.5% in 6-inch diameter columns. Test NO-100 achieved a comparable gold extraction of 95.3% in an 8-inch diameter column, consistent with the results observed for Test NO-25.

Gold recoveries via gravity separation were modest, with Comp 1, Comp 2, and Comp 3 yielding 2.39%, 10.1%, and 14.0%, respectively.

Bottle roll cyanidation tests utilized both feed material and gravity tailings charges for each of the three composites. Comp 1, with a low sulphide grade of 0.34%, achieved high gold extraction rates exceeding 90%. Conversely, Comp 2 and Comp 3 (Deep Sulphide mineralization) yielded less than 44% gold recovery, indicating refractory behavior; gold extraction efficiencies decreased with increasing sulphide content.

Eight rougher flotation tests were executed, four each for Comp 2 and Comp 3. Calculated head grade for Comp 2 was 22.9 g/t and for Comp 3 was 5.3 g/t. Gold recoveries of 75.2% and 91.3% were attained at



mass pulls of 16.7% and 19.1%, respectively. Optimal results were achieved utilizing 100 g/t CuSO₄, 150 g/t PAX, and 40–45 g/t MIBC at an average primary grind size of 130 µm.

13.5.4 Characterization and Comminution Variability Testing on Geometallurgical Samples (Report No. 17176-004, Jul-2020)

Two batches of samples collected from the Coffee Gold project were analyzed. A total of 81 samples, comprising 34 GMCCR samples and 47 GM samples, were selected for this testing program. GMCCR and GM are sample identifications for the two batch of samples.

The results of the Cerchar abrasivity test on the forty-seven GM samples demonstrated considerable variability. Two samples were categorized as having low abrasivity, one as medium, while the remaining 43 samples exhibited highly to extremely abrasive characteristics. The outcomes of the Point Load Test and Bond abrasion test for these same 47 GM samples also showed significant variability. Unlike the Cerchar abrasivity test results, the Bond abrasion test indicated that all samples ranged from very mild to moderately abrasive.

Similarly, both the 47 GM samples and 33 GMCCR samples displayed substantial variation in response to the standard bottle roll cyanidation test. Gold recoveries ranged from approximately 10% (high sulphide sample) to over 98%. Poor recovery rates could not be attributed to preg-robbing, as organic carbon concentrations were consistently low across all samples. There was a notable inverse correlation between gold recovery and sulphide content, suggesting that unrecoverable gold was encapsulated within pyrite, which was present at concentrations of up to 2% in certain samples.

13.5.5 Characterization and Comminution Variability Testing on Geometallurgical Samples (Report No. 17176-005, Sep-2021)

A total of 56 samples (22 half NQ core and 34 RC reject) were used for geometallurgical and hardness characterization studies. Cerchar abrasivity tests on half NQ core samples showed mostly high or extreme abrasiveness, with only two rated as medium. Point Load and Bond abrasion tests also indicated variable results, with most samples classed from mild to medium abrasive.

Gold recovery via bottle roll cyanidation varied widely, from 2% (high sulphide sample) to over 98%, though very low recoveries (<5%) were rare and not linked to preg-robbing. Organic carbon was consistently low. An inverse correlation was observed between gold recovery and sulphide grade, suggesting unrecoverable gold was locked in pyrite, which reached up to 1.5% in some samples.

13.6 Metallurgical Analysis and Discussion

13.6.1 General Analysis

A number of test programs completed using samples of Coffee mineralization by reputable independent laboratories have been completed. The preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. Good metal accountability across a range of testwork results was observed and added confidence to the data review.

Extensive sampling and metallurgical testing have been conducted for Latte and Supremo zones. Sampling and testing were also conducted for Double-Double and Kona samples to a lesser extent. In the QP's opinion, the existing samples and testing overall are adequate for a PEA study and aligns with processing plans.

Gold extraction using cyanide leaching is consistently high for oxide samples and can achieve higher than 90%, while transition samples show lower rates (30–73%) and sulphide samples yield poor recovery (10%



or less), as expected. Cyanide consumption ranges from 0.4 to 1.0 kg/t, varying slightly with oxidation due to soluble species like iron and copper; these levels are reasonable. Based on KCA's experience, actual cyanide use would likely be about 25% of laboratory test values for similar materials. The actual consumption of cyanide can fluctuate depending on operational parameters and various influencing factors.

Temperature impacts on leaching recovery have been tested and the results indicate that minimal variation in leaching kinetics is to be expected between a cold temperature of 4°C and warm ambient temperature of 22°C.

Based on the agglomeration and compacted permeability tests, cement agglomeration was not considered necessary.

Mercury was identified in the head analyses at low concentrations, indicating that the inclusion of a mercury retort should be considered in the process design. Based on the available metallurgical test results, there are no known processing factors or other deleterious elements present that would significantly impact the potential economic extraction of Coffee material.

13.6.2 Gold Recovery and Reagent Consumption

Table 13.24 provides an overview of the final average gold recoveries and the amount of reagents used, based on bottle roll and column leach tests carried out by KCA and SGS. It includes gold recovery data for Latte, Supremo, Double-Double, and Kona, broken down by material type (oxide or transition). Silver recovery is not included in this summary.

Commercial heap leach operations typically achieve lower gold recoveries than laboratory tests suggest. This difference is due to various factors specific to the site and operation, such as local climate, crushing circuit design, heap leach pad design, stacking, and overall operating approach. A recovery discount of approximately 3 to 5% was recommended by KCA based on their extensive experience, which is a general accepted practice. A 3% discount has been applied to most zones and oxide materials, while discounts of 4% and 5% are utilized in areas with limited testing, notably transition zones.

Table 13.24
Estimated Gold Recovery and Reagent Consumption

Description	Gold Recovery (%)	Reagent Consumption	
		NaCN (kg/t)	Lime Addition (kg/t)
Latte			
Oxide	88.6	0.2	1.2
Upper Transition	77.4	0.2	1.4
Middle Transition	60.3	0.3	1.5
Lower Transition	30.0	0.3	1.6
Supremo			
Oxide	87.2	0.3	1.4
Upper Transition	79.3	0.2	1.3
Middle Transition	52.6	0.2	1.5
Lower Transition	34.2	0.2	1.5
Double-Double			
Oxide	89.1	0.2	1.5
Upper Transition	77.0	0.3	1.5
Middle Transition	42.6	0.3	1.6
Lower Transition	30.2	0.3	1.5
Kona			



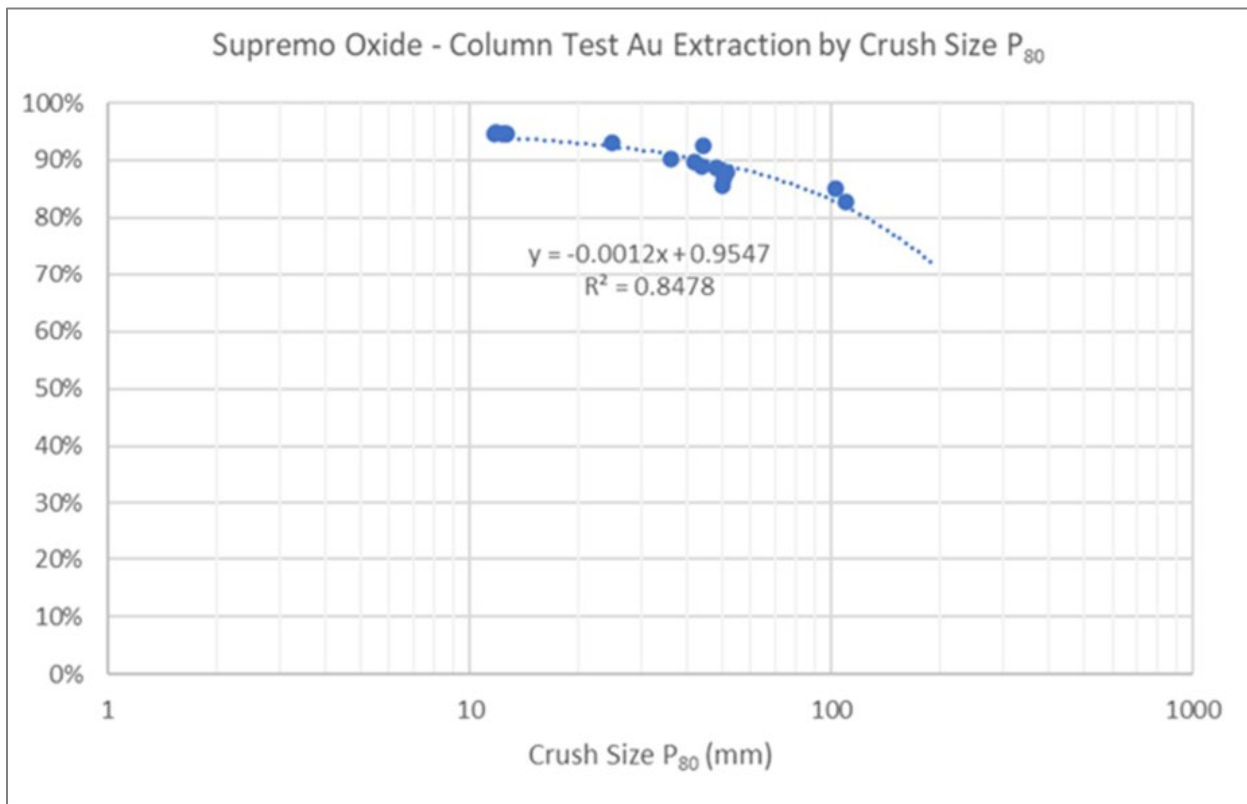
Description	Gold Recovery (%)	Reagent Consumption	
		NaCN (kg/t)	Lime Addition (kg/t)
Oxide	83.0	0.2	1.6
Upper Transition	71.3	0.2	1.5
Middle Transition	57.3	0.3	1.5
Lower Transition	28.6	0.3	1.5

Source: KCA (2024)

13.6.3 Crush Size and Leach Cycle

According to the Internal Engineering Report (JDS, 2024), KCA undertook a review on the selected crush size considering all existing test data. The review was completed by investing the Supremo Oxide test results, as it was considered as the major component of the Coffee Project. Based on KCA's data and analysis, the relationship between gold extraction and crush size P₈₀ was developed as shown in Figure 13.6.

Figure 13.6
Supremo Oxide Column Tests – Gold Extraction by Crush Size P₈₀



Source: JDS/KCA (2024)

Figure 13.6 shows that gold extraction decreases by 6% as P₈₀ crush size increases from 50 mm to 100 mm, with finer sizes improving recovery. Evaluating smaller targets (25.4 mm or 38.1 mm) is recommended, though it may require larger and/or additional equipment and higher costs. Secondary crushing can help minimize circuit disruption, and overall, targeting finer sizes could improve gold recovery.

Laboratory column experiments yielded leach kinetic curves for each material type studied. These curves form the basis for determining the optimal leach cycle time. Each curve comprises two distinct stages: an



initial rapid leaching phase characterized by high extraction rates over a short period, followed by a slower leaching phase as ultimate extraction is approached. The duration of the first phase is primarily influenced by solution-to-leach feed solids contact, while the second phase is largely determined by overall leach duration. Both kinetic curve characteristics were taken into account by KCA when establishing the appropriate leach cycle time.

The leach kinetic curve can be expressed as the function below, where E is the extraction at time t in days, Eu is the gold extraction that can be found in Table 13.24 and k is a kinetic parameter (1/day).

$$E = Eu * (1 - e^{-kt})$$

As listed in Table 13.25, Eu and k parameters for each material type were determined by KCA via analysis of column leach data.

**Table 13.25
Leach Kinetic Parameters**

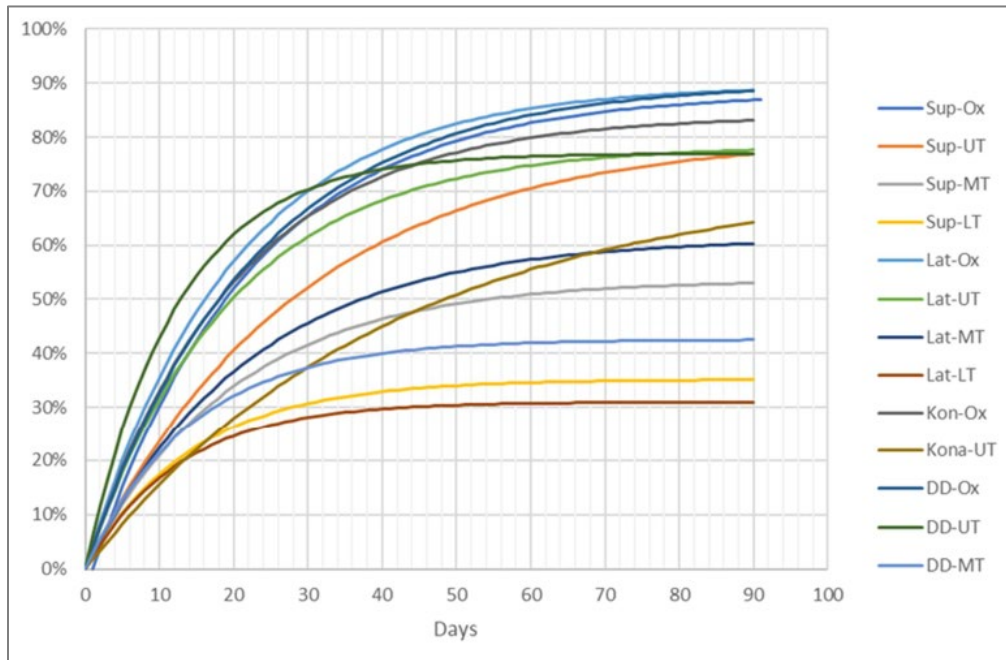
Description	Kinetic Parameter, k (1/d)
Latte	
Oxide	0.051
Upper Transition	0.051
Middle Transition	0.045
Lower Transition	0.078
Double-Double	
Oxide	0.045
Upper Transition	0.082
Middle Transition	0.07
Supremo	
Oxide	0.047
Upper Transition	0.035
Middle Transition	0.05
Lower Transition	0.069
Kona	
Oxide	0.05
Upper Transition	0.024

Source: KCA (2024)

Based on the leach kinetics and parameters, curves were developed as shown in Figure 13.7 to demonstrate gold extraction as compared to time (days) under leach.



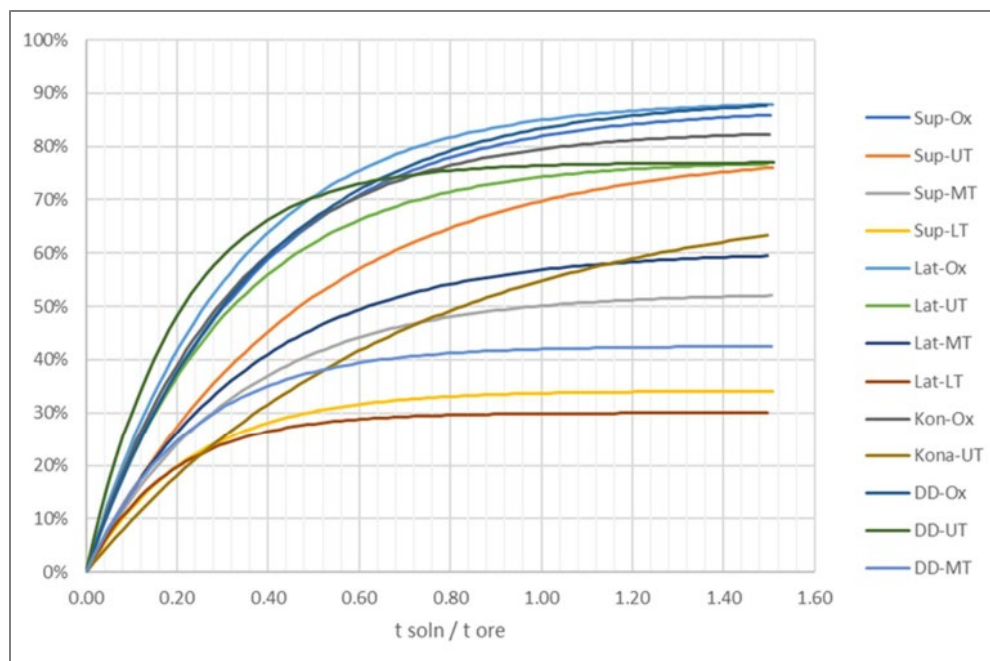
Figure 13.7
Kinetic Leach Curves - Gold Extraction vs Time Under Leach



Source: JDS/KCA (2024)

The kinetic curves for gold recovery as compared to solution/solids ratio is shown in Figure 13.8.

Figure 13.8
Kinetic Leach Curves – Gold Extraction vs Solution/Solids Ratio



Source: JDS/KCA (2024)



In the opinion of the WSP's QP for this Section of the Technical Report, an initial leach cycle time of 60 days is recommended to extract most of the gold through the primary leach cycle. The remaining portion of the gold will be recovered during the secondary leach cycles after material has been stacked over. This is subject to change through the heap leach operation based on the actual recovery during the operation. A more detailed analysis during pre-feasibility and feasibility stages is recommended to gain more confidence in achieving the optimum economic return for the project.



14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The current MRE for the Coffee Gold Project supersedes all previous estimates and reports, regardless of their source or date.

The Coffee Gold Project deposit consists of relatively shallow and continuous, sub-vertical, crossing gold mineralization structures, extending predominantly north-south and east-west continuously near surface for up to 4,000 m and steeply dipping down 150 m on average and exceeding 200 m in a few areas.

14.2 Mineral Resource Statement

Micon's QP has reviewed and accepted the estimated mineral resource for the Coffee Gold Project deposit. The MRE was updated following the existing Newmont's mineralization interpretation wireframes, by Fuerte's geological team and was reviewed and edited by the Micon QP using Leapfrog Geo/EDGE for the mineral resource modelling and subsequently Datamine Studio software to conduct the open pit optimizations. This work was completed in accordance with the CIM Definitions Standards for Mineral Resources and Mineral Reserves adopted by the CIM council on May 10, 2014 (2014 CIM Definitions Standards) and the 2019 CIM Best Practice Guidelines, and includes the resource definitions reproduced below:

Mineral Resources

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors.



Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

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

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drillholes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the Project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.



Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

14.3 CIM Estimation of Mineral Resources Best Practice Guidelines

When conducting the mineral resources update for the Coffee Gold Project deposit, Micon and its QPs have used the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines which were adopted by the CIM Council on November 29, 2019.

Assuming open pit mining methods and heap leaching process, the MRE statement allocated to each deposit of the Coffee Gold Project is presented in Table 14.1

**Table 14.1
Coffee Gold Project Mineral Resource Statement**

Area	Category	Tonnage (kt)	Gold Grade (g/t)	Gold Ounces (koz)	Strip Ratio
Supremo	Measured	1,200	1.80	69	5.1
	Indicated	52,558	1.10	1,857	
	Measured + Indicate	53,758	1.11	1,926	
	Inferred	3,933	1.11	141	
Latte	Indicated	15,455	1.24	616	5.1
	Inferred	3,083	0.99	98	
Double-Double	Indicated	2,281	1.81	133	5.1
	Inferred	669	1.43	31	
Supremo Extension	Indicated	2,437	1.18	92	8.1
	Inferred	6,059	1.72	335	
Arabica	Indicated	1,771	0.77	44	4.5
	Inferred	4,687	0.73	110	
Cappuccino	Inferred	396	0.99	13	5.9
French Press	Inferred	172	0.83	5	4.9
Kona	Indicated	4,344	1.05	146	2.8
	Inferred	2,202	0.95	67	
All Areas	Measured	1,200	1.80	69	5.1
	Indicated	78,846	1.14	2,888	
	Measured + Indicated	80,046	1.15	2,957	
	Inferred	21,200	1.17	800	

Notes:

- Economic parameters used in the resource are a gold price of US\$2,500/oz; heap leach average recoveries for the individual metallurgical domains of 86.3% for Oxide, 76.0% for Upper Transition, 54.5% for Middle Transition and 31.4% for Lower Transition; a mining cost of \$CA3.27-\$3.50/t, processing costs of \$CA6.64/t, and general and administrative costs of \$CA6.00/t. A US\$:CA\$ exchange rate of 1.35 was also assumed.



Area	Category	Tonnage (kt)	Gold Grade (g/t)	Gold Ounces (koz)	Strip Ratio
2.	The calculated cut-off grades vary between 0.13 g/t Au and 0.48 g/t Au, depending on the metallurgical domain. The global weighted average cut-off grade is 0.18 g/t Au, with domain tonnage contributions comprising 64% Oxide, 18% Upper Transition, 5% Middle Transition, and 13% Lower Transition.				
3.	Pit slope angles vary between 45.0 and 48.8 degrees depending on the pit area.				
4.	Pit optimization was done on 12x12x10 m re-block model with a minimum of 4x4x5 m regularized SMU.				
5.	Numbers have been rounded to the nearest thousand for tonnes and ounces. Differences may occur in totals due to rounding.				
6.	The mineral resources described above have been prepared in accordance with 2019 CIM Best Practice Guidelines and 2014 CIM Definitions Standards.				
7.	Charley Murahwi, P.Geo., FAusIMM, from Micon is the QP for this MRE.				
8.	Mineral resources unlike mineral reserves have no demonstrated economic viability. The tonnages and grade of the reported Indicated and Inferred mineral resources are uncertain in nature; however, it is reasonably expected that significant portions of the Indicated and Inferred Mineral Resources can be upgraded into Measured and Indicated Mineral Resources, respectively, with further exploration field work.				
9.	Micon's QP has not identified any legal, political, environmental, or other factors that could materially affect the potential development of this mineral resource other than what is indicated in the Report.				

The Coffee mineral resource contains significant amounts of high grade material within the pits, a cut-off grade sensitivity is presented in Table 14.2, a chart version of the sensitivity can be seen in Figure 14.1 and the spatial distribution of the high grade at various cut-off is shown in Figure 14.2.

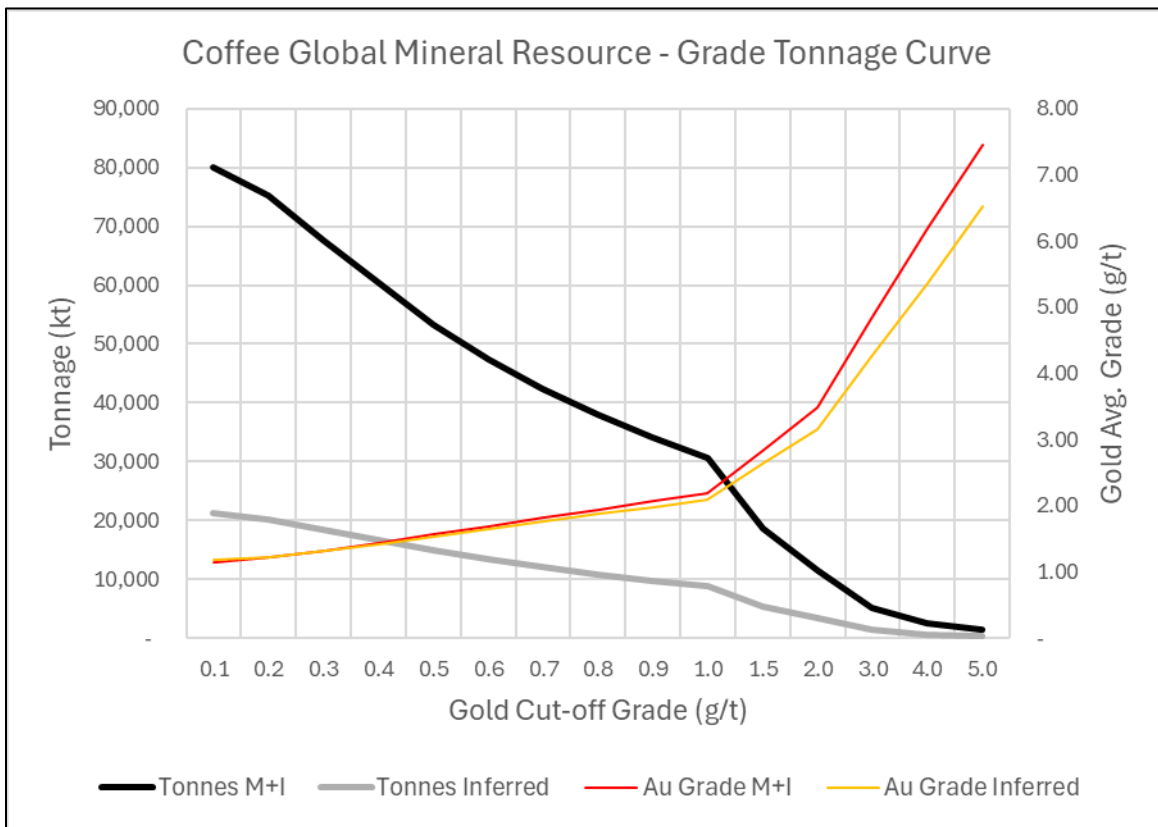
Table 14.2
Coffee Mineral Resource Gold Cut-off Grade Sensitivity

Cut-off Grade	Measured + Indicated			Inferred		
	Tonnage	Avg. Au Grade	Metal Content	Tonnage	Avg. Au Grade	Metal Content
<i>g/t</i>	<i>kt</i>	<i>g/t</i>	<i>koz</i>	<i>kt</i>	<i>g/t</i>	<i>koz</i>
5.0	1,524	7.46	365	270	6.53	57
4.0	2,613	6.20	521	595	5.37	103
3.0	5,049	4.86	789	1,370	4.27	188
2.0	11,519	3.49	1,291	3,396	3.16	345
1.5	18,557	2.82	1,682	5,447	2.62	460
1.0	30,720	2.19	2,162	8,833	2.09	593
0.9	34,136	2.07	2,267	9,761	1.98	622
0.8	37,933	1.94	2,370	10,806	1.87	650
0.7	42,307	1.82	2,476	11,993	1.76	679
0.6	47,357	1.70	2,581	13,390	1.64	708
0.5	53,362	1.57	2,687	14,998	1.53	736
0.4	60,445	1.44	2,789	16,662	1.42	760
0.3	67,671	1.32	2,871	18,351	1.32	779
0.2	75,209	1.21	2,931	20,190	1.22	794
0.1	80,046	1.15	2,957	21,200	1.17	800

The lowest effective gold cut-off grade of the mineral resource statement is 0.13 g/t Au. Table 14.2 shows the various cut-off grades sensitivity using two intervals sizes, 0.1 from 0.1 g/t Au to 1 g/t Au and then 0.5 thereafter up to 5 g/t Au, this interval size difference changes the slope of the curves in the grade-tonnage plot shown in Figure 14.1.

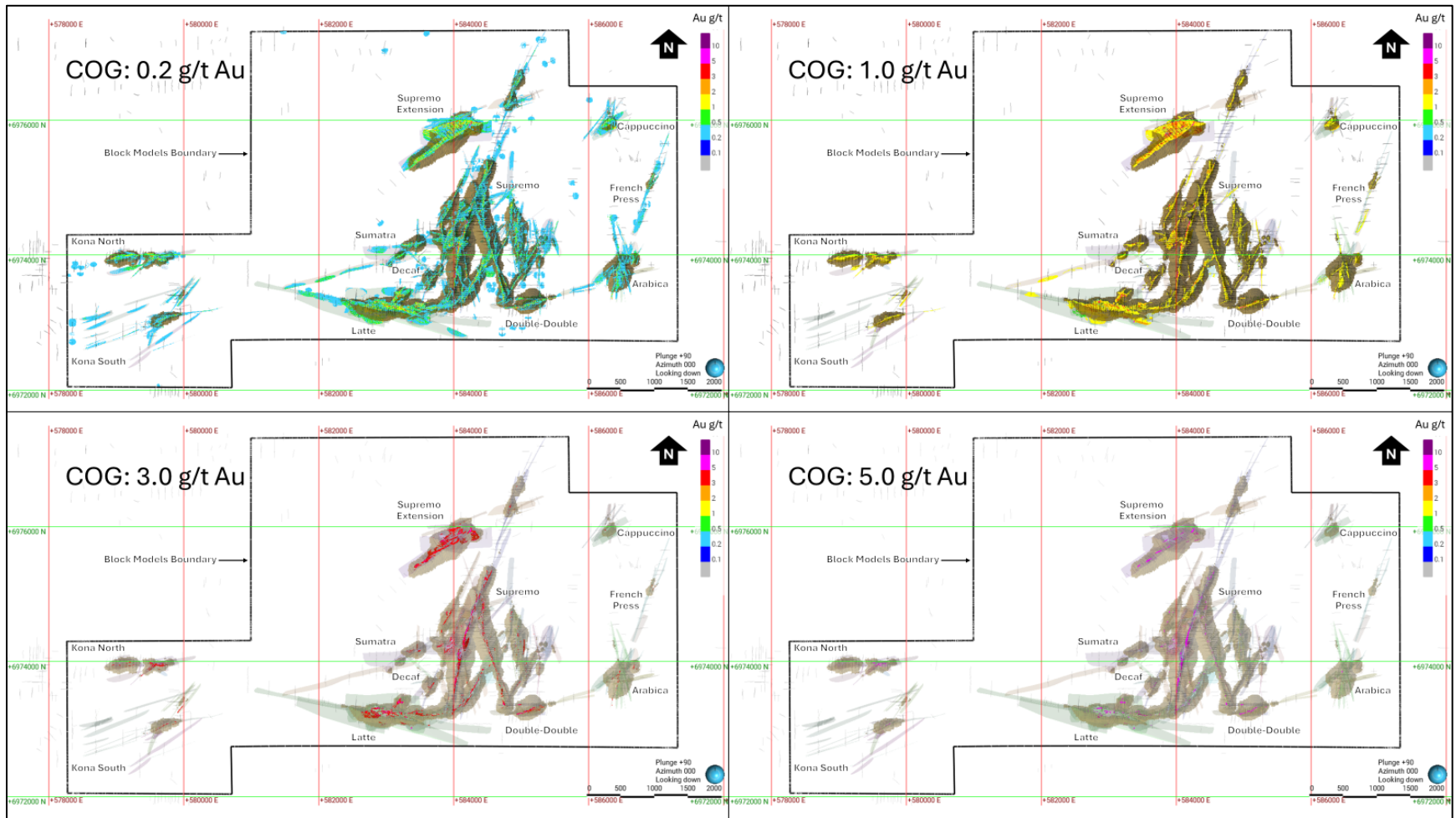


Figure 14.1
Coffee Gold Project Cut-off Sensitivity Chart



Source: Micon, 2025

Figure 14.2
Coffee Gold Project Sensitivity Maps at various Gold Cut-off Grades



Source: Micon, 2025 Note: COG = cut-off grade



14.4 Geological Database

The Project database is maintained in an acquire SQL database management system, which contains collar locations, downhole survey data, qualitative logging information, and assay and multielement geochemical data, among other items. Data for geological modelling and resource estimation purposes were exported as CSV files and then imported into Leapfrog Geo v.2024.1.3 and ioGAS v.8.3 for analysis. The geological database that supports this MRE includes drillholes completed by Kaminak, Goldcorp, and Newmont between 2010 and 2023.

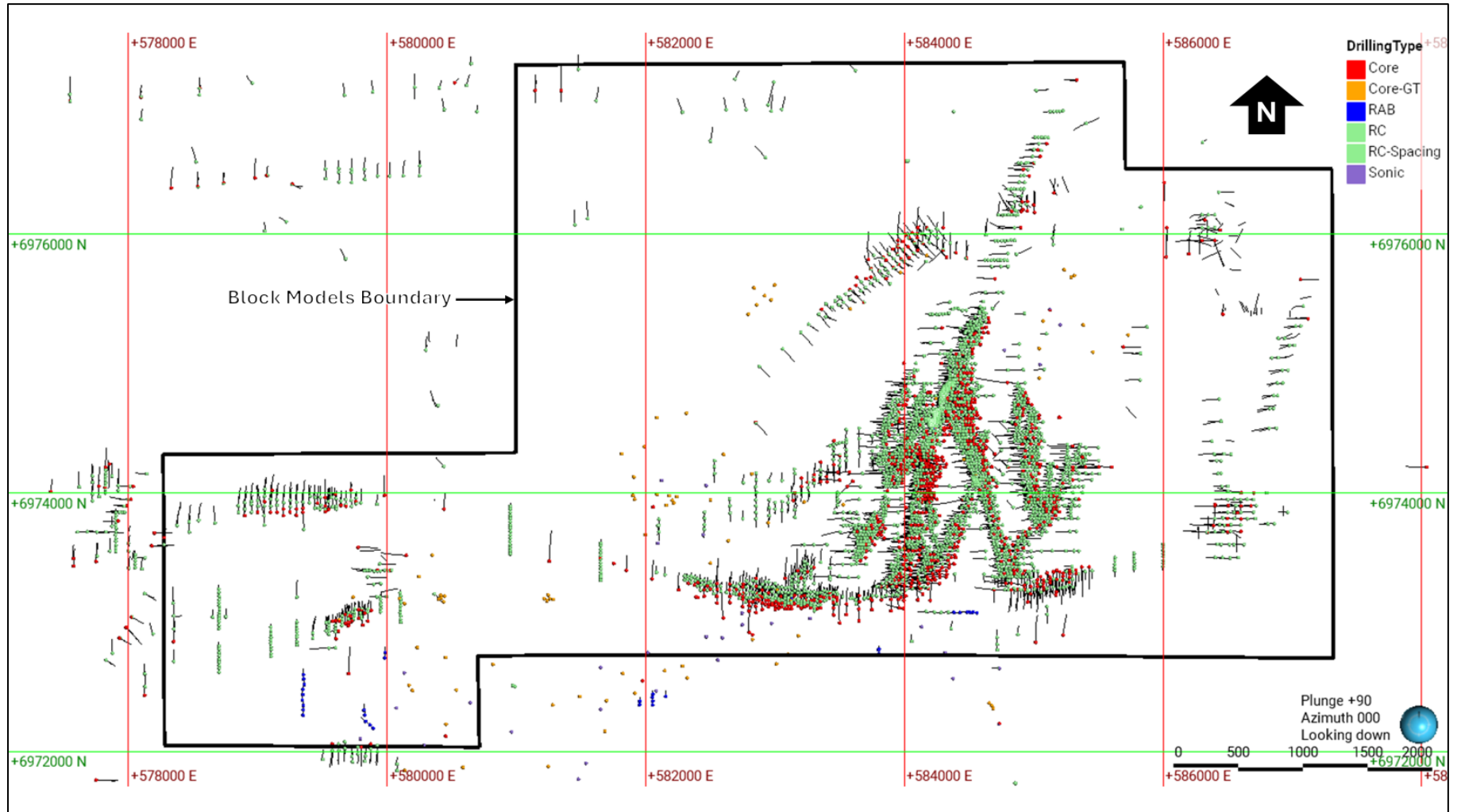
The full geological database covering the Coffee Gold Project areas contain 4,638 drillholes, of which 4,029 drillholes totaling 379,130 m of sampling were included in the resource estimation, see Figure 14.3 for drillhole location details. The database used for mineral resource estimation includes drillholes completed by Kaminak, Goldcorp, or Newmont between 2010 and 2023. Of these drillholes, 27.1% are diamond drillholes (Core) and the remainder (72.9%) are reverse circulation drillholes (RC). In total, 609 holes, or 13.1% of drillholes were excluded from the MRE, 410 holes due to the drillhole location falling outside the resource model extents and the rest for not contributing to definition of the 39 estimation domains. An additional 28 RAB drillholes and 22 sonic drillholes lie within the model extents but were also excluded from estimation due to the generally poor sample quality associated with RAB and sonic drilling nature.

The final selected geological database supporting the mineral resource estimation contains a total of 416,022 accepted gold fire assay (AU_FA) records, and a total of 34,969 accepted cyanide-soluble gold assay (AU_AA) records. Drilling statistics for the final estimation dataset are presented in Table 14.3.

Table 14.3
Entire Coffee Gold Project Drillhole Count and Meterage by Type by Year

Year	Core		Core-GT		RAB		RC		RC-Spacing		Sonic		Total	
	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)	Count	Length (m)
2010	76	16,105											76	16,105
2011	111	29,963					145	19,518					256	49,481
2012	124	29,609					223	39,451					347	69,060
2013	62	12,273					240	43,204					302	55,477
2014	145	26,794	2	100			206	25,867					353	52,760
2015	103	15,840			35	2,198	197	23,702			35	156	370	41,895
2016	26	6,703	35	370			261	20,473					322	27,546
2017	74	17,697	43	723			329	48,197	219	5,535			665	72,152
2018	195	36,691	7	1,148			590	60,655	90	5,823			882	104,317
2019	194	30,203					550	60,403					744	90,606
2021	1	161					49	8,263					50	8,424
2022	16	3,456					39	6,820					55	10,276
2023	43	9,780	4	669			169	8,381					216	18,830
Total	1,170	235,274	91	3,010	35	2,198	2,998	364,933	309	11,358	35	156	4,638	616,930

Figure 14.3
Coffee Gold Project Drillhole Location Map



Source: Micon, 2025



14.5 Mineral Resource Estimate Methodology

The MRE modelling work and construction of the block models was prepared internally by Fuerte Metals Corporation technical staff, once the work was completed it was delivered to Micon International Limited for review, edition, and acceptance. Messrs. Alan J. San Martin, P.Eng. and Charley Murahwi, P.Geo. are the qualified person (QPs) of the Coffee Gold Project Mineral Resource Estimation under NI 43-101 Standard, both QPs are independent of Fuerte Metals Corporation. Messrs. San Martin and Murahwi are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the present mineral resources as of the date of this report.

The present Technical Report presents a gold mineral resource for the Coffee Gold Project with an effective date of August 21st, 2025, the date which Micon completed its work. The geological domains were constructed originally by Newmont, subsequently Fuerte used Leapfrog Geo v. 2024.1.3, including input from analyses completed in ioGAS v.8.3 to compile the entire Coffee Gold Project and reconstruct the model. Geostatistical evaluations and exploratory data analysis (EDA), including grade capping selection, declustering, and variography, were reassessed and completed using Snowden Supervisor v.9.0. The interpolation of grade and the construction of the three block models for the resource estimation was prepared using Leapfrog EDGE v.2024.1.3. Pit optimization to determine reasonable prospects for eventual economic extraction (RPEEE) was completed using Datamine NPVS software.

14.5.1 Data Preparation

Fuerte was provided various exploration datasets related to the Coffee Gold Project, including summary reports, geologic maps, LiDAR topography, and drillhole database which includes collar locations, downhole surveys, lithology logs, gold assay data, and multi-element geochemistry, among other items. All these data and information was compiled and processed by Fuerte and then submitted to Micon for independent review and sign-off.

14.5.1.1 Drilling

Drillhole data used in the MRE were checked for overlapping sample intervals, negative or invalid values, and irregular downhole survey deviation in Leapfrog Geo. All errors were assessed and corrected prior to completing statistical analysis and estimation.

Drillhole collars were also visually checked against the most current topographic surface. Most collars are set to topography, with minor deviations in some collars (<1m) attributed to local variations in the topography.

Gold assay values less than the detection limit were assigned a value equal to half of the detection limit value prior to use in modelling and estimation. Null values for gold were assigned for all intervals with no recovery or where gaps were found.

14.5.2 Wireframes

All wireframes described in this section, aside from those outlined in the “Oxidation” subsection, were completed by Newmont Mining Corporation, using the current dataset. Detailed review and validation were completed for all wireframes by Fuerte prior to use in mineral resource estimation.

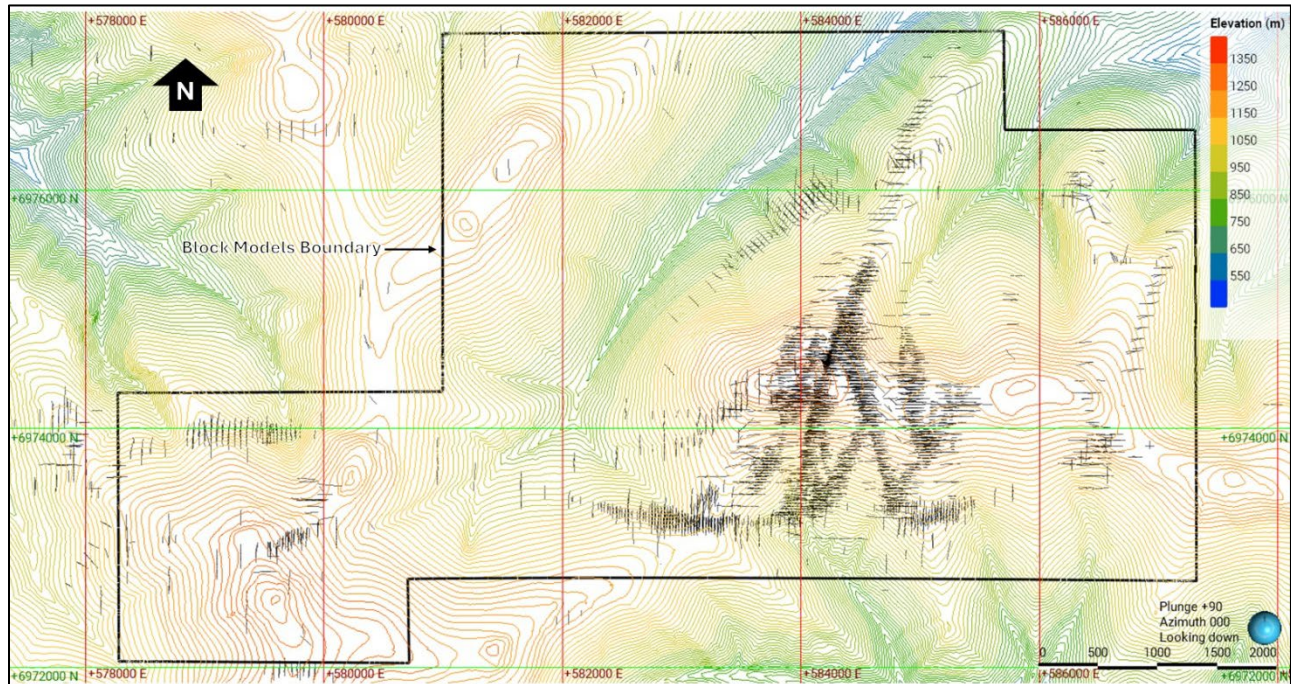
14.5.2.1 Topography

Topography was supplied in UTM NAD83, Zone 7 coordinates as LiDAR with contour intervals of 2 m. These data were derived from a LiDAR survey of the area conducted by Eagle Mapping Ltd. in 2010. A



single integrated topography mesh was generated and used across all input Leapfrog modelling projects to ensure consistency. Figure 14.4 shows the Coffee Gold Project Topography.

Figure 14.4
Coffee Gold Project Topography Map at 10m Contours



Source: Micon, 2025

14.5.2.2 Lithology, Breccia, and Dikes

The lithology model depicts the distribution of schist, gneiss, meta-basalt and ultramafic units across the Project, and is used specifically to inform density estimates during resource modelling. Overall, the model comprises 20 solids (Figure 14.5). Modelled lithologies include Biotite Schists, Felsic Gneiss, Mixed Mafic Gneiss, Granite, Meta-basalt, Amphibolite and Ultramafic. All fault and lithology wireframes were snapped to drillhole data and were checked for closure and consistency prior to resource estimation. Eighteen major lithologies were modelled and are outlined in Table 14.4. Breccia units were primarily interpreted based on logged lithology; 126 separate breccia zones were modelled implicitly in Leapfrog utilizing the vein modelling workflow.

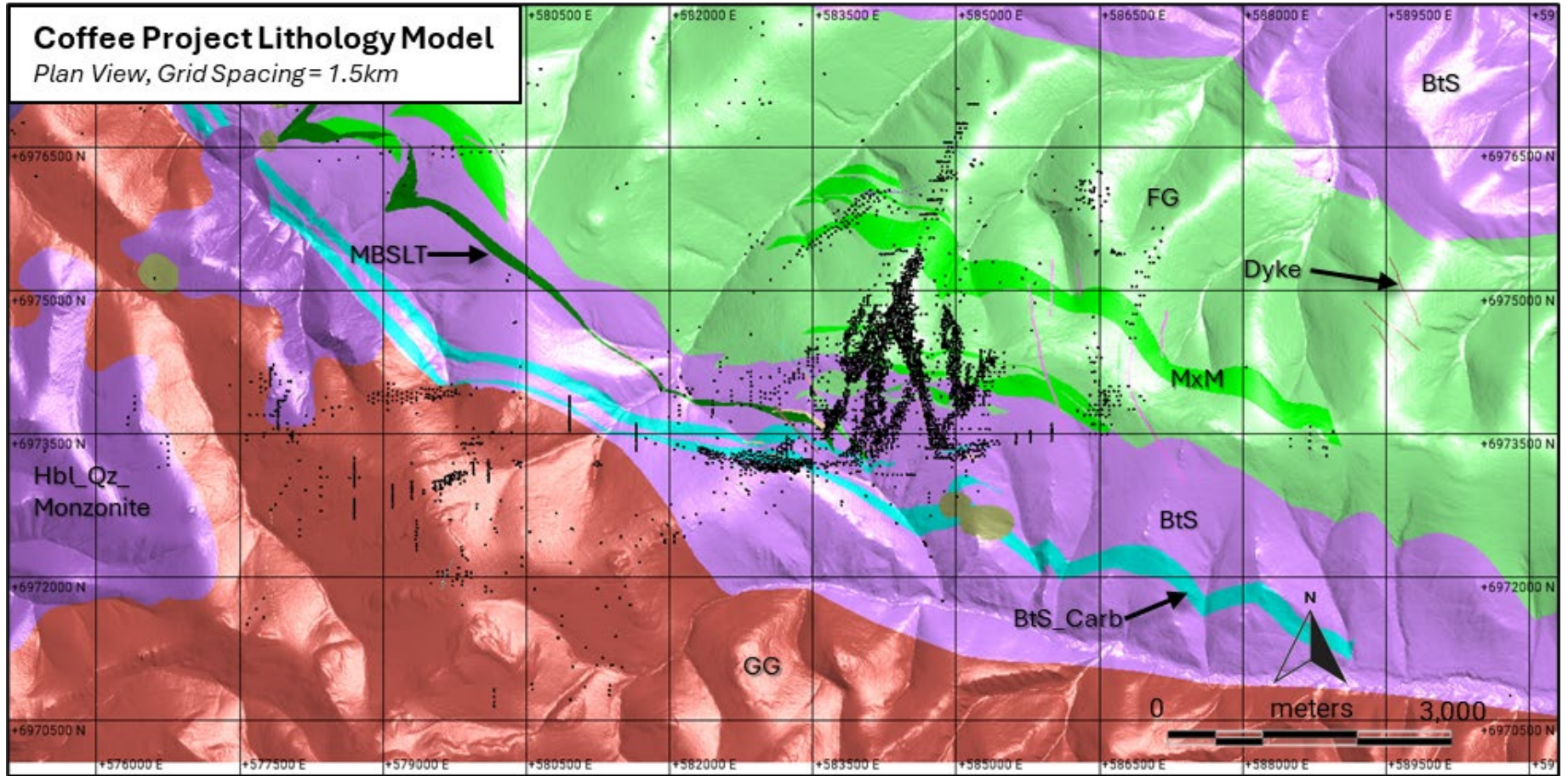
Dikes are spatially correlated to Breccia and mineralized zones; intervals were primarily interpreted on the basis of logged lithology, 127 separate dyke solids were modelled implicitly in Leapfrog utilizing the vein modelling workflow and were included in the set of lithology solids flagged to the block models.



Table 14.4
Coffee Gold Project Lithology Codes

Code	Lithology	Description
1	Intrusion_Undifferentiated	Undifferentiated intrusive rock
2	Undifferentiated_Granitoid	Undifferentiated granitoid
3	Hbl_Qz_Monzonite	Hornblende-quartz monzonite
4	Undifferentiated_Dyke	Undifferentiated dike
5	mGG	Microgranite
6	UM	Mafic-ultramafic
7	GDIORa	Granodiorite
8	GBRO	Gabbro
9	Amph	Amphibolite
10	MxM	Mixed mafic gneiss
11	MBSLT	Metabasalt
12	IV	Dike
13	GG	Granite
14	FG	Felsic gneiss
15	FC	Dyke
16	DIOR	Diorite
17	BtS	Biotite schist
18	BtS_Carb	Carbonaceous biotite schist

Figure 14.5
Coffee Gold Project Lithology Model with Drillholes



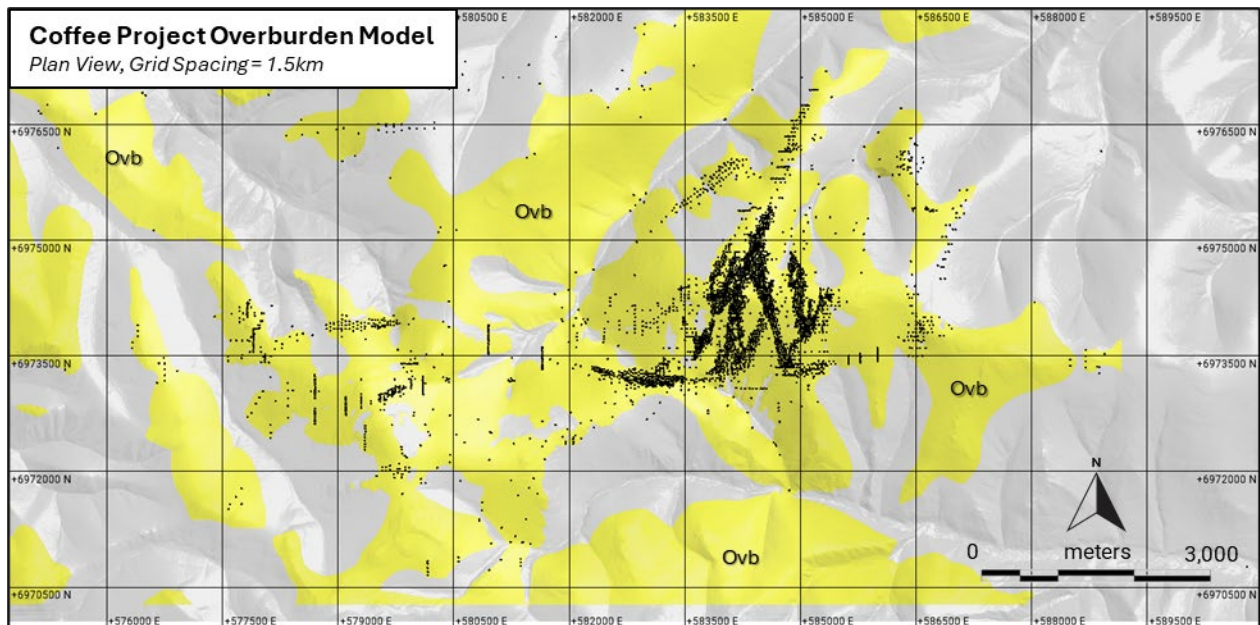
Source: Fuerte, 2025



14.5.2.3 Overburden

An overburden (Ovb) solid was generated for the Coffee Gold Project area in Leapfrog Geo, using all available diamond and RC drillholes completed across the Project from 2010 to 2023. The resulting solid has an average vertical thickness of about 4.5 m and a background grade of 0.001 g/t Au and a density of 1.9 g/cm³ was assigned to the entire unit.

Figure 14.6
Coffee Gold Project Overburden Model with Drillholes



Source: Fuerte, 2025

14.5.2.4 Mineralization

Gold mineralization is located within a series of steeply dipping structures that crosscut all rock units. The structural zones are identified in the drill core, from surface mapping and trenching, and recently from VLF surveys. Soil sampling has also located gold-in-soil anomalies in many areas which were subsequently drilled. Although these structural zones can exhibit a variety of characteristics, including faulting, brecciation, silicification, alteration, and local sulphide veining, they can be traced over strike lengths up to 2.5 km.

Lithology and structure models were completed initially and formed the basis of all subsequent mineralization models. The structural model depicts the trend of mineralized corridors based on drillhole logging of dike and breccia lithologies. Gold mineralization has a strong spatial correlation with dike and breccia and are often mineralized themselves. The dike and breccia models are therefore used to inform the trend of mineralization.



Gold mineralization was modelled using the Vein System tool in Leapfrog Geo, considering a cut-off grade of 0.2 g/t Au for mineralized domain construction. Individual veins were modelled using interval selection completed on a merged table which contains gold assay data, multi-element geochemistry, and qualitative logging attributes such as lithology. Mineralization was modelled in four separate Leapfrog projects:

- Supremo, Latte and Double-Double;
- Kona and Kona North;
- Arabica, and
- Supremo Extension.

Veins modelled in these individual projects were then consolidated into a single Leapfrog project for estimation purposes.

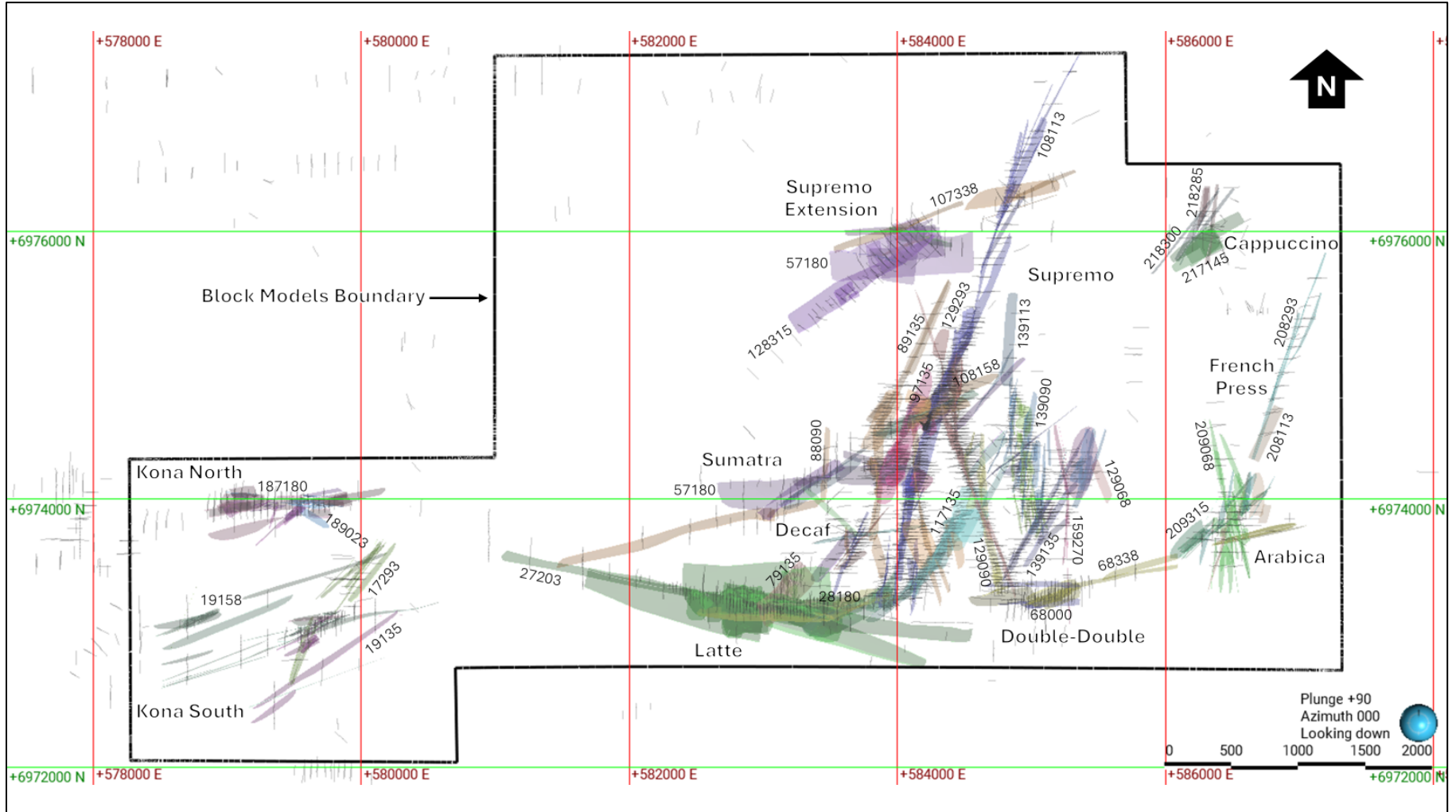
The Vein System tool was chosen to best represent the distribution of structurally controlled gold. Vein pinch outs were used to identify drillhole intervals included in individual vein solids that were lacking composite values. In these instances, a case-by-case assessment was made whether to include an interval of waste to maintain structural continuity.

The gold mineralization modelling workflow produced a total of 478 individual vein solids across the Project. These individual solids were then grouped according to their average strike and dip, resulting in a total of 38 mineralized domains which were then used for variography, contact analysis, grade capping analysis, and other estimation parameters (see Figure 14.7). Many of these domains show similar grade distributions but were separated due to significant differences in orientation or due to being spatially distinct.

In addition to the grouped vein wireframes, a low-grade halo domain (Code 2) was constructed based on a 15 m distance buffer zone around the mineralized domains (see Figure 14.8). A non-mineralized background domain was also constructed based on a 150vm distance buffer to drill traces within the final estimation database extents.



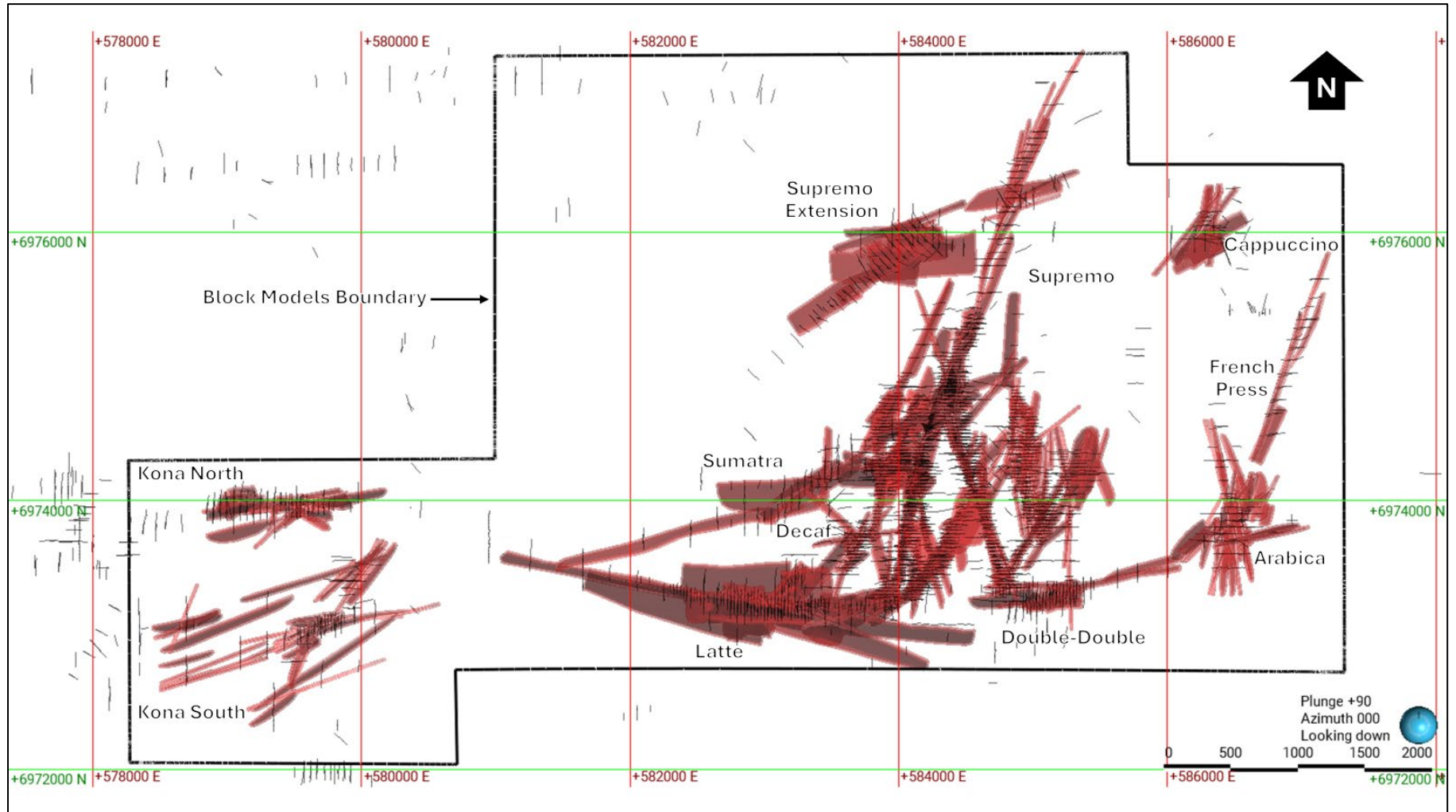
Figure 14.7
Coffee Gold Project Estimation Domains



Source: Micon, 2025



Figure 14.8
Coffee Gold Project Gold Low-Grade Halo Boundaries



Source: Micon, 2025



Table 14.5
Coffee Gold Project Gold Estimation Domains

Domain Group	Domain	Wireframes	Area(s)	General Strike	General Dip Direction
Mineralized (>0.2 g/t Au vein domains)	78045	1	Supremo	NW	NE
	17293	15	Kona	NNE	ESE, WSW
	19135	28	Kona	NE	SE
	19158	21	Kona	ENE	SSE
	27203	29	Latte, Supremo Extension	ESE	SSW
	28180	5	Latte	EW	S
	57135	5	Supremo	NE	SW
	57180	13	Supremo, Supremo Extension	EW	S
	68000	6	Double-Double, Supremo Extension	EW	N
	68338	19	Latte, Double-Double, Arabica	ENE	NNW
	78338	2	Supremo	ENE	NNW
	79135	3	Latte, Supremo	NE	SW
	88090	23	Supremo	NS	E
	89135	5	Supremo	NE	SW
	97135	13	Supremo	NE	SW
	107338	12	Supremo, Supremo Extension	ENE	NNW
	108113	29	Supremo	NNE	ESE
	108158	10	Latte, Supremo	ENE	SSW
	117135	19	Supremo	NE	SW
	128315	9	Supremo, Supremo Extension	NE	NW
	129068	15	Supremo	NW	NE
	129090	11	Supremo	NS	E
	129293	5	Supremo	NS	W
	139090	26	Supremo	NS	E
	139113	28	Supremo	NNE	ESE
	139135	9	Supremo	NE	SW
	156270	2	Supremo	NS	W
	159270	6	Double-Double, Supremo	NS	W
	187180	21	Kona	EW	S
	188338	9	Kona	ENE	NNW
	189023	10	Kona	EW	N
	208113	8	Arabica, French Press	NE	SE
	208293	9	Arabica, French Press	NE	NW
209068	29	Arabica	NNW	ENE	
209315	12	Arabica	NE	NW	
217145	4	Cappuccino	ENE	SSE	
218285	3	Cappuccino	NS	W	
218300	4	Cappuccino	NE	NW	
Low-grade halo	2	1	All	Multiple	Multiple
Background	0	1	All	Multiple	Multiple

14.5.2.5 Oxidation

Cyanide (CN)-soluble gold assays (AU_AA) have been routinely collected in drilling where the original fire assay result (AU_FA) returned values greater than 0.3 g/t, meaning that most of the available CN-soluble assays fall either within the mineralized domain solids or the low-grade 'halo' domain. The AU_AA to AU_FA ratio was calculated for all samples where both values are available, and the oxide classification for each sample was derived from this ratio according to the criteria presented in Table 14.6. In some instances



where individual samples where the CN-soluble assay was greater than the original fire assay, the ratio was set to 1.

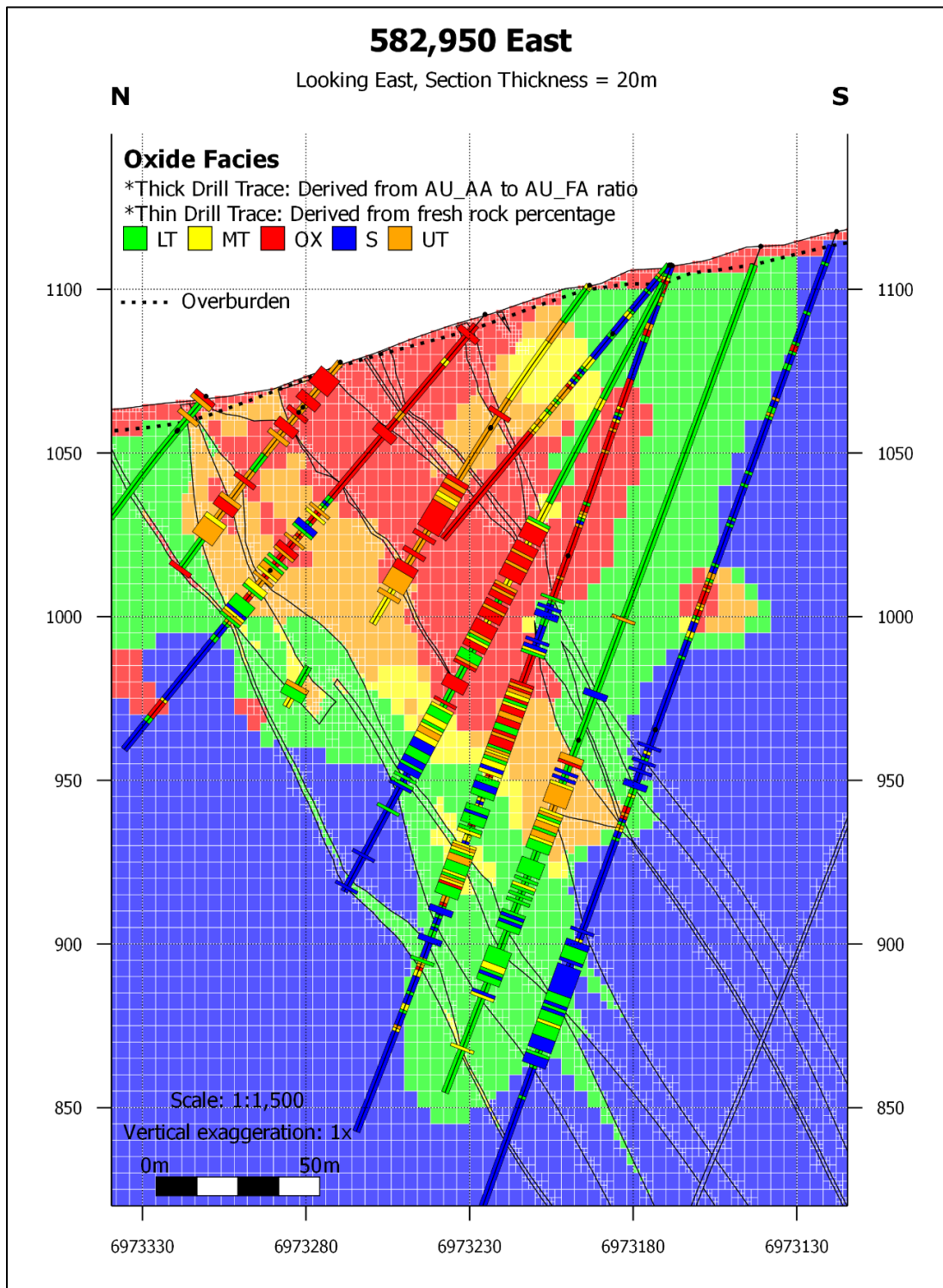
Table 14.6
Cyanide/Fire Assay Gold Ratios for Oxide Types in Mineralized Domains

Au_AA/Au_FA (AAFA)		Oxide Type	
Upper Limit	Lower Limit	Text Code	Numeric Code
1	0.9	OX (Oxide)	1
0.9	0.7	UT (Upper Transitional)	2
0.7	0.5	MT (Middle Transitional)	3
0.5	0.1	LT (Lower Transitional)	4
0.1	0	SU (Sulphide)	5

For mineralized domains, an indicator model was generated for each derived oxide category, using the Indicator Interpolant tool and Spherical Interpolant function in Leapfrog Geo. The geometry and continuity of the indicator volumes for each category were controlled using a structural trend derived from the mineralized domain solids, and final Iso values were selected for each interpolant to best reflect the interpreted geological continuity of oxidation. For non-mineralized domains, where AU_AA data are generally unavailable, oxidation was modelled based on the percentage of fresh rock logged (Fresh_pct). Fresh rock percentages used to assign oxide categories for drill intervals in the non-mineralized domains were determined using the same thresholds as shown in the AU_AA to AA_FA ratio classification scheme for mineralized domains in Table 14.6. For example, an interval which is logged as Fresh_pct = 10% (i.e. 90% oxidized) is classified as OX (oxide). An indicator model was then generated for each category using a similar process as described above for the mineralized domains. The final 'oxide' variable is populated using the set of indicator models based on the AU_AA to AU_FA ratio classification for mineralized domains and is populated using the set of indicator models based on the fresh rock percentage outside mineralized domains. Modelled overburden is flagged to the model as OX (oxide). An example section showing oxide flagging is shown in Figure 14.9.

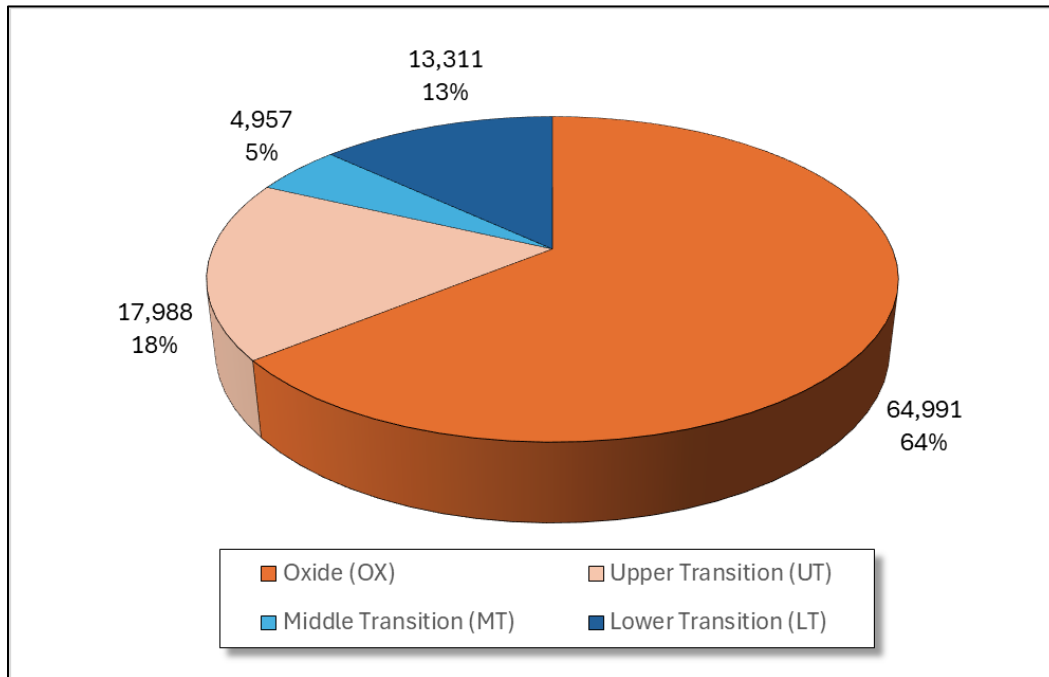


Figure 14.9
Vertical Section 582,950mE - Oxide Facies Model



Source: Fuerte, 2025
Note: S = sulphide

Figure 14.10
Percentages of Oxide Types in Mineralized Domains



14.5.3 Exploratory Data Analysis

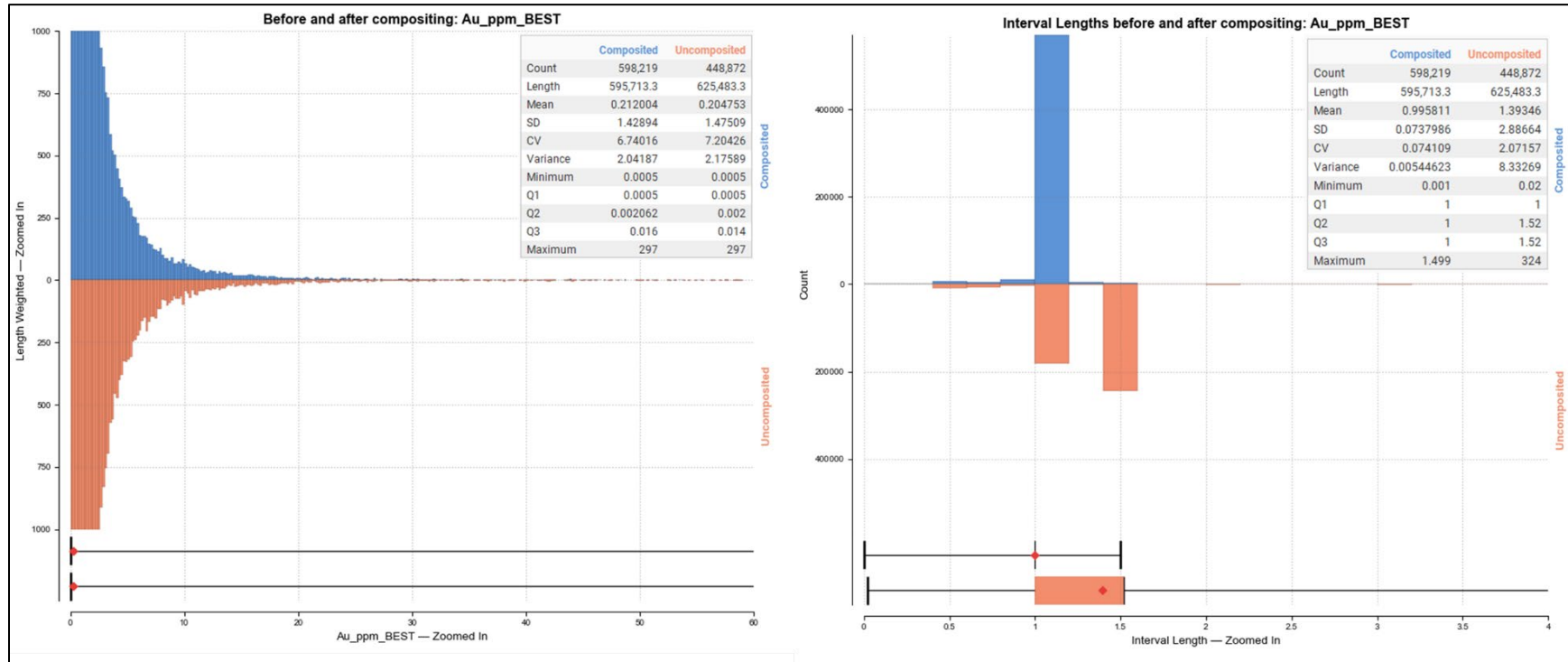
14.5.3.1 Composites

The most frequent sample interval in the assay data Table is 1.524 m, corresponding to the standard 5 ft sampling length used in RC drilling. Most core samples were taken at 1 m intervals, and this value corresponds to the quartile 1 (Q1), quartile 2 (Q2), and quartile 3 (Q3) values in the core-only subset of the drillhole database. As a result, a 1 m composite interval was selected for mineral resource estimation.

Composites were generated within the mineralized domain boundaries shown in Figure 14.7. A global comparison between raw assay interval lengths and composited data used for estimation is shown in Figure 14.11.



Figure 14.11
Global Compositing Interval Length Statistics for Composites Used in Estimation



Source: Fuerte, 2025

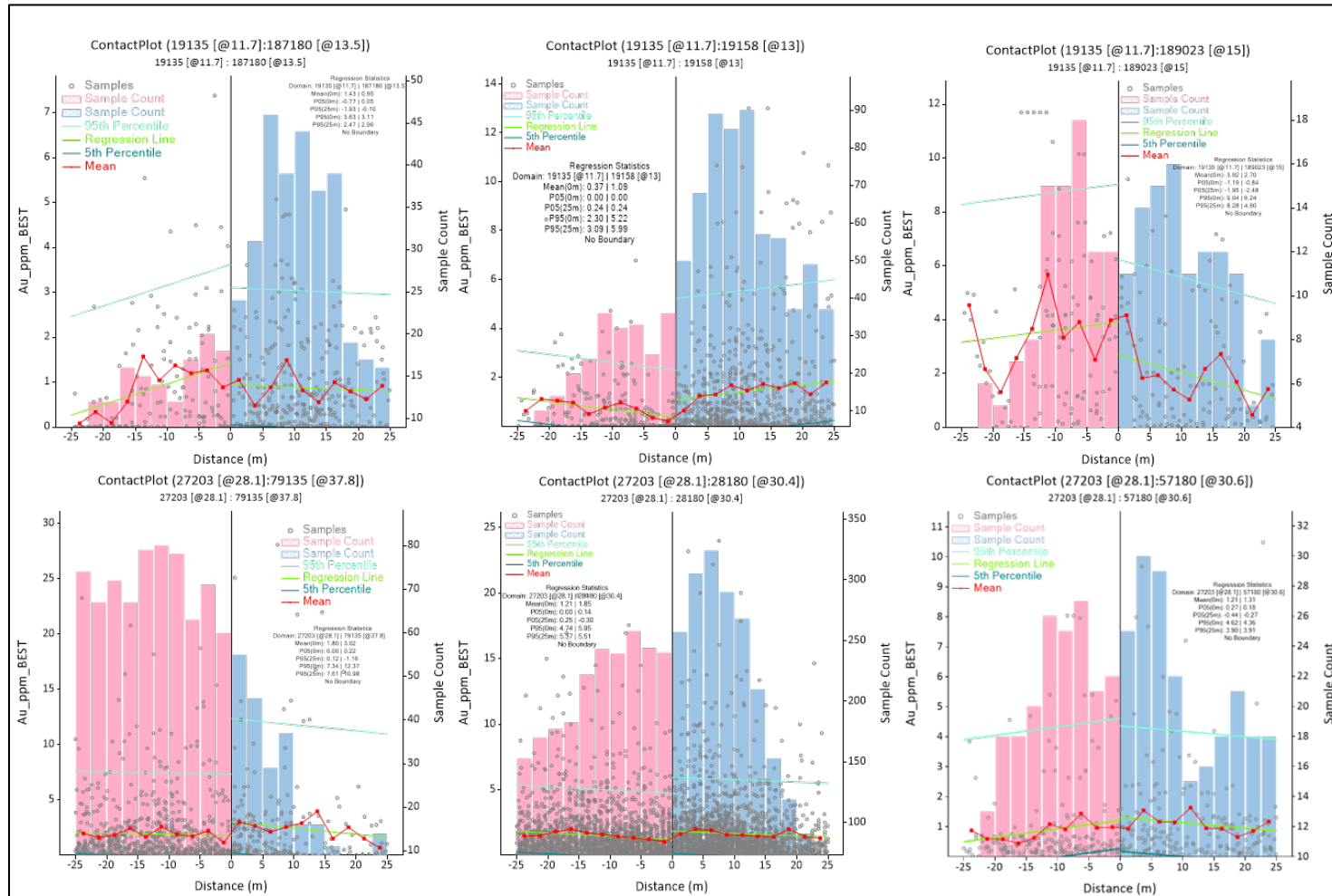


14.5.3.2 *Contact Analysis*

Contact profiles were generated for gold across all estimation domains to assess grade interpolation limits between adjacent domains. Based on the analysis of composited data, all contacts between mineralized domains and the low-grade 'halo' domain (Code 2) were treated as a hard boundary. Similarly, all contacts between mineralized domains and the Background domain (Code 0) were treated as hard. Soft boundaries were applied between contacting mineralized domains, with a maximum soft boundary search distance set to $\frac{1}{4}$ of the direction one search distance of the primary domain. Contact analysis examples from several domains are shown in Figure 14.12.



Figure 14.12
Contact Plot Examples from Domains 19135 (upper) and 27203 (lower)



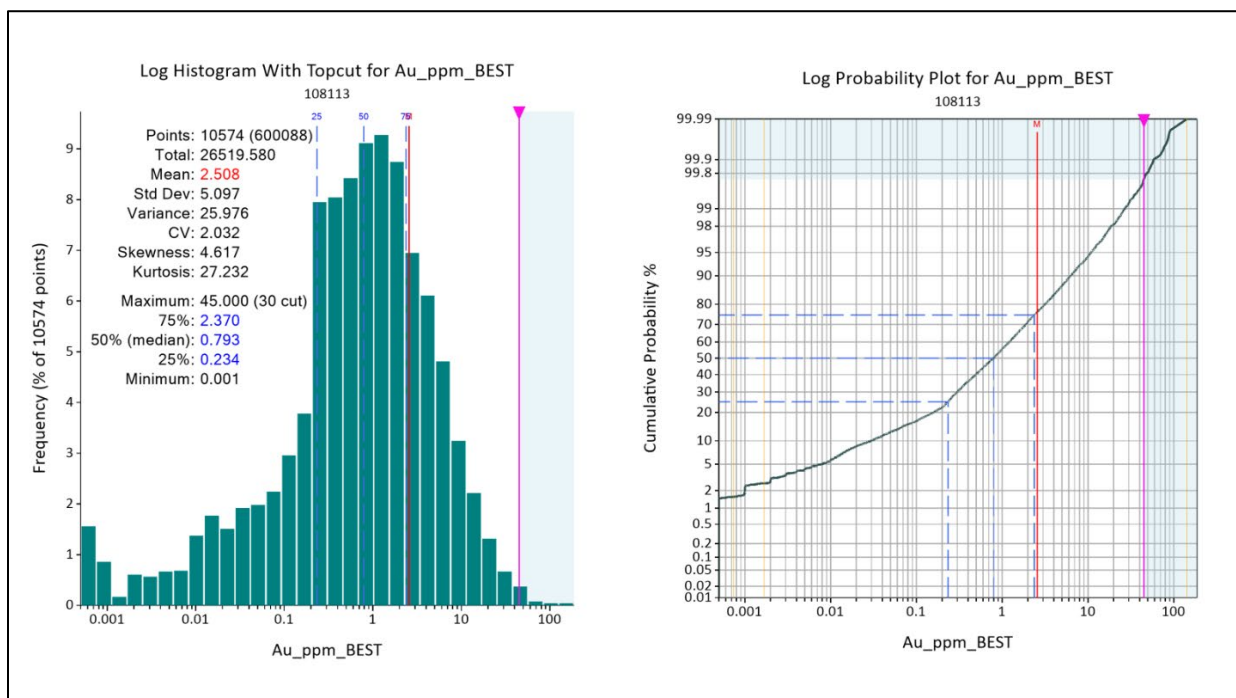
Source: Fuerte, 2025



14.5.3.3 Outlier Management and Grade Capping Strategy

Capping analysis was completed on composited data for gold across all estimation domains using histograms, mean-variance plots, cumulative metal plots, and disintegration analysis considering step changes of 10% and 15% between the assay values of adjacent data points on log-probability plots (Figure 14.13 through Figure 14.16). Capped samples were then evaluated in 3D within each domain to ensure that the samples were not clustered and represented true outliers. Ordinary Kriging (OK) estimates were also completed within each mineralized domain, using both the capped and uncapped datasets to assess the impact to average grade and contained metal (Table 14.7).

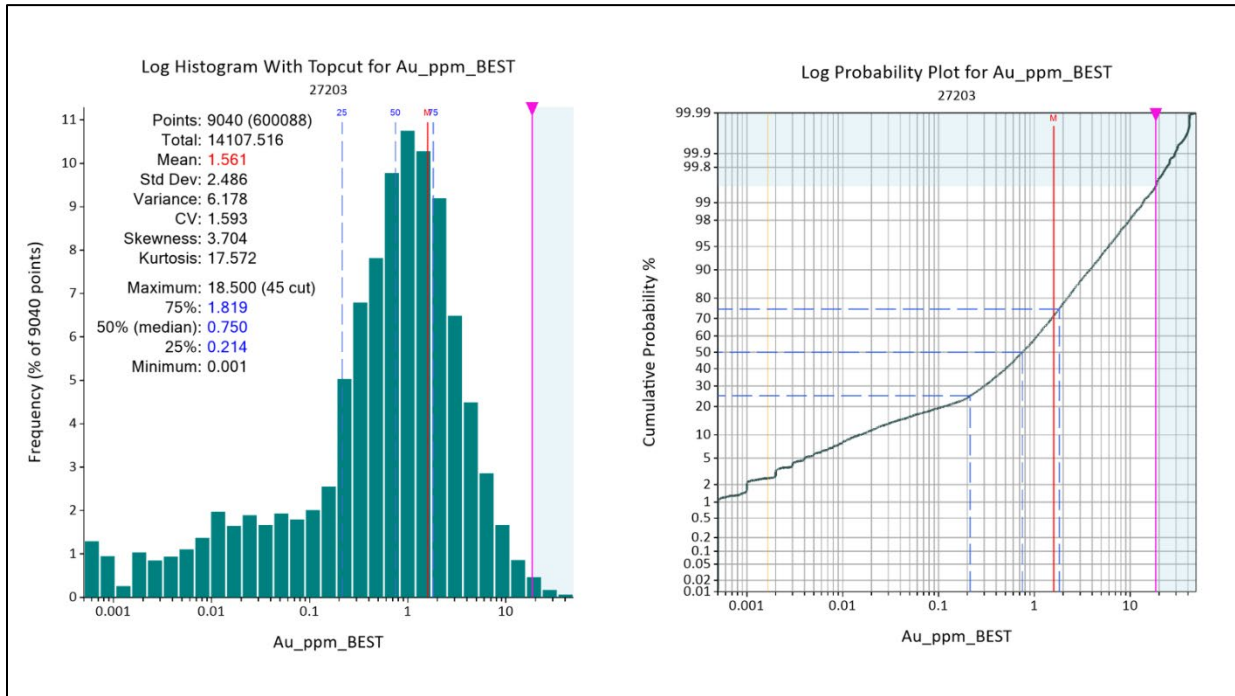
Figure 14.13
Grade Capping Analysis for the 108113 Estimation Domain With Pink Line Indicating Capping Value



Source: Micon, 2025

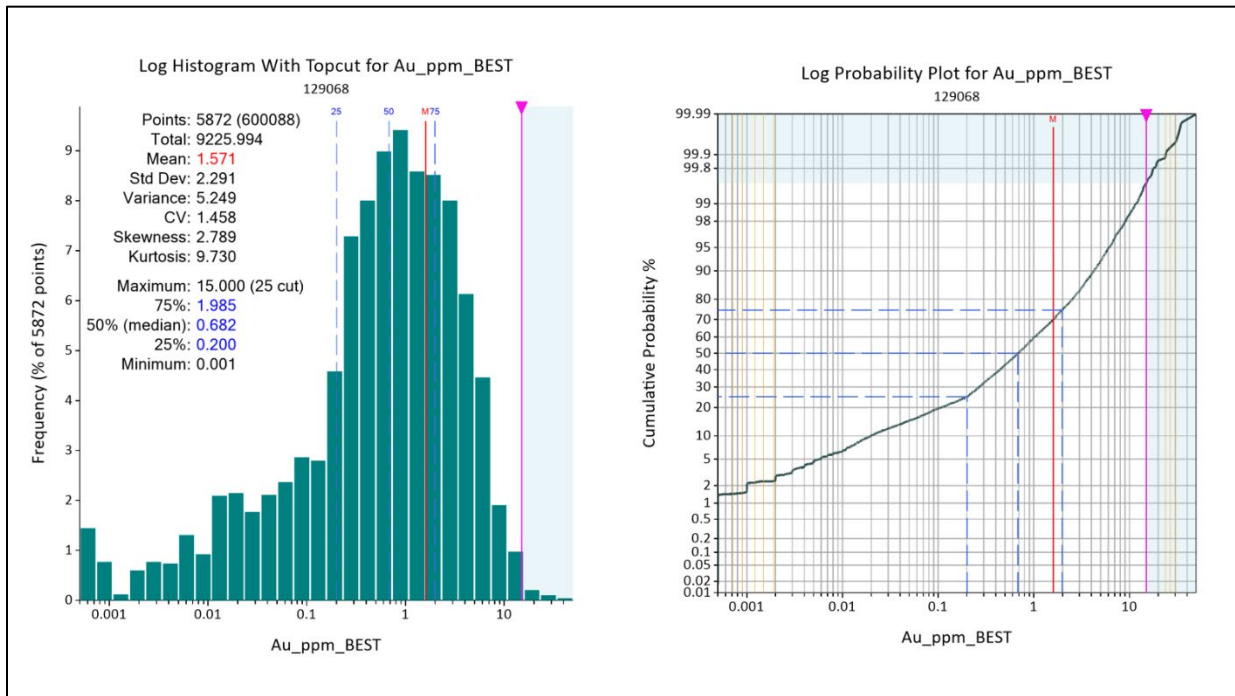


Figure 14.14
Grade Capping Analysis for the 27203 Estimation Domain



Source: Micon, 2025

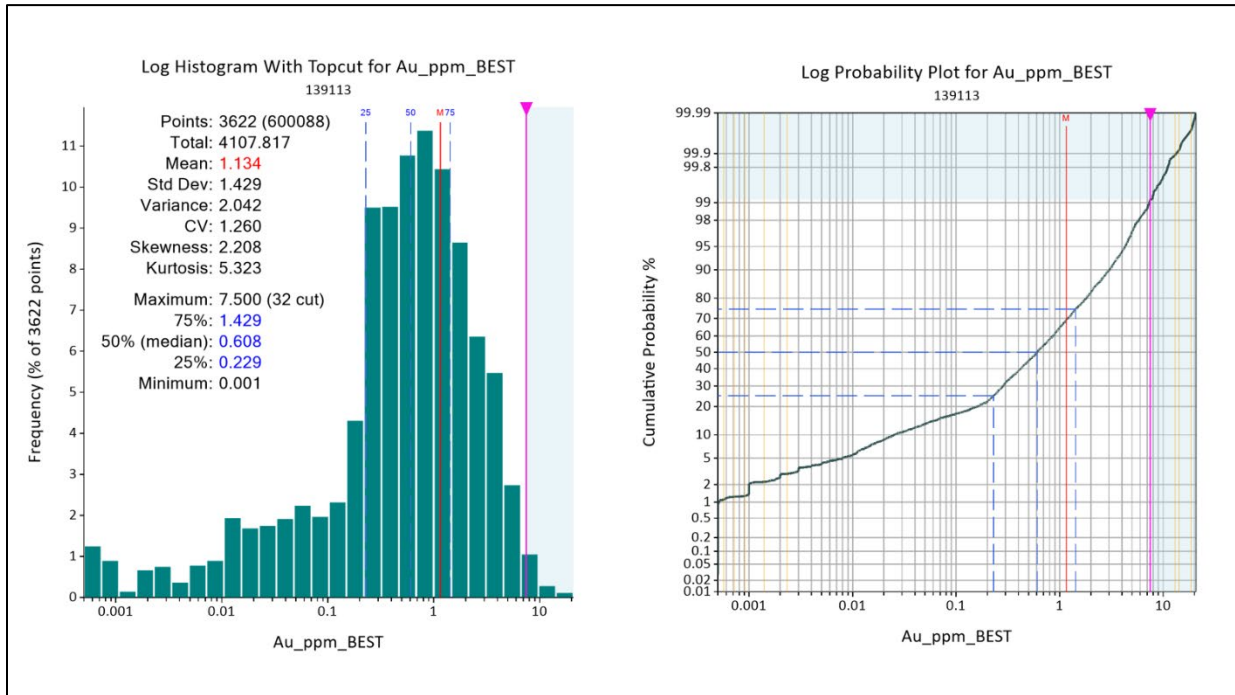
Figure 14.15
Grade Capping Analysis for the 129068 Estimation Domain



Source: Micon, 2025



Figure 14.16
Grade Capping Analysis for the 139113 Estimation Domain



Source: Micon, 2025



Table 14.7
Uncapped and Capped 1 m Gold (g/t) Composites Statistics by domains used in estimation

Domain	Count	Length	Uncapped 1m Composites									Capped 1m Composites								
			Mean	SD	CoV	Var	Min	Q1	Median	Q3	Max	Mean	SD	CoV	Var	Min	Q1	Median	Q3	Max
All	598,219	595,713	0.21	1.43	6.74	2.04	0.00	0.00	0.00	0.02	297.00	0.20	1.14	5.77	1.30	0.00	0.00	0.00	0.02	45.00
2	315,807	314,971	0.04	0.39	8.90	0.15	0.00	0.00	0.00	0.01	106.00	0.03	0.11	3.71	0.01	0.00	0.00	0.00	0.01	1.00
108113	10,574	10,323	2.61	5.97	2.29	35.62	0.00	0.24	0.81	2.43	187.46	2.55	5.14	2.02	26.45	0.00	0.24	0.81	2.43	45.00
27203	9,040	8,934	1.60	2.80	1.74	7.81	0.00	0.22	0.76	1.83	48.66	1.57	2.49	1.59	6.22	0.00	0.22	0.76	1.83	18.50
129068	5,872	5,718	1.63	2.56	1.58	6.56	0.00	0.21	0.70	2.03	49.70	1.60	2.31	1.45	5.34	0.00	0.21	0.70	2.03	15.00
139113	3,622	3,495	1.17	1.58	1.36	2.50	0.00	0.23	0.61	1.46	20.80	1.14	1.43	1.25	2.05	0.00	0.23	0.61	1.46	7.50
117135	3,246	3,157	1.67	2.91	1.75	8.45	0.00	0.19	0.66	2.00	62.99	1.61	2.40	1.49	5.75	0.00	0.19	0.66	2.00	14.00
57180	2,832	2,799	1.50	7.29	4.88	53.21	0.00	0.05	0.37	1.22	297.00	1.21	2.19	1.81	4.81	0.00	0.05	0.37	1.22	12.00
88090	2,714	2,578	1.52	3.84	2.52	14.73	0.00	0.14	0.50	1.40	81.11	1.36	2.48	1.82	6.17	0.00	0.14	0.50	1.40	15.00
97135	2,600	2,539	1.59	2.52	1.58	6.35	0.00	0.25	0.77	1.95	34.85	1.53	2.09	1.37	4.38	0.00	0.25	0.77	1.95	12.00
28180	2,529	2,503	1.58	2.66	1.68	7.06	0.00	0.13	0.75	1.89	34.40	1.53	2.23	1.46	4.95	0.00	0.13	0.75	1.89	13.00
107338	2,292	2,270	0.58	1.76	3.05	3.08	0.00	0.00	0.02	0.31	20.98	0.56	1.62	2.89	2.61	0.00	0.00	0.02	0.31	12.70
68338	1,747	1,696	2.38	8.66	3.63	75.07	0.00	0.01	0.26	1.39	120.86	2.11	6.11	2.90	37.35	0.00	0.01	0.26	1.39	45.00
139135	1,630	1,571	1.22	1.86	1.52	3.45	0.00	0.14	0.53	1.51	17.95	1.18	1.62	1.38	2.63	0.00	0.14	0.53	1.51	8.00
129090	1,621	1,555	1.59	2.72	1.71	7.40	0.00	0.12	0.59	1.82	23.70	1.57	2.58	1.65	6.63	0.00	0.12	0.59	1.82	17.00
139090	1,570	1,462	1.01	1.35	1.34	1.82	0.00	0.23	0.55	1.30	14.00	0.99	1.22	1.23	1.48	0.00	0.23	0.55	1.30	6.50
108158	1,498	1,456	1.14	2.24	1.97	5.01	0.00	0.01	0.34	1.24	35.50	1.10	1.93	1.75	3.74	0.00	0.01	0.34	1.24	13.00
129293	1,404	1,378	1.75	2.79	1.60	7.78	0.00	0.15	0.63	2.09	27.98	1.74	2.72	1.57	7.40	0.00	0.15	0.63	2.09	17.00
19158	1,347	1,319	1.42	2.66	1.87	7.07	0.00	0.05	0.48	1.62	36.24	1.36	2.24	1.65	5.03	0.00	0.05	0.48	1.62	13.00
128315	1,276	1,243	1.85	2.86	1.54	8.18	0.00	0.02	0.65	2.36	27.09	1.80	2.63	1.46	6.89	0.00	0.02	0.65	2.36	11.00
187180	1,159	1,132	1.15	1.40	1.22	1.95	0.00	0.27	0.69	1.67	16.35	1.10	1.13	1.03	1.28	0.00	0.27	0.69	1.67	4.70
189023	975	966	1.91	2.76	1.45	7.63	0.00	0.27	0.83	2.25	20.71	1.90	2.68	1.41	7.19	0.00	0.27	0.83	2.25	15.00
89135	801	778	1.53	2.85	1.86	8.13	0.00	0.22	0.57	1.68	35.84	1.47	2.39	1.62	5.70	0.00	0.22	0.57	1.68	15.00



Domain	Count	Length	Uncapped 1m Composites									Capped 1m Composites								
			Mean	SD	CoV	Var	Min	Q1	Median	Q3	Max	Mean	SD	CoV	Var	Min	Q1	Median	Q3	Max
209068	768	740	0.71	0.96	1.36	0.93	0.00	0.19	0.38	0.90	9.25	0.70	0.92	1.31	0.84	0.00	0.19	0.38	0.90	6.00
19135	644	620	1.41	2.25	1.59	5.05	0.00	0.07	0.58	1.68	20.66	1.38	2.07	1.50	4.28	0.00	0.07	0.58	1.68	11.70
159270	640	612	0.90	1.75	1.94	3.06	0.00	0.02	0.36	1.16	19.92	0.88	1.58	1.78	2.48	0.00	0.02	0.36	1.16	12.80
79135	588	564	2.49	4.99	2.01	24.86	0.00	0.14	0.77	2.92	60.79	2.19	3.28	1.50	10.77	0.00	0.14	0.77	2.92	14.00
68000	465	449	1.51	3.87	2.56	14.94	0.00	0.01	0.20	1.03	43.80	1.45	3.33	2.30	11.06	0.00	0.01	0.20	1.03	23.50
209315	418	414	1.11	1.72	1.55	2.95	0.00	0.21	0.54	1.19	12.56	1.09	1.63	1.49	2.67	0.00	0.21	0.54	1.19	9.80
57135	387	376	1.73	4.13	2.38	17.04	0.00	0.20	0.61	1.67	57.59	1.49	2.25	1.51	5.06	0.00	0.20	0.61	1.67	11.00
78338	374	370	0.55	1.64	2.97	2.69	0.00	0.00	0.01	0.26	12.96	0.54	1.53	2.84	2.33	0.00	0.00	0.01	0.26	10.20
208293	262	254	1.12	1.84	1.63	3.37	0.00	0.29	0.55	1.12	16.44	1.06	1.49	1.39	2.21	0.00	0.29	0.55	1.12	7.80
208113	225	216	1.99	3.73	1.87	13.92	0.00	0.32	1.09	2.13	35.70	1.91	3.13	1.64	9.78	0.00	0.32	1.09	2.13	19.70
17293	192	182	1.07	1.40	1.30	1.95	0.00	0.05	0.70	1.36	10.45	1.00	1.09	1.09	1.18	0.00	0.05	0.70	1.36	4.00
188338	121	115	0.64	0.66	1.03	0.43	0.00	0.23	0.42	0.87	4.45	0.62	0.57	0.92	0.32	0.00	0.23	0.42	0.87	2.50
156270	106	104	0.27	0.66	2.44	0.44	0.00	0.00	0.02	0.32	5.23	0.27	0.66	2.44	0.44	0.00	0.00	0.02	0.32	5.23
217145	97	97	1.10	2.28	2.08	5.19	0.00	0.06	0.38	0.99	17.83	1.01	1.74	1.72	3.02	0.00	0.06	0.38	0.99	9.60
78045	69	64	1.30	3.12	2.40	9.72	0.00	0.01	0.23	0.80	20.57	1.14	2.26	1.99	5.09	0.00	0.01	0.23	0.80	10.20
218300	58	57	0.92	1.17	1.27	1.37	0.00	0.09	0.45	1.33	5.46	0.87	1.03	1.18	1.06	0.00	0.09	0.45	1.33	3.50
218285	55	54	0.71	1.01	1.42	1.02	0.00	0.03	0.40	0.80	5.24	0.71	1.01	1.42	1.02	0.00	0.03	0.40	0.80	5.24

Source: Micon, 2025

Note: SD = standard deviation; CoV = coefficient of variation; Var = variance



14.5.4 Variography

Variography was completed for gold within mineralized estimation domains using Snowden Supervisor v.9.0. Variogram modelling was completed on normal scores-transformed data and modelled using a nugget obtained from downhole variograms and generally two spherical structures. The back-transformation of normal scores variograms to original units was then completed for variograms in each domain using 90 Hermite polynomials, and the orientation of the variograms were checked against the mineralization orientation for each domain in 3D prior to use in estimation. Search orientations determined from variography were used in both the inverse distance cubed (ID3) and final OK estimates used for resource reporting. Figure 14.17 shows the location of the major mineralized domain structures, 108113, 27203, 117135, 129068 and 139113 as they are the most significant domains with meaningful variograms. Plots of these major domains variograms are shown in Figure 14.18 through Figure 14.21, with results for all mineralized domains presented in Table 14.8.

Figure 14.17
Coffee Gold Project Major Geological Domains Supporting the Variographic Analysis

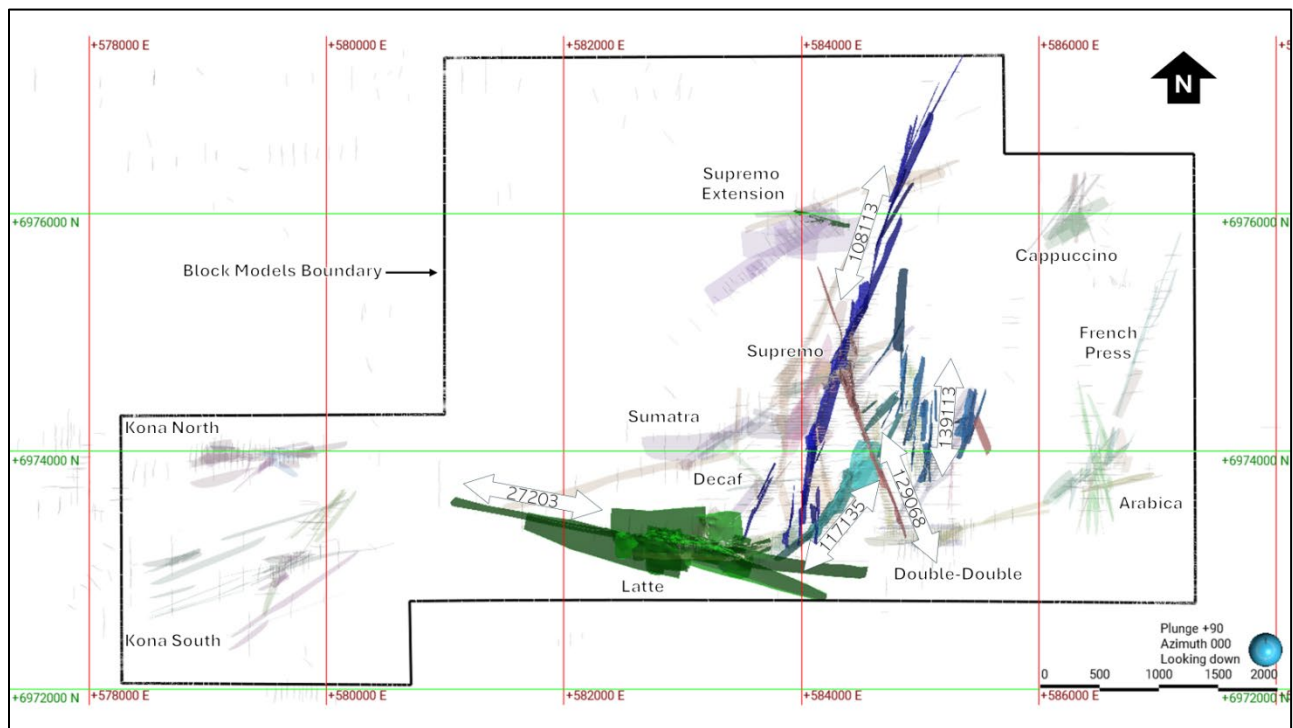
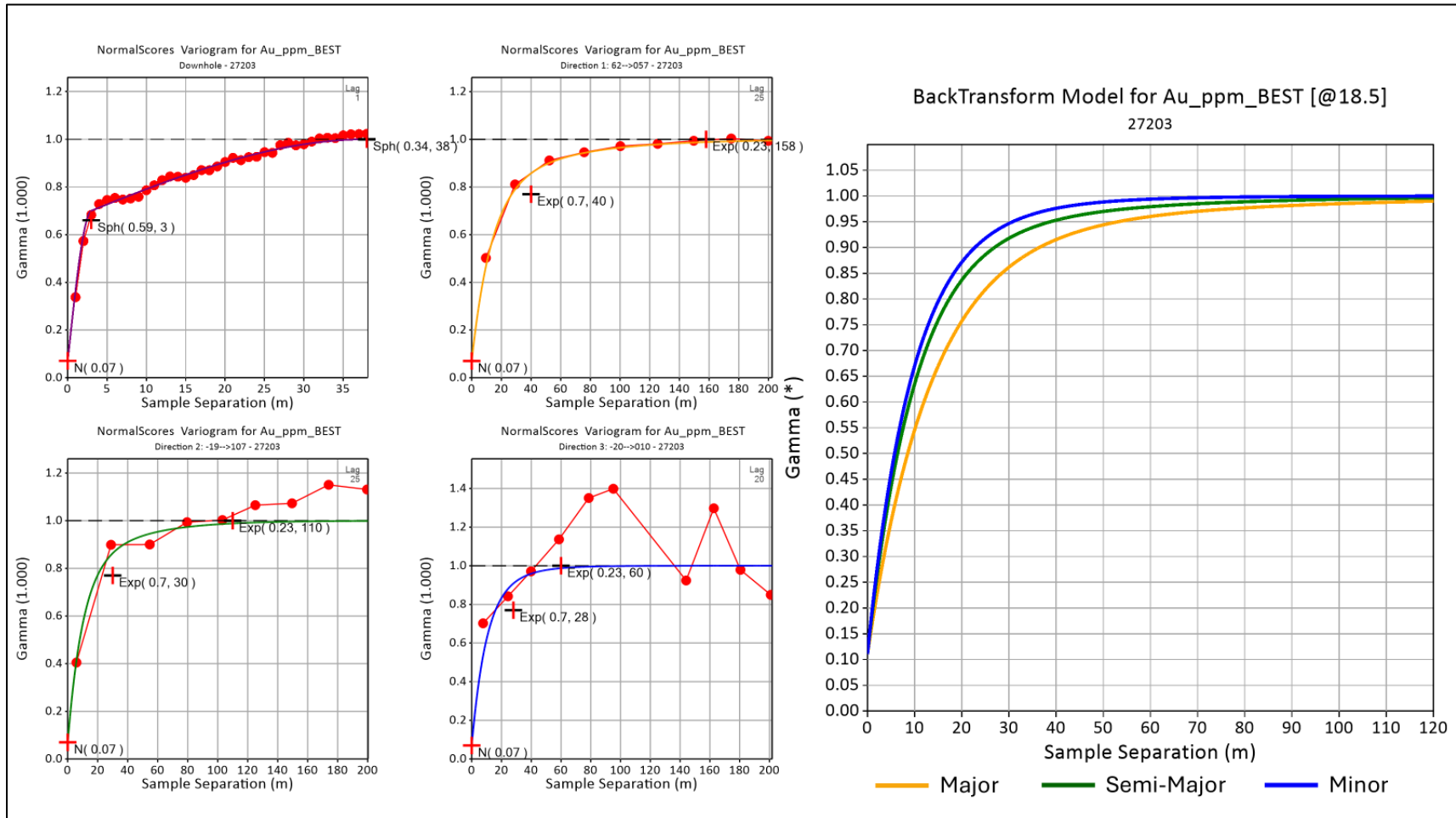




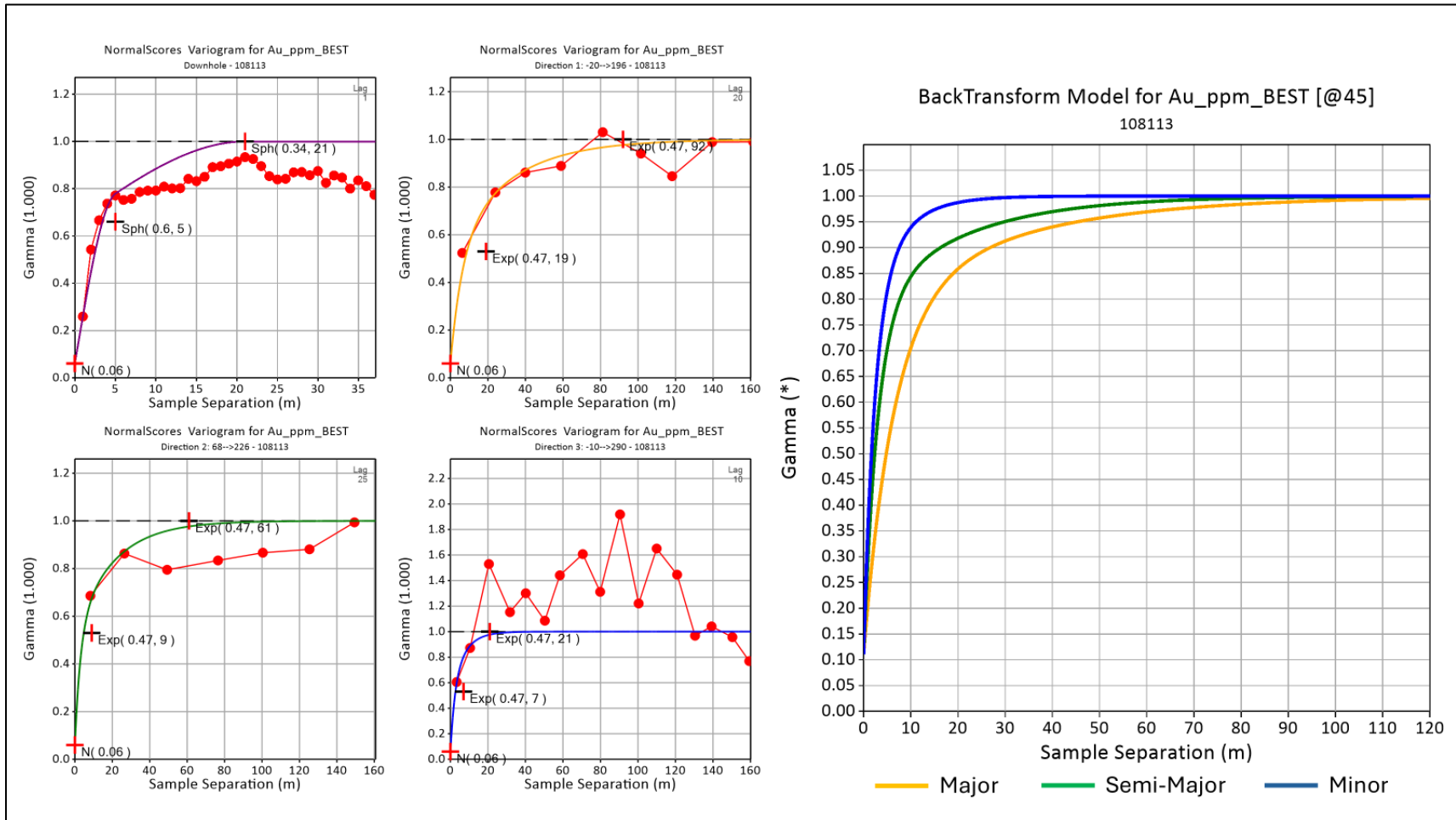
Figure 14.18
Normal Scores Variography and Back-Transform Model for Gold Estimation in the 27203 Domain



Source: Micon, 2025



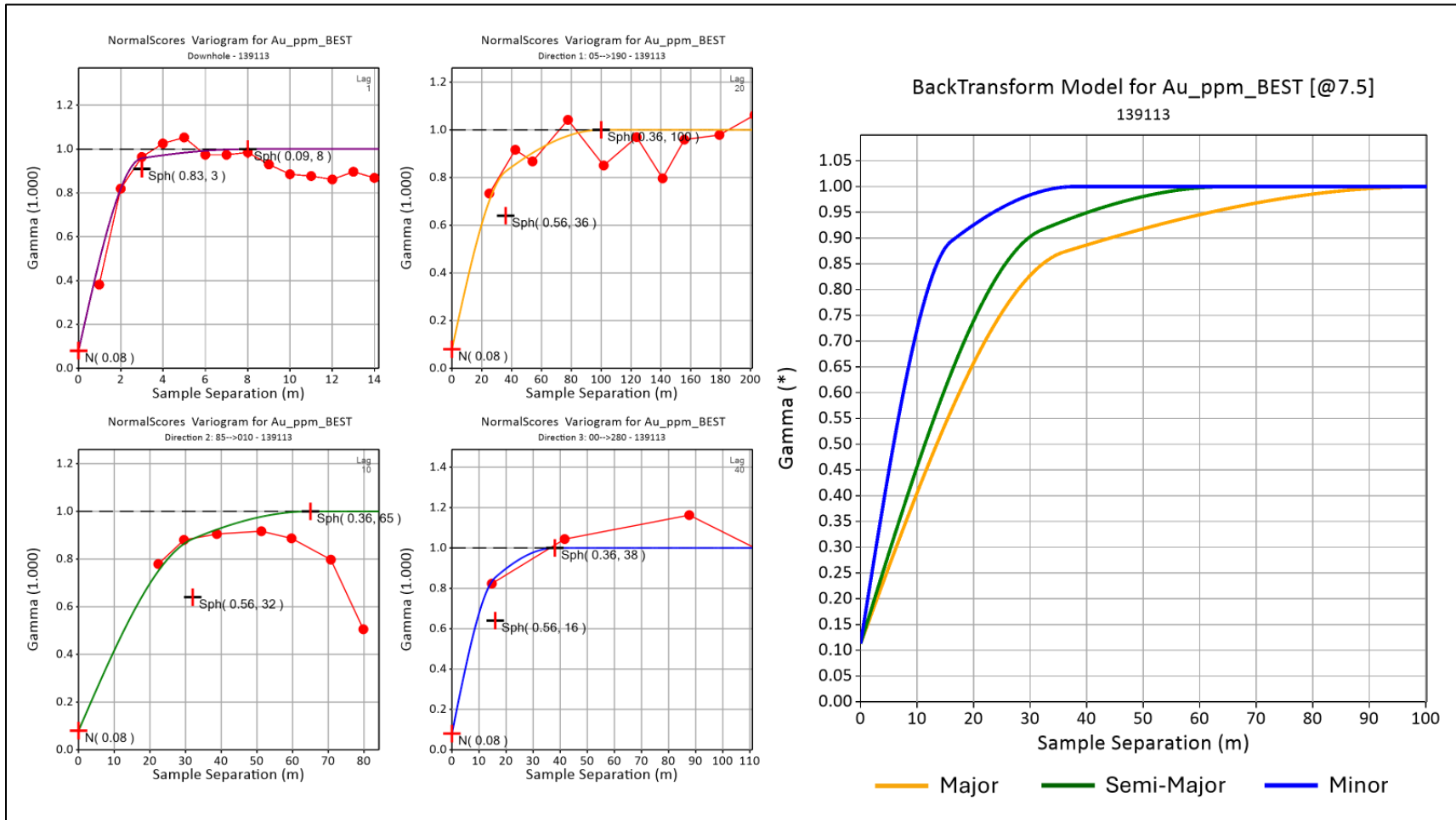
Figure 14.19
Normal Scores Variography and Back-Transform Model for Gold Estimation in the 108113 Domain



Source: Micon, 2025



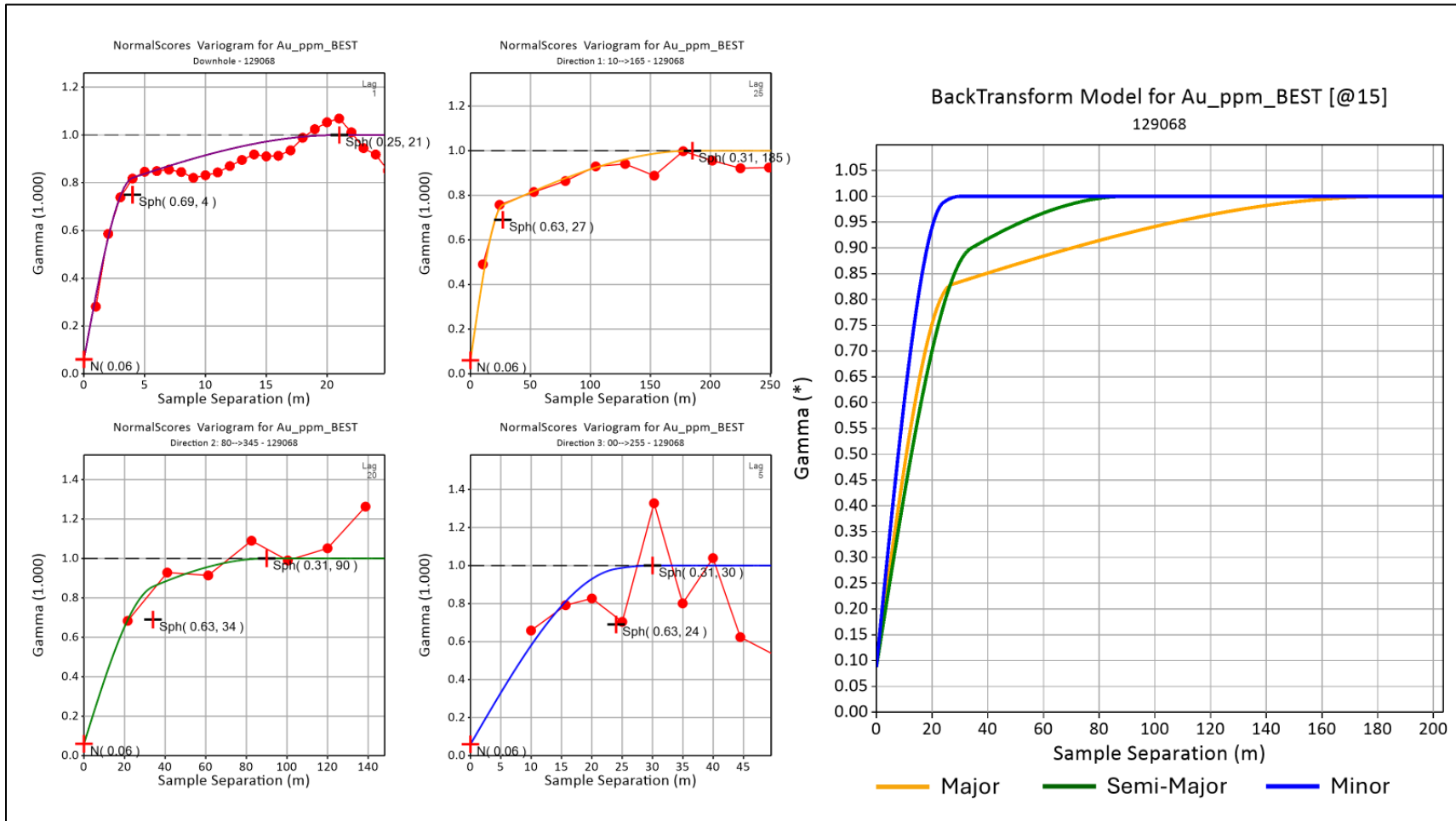
Figure 14.20
Normal Scores Variography and Back-Transform Model for Gold Estimation in the 139113 Domain



Source: Micon, 2025



Figure 14.21
Normal Scores Variography and Back-Transform Model for Gold Estimation in 129068 Domain



Source: Micon, 2025



Table 14.8
Variogram Parameters for Mineralized Domains Used in Estimation

Domain	Rotation – Snowden Supervisor			Nugget	Structure 1					Structure 2				
	Horizontal	Across Strike	Dip Plane		Type	Normalized Sill	Major (m)	Semi-Major (m)	Minor (m)	Type	Normalized Sill	Major (m)	Semi-Major (m)	Minor (m)
17293	210	185	85	0.079	Spherical	0.436	113	36	2	Spherical	0.485	185	81	10
19135	60	175	95	0.118	Spherical	0.439	28	27	43	Spherical	0.444	91	69	57
19158	75	170	95	0.105	Spherical	0.629	50	45	9	Spherical	0.266	114	109	18
27203	100	200	160	0.112	Exponential	0.802	40	30	28	Exponential	0.0863	158	110	60
28180	90	190	90	0.114	Spherical	0.518	39	47	57	Spherical	0.368	438	93	65
57135	20	225	90	0.146	Spherical	0.447	47	20	4	Spherical	0.407	89	45	10
57180	85	225	300	0.0849	Spherical	0.379	139	16	10	Spherical	0.379	173	158	45
68000	270	185	90	0.131	Spherical	0.488	146	95	13	Spherical	0.381	277	100	35
68338	255	175	295	0.14	Spherical	0.652	50	40	27	Spherical	0.208	151	96	33
78045	315	185	90	0.11	Spherical	0.89	149	149	10	-	-	-	-	-
78338	260	185	90	0.117	Spherical	0.883	132	46	12	-	-	-	-	-
79135	45	205	290	0.107	Spherical	0.537	39	45	7	Spherical	0.355	114	50	15
88090	0	185	80	0.121	Spherical	0.612	63	39	27	Spherical	0.266	260	83	41
89135	25	190	75	0.114	Spherical	0.639	53	57	35	Spherical	0.247	287	111	50
97135	20	195	75	0.0919	Spherical	0.795	33	25	22	Spherical	0.113	174	134	59
107338	250	190	170	0.0894	Spherical	0.548	26	13	5	Spherical	0.362	158	53	30
108113	20	190	290	0.111	Exponential	0.674	19	9	7	Exponential	0.215	103	61	21
108158	65	190	80	0.123	Spherical	0.609	34	49	7	Spherical	0.268	123	65	38
117135	70	190	80	0.0766	Spherical	0.507	80	49	10	Spherical	0.417	105	67	29
128315	235	205	80	0.112	Spherical	0.483	89	62	19	Spherical	0.405	432	262	54
129068	345	180	260	0.0886	Spherical	0.701	25	33	24	Spherical	0.21	185	99	30
129090	350	190	285	0.0799	Spherical	0.691	58	39	25	Spherical	0.229	324	80	62
129293	190	190	275	0.116	Spherical	0.591	23	23	8	Spherical	0.293	133	81	20



	Rotation – Snowden Supervisor				Structure 1					Structure 2				
139090	0	190	75	0.0667	Spherical	0.711	43	43	46	Spherical	0.233	145	108	62
139113	10	180	265	0.116	Spherical	0.641	36	32	16	Spherical	0.243	100	65	38
139135	210	185	70	0.0842	Spherical	0.651	63	27	32	Spherical	0.264	127	148	39
156270	195	190	75	0.0729	Spherical	0.927	89	89	15	-	-	-	-	-
159270	195	190	95	0.0859	Spherical	0.914	129	129	15	-	-	-	-	-
187180	85	195	80	0.0747	Spherical	0.599	66	15	8	Spherical	0.327	179	52	15
188338	85	175	90	0.0885	Spherical	0.911	94	68	15	-	-	-	-	-
189023	90	175	90	0.0917	Spherical	0.568	102	10	4	Spherical	0.34	153	40	19
208113	15	185	285	0.102	Spherical	0.898	108	51	20	-	-	-	-	-
108158	65	190	80	0.123	Spherical	0.609	34	49	7	Spherical	0.268	123	65	38
117135	70	190	80	0.0766	Spherical	0.507	80	49	10	Spherical	0.417	105	67	29
208293	200	185	280	0.105	Spherical	0.337	108	33	17	Spherical	0.559	284	62	20
209068	170	180	90	0.111	Spherical	0.429	59	31	14	Spherical	0.46	145	90	15
209315	50	180	90	0.122	Spherical	0.406	30	31	4	Spherical	0.472	116	69	15
217145	55	200	90	0.121	Spherical	0.879	102	102	15	-	-	-	-	-
218285	190	185	90	0.119	Spherical	0.881	100	100	15	-	-	-	-	-
218300	215	185	90	0.0739	Spherical	0.926	164	110	15	-	-	-	-	-

Notes:

1. Nugget and normalized sill values from back-transformed normal scores variograms.
2. Discretization of 2x2x2 (x/y/z) was selected for all OK estimates

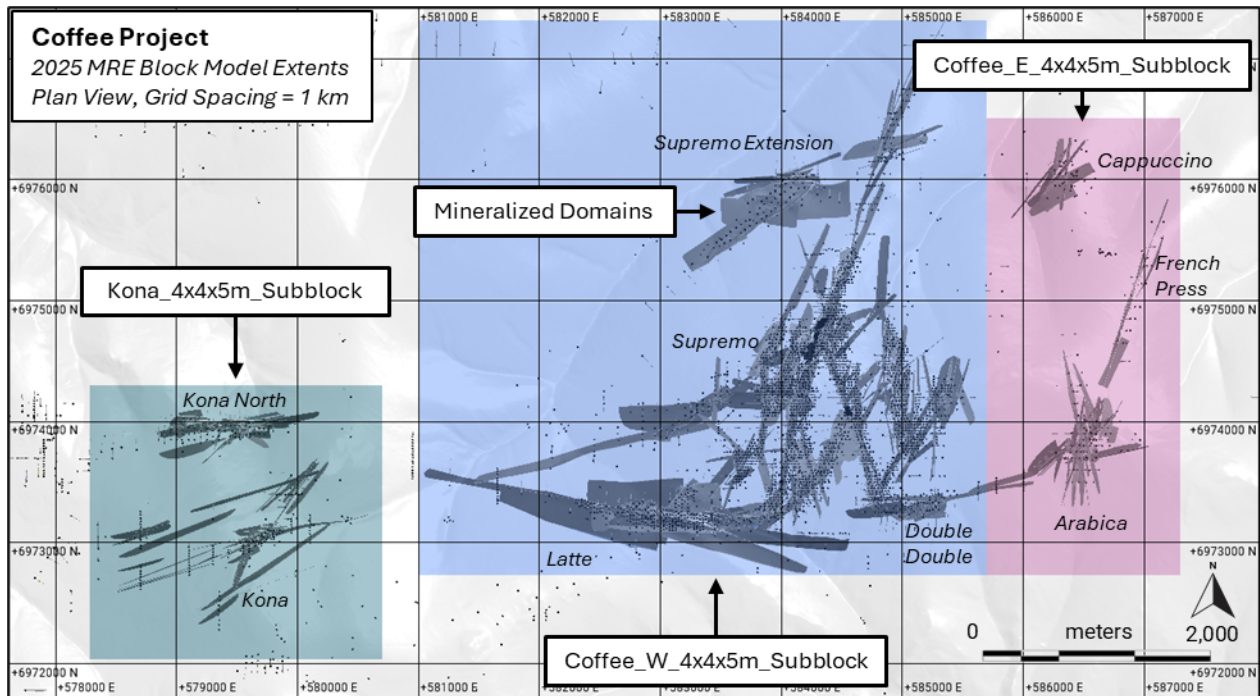


14.5.5 Block Model Set Up

14.5.5.1 Block Model

Three non-rotated 4 m x 4 m x 5 m block models were constructed in Leapfrog EDGE v.2024.1.3 in the UTM NAD83, Zone 7 N coordinate system (Figure 14.22). All models are sub-blocked along mineralized domain boundaries, with the smallest allowable sub-block measuring 1 m x 1 m x 1.25 m. The Kona_4x4x5m_Subblock model covers the Kona and Kona North targets, the Coffee_E_4x4x5m_Subblock model covers the Arabica, Cappuccino, and French Press targets, and the Coffee_W_4x4x5m_Subblock model covers the Latte, Double-Double, Supremo, and Supremo Extension targets. Block models were regularized to the 4 m x 4 m x 5 m parent block size prior to pit optimization.

Figure 14.22
Coffee Gold Project Block Model Extents



Source: Fuerte, 2025



**Table 14.9
Coffee Gold Project Block Model Parameters**

Model Build:	Leapfrog EDGE v.2024.1.3								
Coordinate System:	NAD83 / UTM Zone 7 N								
Model:	Kona_4x4x5m_Subblock			Coffee_E_4x4x5m_Subblock			Coffee_W_4x4x5m_Subblock		
Rotation:	No Rotation								
Coordinate:	East (X)	North (Y)	Elev. (Z)	East (X)	North (Y)	Elev. (Z)	East (X)	North (Y)	Elev. (Z)
Parent Block Size (m)	4	4	5	4	4	5	4	4	5
Sub-blocks (m)	1	1	1.25	1	1	1.25	1	1	1.25
Origin Centroid (m)	578,282	6,972,042	772.5	585,702	6,972,732	502.5	581,002	6,972,732	502.5
Number of Parent Blocks	605	565	126	400	944	170	396	74	166

14.5.6 Grade Interpolation

Gold grades were estimated by (ID3, OK, and nearest neighbour (NN) in all mineralized domains. Search ellipse orientation and radii were selected based on variogram models for each individual estimation domain, with variable search orientation (VO) applied according to the nearest vein midpoint surface. Initial search parameters were selected using Kriging Neighborhood Analysis and were then refined based on results from preliminary model validation checks. A two-pass search strategy was applied for mineralized domains, with search ellipse distances doubled in the second estimation pass. A single pass was applied for the Background domain. Estimation parameters for all domains are summarized in Table 14.10.

Ordinary Kriging was selected as the final estimation method because it reconciles well with NN and ID3 estimates and shows favourable results in the model validation checks.



**Table 14.10.
Coffee Gold Project Estimation Parameters**

Domain	Leapfrog Search Orientation			Pass 1 Data Search						Pass 2 Data Search						High-Grade Restriction			
	Dip	Dip Azi.	Pitch	Major	Semi-Maj.	Minor	Min. Samples	Max. Samples	Max Samples/Hole	Major	Semi-Maj.	Minor	Min. Samples	Max. Samples	Max Samples/Hole	Threshold (g/t)	Major	Semi-Maj.	Minor
17293	85	300	5	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
19135	85	330	5	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
19158	80	345	5	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
27203	70	190	110	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
28180	80	180	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
57135	45	110	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
57180	45	175	150	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
68000	85	180	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
68338	85	165	25	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
78045	85	45	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
78338	85	350	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
79135	65	135	160	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
88090	85	90	10	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
89135	80	115	15	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
97135	75	110	15	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
107338	80	340	100	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
108113	80	110	160	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
108158	80	155	10	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
117135	80	160	10	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
128315	65	325	10	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
129068	90	255	170	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
129090	80	80	165	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-



Domain	Leapfrog Search Orientation			Pass 1 Data Search						Pass 2 Data Search						High-Grade Restriction			
	Dip	Dip Azi.	Pitch	Major	Semi-Maj.	Minor	Min. Samples	Max. Samples	Max Samples/Hole	Major	Semi-Maj.	Minor	Min. Samples	Max. Samples	Max Samples/Hole	Threshold (g/t)	Major	Semi-Maj.	Minor
129293	80	280	175	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
139090	80	90	15	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
139113	90	280	175	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
139135	85	300	20	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
156270	80	285	15	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
159270	80	285	175	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
187180	75	175	10	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
188338	85	355	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
189023	85	0	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
208113	85	105	165	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
208293	85	290	170	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
209068	90	80	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
209315	90	320	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
217145	70	145	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
218285	85	280	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
218300	85	305	0	80	80	10	9	20	4	160	160	20	1	12	4	-	-	-	-
0	0	0	90	50	50	50	4	12	4	-	-	-	-	-	-	0.5	10	10	10
2	0	0	90	80	80	10	9	20	4	160	160	20	1	12	4	1	10	10	10

Notes:

1. The search ellipse orientations shown above are the global plunge direction for each domain. Local search orientation is determined from variable orientation models.
2. All search distances in metres.



14.5.7 Bulk Density Modelling

Samples for SG measurements were typically selected at 10 m intervals down most of the diamond drillholes. In some drillholes the frequency of SG measurements increases within the interpreted mineralized domains. SG data are only collected from diamond drill (core) holes. Although the SG database is relatively large, the distribution of SG data is locally sparse and therefore is insufficient to support direct interpolation into model blocks.

The primary controls on density are a combination of the host lithology, oxidation, and whether within a mineral domain. A relationship also exists between the density of rocks and the intensity of oxidation, a common feature in deposits of this type. Average densities have been determined by oxide type, lithology, and mineral domain, and are assigned to model blocks as shown in Table 14.11. The values shown are based on a total of 10,262 samples calculated for SG.

Table 14.11
Density values (g/cm³) assigned to Coffee Gold Project block models

Domain	Biotite Schist (BtS)		Felsic and Mixed Mafic Gneiss (FG+MxM)		Dike and Breccia (DK+BX)		Granite (GG)	Meta-basalt and Amphibolite (MBSlt+Amph)	Mafic-ultramafic (UM)
	Inside Mineral Domains	Outside Mineral Domains	Inside Mineral Domains	Outside Mineral Domains	Inside Mineral Domains	Outside Mineral Domains	Inside and Outside Mineral Domains	Inside and Outside Mineral Domains	Inside and Outside Mineral Domains
Oxide	2.58	2.57	2.56	2.55	2.55	2.52	2.50	2.56	2.57
Upper Trans	2.58	2.59	2.61	2.56	2.59	2.56	2.51	2.63	2.63
Middle Trans	2.62	2.6	2.61	2.57	2.63	2.58	2.52	2.64	2.69
Lower Trans	2.67	2.66	2.62	2.59	2.65	2.60	2.53	2.67	2.72
Sulphide	2.67	2.67	2.66	2.65	2.67	2.65	2.56	2.71	2.82

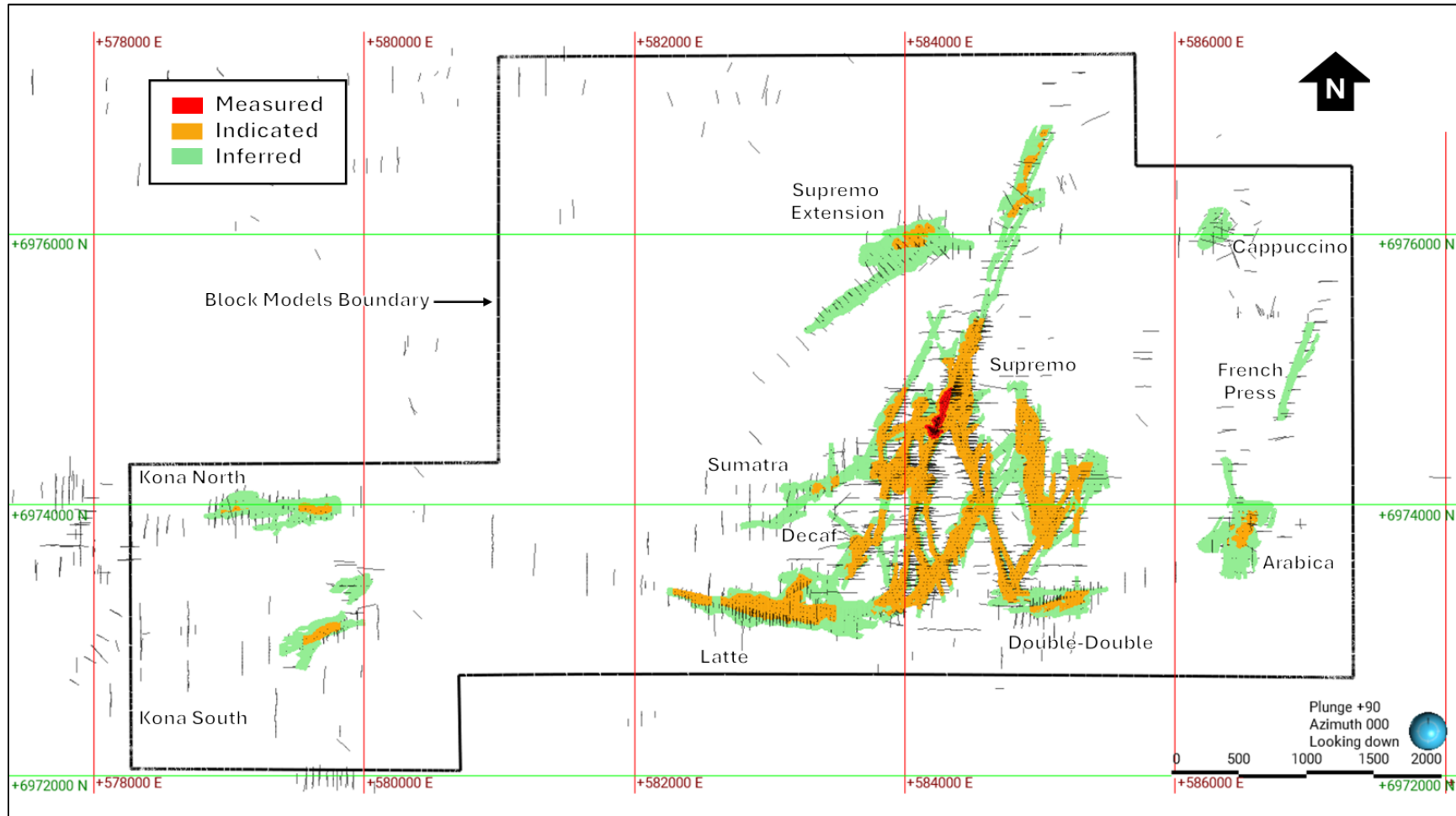


14.5.8 Mineral Resource Classification

Mineral resources are classified under the Measured, Indicated, and Inferred categories, in accordance with the 2019 CIM Definition Standards. Data spacing sufficient for each category was determined by calculating the weighted average distance at which the direction 1 variogram models for mineralized domains reach 50% of the normalized sill ($\text{Gamma} = 0.5$) for Measured resources, 80% of the normalized sill ($\text{Gamma} = 0.8$) for Indicated resources, and 95% of the normalized sill ($\text{Gamma} = 0.95$) for Inferred resources. Weights for each domain were assigned according to their total gold ounce inventory reported from the block models. As a result of this analysis, Inferred resources were categorized based on a three-hole drill spacing of 60 m or less, Indicated resources were categorized based on a three-hole drill spacing of 30 m or less, and Measured resources were categorized based on a three-hole drill spacing of 15 m or less. Blocks which meet the Measured mineral resources data spacing threshold, but which are not supported by the RC-Spacing grade control test drill program completed in 2017 and 2018 were downgraded to Indicated. All blocks in the Background domain (Code0) were downgraded waste for pit optimization. Figure 14.23 shows classified blocks for all three block models.



Figure 14.23
Coffee Gold Project Mineral Resource Classification Map



Source: Micon, 2025



14.5.9 Model Validation

Validation checks are focused on mineralized domains, which are the foundation of the entire grade interpolation boundaries. The models were validated using the following methods:

- Statistical comparison (ID3, NN, and OK).
- Sectional validation – visual comparison between block grades and composite grades.
- Swath plots.

14.5.9.1 Statistical Comparison

Statistics for the final OK estimates were compared to the NN and OK estimates, globally and by domain. The difference in average estimated grade between the OK and NN estimates is less than 5% for 25 of the 38 mineralized domains and is less than 10% for 36 of the mineralized domains. Two domains, 78338 and 156270, show variance greater than 10%, but this is not considered material given that these domains collectively represent only 0.3% of the total mineralized domain tonnage across the Project. The difference in average estimated grade between the OK and ID3 estimates is less than 5% for 29 of the 38 mineralized domains and is less than 10% for all domains but one (78045).

The final OK estimates were also compared against estimates prepared using the uncapped composite dataset, to evaluate metal loss. The search parameters for the uncapped estimate were otherwise kept identical to the final OK estimates. Metal loss due to capping is less than 5% for most mineralized domains and is less than 10% for all mineralized domains aside from 78338, a sparsely drilled, low-tonnage domain in which the metal loss is driven by one extremely high-grade outlier in the uncapped dataset. Table 14.12 shows the comparison between the various estimation methods for gold by domain.

Table 14.12
Estimated Mean Comparison between Capped and Uncapped (OK) Models

Domain	Capped Comp. Mean (g/t)	Comp. Count	Au_OK Capped (g/t)	Au_OK Uncapped (g/t)	OK Capped vs. OK Uncapped	Au_NN Capped (g/t)	OK Capped vs. NN Capped	Au_ID3 Capped (g/t)	OK Capped vs. ID3 Capped	Domain Tonnes
17293	1.01	192	0.72	0.75	-3.90%	0.77	-7.10%	0.77	-7.10%	4,894,350
19135	1.36	644	1	1.01	-1.10%	1.03	-3.00%	1.05	-4.80%	7,322,488
19158	1.34	1347	0.53	0.53	-0.50%	0.56	-5.50%	0.55	-4.50%	2,496,244
27203	1.58	9040	0.82	0.83	-0.40%	0.84	-2.10%	0.84	-2.20%	9,861,747
28180	1.58	2529	1.23	1.23	-0.30%	1.27	-3.10%	1.27	-3.00%	2,243,347
57135	1.57	387	0.88	0.9	-2.40%	0.87	1.00%	0.86	1.70%	2,604,503
57180	1.3	2832	1.21	1.25	-2.90%	1.21	0.00%	1.2	0.80%	4,681,459
68000	1.42	465	1.04	1.14	-8.60%	1	4.30%	1.03	1.30%	1,297,316
68338	2.22	1747	1.04	1.07	-2.30%	1.13	-7.50%	1.15	-8.90%	2,476,147
78045	1.1	69	0.38	0.39	-2.30%	0.37	2.40%	0.49	-22.00%	509,800
78338	0.53	374	0.65	0.84	-22.60%	0.52	24.70%	0.65	0.80%	590,913
79135	2.37	588	1.53	1.56	-1.90%	1.66	-7.50%	1.65	-7.30%	1,935,797



Domain	Capped Comp. Mean (g/t)	Comp. Count	Au_OK Capped (g/t)	Au_OK Uncapped (g/t)	OK Capped vs. OK Uncapped	Au_NN Capped (g/t)	OK Capped vs. NN Capped	Au_ID3 Capped (g/t)	OK Capped vs. ID3 Capped	Domain Tonnes
88090	1.44	2714	1.02	1.04	-2.10%	0.95	6.90%	1.08	-5.20%	8,495,563
89135	1.45	801	1.04	1.12	-6.90%	1.04	0.40%	1.03	1.40%	4,656,353
97135	1.57	2600	1.24	1.24	0.10%	1.2	3.60%	1.27	-2.10%	4,700,856
107338	0.56	2292	0.87	0.9	-2.60%	0.82	6.80%	0.88	-1.00%	9,712,513
108113	2.53	10547	1.51	1.53	-1.00%	1.47	2.50%	1.51	0.40%	0,067,094
108158	1.1	1498	1.13	1.14	-1.00%	1.22	-7.50%	1.21	-6.70%	4,468,225
117135	1.61	3246	1.37	1.38	-1.00%	1.34	2.30%	1.4	-2.30%	8,570,841
128315	1.75	1276	1.83	1.84	-0.30%	1.86	-1.60%	1.85	-0.90%	6,730,613
129068	1.59	5318	1.26	1.27	-0.90%	1.31	-3.90%	1.3	-3.70%	4,049,613
129090	1.53	1621	1.33	1.36	-1.70%	1.38	-3.10%	1.4	-4.40%	3,780,234
129293	1.71	1404	1.36	1.36	-0.10%	1.4	-2.90%	1.39	-2.20%	1,744,119
139090	0.98	1570	0.79	0.79	-0.10%	0.84	-6.00%	0.84	-6.30%	5,030,622
139113	1.16	3622	0.79	0.79	0.10%	0.78	1.30%	0.8	-0.70%	8,557,275
139135	1.19	1630	0.86	0.86	-0.60%	0.85	1.40%	0.89	-3.40%	7,145,481
156270	0.28	106	0.49	0.5	-0.10%	0.57	-12.50%	0.55	-9.70%	208,503
159270	0.88	640	0.97	0.98	-0.50%	0.99	-1.50%	0.96	1.10%	5,089,134
187180	1.14	1159	0.94	0.95	-1.00%	0.89	5.30%	0.94	0.00%	1,262,644
188338	0.64	121	0.62	0.62	-0.50%	0.62	-1.20%	0.62	-0.50%	603,128
189023	1.88	975	1.63	1.64	-0.60%	1.59	2.60%	1.68	-2.70%	4,636,438
208113	1.88	225	1.33	1.34	-1.00%	1.45	-8.30%	1.43	-7.10%	3,587,606
208293	1.12	262	0.75	0.77	-2.80%	0.78	-3.40%	0.77	-2.10%	7,035,100
209068	0.7	768	0.71	0.73	-3.70%	0.71	-0.70%	0.71	-0.50%	3,219,044
209315	1.09	418	0.81	0.82	-0.50%	0.86	-5.30%	0.84	-3.20%	5,255,294
217145	1	97	1.03	1.09	-5.80%	1.02	0.40%	1.06	-3.40%	1,736,297
218285	0.7	55	0.63	0.63	-0.10%	0.62	1.50%	0.63	-0.80%	2,452,681
218300	0.87	58	0.53	0.55	-4.20%	0.52	1.70%	0.52	1.90%	2,241,606

Notes:

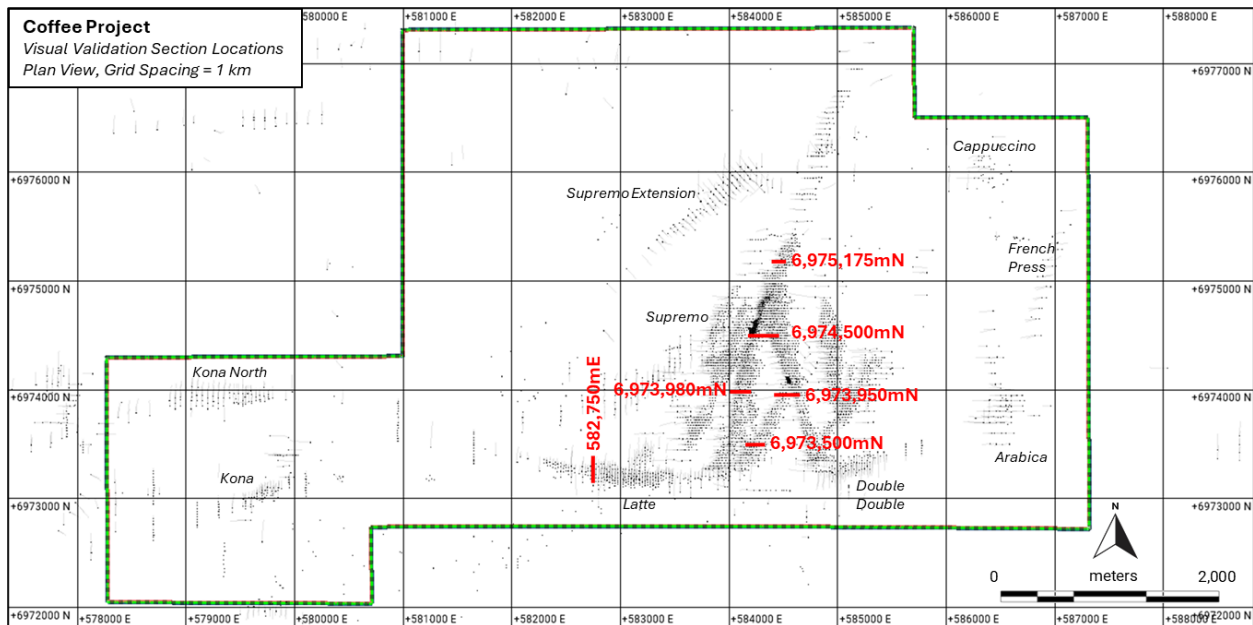
1. Au_ID3, Au_NN, and Au_OK values are estimated using the capped composite dataset.
2. Au_OK Uncapped is estimated using uncapped composites, with the same search parameters as for Au_OK.
3. Numbers are global values, no resource classification constraint or pit constraint applied.



14.5.9.2 Sectional Validation

Estimated gold block grades, resource classification, lithology model and mineralization wireframe assignment to blocks, and drillhole composite data were compared visually in plan and cross section for all domains. Visual validation demonstrates that OK-estimated block grades reproduce the composite grades well. Figure 14.24 through Figure 14.30 show locations for the several examples comparing estimated block grades to the informing composites.

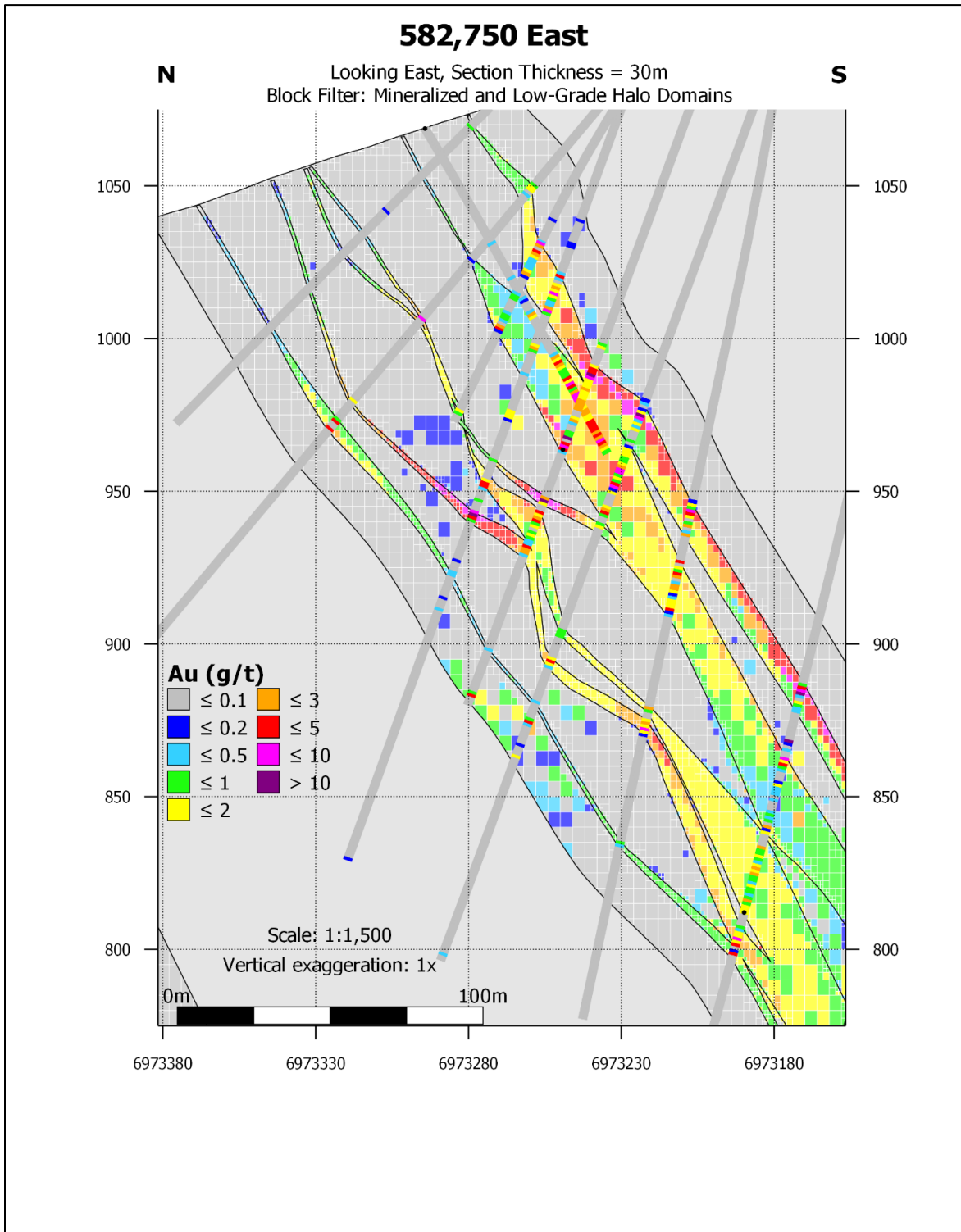
Figure 14.24
Coffee Gold Project – Visual Validation Vertical Section locations Map



Source: Fuerte, 2025



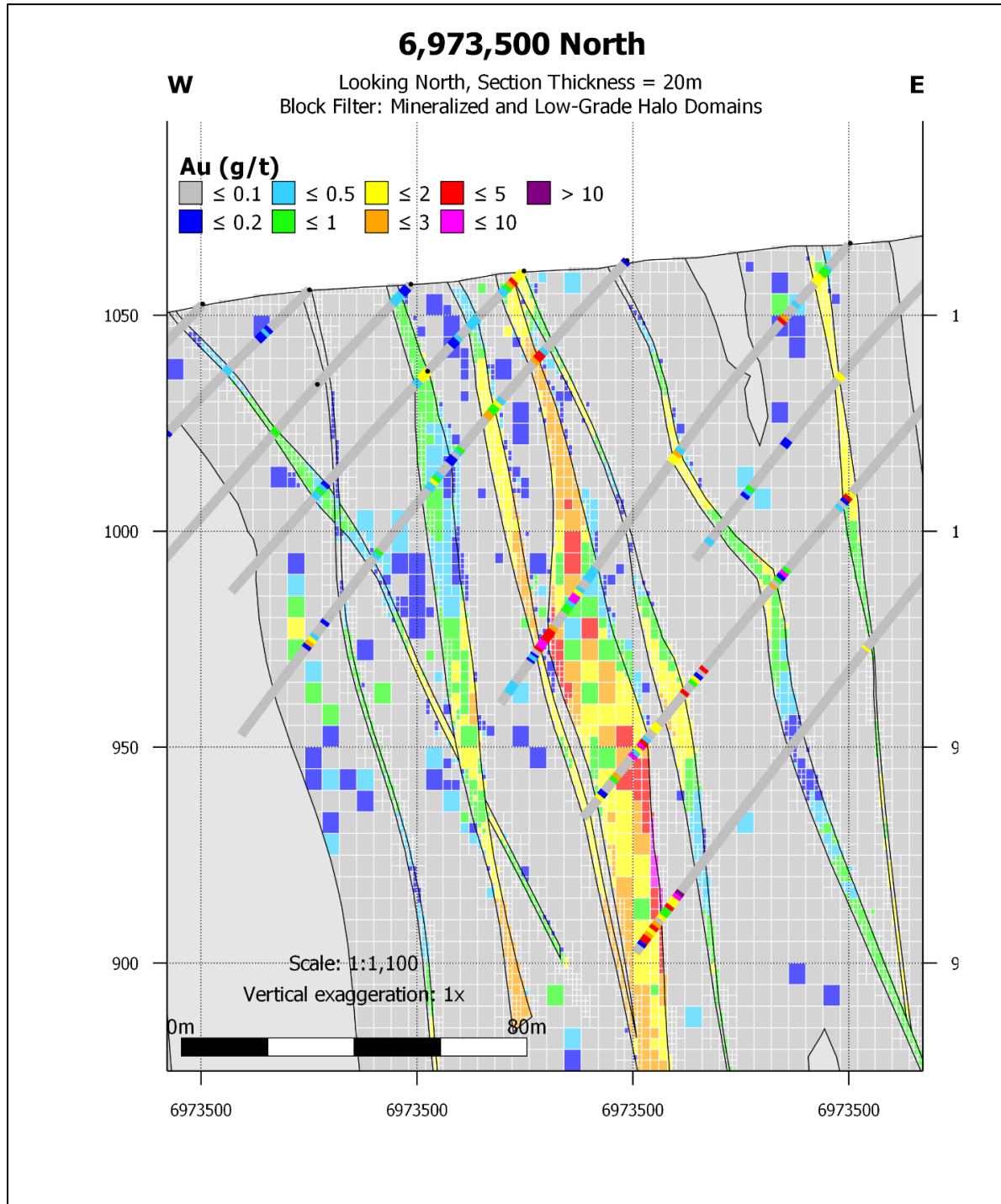
Figure 14.25
Validation Vertical Section 582,750mE



Source: Fuerte, 2025



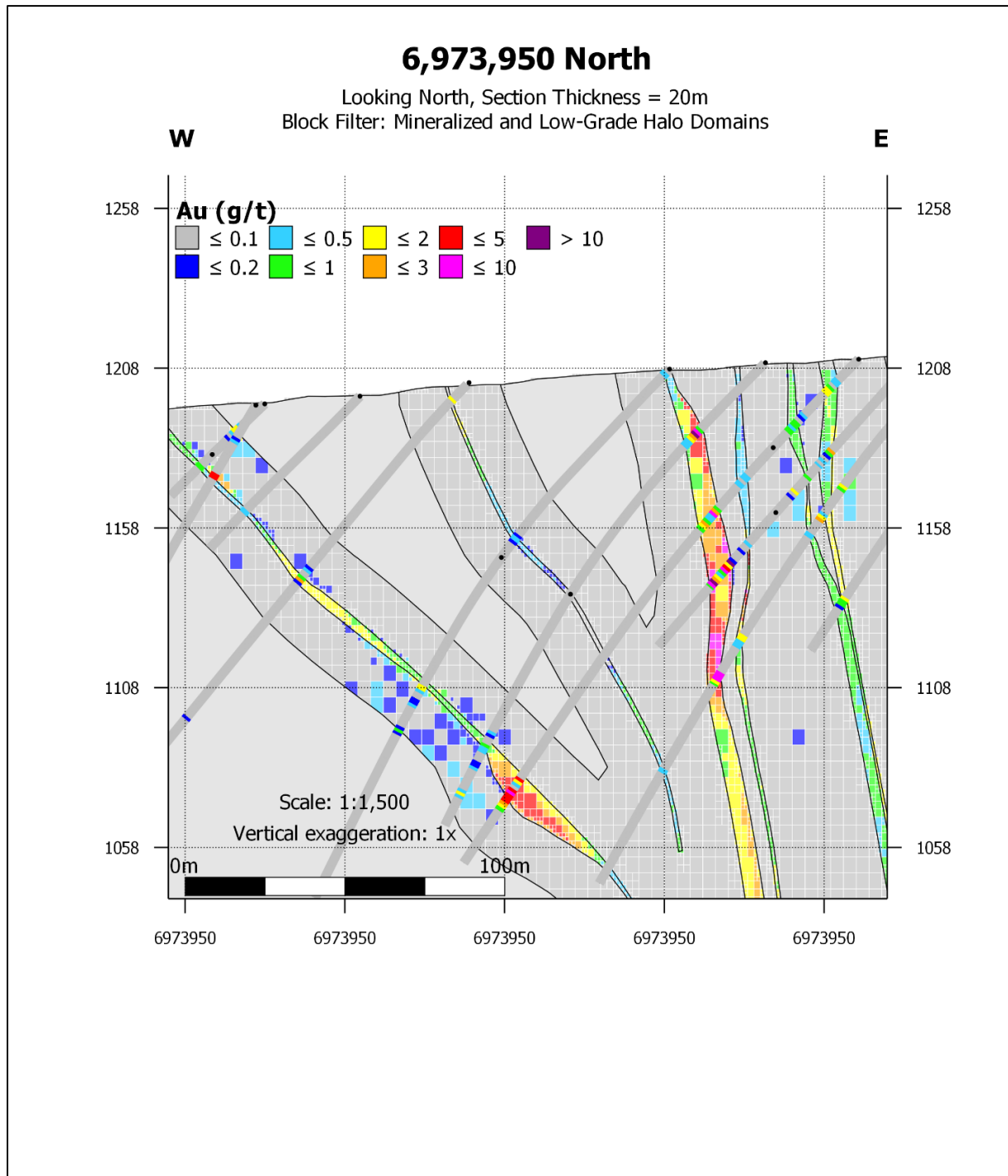
Figure 14.26
Validation Vertical Section 6,793,500mN



Source: Fuerte, 2025



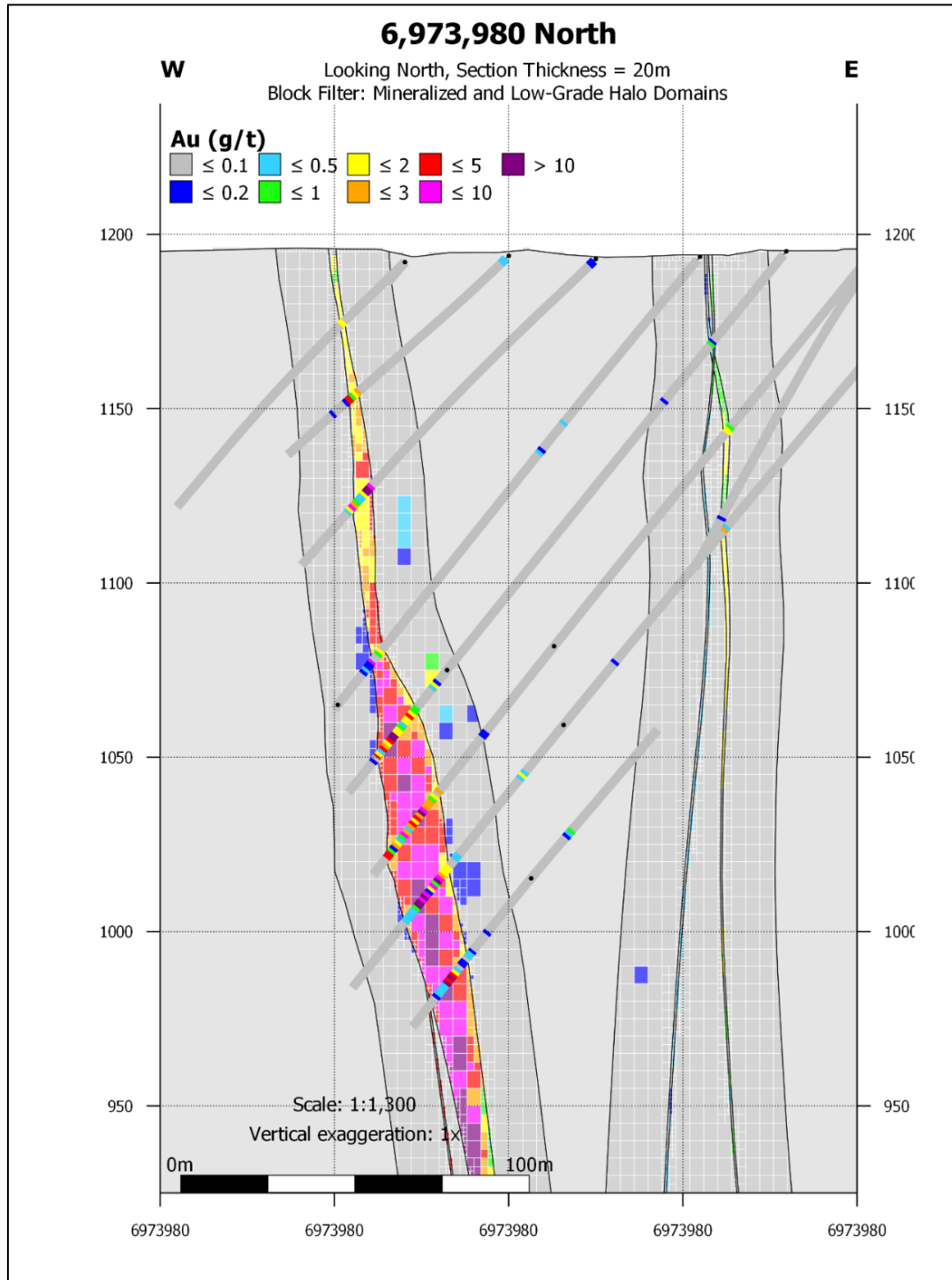
Figure 14.27
Validation Vertical Section 6,973,950mN



Source: Fuerte, 2025



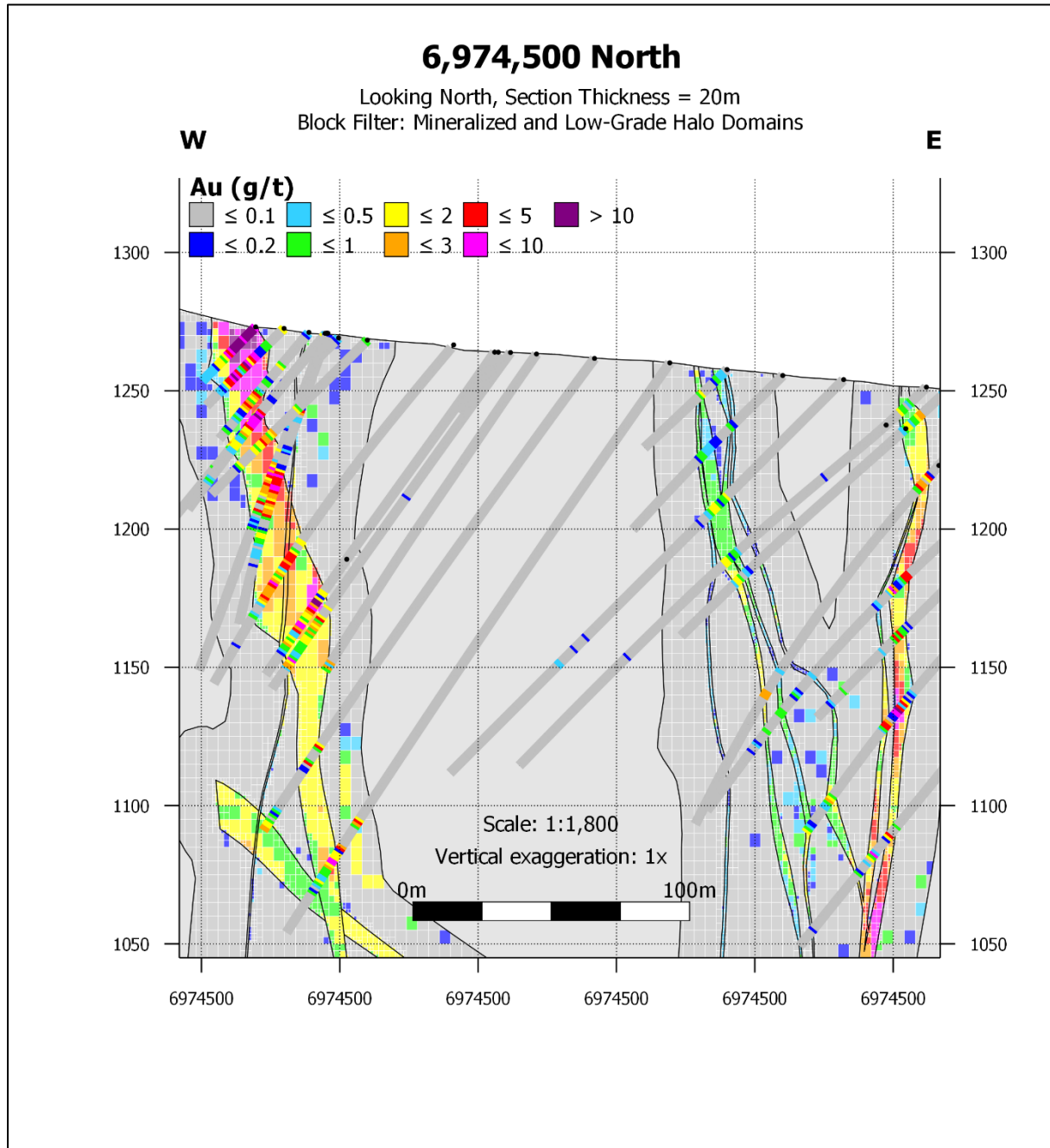
Figure 14.28
Validation Vertical Section 6,973,980mN



Source: Fuerte, 2025



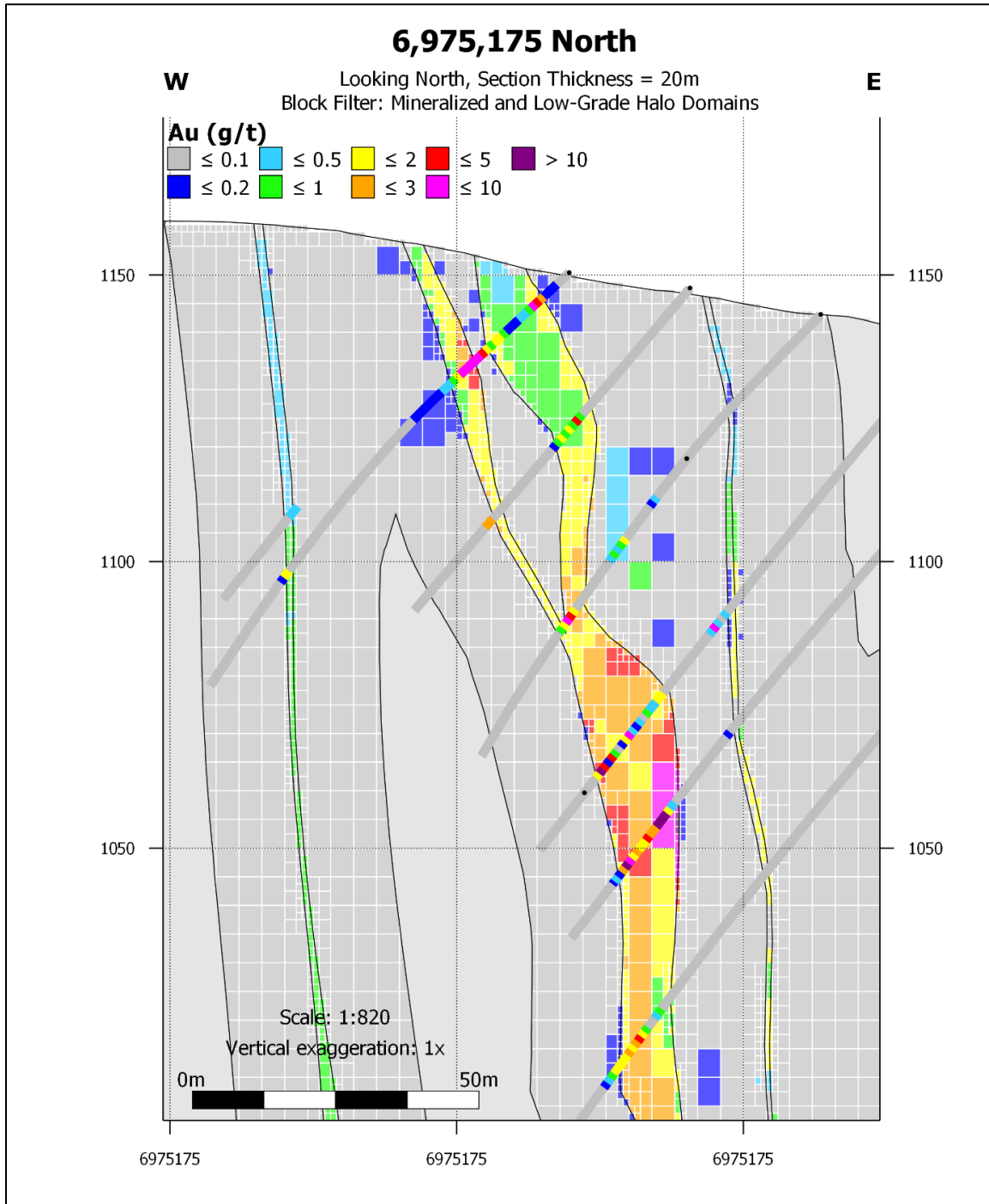
Figure 14.29
Validation Vertical Section 6,974,500mN



Source: Fuerte, 2025



Figure 14.30
Validation Vertical Section 6,975,175mN



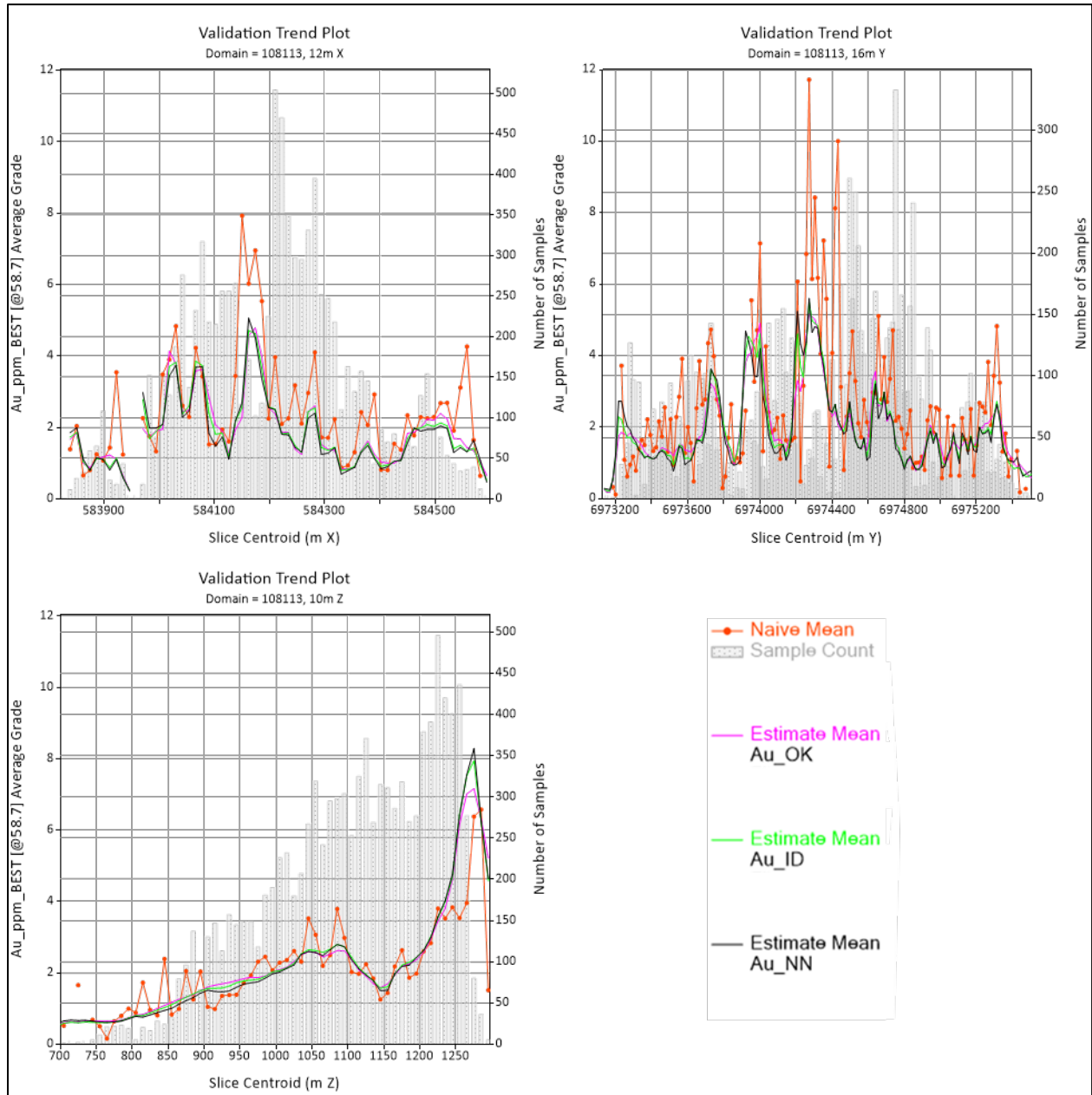


14.5.9.3 *Swath Plots*

Swath plots were generated for each mineralized domain to compare the OK, ID3, and NN estimates against one another and against composite grades. Results demonstrate that the OK estimates for gold in mineralized domains do not show a systematic high or low bias against the NN estimate or composites, and that the estimated grades for all three methods reflect the composite grades well in easting, northing, and elevation. Figure 14.31 through Figure 14.34 show examples from several mineralized domains.



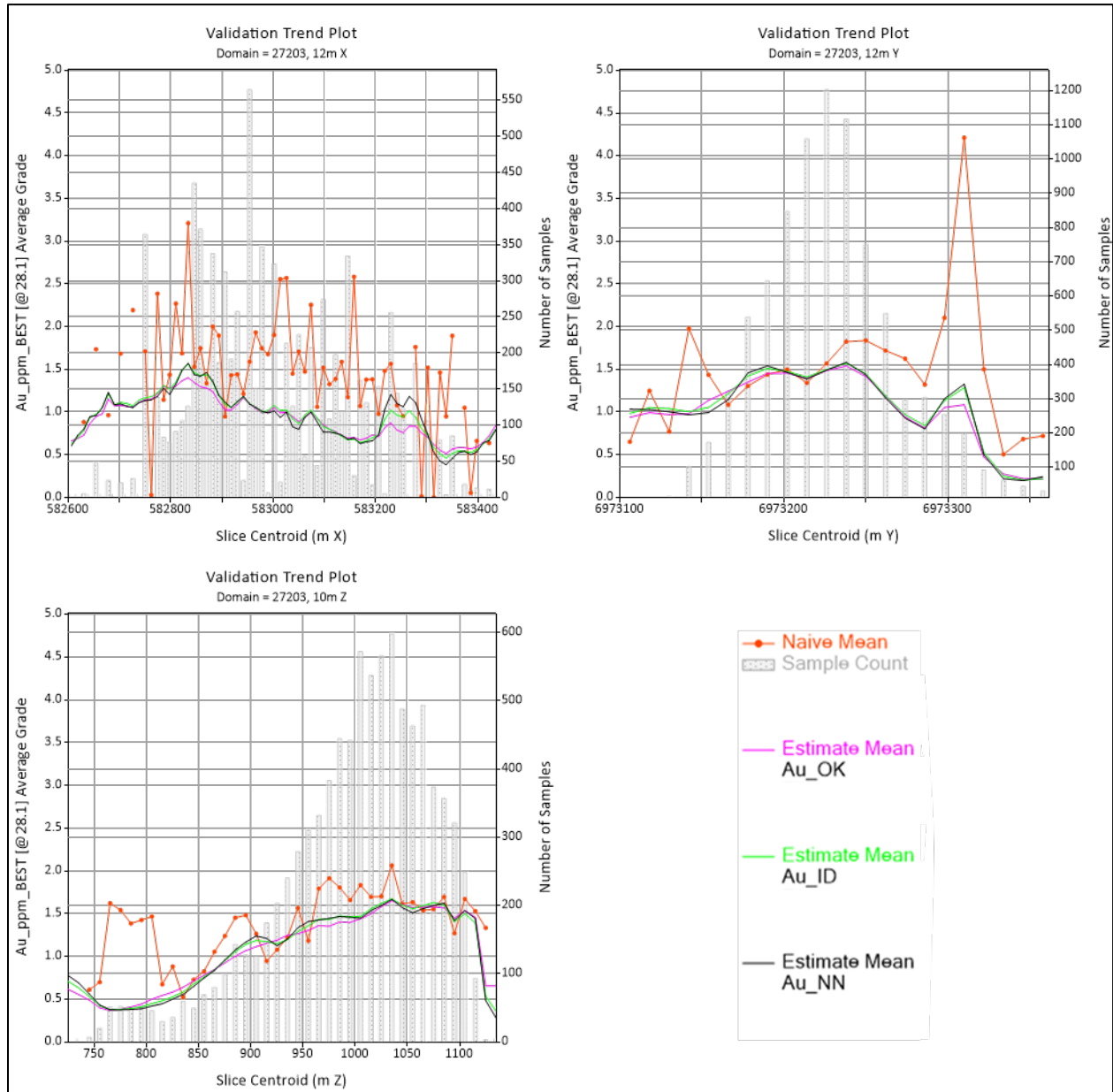
Figure 14.31
Swath Plots from Domain 108113



Source: Fuerte, 2025



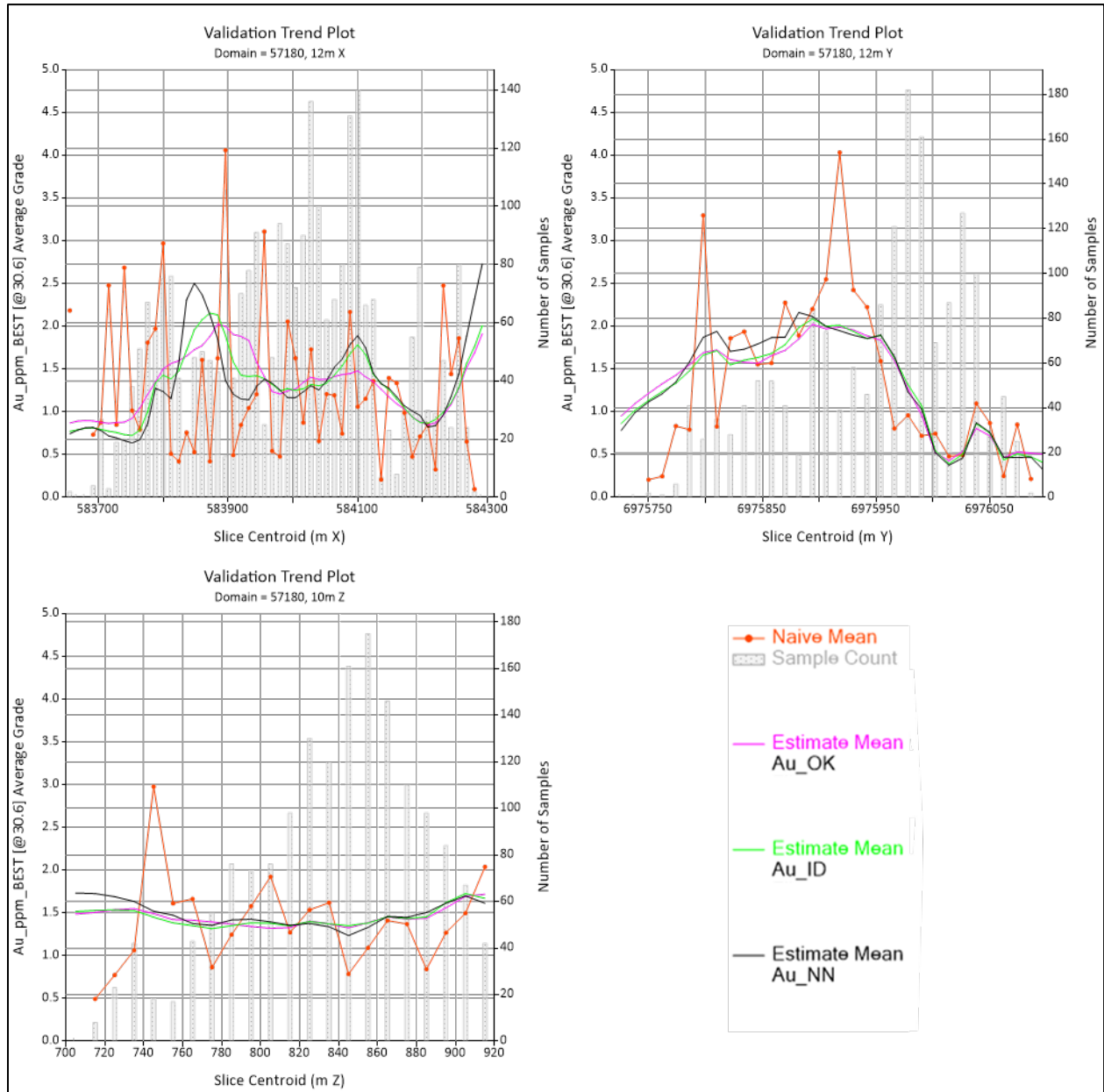
Figure 14.32
Swath Plots from Domain 27203



Source: Fuerte, 2025



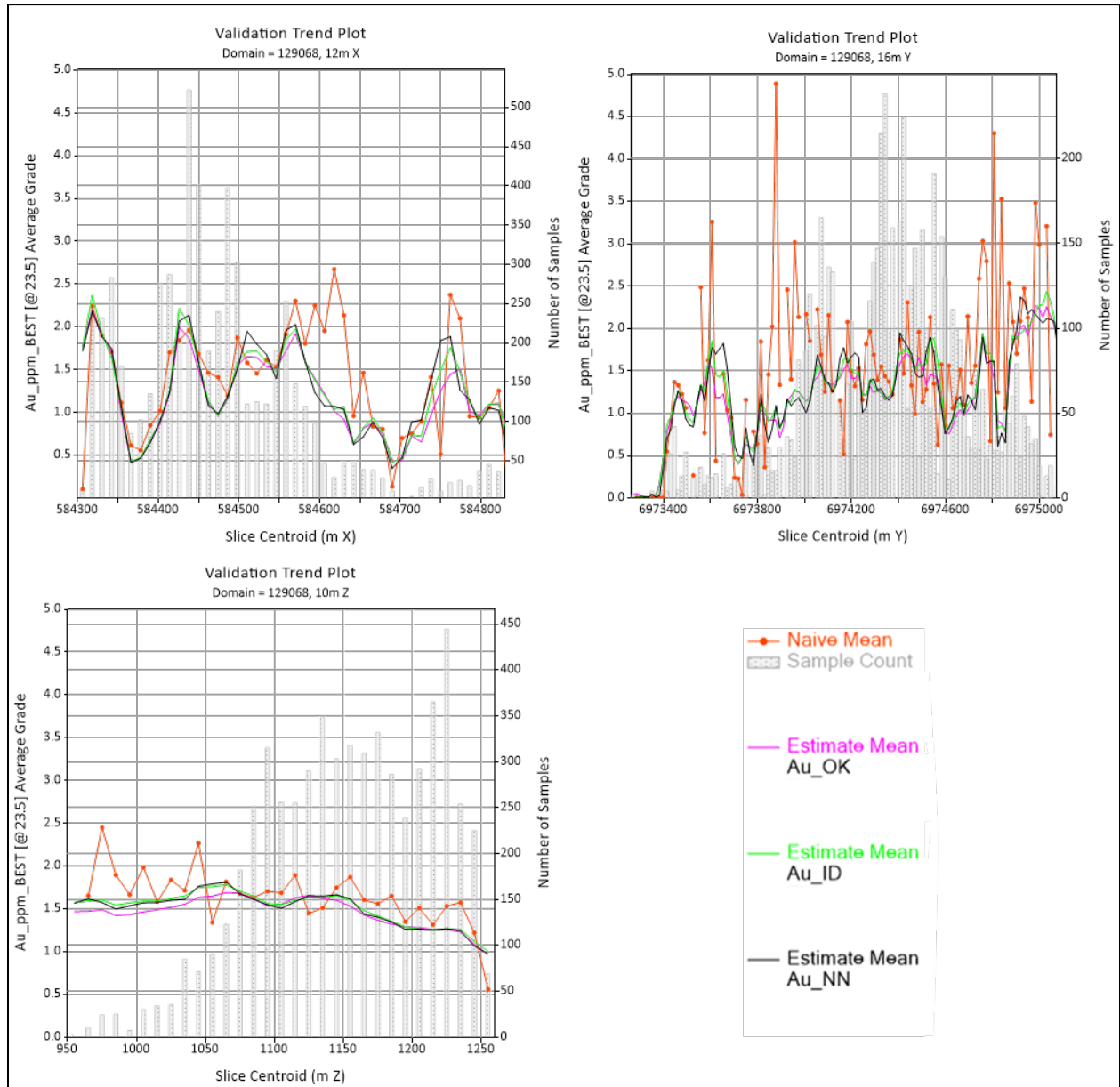
Figure 14.33
Swath Plots from Domain 57180



Source: Fuerte, 2025



Figure 14.34
Swath Plots from Domain 129068



Source: Fuerte, 2025



14.6 Reasonable Prospects for Eventual Economic Extraction

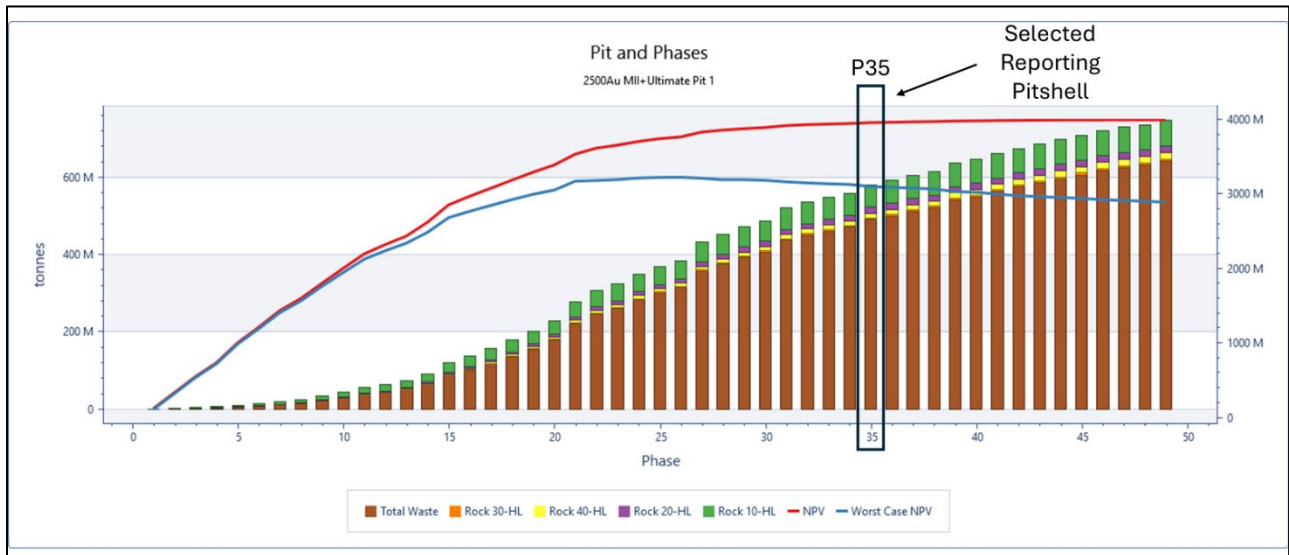
To determine RPEEE, a pit shell is used to constrain each of the three block models. The technical and economic assumptions used to determine the constraining pit shell and cut-off grades are summarized in Table 14.13. The nested pit-by-pit graphs and the selected pit shell for Coffee West block model is shown in Figure 14.35, Coffee East block model is shown in Figure 14.36 and the Kona block model is shown in Figure 14.37.

Table 14.13
Coffee Gold Project Economic Assumptions and Gold Cut-off Grades

Item	Unit	Coffee Gold Project Areas				
		Supremo	Latte	Double-Double	Kona	Other Areas
Exchange Rate	US\$/CA\$	1.35				
Gold Price	US\$/oz	2,500				
Processing Cost	CA\$/t	6.64				
G&A Cost	CA\$/t	6.00				
Transportation & Refining Cost	CA\$/oz	1.30				
Royalty	%	2.38				
Gold Payability	%	99.80				
Mining Cost	CA\$/t	3.27	3.30	3.32	3.50	3.50
Gold Metallurgical Recoveries						
Oxide (OX)	%	87.2	88.6	89.1	83.0	80.0
Upper Transition (UT)	%	79.3	77.4	77.0	71.3	70.0
Middle Transition (MT)	%	52.6	60.3	42.6	57.3	50.0
Lower Transition (LT)	%	34.2	30.0	30.2	28.6	25.0
Open Pit Optimization Parameters						
Overall Slope Angle	degrees	48.8	46.5	47.8	47.8	45.0
Block Model re-blocked (X, Y, Z)	m	12 x 12 x 10				
Regularized SMU (X, Y, Z)	m	4 x 4 x 5				
Mining Recovery	%	100				
Mining Dilution	%	0				
OP minimum mining width (bottom)	m	12				
Gold Cut-off Grades						
Oxide (OX)	g/t	0.14	0.14	0.13	0.14	0.15
Upper Transition (UT)	g/t	0.15	0.15	0.16	0.17	0.17
Middle Transition (MT)	g/t	0.23	0.20	0.28	0.21	0.24
Lower Transition (LT)	g/t	0.35	0.40	0.40	0.42	0.48

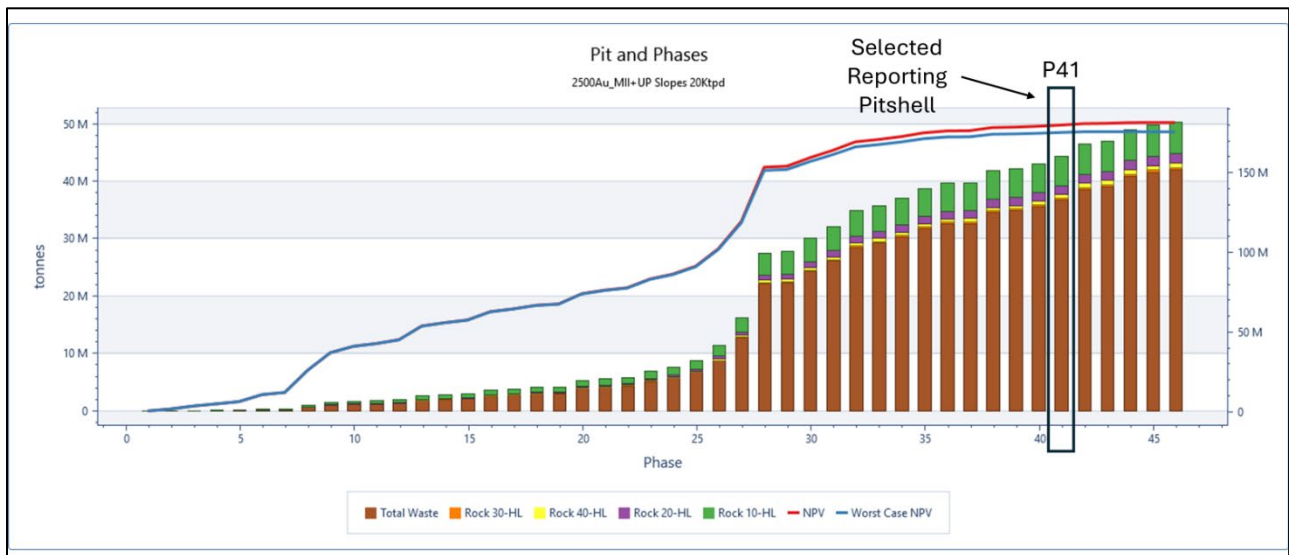


Figure 14.35
Coffee West Block Model Pit by Pit Graph and Selected Pitshell



Source: Micon, 2025

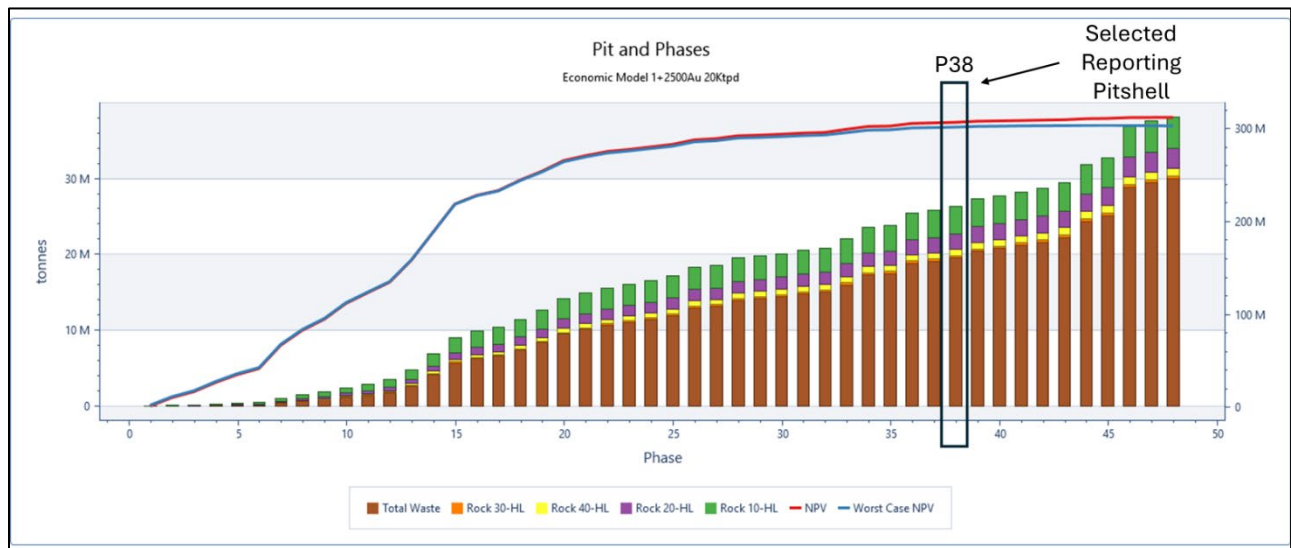
Figure 14.36
Coffee East Block Model Pit by Pit Graph and Selected Pitshell



Source: Micon, 2025



Figure 14.37
Kona Block Model Pit by Pit Graph and Selected Pitshell

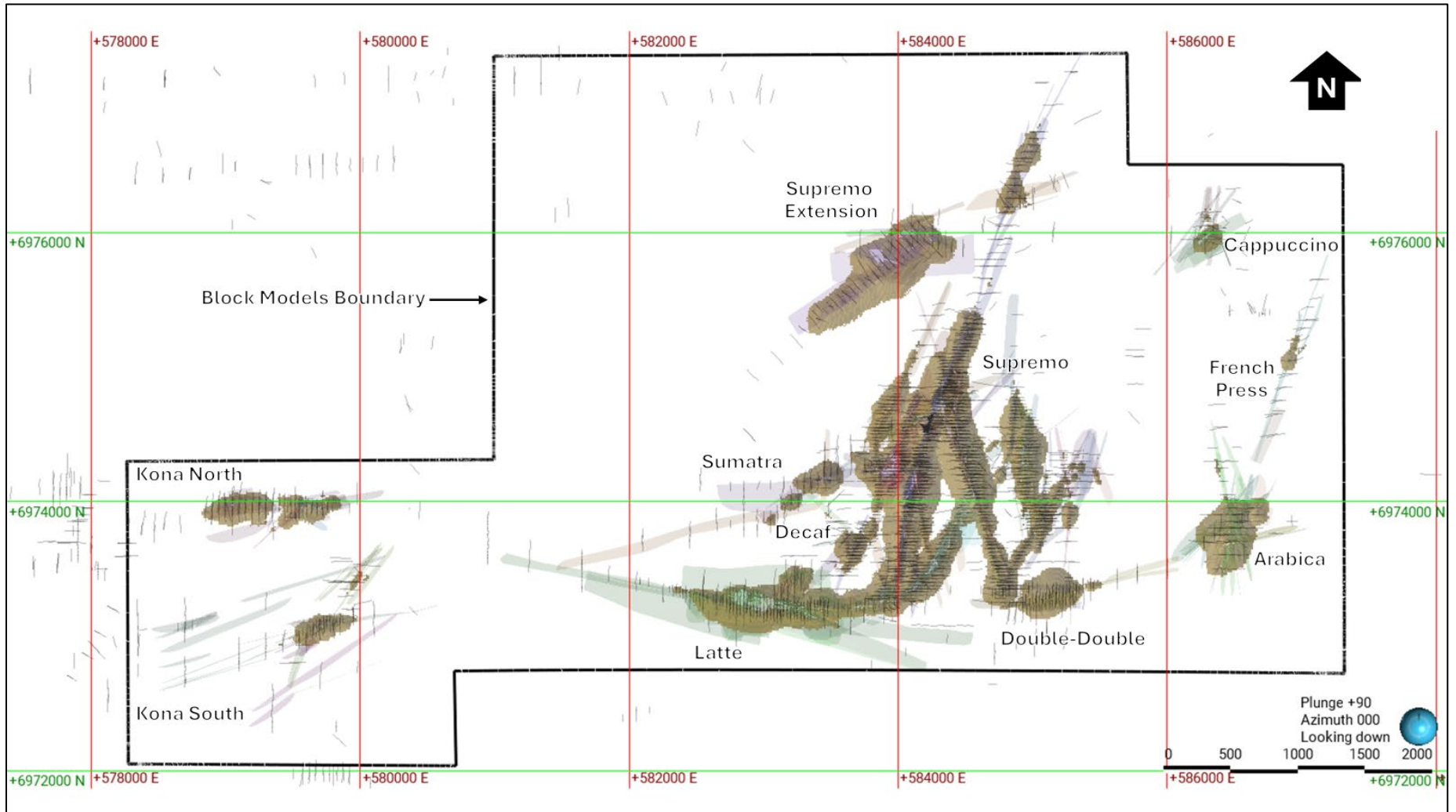


Source: Micon, 2025

After the selecting pits hells 35, 41 and 38 for each optimization, the pitshells were groomed to eliminate any scattered or isolated pits that most likely would not be developed in a real-world mining scenario. Figure 14.38 shows the resulting selected pit shell for the Coffee Gold Project.



Figure 14.38
Coffee Gold Project Selected PitsHELLS Boundaries



Source: Micon, 2025



14.7 Factors that Could Materially Affect Mineral Resources

Technical risk is also present in the metallurgical recovery assumptions. The resource model incorporates variable heap leach recoveries, which are relatively high for Oxide material (86.3%) but decline sharply to 31.4% for Lower Transition material. Changes in key cost assumptions such as mining (C\$3.27–C\$3.50/t), processing (C\$6.64/t), and general and administrative (C\$6.00/t) as well as potential increases in government levies or royalties (currently 2.38%) could further influence cut-off grades and materially affect the tonnage of material meeting RPEEE criteria.

Conversely, there are upside opportunities. Enhancing metallurgical recoveries in the four oxide facies domains, could positively impact the overall mineral resource potential.

The Micon QP is not aware of any other known risks that could materially affect the MRE.

14.8 QP Opinion and Comments

The Coffee Gold Project Mineral Resource Estimate was completed in accordance with industry best practices. Following a comprehensive technical review of the initial work conducted by Fuerte, the Qualified Person (QP) made targeted adjustments to the gold grade capping and resource classification to ensure full confidence in signing off on the estimate.

The QP independently selected the economic parameters and conducted the assessment of reasonable prospects for eventual economic extraction using surface mining methods.

The resulting Mineral Resource Statement is robust, underpinned by a substantial volume of high-quality, validated data. The QP is fully satisfied with the integrity and completeness of the work.

The Coffee Gold Project Mineral Resource Statement have been derived by testing the concept of Reasonable Prospects for Eventual Economic Extraction (RPEEE), based on defined technical and economic parameters. To ensure transparency for investors, it is important to discuss the extent to which these Estimate may be materially impacted by external factors or underlying technical assumptions.

At present, the Qualified Persons (QP) has not identified any known legal, political, environmental, title, taxation, socio-economic, or marketing constraints that would preclude the potential development of the Mineral Resource. However, the economic model underpinning the RPEEE is sensitive to market conditions. Specifically, it assumes a gold price of US\$2,500/oz and a US\$:CA\$ exchange rate of 1.35. Any adverse shifts in these assumptions would directly affect the calculated cut-off grades, which currently vary between 0.13 g/t Au to 0.48 g/t Au, depending on the metallurgical domain the material comes from.

The sensitivity analysis in Table 14.2 illustrates the impact of changing cut-off grades. For example, increasing the cut-off from the base case average of 0.18 g/t Au to 1.0 g/t Au results in a significant reduction in mineral resource material.



15.0 MINERAL RESERVE ESTIMATES

Currently, there are no mineral reserves on the Coffee Gold Project.



16.0 MINING METHODS

16.1 Geotechnical Analysis and Recommendations

The WSP QP has reviewed the geotechnical analysis of the Internal Engineering Report (JDS 2019) and, confirms the recommendations presented are adequate for the PEA-level analysis. The following discussion summarizes content from that report.

16.1.1 Geotechnical Data Collection

Open pit geotechnical design criteria were initially developed for the pits by SRK (2016) and then updated by JDS in 2018 based on additional geotechnical core drilling, strength testing and downhole televiewer surveying. A total of 13 HQ3 diameter core holes were logged and tested between the two programs totalling 2,105 m in length. Both geotechnical drilling campaigns consisted of detailed geotechnical core logging and discontinuity orientation, field point load testing, and laboratory strength testing of core samples.

A total of 112 laboratory tests were conducted on core samples selected to represent the range of the conditions observed in the core. The laboratory testing programs consisted of uniaxial compressive, triaxial compressive, indirect tensile (Brazilian), and natural joint direct shear strength tests, as well as measurements of unit weight and elastic properties. Results of the laboratory testing program are summarized in Table 16.1 by rock type.

Table 16.1
Average Intact Rock Properties Derived from Laboratory Test Results

Lithology	Oxide	g (kN/m ³) ¹	BTS ¹ (MPa)	UCS ^{1,2} (MPa)	E _i (GPa) ¹	n ¹	m _i	σ _{ci} ² (MPa)
Gneiss	All	25.4 (56)	11.9 (19)	81 (31)	26 (22)	0.25 (11)	18	89
	Oxide	24.7 (2)	-	67 (2)	-	-		
	Transition	25.4 (19)	8.4 (5)	77 (12)	30 (7)	0.26 (5)		
	Fresh	25.5 (35)	13.2 (14)	82 (18)	24 (15)	0.25 (6)		
Schist	All	25.4 (31)	7.7 (8)	69 (13)	16 (10)	0.24 (6)	19	54
	Oxide	26.1 (3)	-	58 (2)	12 (1)	0.11 (1)		
	Transition	24.6 (15)	4.6 (5)	66 (6)	16 (7)	0.26 (5)		
	Fresh (Valid Tests)	26.3 (31)	12.7 (3)	76 (5)	19 (2)	-		
	Fresh (All Tests)	26.3 (31)	12.7 (3)	51 (9) ³	19 (2)	-		
Granite	All	24.6 (8)	7.1 (3)	70 (6)	29 (3)	0.19 (3)	-	-
	Oxide	-	-	-	-	-		
	Transition	23.9 (4)	2.6 (1)	41 (4)	15 (2)	0.14 (2)		
	Fresh	25.2 (4)	9.4 (2)	130 (2)	55 (1)	0.30 (1)		

Notes:

¹ Numbers in parentheses indicate the number of test results available to calculate the average intact rock parameters.

² 'UCS' indicates laboratory UCS test results while 'σ_{ci}' represents the intercept of the triaxial strength envelope with the σ₃ axis.

³ Excludes breaks where the sample broke along a pre-existing weakness such as healed joints or foliation.

Source: JDS (2019)

An additional 20 core samples were selected for small scale direct shear (SSDS) testing to characterize shear strengths of natural discontinuities. Samples with natural fractures preserved in the field were used



for the tests. The range of normal stresses applied during testing ranged from approximately 170 to 4,880 kPa. Results of the direct shear testing of natural fractures are summarized by rock type in Table 16.2.

**Table 16.2
Direct Shear Test Results**

Fracture Type	No.	Friction Angle (°)	Cohesion (kPa)
Schist Foliation	7	29	28
Gneiss Foliation	4	31	47
Gneiss Joint	6	29	40
Schist Joint	1	29	53
Granite Joint	1	38	53

Source: JDS (2019)

16.1.2 Rock Mass Characterization

Core logging and laboratory test data obtained from the 13 geomechanical drillholes was used for empirical rock mass characterization according to the Bieniawski (1989) Rock Mass Rating (RMR) system. The RMR system consists of five primary parameters: unconfined compressive strength (UCS), RQD, discontinuity spacing, joint conditions (Jc), and groundwater. Dry conditions were assumed for the RMR calculations as groundwater pressures are commonly accounted for during slope stability analysis using effective stress type analyses. Similarly, the RMR values were not discounted for adverse joint set orientations as potential planar, or wedge type failures were evaluated independently using kinematic analyses. The RMR value for a given interval is equal to the sum of the five parameters with the maximum possible (most competent rock mass) being 100.

The geotechnical properties were analyzed by primary lithology type and oxidation group. It was concluded from the analyses that, while the three primary lithologies (i.e., schist, gneiss or granite) were generally similar in their characteristics, rock mass quality is best delineated by the oxidation zones within each lithology (i.e., oxide, transition and fresh).

Results of the rock mass characterization suggest that, except for the oxide materials which will be mostly mined and processed, the rock mass at Coffee is generally of 'Fair' to 'Good' geomechanical quality according to the Bieniawski (1989) criteria. Table 16.3 summarizes the average and range of RMR values derived from the geotechnical logging data for each of the three primary rock types and oxidation zones.

**Table 16.3
Summary of Rock Mass Classification Data**

Pit	Lithology	Oxide Zone	No. of Tests	RMR (Bieniawski, 1989)			
				Avg	Std. Dev.	Min	Max
Supremo	Gneiss (MXF/MxM/FG)	Oxide	27	59	9.9	39	76
		Upper Trans	32	61	11.1	35	77
		Middle Trans	87	61	9.1	34	78
		Lower Trans	180	66	11.2	34	91
		Fresh	391	69	10.8	31	91
Double-Double	Gneiss (MxF/MxM)	Oxide	-	-	-	-	-
		Upper Trans	-	-	-	-	-
		Middle Trans	5	58	6.3	50	68
		Lower Trans	13	66	11.5	42	82
		Fresh	20	62	8.7	48	80



Pit	Lithology	Oxide Zone	No. of Tests	RMR (Bieniawski, 1989)			
				Avg	Std. Dev.	Min	Max
	Schist (BtS)	Fresh	21	64	9.7	42	93
Latte	Schist (BtS/BtS_carb)	Oxide	28	54	8.2	32	73
		Upper Trans	16	56	13.0	36	78
		Middle Trans	34	69	9.5	37	83
		Lower Trans	100	67	10.5	33	91
		Fresh	90	61	10.2	28	82
Kona	Granite (GG)	Oxide	-	-	-	-	-
		Upper Trans	10	58	16.0	30	77
		Middle Trans	27	69	8.1	38	77
		Lower Trans	6	77	9.2	61	85
		Fresh	34	77	8.1	81	93

Source: JDS (2019)

Major fault structures and other weak features observed in the core were logged and analyzed independent of the RMR data to avoid potentially obscuring important structures by the combined overall RMR value calculated for the interval.

16.1.3 Rock Mass Shear Strength

Based on the rock mass characterization and laboratory test data, rock mass shear strength/normal stress relationships were developed for each unit according to the Generalized Hoek-Brown criterion (Hoek, et al., 2002). The Generalized Hoek-Brown criterion defines curvilinear shear strength envelopes considered effective representations of intact rock and heavily jointed rock mass behavior.

Inputs to the rock mass strength criteria include Geologic Strength Index (GSI), Uniaxial Compressive Strength (UCS), Intact Material Constant (m_i) and Disturbance Factor (D). Rock mass strength parameters were derived from the geotechnical logging and laboratory test data and are summarized in Table 16.4 for each of the primary rock types and oxidation zones.



**Table 16.4
Design Rock Mass Parameters**

Pit	Rock Type	Oxidation Group	Intact Rock		Rock Mass		
			m_i	UCS (MPa)	GSI	D	Density (kN/m ³)
Supremo	Gneiss	Oxide	18	49	54	0.7	26.0
		Upper & Middle Trans	18	70	56		
		Lower Trans & Fresh	18	87	63		
Latte	Schist	Oxide	19	46	49	0.7	26.0
		Upper Trans	19	58	51		
		Middle & Lower Trans	19	52	62		
Double-Double	Gneiss & Schist	Middle & Lower Trans	18	34	62	0.7	26.0
		Fresh	18	54	58		
Kona	Granite	Middle Trans	23 ¹	60	64	0.7	25.1
		Fresh	23 ¹	97	72		

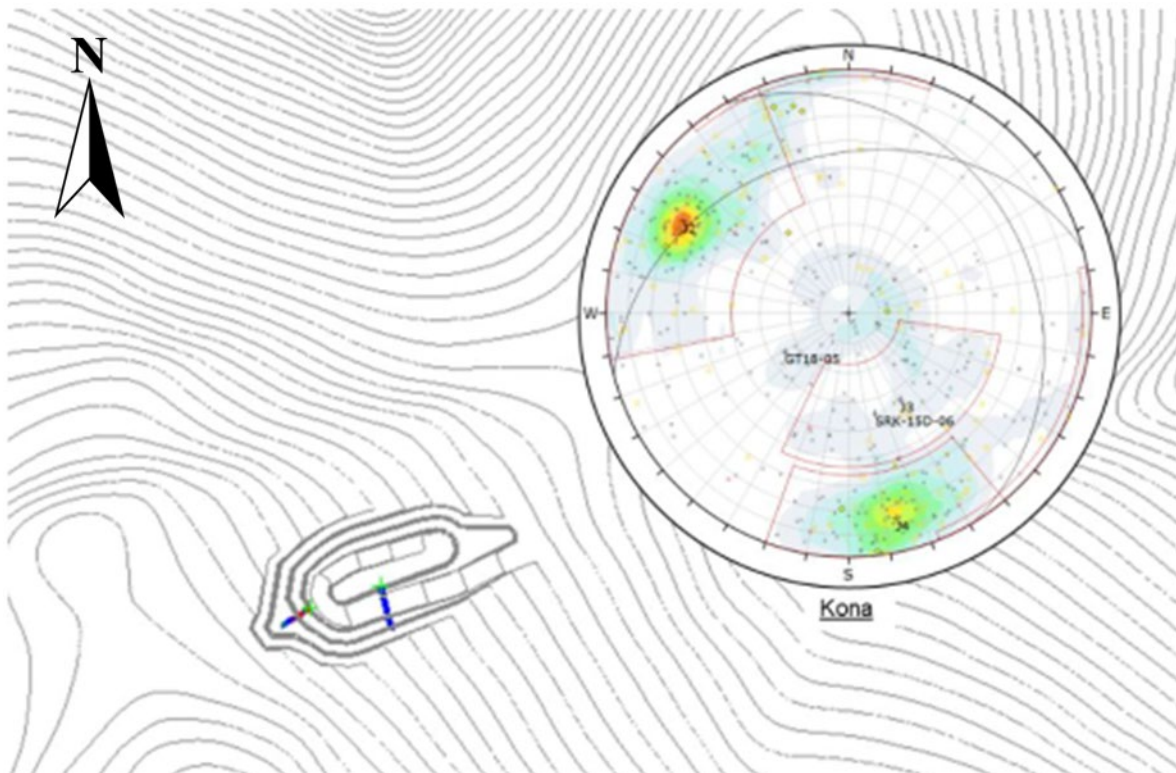
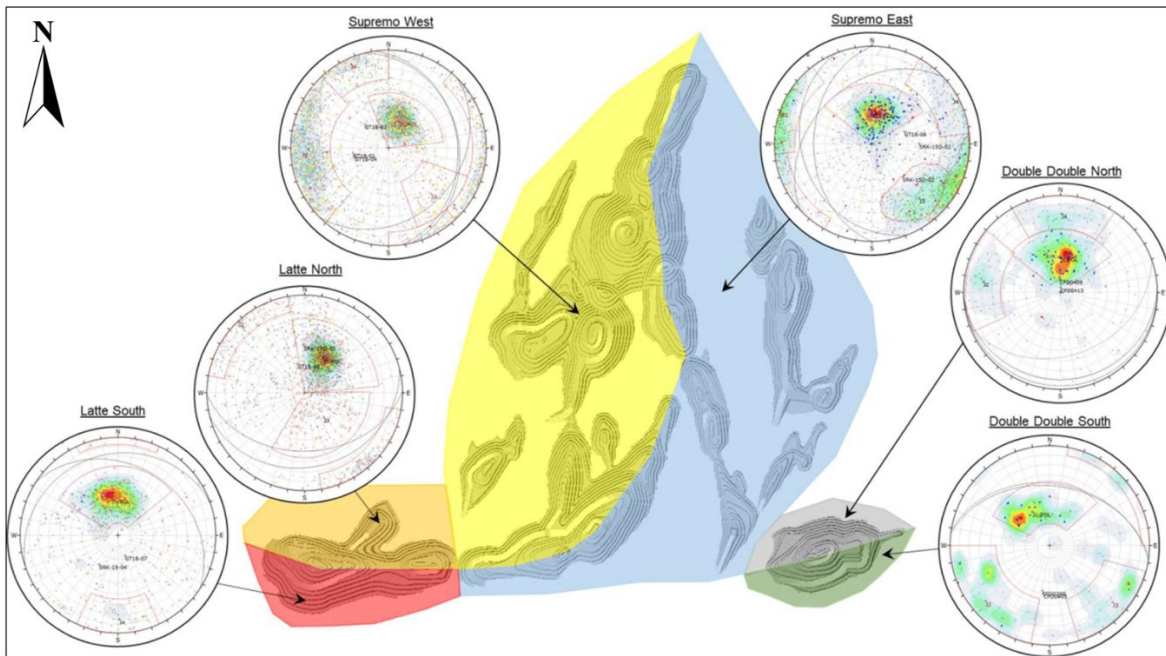
Source: JDS (2019)

Disturbance Factor depends on the degree of disturbance that the rock mass is expected to be subjected to as a result of blast damage and stress relaxation caused by excavation of the pit. A factor of 0.7 is commonly accepted in the case of carefully controlled blasting and with 1.0 assumed for heavy large-scale production blasting. The disturbance zone typically extends on the order of 10 m to 50 m behind the face and does not apply to the entire rock mass behind the slope. Additional discussion regarding the Disturbance Factor selected is contained in Section 16.3.5 (Interramp/Overall Slope Stability Analyses).

16.1.4 Geological Structure and Bench Design

A total of seven separate structural domains were developed based on dominant discontinuity orientations obtained from the combined 13 geotechnical drillholes along with the SRK (2017) 3D structural model for the project. The Supremo, Latte and Double-Double pits were divided into two separate domains each as shown on Figure 16.1. Kona was considered as a single domain due to its small size.

Figure 16.1
Structural Domains and Respective Discontinuity Pole Plots



Source: JDS, 2019



Kinematic analyses were completed for each domain to assess the potential of discontinuity sets, or combinations of sets, to adversely impact bench stability for the various slope orientations in each pit. Using stereoplots, the orientation and variability of each discontinuity set was graphically assessed to determine which sets, or combinations of sets, would have the potential to form 3D (planar or wedge) blocks that could displace under gravitational forces for a given pit wall orientation. Details of the kinematic assessment are summarized below:

- The dominant, shallow to moderately south-southwest dipping foliation has a high potential to create planar instabilities and will likely control achievable bench face angles in the Latte pit north wall. The foliation has similar potential for planar instabilities in the Double-Double north wall but is only considered moderate due to foliation being less intense in the gneiss. The SRK (2016) probabilistic bench design analyses indicate a maximum achievable bench face angle of approximately 65° for these walls;
- Due mostly to the north-south orientation of the pit, the east and west pit wall benches at Supremo are not anticipated to be significantly impacted by structurally controlled instabilities. However, the sub-vertical, north to north-northeast dipping joint set (J2) may cause local sloughing where persistent discontinuities strike oblique to bench faces. As such, a maximum achievable bench face of 75° was estimated for Supremo east and west pit walls based on operational constraints;
- Stability of the Supremo, Latte and Double-Double south wall benches is not anticipated to be controlled by geologic structure. A maximum achievable bench face of 75° was therefore selected based on operational considerations; and
- Benches on the northwest Kona pit have a slightly higher probability of structurally controlled instabilities due to the presence of a moderate to steeply southeast dipping joint set (J2). The analyses indicate a design bench face angle of approximately 65° is appropriate for this sector of the pit. The remaining pit areas are not anticipated to be significantly impacted by geologic structure-controlled instabilities and are therefore estimated to have maximum achievable bench face angles of 75° based on operational constraints.

Based on the Modified Ritchie Criteria as described by Call (1992), a minimum acceptable catch bench width of 8.5 m has been assumed for the bench design.

Bench design and kinematic analyses are based solely on orientations of geologic structure and do not directly consider effects of weathering, alteration, blasting or excavation techniques. Recommendations for south facing pit walls assume that the achievable bench face angles are controlled by an assumed amount of partial backbreak along foliation sets. This assumption will require follow-up assessment in the next stage of geotechnical study.

To achieve the recommended bench face angles of 70° or better requires high quality wall control blasting including free-faced pre-split (70° to 75° or greater) and/or trim blasting adjacent to final walls. Depending on the quality of blasting, scaling and excavation practices, achievable bench face angles may be significantly reduced from the theoretical angles determined by these analyses.

Field trials of various wall control basting techniques should be performed, carefully documenting the results to confirm that the actual slope designs are being achieved or, if necessary, to serve as the basis of slope angle refinements.

16.1.5 Interramp/Overall Slope Stability Analyses

A total of seven critical sections were analyzed between the four pits to verify stability of the interramp and overall slope angles developed based on achievable bench face angles. Locations of the sections were



selected at locations where slope stability conditions are anticipated to be the most adverse such as where the slope height is at its maximum and/or discontinuities are adversely oriented.

The seven critical sections were analyzed using the limit equilibrium slope stability analysis software, Slide (Rocscience, 2018). Spencer's method of slices was selected for the analyses due its consideration of both force and moment equilibrium and typically less conservative results compared to methods that consider only force equilibrium.

The respective average properties were selected to represent the rock mass and joint strengths for each of the primary lithologies and oxidation types as summarized in Table 16.3 and Table 16.4, respectively. Slide's Generalized Anisotropic shear strength model was used to simulate the inherent weaker rock strength in the direction parallel to foliation and, to a lesser extent, the less dominant non foliation joint sets. The model assigns a weaker strength to any portion of a potential failure surface that is sub-parallel to the joint of foliation planes. The strength of foliation and joint parallel surfaces were developed using an equivalent strength material approach suggested by Jennings (1972). The method effectively averages the shear strength of the discontinuity and intact rock bridges, weighted by the percentage of the surface along a discontinuity and through intact rock bridges. The percentage of the surface along the discontinuity is represented by a Continuity Coefficient (k) ranging between 1 (100% discontinuity) and 0 for (0% discontinuity).

Given the lack of discontinuity length information, the analyses were conducted assuming conservative discontinuity lengths using Continuity Coefficients of 0.7 for joints (70% of surface is along existing discontinuity), 0.80 for schist foliation and 0.75 for the gneiss foliation. The isotropic, Hoek-Brown rock mass shear strength was used to represent the intact rock bridge strength. It is recommended to collect discontinuity persistence through future field investigations for both rock bridging and bench back break purposes.

Based on accepted engineering experience, interramp/overall slope designs that yield factors of safety (FOS) of 1.3 for slopes with high failure consequences and 1.2 for low failure consequences are appropriate for most open pit mines. Slopes of high failure consequence are generally those slopes that are critical to mine operations, such as those on which major haul roads are established, those providing ingress or egress points to the pit, or those underlying infrastructure such as processing facilities or structures. The results of the stability analyses for the overall slope angle (OSA) and interramp slope angle (ISA) are summarized in Table 16.5 in terms of a safety factor for each critical section.



**Table 16.5
Results or Interramp/Overall Slope Stability Analyses**

Pit	Section	Slope Design			Safety Factor	
		Max Height (m)	OSA (°)	ISA (°)	ISA	OSA
Supremo	A	190	56	55	3.8	2.8
	B	170	49	51	3.3	2.9
	C	180	53	55	2.1	2.3
Latte	D	150	44	47	1.3	1.3
	E	180	53	55	2.1	2.0
Double-Double	F	140	44	46	1.7	1.9
Kona	G	90	47	52	3.4	3.9

Source: JDS (2019)

The Disturbance Factor of 0.7 for the entire rock mass rather than only 10 m to 50 m next to the face could be considered conservative. However, given that the critical failure surfaces produced by the analyses are typically within 50 m from the face (approximate limits of the disturbed zone), more detailed modelling or zoning of the Disturbance Factor be unlikely to have a significant impact on the results.

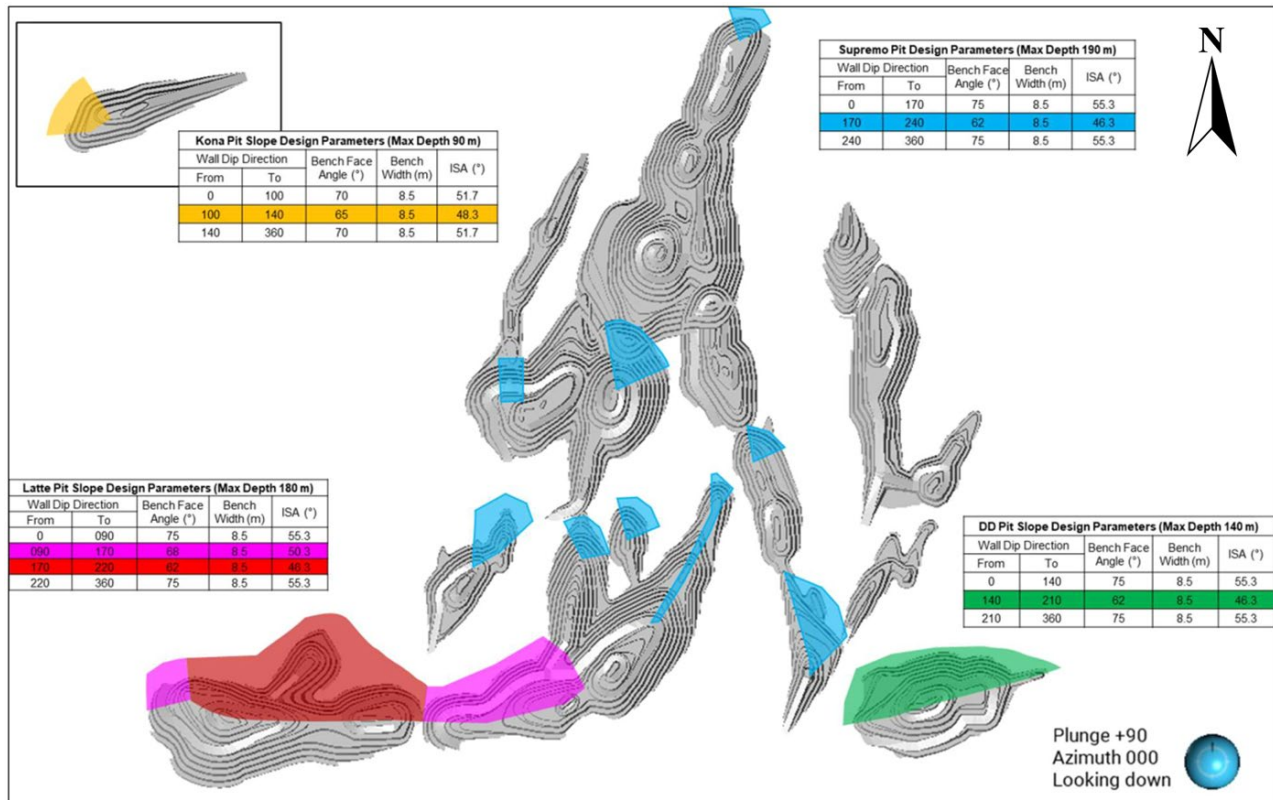
Results of the overall slope stability analyses indicate the bench configuration-based slope angles meet or exceed the minimum acceptable safety factor of 1.3 for slopes with high failure consequences. Critical failure surfaces were typically non-circular, partially through intact rock mass or rock bridges and partially along discontinuities.

16.1.6 Pit Geotechnical Design Recommendations

Recommendations for geotechnical pit slope design parameters are summarized in Table 16.6 and shown graphically on Figure 16.2. Continuous bench stacks or interramp slope heights should not exceed a height of approximately 100 m (vertically) without a minimum 15 m wide geotechnical berm, haul road, or other wide step-out for mine infrastructure.



**Figure 16.2
Pit Slope Design Recommendations**



Source: JDS 2019

**Table 16.6
Pit Slope Design Recommendations**

Pit	Max. Slope Height (m)	Wall Dip Direction		Bench Face Angle (°)	Bench Height (m)	Min. Bench Width (m)	Max. ISA (°)
		From (°)	To (°)				
Latte	170	0	90	75	20	8.5	55.3
		90	170	68	20	8.5	50.3
		170	220	62	20	8.5	46.3
		220	360	75	20	8.5	55.3
Supremo	180	0	170	75	20	8.5	55.3
		170	240	62	20	8.5	46.3
		240	360	75	20	8.5	55.3
Double-Double	135	0	140	75	20	8.5	55.3
		140	210	62	20	8.5	46.3
		210	360	75	20	8.5	55.3
Kona	100	0	100	70	20	8.5	51.7
		100	140	65	20	8.5	48.3
		140	360	70	20	8.5	51.7

Source: JDS 2019



16.2 Pit Optimization

Pit optimization was carried out using a Pseudoflow algorithm used specifically for mining applications. The pit optimization was performed using the geologic and economic assumptions summarized in Table 16.7.

Table 16.7
Coffee Gold Project Economic Assumptions and Gold Cut-off Grades

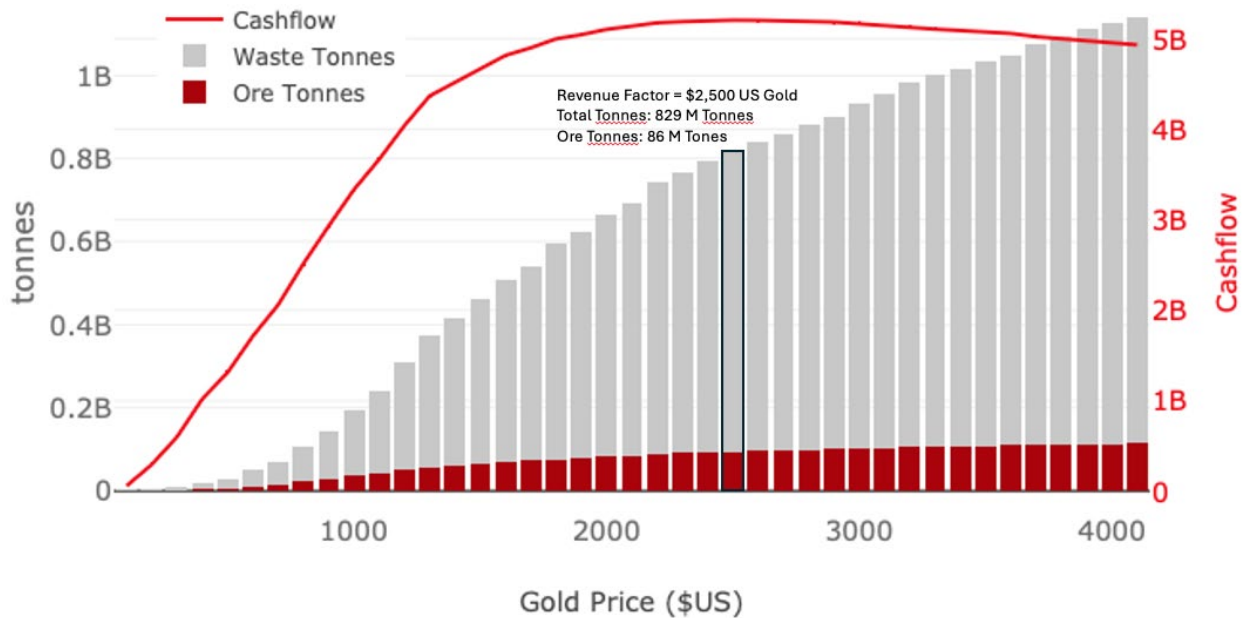
Parameter	Units	Coffee Gold Project Areas				
		Supremo	Dbl-Dbl	Kona	Other	Latte
Revenue, Smelting & Refining						
Gold price	US\$/oz Au	2,500				
Exchange Rate	\$US:\$CA	1.35				
Payable metal	%	99.8				
Transport & Refining Cost	\$CA/oz Au	1.30				
Royalty @ 2.53% NSR	\$CA/oz Au	85.22				
Net gold value	\$CA/oz	3,288				
Net gold value per gram	\$CA/g	105.73				
Operating Cost Estimates						
Mining Cost	\$CA/t mined	3.27	3.32	3.50	3.50	3.30
Crush Leach Operating Cost Estimates						
Process Cost	\$CA/t leached	7.19				
G&A	\$CA/t leached	5.55				
Site Services	\$CA/t leached	2.60				
Total Crush Leach Cost	CA\$/t leached	15.34				
Recovery and Dilution						
External Mining Dilution	%	0.0	0.0	0.0	0.0	0.0
Mining Recovery	%	100				
Crush Leach (P80 of 50mm) Gold Recovery						
Oxide (OX)	%	87.2	89.1	83.0	80.0	88.6
Upper Transition (UT)	%	79.3	77.0	71.3	70.0	77.4
Middle Transition (MT)	%	52.6	42.6	57.3	50.0	60.3
Lower Transition (LT)	%	34.2	30.2	28.6	25.0	30.0
Crush Leach (P80 of 50mm) Grade Calculations						
Oxide (OX)	g/t Au	0.20	0.20	0.21	0.22	0.20
Upper Transition (UT)	g/t Au	0.22	0.23	0.25	0.25	0.23
Middle Transition (MT)	g/t Au	0.33	0.41	0.31	0.36	0.29
Lower Transition (LT)	g/t Au	0.51	0.58	0.62	0.71	0.59



Parameter	Units	Coffee Gold Project Areas				
		Supremo	Dbl-Dbl	Kona	Other	Latte
Geotechnical						
ISA	degree	55.3	46.3 / 55.3	52.0	45.0	46.3 / 55.3
OSA	degree	45.0	35.0 / 45.0	45.0	35.0	35 / 45

The optimization generated pit shells across revenue factors ranging from 0 to 1.6 (US\$0-\$4,000) at \$100 intervals. The revenue factor 1.0 shell, corresponding to a gold price of \$2,500/oz, was selected as the basis for the mine plan designs. Revenue factor shells as low as \$500 were used to design internal higher margin phases. Figure 16.3 summarizes the optimization results.

**Figure 16.3
Optimization Results**



Source: Opencontour Optimization Summary 2026

16.3 Mine Dilution and Recovery

The mine plan and metal accounting does not include dilution or recovery factors (100% mine recovery and 0% dilution). The block model was sized 4 m long x 4 m wide x and 5 m in the vertical direction. This block size SMU (smallest mining unit) is sufficient to mitigate dilution and mine recovery given the fleet size (16 m³ shovel bucket).

Earlier technical work (KAM-NI-43101_Feasibility_Report 2016) did apply dilution factors by adding a 0.5-m waste skin along the interfaces as mineralized material using specialized software. The block size used in the models measured 10 m along strike x 2.5 m across strike x 5 m vertical. Since that time, the block model was updated (2018) to more regular larger 4 m x 4 m x 4 m block size. During this time, metal accounting and resource work was done without any dilution or recovery factors as it was believed the larger block size would mitigate the dilution/recovery factors. The larger block size would make up for the



~5% mine loss applied if dilution/recovery factors were applied. This technical study uses the latest 4 m x 4 m x 4 m model size and therefore does not utilize dilution/recovery factors.

If later in the mine life, poor model to mine reconciliation is discovered due to dilution (perhaps due to grade control methods or blast control issues) dilution/recovery factors can be added without having to adjust model block size.

16.4 Pit Design

Open pit mine design parameters were developed to support a preliminary mineable shape. The design criteria reflect conventional open pit mining practices for a gold deposit and taken from previous technical work done on the Project. The parameters are intended to provide a practical basis for phase design, mine planning, and equipment selection.

The pit has been designed using a **10 m mining bench height**, with a **20 m final double-bench configuration** adopted in the ultimate wall design. This approach is consistent with conventional drill-and-blast open pit operations. Mining is conducted on single 10 m benches and final walls are established as paired benches to improve wall stability and overall geotechnical performance while providing a **catch bench berm width of 8.5 m** to provide rockfall retention capacity and access for wall scaling and inspection where required.

Final wall **bench face angles** range from **62° to 75°**, reflecting sector-specific geotechnical conditions and lithologic or structural domains within the deposit area. Overall wall configuration was governed by **inter-ramp angles ranging from 46.3° to 55.3°**. The variation in inter-ramp angle reflects differing rock mass conditions and slope performance expectations by sector. These design angles were used in the generation of the final pit shell geometry and established in previous technical studies.

Pit access is provided by haul ramps designed at a width of **28.2 m**, which is suitable for the planned mining fleet (150 tonne class) and allows for safe two-way truck traffic under standard operating conditions. Ramps were designed at a maximum **gradient of 10%**, consistent with conventional open pit haulage practice and intended to balance haul truck productivity, operating safety, and road maintenance requirements.

The principal open pit design criteria applied are as follows:

- Mining bench height: **10 m**
- Final design bench configuration: **20 m**
- Bench face angle: **62° to 75°**
- Catch bench berm width: **8.5 m**
- Inter-ramp angle: **46.3° to 55.3°**
- Ramp width: **28.2 m**
- Ramp gradient: **10%**

The design criteria should be reviewed and refined as additional geotechnical, hydrogeological, and operational data become available during future engineering studies.

A total of 15 phases were utilized, incorporating 10 m benches with double benching at 20 m intervals. The work focused on the highest-value nested shells, with approximately \$500–\$600 shells used in the early



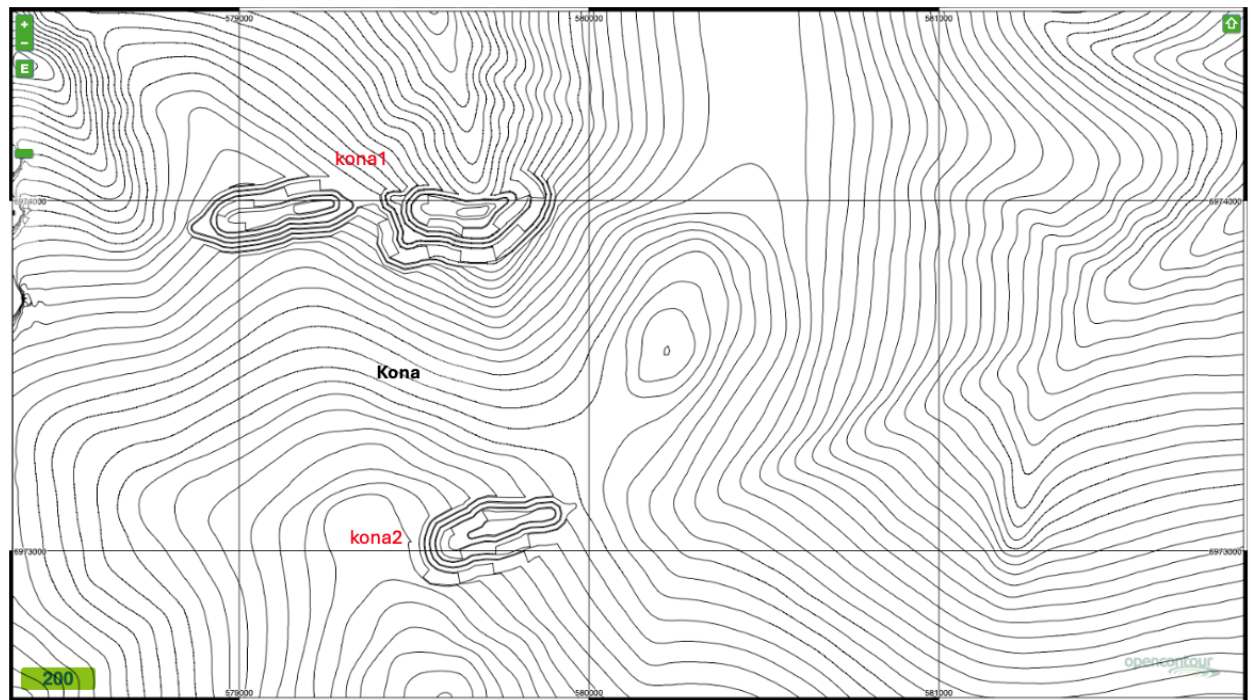
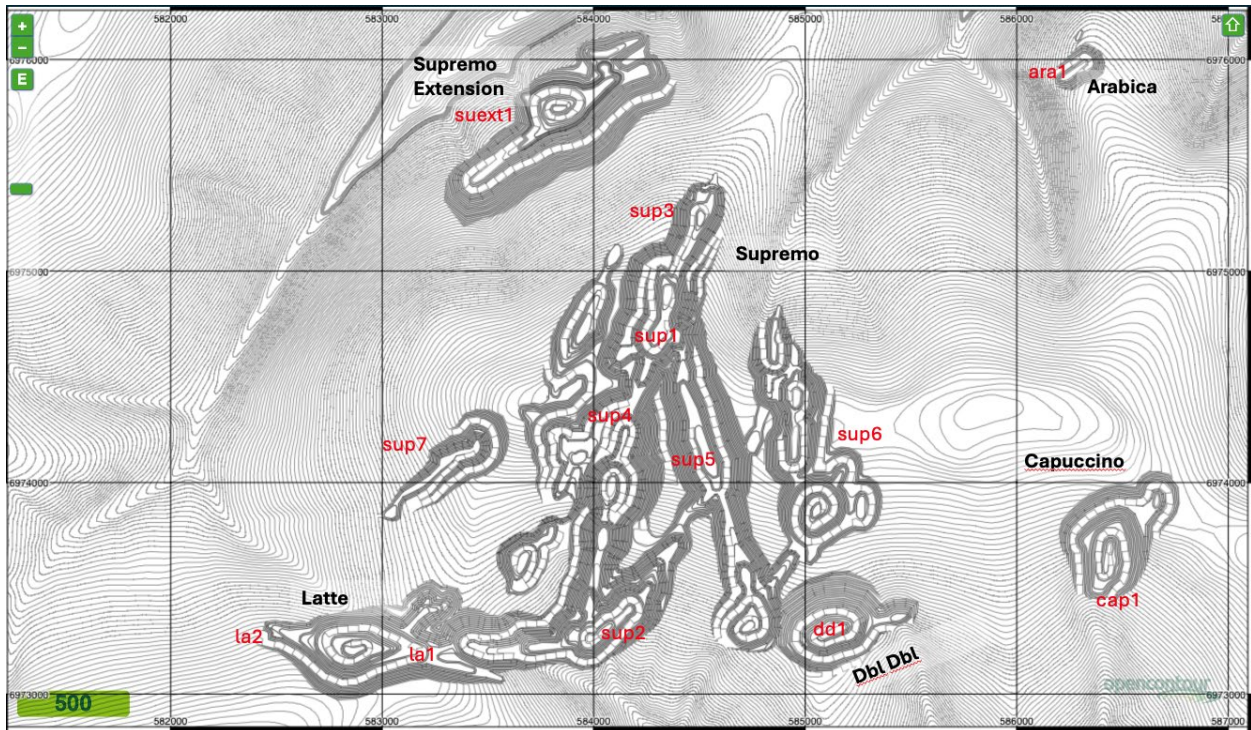
Latte and Supremo design configurations. Table 16.8 summarizes the value ranking of each designed phase while Figure 16.4 includes the designed phases.

Table 16.8
Phase Value Ranking

Phase Name	Total Tonnes (x1,000)	Ore Tonnes (x1,000)	Contained Ounces (x1,000)	Recovered Ounces (x1,000)	Head Grade (g/t)	Cost/Cont Ounce (\$US)	Pretax Cash flow (US\$Million)	Year Mined
sup1*	17,391	6,495	209	179	2.47	\$518	\$480	1
la1*	29,907	11,977	385	318	1.41	\$640	\$798	1
ko1	23,058	4,401	141	99	1.24	\$1,021	\$182	2
la2*	47,864	8,232	265	160	1.29	\$1,074	\$241	2
sup2*	103,426	16,726	538	432	1.21	\$1,144	\$805	3
sup3*	43,321	6,828	220	184	1.21	\$1,164	\$351	4
suext1	96,428	12,587	405	324	1.67	\$1,216	\$573	5
dd1*	41,791	4,658	150	118	2.11	\$1,315	\$192	6
ko2	9,907	1,271	41	32	0.87	\$1,494	\$45	7
sup4*	159,786	16,968	546	424	1.20	\$1,541	\$553	7
sup5*	100,450	10,320	332	269	1.10	\$1,616	\$348	8
cap1	3,318	333	11	9	1.07	\$1,656	\$10	10
ara1	38,402	4,055	130	104	0.84	\$1,718	\$119	10
sup6*	76,832	7,452	240	195	0.82	\$1,834	\$202	11
sup7	16,972	1,274	41	34	0.89	\$2,153	\$24	12
Total	808,855	91,001	3,652	2,880	1.29	\$1,340	\$4,922	



Figure 16.4
Phase Design



Source: Opencontour 2026



16.5 Production Scheduling

The plan was ranked across 15 designed phases, with sequencing driven by value optimization. Table 16.8 shows phases with higher profit margins (Cost/Contained Ounce) are mined earlier in the schedule increasing metal and cash flow in the early years. A process rate of 7.4 Mt/a drove the mine schedule so that the crusher had adequate feed while stockpile tonnage was minimized. Truck hours were used as the primary scheduling metric. Maintaining relatively constant truck hours to reduce variability in annual fleet size provided a more realistic schedule and reduced capital spending on trucks. Haul road layouts were kept consistent with previous technical studies. Table 16.9 summarizes mined physicals on an annual production basis. A summary of the production schedule fleet requirements is shown in Table 16.10.



**Table 16.9
Annual Mine Production Schedule**

	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Total Tonnes Mined	Mt	778.3	31.0	45.8	82.4	55.6	67.7	71.3	80.7	84.7	82.5	57.9	55.4	63.3
Mill Feed Mined Tonnes	Mt	90.5	7.6	7.3	8.4	8.8	7.4	4.8	9.3	7.1	8.5	6.5	6.9	8.0
Waste Tonnes	Mt	687.7	23.5	38.4	74.1	46.7	60.3	66.5	71.4	77.5	74.0	51.4	48.5	55.3
Contained Ounces	koz	3,644.2	385.9	353.5	334.2	353.9	307.7	261.5	407.5	221.4	372.3	199.6	227.9	218.8
Mined Grade	g/t Au	1.26	1.59	1.50	1.24	1.25	1.29	1.70	1.36	0.97	1.36	0.96	1.03	0.85
Strip Ratio	-	7.81	3.11	5.25	8.86	5.29	8.15	13.94	7.68	10.89	8.72	7.92	7.03	6.93
Stockpile Tonnes	Mt	21.1	0.55	0.48	1.44	2.87	2.88	0.25	2.14	1.86	2.95	2.04	1.55	2.13
Truck Calendar Hours	hrs x1000	3658	80	300	240	240	300	360	360	360	360	360	360	338



**Table 16.10
Annual Equipment Fleet Requirements**

Equipment	# Equip Required	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
150 t Class Trucks	45.0	11.0	38.0	31.0	31.0	38.0	45.0	45.0	45.0	45.0	45.0	45.0	42.0
Shovel (16.0 m ³ 6030/PC3000)	8.0	3.0	4.0	7.0	5.0	6.0	7.0	7.0	8.0	7.0	5.0	5.0	6.0
FEL (13.0 m ³ , 993K)	3.0	3.0	3.0	3.0	3.0	3.0	2.0	3.0	3.0	3.0	2.0	3.0	3.0
Drill (172-254 mm, Rotary, MD6250)	6.0	3.0	4.0	6.0	5.0	5.0	6.0	6.0	6.0	6.0	5.0	5.0	5.0
Drill (89-152mm, TH, DP1500i)	3.0	2.0	2.0	3.0	2.0	3.0	3.0	3.0	3.0	3.0	3.0	2.0	3.0

Note: FEL = front-end-loader

16.6 Operations

Open pit mining is assumed to be carried out by the owner using conventional drill, blast, load, and haul methods, although autonomous mining methods may be an option. Bulk material excavation will primarily be performed using hydraulic excavators for waste rock and more precise control on mineralization with a front-end loader available for mining mineralized material and backup. The diesel-powered mining fleet is expected to include a 16 m³ excavator paired with 150-t class haul trucks. Drilling operations will utilize 172-254 mm diameter rotary drills for production and 89 mm to 152 mm diameter drills for pre-split drilling for wall stability and secondary drilling as required. Considering the scale of the operation and equipment requirements, the mining fleet will operate entirely on diesel power.

The open pits are designed predominantly with 10 m bench heights. Mining will progress sequentially through pit pushbacks, typically with two or more mining phases active simultaneously. With the selected equipment and mining sequence, a maximum mining rate of 85 Mt/ar (approximately 230 kt/d) can be achieved.

Mineralized material will be transported to the crusher area, where it may either be fed directly into the crusher or placed on designated run-of-mine (ROM) stockpiles. Mineralized material is assumed to be stockpiled during the first quarter of the year (Q1), as heap leach stacking is not expected to occur during the coldest months. This stockpiled material will subsequently be reclaimed and rehandled to the crusher over the remaining three quarters to maintain the targeted annual processing rate of 7.4 Mt/a.

Grade control remains an important function given the deposit geometry. Gold mineralization is located within a series of steeply dipping structures that cut through all rock units on the property. The orientation and distribution of the gold mineralization above defined gold cut-off grades may prove difficult to define boundaries with the typical planned production blast pattern. Conventional vertical blast hole drilling and sampling methods may miss many of the narrow, strongly mineralized zones or be biased if drilled down a mineralized zone.

As such, to better define the location and grade of the narrow mineralized zones, angled RC drilling is proposed. An angle RC grade control program (assumed to be undertaken by a drilling contractor) would feature drilling rows or fences of angled drillholes oriented perpendicular to the strike of the gold system. The spacing of the drill fences and holes along each fence should initially be designed to intersect the mineralized zones in order to confidently be able to confirm the location of the mineralized zones. An



effective RC grade control program would need to be designed to achieve different coverage for various locations within the various deposits.

Blasting was assumed to be by a contractor, including, production, delivery, hole loading and blast initiation. Blasting will utilize ANFO and/or hole liners or emulsion to mitigate water-in-the-whole conditions. Powder factors range from 0.65 – 0.70 kg/m³ depending on the geology.

16.7 Infrastructure

The Project will include a range of surface infrastructure to support mining, processing, maintenance, storage, and administrative functions. These facilities will be constructed primarily using steel-framed, fabric-covered structures and modular buildings designed for rapid installation, durability in remote environments, and operational flexibility. The infrastructure layout has been planned to support efficient operations, reduce personnel travel time between key areas, and provide adequate maintenance and storage capacity for both the mining fleet and process plant.

The ADR process plant building will house the adsorption–desorption–recovery circuit and associated processing equipment. The structure will consist of a steel-framed, insulated, fabric-covered building with a peaked roof and straight eave configuration. The facility will include overhead crane capacity to support equipment installation and maintenance activities. Multiple vehicle access doors and personnel entrances will be provided to facilitate equipment access and routine operations. The building will be founded on cast-in-place concrete foundations designed with containment features and sumps to manage potential process spills. A dedicated lean-to structure will be incorporated to accommodate cyanide delivery and handling for the process plant.

An assay laboratory will be located adjacent to the process plant to support plant operations, metallurgical testing, and environmental monitoring requirements. The laboratory will consist of prefabricated modular units equipped with sample preparation and analytical equipment. Supporting systems such as drying ovens, heating systems, and dust and fume control equipment will be included to ensure safe and effective sample handling.

The mine maintenance area will include a truck shop and wash bay designed to support the servicing and repair of the primary mining fleet (45 x 150 tonne class haul trucks) and light vehicles. The truck shop will be a steel-framed, fabric-covered building containing multiple maintenance bays, spare parts storage, and service areas equipped for routine equipment maintenance and repair. The facility will be equipped with an overhead crane capable of servicing all maintenance bays and will include integrated systems for dispensing lubricants, coolants, and other maintenance fluids. The building will be heated to maintain safe and effective working conditions during colder periods. Tire repair activities will typically be conducted outdoors when conditions permit, with indoor provisions available during inclement weather.

A dedicated wash bay will be constructed adjacent to the truck shop to allow for cleaning of mining equipment and vehicles prior to maintenance activities. The wash bay will be a fabric-covered structure with insulation and vapor barrier systems to manage moisture and maintain building integrity.

Warehouse storage infrastructure will be provided through a combination of heated and cold storage buildings constructed as steel-framed fabric structures. These facilities will house spare parts, consumables, and equipment required for the process plant, crushing facilities, maintenance operations, and the mining fleet. Additional covered and open laydown areas will be established for storage of large materials and equipment that do not require environmental protection.

Personnel facilities will include modular mine dry complexes and administrative offices. These modular units will provide locker rooms, showers, and washroom facilities for both male and female personnel and will support workers during construction and operations. Separate mine dry facilities will be located near the open pit operations and the process plant to reduce travel time during shift changes. The site office



complex will consist of modular units providing private offices, shared workspaces, meeting rooms, and operational coordination areas used for daily mine planning and workforce management.

16.8 Workforce Requirements

The Project is planned to operate as a continuous mining operation, running 24 hours per day, 7 days per week, and 365 days per year. Operational and maintenance personnel required to support continuous production will work two 12-hour shifts per day. Staffing will follow a rotating four-shift schedule to ensure full coverage while allowing for appropriate rest periods. Technical Services and Management personnel will follow the same shift rotation as the production crews to maintain alignment between operational and technical decision-making.

Labour requirements for equipment operators are based on the number and type of equipment units in operation, their utilization, and the shift rotation schedule. Maintenance staffing levels are determined from the number of equipment units in the fleet, expected mechanical availability targets, and typical labour ratios required to maintain each type of mining equipment.

The mine operations department will oversee all mining activities, including drilling, loading, and hauling of material, as well as operation of the waste rock storage facility, haul road construction and maintenance, and mine dewatering. Each shift crew will be supervised by a mine shift foreman responsible for coordinating production activities and ensuring safe operations. Explosives delivery and blast hole loading will be completed by a specialized contractor.

Overall workforce requirements include personnel for mine supervision, operations, maintenance, and technical services. Total staffing levels are expected to increase as mining ramps up, reaching a peak workforce of approximately 480 personnel in the Mine Department excluding contractor support. Table 16.11 lists all mine workforce requirements for each year.

**Table 16.11
Annual Personnel Requirements**

Mine Workforce Requirements	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mine operations	99	247	245	230	261	284	292	295	293	276	278	274	29
Mine maintenance	54	141	141	141	141	141	141	141	141	141	141	141	54
Technical Services	33	44	44	44	44	44	44	44	44	44	44	44	12
Total	186	432	430	415	446	469	477	480	478	461	463	459	95



17.0 RECOVERY METHODS

17.1 Introduction

This section describes the recovery methods used for the Coffee Gold Project, including crushing, heap leaching, and ADR facilities. The flowsheet selection, operating parameters, and design criteria are based on metallurgical test results discussed in Section 13.0. The gold recovery process is designed to treat 7.4 Mt/a with an average gold head grade of 1.25 g/t based on the life of mine resource and an overall gold recovery rate of 77.5%.

The two-stage crushing plant is designed to operate at an average hourly throughput of 1,495 t/h, for 18 h/d and 275 d/a operation. The crushing and heap leach pad loading are planned to be suspended during the coldest period of the year. The process plant will be situated adjacent to and downstream from the HLF, thereby minimizing the pumping and pipeline requirements for pregnant and barren solutions transportation. The leaching irrigation will operate year-round except during scheduled maintenance downtime. The PLS will be collected and delivered to the plant at a nominal flow rate designed to be 1,100 m³/h. The ADR plant is designed to utilize 6-tonne carbon circuits to extract gold from the PLS and produce gold doré as a final product.

17.1.1 Background

In 2018, Hatch delivered engineering and procurement services to Goldcorp to support the development of process and infrastructure for the Coffee Gold Project. In 2019, a throughput update was conducted for the process design, increasing the feed capacity from a nominal 5 Mt/a to 7.4 Mt/a. In 2024, JDS updated the design for an internal study (Internal Engineering Report) by revising the pricing and integrating new engineering technology. The report reflects a similar capacity, with only minor variations.

The WSP QP has completed a technical review of the JDS process design, focusing on mineral processing and recovery, with a nominal throughput of 7.4 Mt/a.

17.2 Process Design

17.2.1 Overall Process

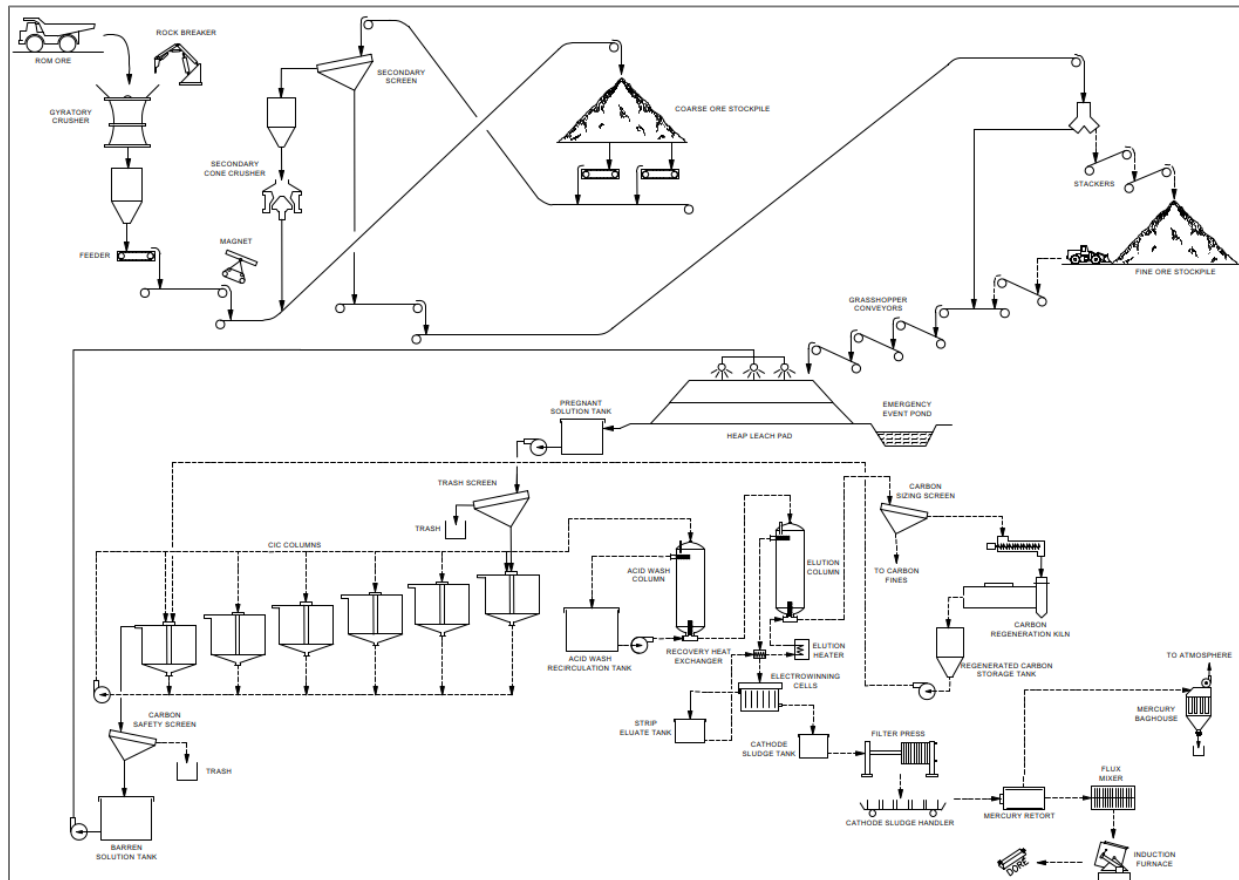
The process design subsections below outline the annual throughput, main mass flows, key process parameters, and operating conditions.

The ROM material will be processed at a nominal capacity of 7.4 Mt/a. The crushing facility is designed to handle an average of 1,495 t/h, operating at 75% availability for 275 d/a. The plant will utilize two stages of crushing to achieve a product size P80 of 50 mm. The primary crushing will use a gyratory crusher, followed by secondary crushing via a cone crusher. The crushed material will be transferred to the heap leach pad using overland and grasshopper conveyors.

Barren leach solution containing cyanide at a concentration approximately 200 ppm and pH maintained at 10.5~11.5 by lime will be pumped to the heap and percolate downward, dissolving gold into the solution. The PLS from the heap leach pad will be collected and transferred to the PLS storage tanks with a holding capacity of approximately two hours. The PLS will be pumped at a constant rate through six stage of adsorption by CIC process, followed by desorption of loaded carbon through an elution circuit. The eluted gold rich solution is treated through electrowinning and refining to produce gold doré as the final product. Barren carbon will be regenerated through a regeneration kiln and files removed by a screen.

Figure 17.1 presents a conceptual overall process flow diagram illustrating the main process.

Figure 17.1
Conceptual Overall Process Flow Diagram



Source: WSP (2026)

Based on the Internal Engineering Report (JDS, 2024), several measures have been considered in the design to help manage severe winter conditions:

- No stacking during the coldest period of the year (typically January through March);
- The barren solution will be heated as necessary to ensure a minimum discharge temperature that can sufficiently ensure a solution temperature well above freezing condition when entering and percolating through the heap;
- Drip emitter lines will be buried during winter months to enhance insulation;
- Dedicated standby generators will be provided to supply backup power for the barren solution pumps;
- The process plant will be housed within a heated, insulated facility.



17.2.2 Key Process Design Criteria

17.2.2.1 Crushing and ADR Plant Key Process Design Criteria

Table 17.1 summarizes the key design criteria for the crushing and ADR plant.

Table 17.1
Key Process Design Criteria – Crushing and ADR Plant

Description	Unit	Value
General		
Annual Throughput	t/a	7,400,000
Nominal Daily Throughput	t/d	26,909
Operating Days per Year	d/a	275
Crushing Plant Availability	-	75%
Average LOM Feed Grade	g/t Au	1.25
ROM Moisture	% w/w	4.5
Specific Gravity (Average)	t/m ³	2.7
Crushing and Handling		
Crushed Feed Bulk Density	t/m ³	1.6
Crushing Feed Size F100	mm	1,000
Crushing Feed Size F80	mm	619
Primary Crusher Type	-	Gyratory
Primary Crusher Model	-	Superior MK-III 50-65, or equivalent
Secondary Crusher Type	-	Cone Crusher
Secondary Crusher Model	-	MP800, or equivalent
Secondary Crushing Product P80	mm	50
Stockpile Capacity (fine material)	t	10,000
Heap Loading and Spreading Method	-	Conveyor with Radial Stacker
Adsorption, Desorption and Recovery (ADR)		
Pregnant Solution Flow Rate	m ³ /h	1,100
Number CIC Trains	-	1
Number of Columns per CIC Train	-	6
Carbon Capacity per CIC Column	t	6.0
Elution Method	-	Modified Zadra
Carbon Regeneration Capacity	-	100%
Number of Electrowinning Cells	-	2
Mercury Retort	-	Required
Final Product	-	Doré Bar

17.2.2.2 Heap Leach Facility

Table 17.2 presents the engineering design criteria for the heap leach pad. Designs and analyses will be further developed in subsequent Project phases to meet criteria defined by the Yukon Guidelines for Mine Waste Management Facilities, February 2023 and other regulatory requirements.



**Table 17.2
Heap Leach Facility Key Design Criteria**

Description	Unit	Value
General		
Heap Capacity	Mt	91.5 +
Heap Loading and Spreading Method	-	Conveyor with Radial Stacker and Dozing
Barren Solution Flow Rate	m ³ /h	1,100
Heap Leach Pad Liner System		
Geosynthetic Liner	-	2.0mm Linear Low Density polyethylene (LLDPE) liner material to provide required puncture resistance, elastic strain range and resistance to solution attack and chemical breakdown along with cold weather performance for the Project's climate conditions.
Geosynthetic Clay Liner (GCL)	-	Geosynthetic clay liner below the geosynthetic liner to provide a composite liner to minimize leakage. Objective maximum permeability of 5x10 ⁻⁹ cm/s.
Overliner Drain Fill (ODF)	-	Crushed clean rock to provide a free draining layer under the placed material and to protect the lining system from damage by material placement while not impacting the conveyance of solution to the recovery wells.
Event Ponds Liner System		
Liner System	-	Lining to comprise a double LLPDE geomembrane liner system with leak detection and recovery system (LDRS) between the layers for Event Pond 1 (north and south) and the Event Pond 2 Process Segregation. A single layer of LLPDE will be used to line Event Pond 2 outside the Process Segregation. A GCL layer will be placed below the liner of all ponds and will have an objective maximum permeability of 5x10 ⁻⁹ cm/s.
Leak Detection and Recovery System (LDRS)	-	A system within the Event Pond 1 (north and south) and Event Pond 2 Process Segregation to collect any leakage through the composite liner and convey it to monitoring points. The system is to comprise of geonet or a similar synthetic drainage product to collect and convey any leaked solution to a gravel filled sump and pumping system.

17.2.3 Crushing and Handling

The crushing plant is scheduled to operate 275 d/a. During the cold winter (typically January to March), the heap leach pad will not be available for stacking from the crushing circuit. Consequently, the crushing plant will be shut down and ROM material will be stockpiled.



During normal operations, ROM material will be transported by haul trucks and dumped into the primary crusher dump pocket. A gyratory crusher is utilized for the initial size reduction to a P90 of approximately 145 mm. The crushed product will be transferred via conveyor to a coarse stockpile, from which it will be reclaimed using feeders and feed to the secondary crushing cone crusher closed with a sizing screen to produce a fine product with a P80 of approximately 50 mm.

The fine product will be conveyed via transfer and overland conveyors to the heap leach pad. The material will be stacked on the heap leach pad using a series of grasshoppers, a bridge conveyor, and a stacker.

A fine stockpile will be constructed to provide an alternate temporary storage when there is an issue from stacking. Feed will be reclaimed via front end loaders when needed.

Lime will be added directly onto a transfer conveyor to maintain the pH.

17.2.4 Heap Leach Facility

The Coffee Gold Project utilizes a conventional multi-lift, free-draining, ridge-top heap leach pad, planned to be developed in five stages. Material is delivered from the crushing circuit to the toe of the heap leach pad via an overland conveyor, then continues to be placed on the heap leach pad using a system of mobile conveyors and a radial stacker. A dozer will then be used to spread the material and maintain a nominal lift setback and in preparation for leaching.

The heap leach pad is designed to accommodate 91 Mt of leach material and will include solution delivery and collection systems, an engineered liner system, perimeter channels, and a system of event ponds to contain meteoric water that enters the process circuit and to supply the process operation.

17.2.4.1 Leaching & Solution Handling

After each leach cell has been stacked in nominal lifts and spread by dozer, drip emitters will be used to apply a dilute cyanide solution (barren solution) to the stacked material. Barren solution will be delivered to the drip emitters from a barren solution tank located at the ADR plant, followed by conveyance through two insulated and heat-traced pipelines to the north and south perimeters of the heap, based on the location of the actively leaching cell. The barren solution distribution system proceeds up the side slope of the heap leach pad to the active cell on the top lift of the heap leach pad, where the solution is applied to the material via drip emitters. Barren solution will percolate through the stacked material by gravity, dissolving cyanide soluble gold. To mitigate extreme winter temperatures, a boiler will heat the barren solution and drip emitters will be buried to maintain thermal integrity.

The resulting gold-bearing solutions are collected via gravity through a network of corrugated drainage piping and collection pipes atop an engineered liner system. The liner system includes a sufficient layer of crushed overliner to protect the Linear Low-Density Polyethylene (LLDPE) primary geomembrane liner. The piping network continues to convey solution through header pipes within the lined perimeter channels, which then flow by gravity to two pregnant solution tanks located within the insulated ADR process plant building for recovery of the gold from solution.

Meteorological water that enters the process system during precipitation, freshet events, or upset conditions is managed by the lined heap leach pad event ponds. This collected water will be used as make up water to meet process water flow demands, with excess water being delivered to water treatment.



17.2.5 ADR Process Plant

17.2.5.1 Carbon Adsorption

The PLS collected from the heap leach pad will be stored in a PLS storage tank with approximately two hours of holding capacity, followed by being pumped to the CIC circuit in the ADR process plant. The CIC circuit consists of six sequential, casCAing carbon columns. A trash screen will be installed just upstream of the CIC columns to prevent trash from entering the columns. Gold contents in the PLS will be adsorbed by the activated carbon in the columns. Overflow from the final column is discharged onto a recovery screen to capture any carbon particles carried forward, after which the solution gravitates to the barren solution tank for preparation and recirculation to the heap leach pad.

Sodium cyanide solution, caustic solution, antiscalant, and make-up water are added to the barren solution tank as required. The heating of barren solution is planned during cold weather to maintain the thermal integrity of the heap.

On average, 6 tonnes of loaded carbon are withdrawn daily from the first carbon column and transferred to acid wash and stripping circuits. Carbon from each subsequent column advances forward, with newly reactivated carbon replenishing the sixth column.

17.2.5.2 Desorption and Gold Refining

Carbon Acid Wash

Loaded carbon from the CIC circuit is processed in an acid wash vessel with a 3% hydrochloric acid (HCl) solution to remove calcium, magnesium, sodium salts, silica, and fine iron contaminants. Organic foulants, such as oils and fats, unaffected by acid treatment, are removed during subsequent thermal reactivation in a kiln.

Typically, one acid wash/elution cycle occurs daily during normal operation.

Carbon Stripping (Elution)

Following acid washing and pH adjustment, loaded carbon will be transferred to the elution circuit and gold will be stripped from it using a modified ZADRA process. Elution utilizes a solution containing roughly 1% sodium hydroxide and 0.1% sodium cyanide, operating at an elevated temperature and pressure of 130~150°C and 400~600 kPag. The pregnant solution from the elution columns, after cooling, will gravity feed the electrowinning cells. After elution, the stripped carbon will be transferred to the regeneration circuit for carbon reactivation.

Carbon Regeneration

Stripped carbon from the elution column will feed a dewatering screen for water removal and fines elimination, upstream of the regeneration kiln. The carbon regeneration normally operates daily at temperatures between 650°C and 750°C.

Regenerated carbon will be quenched in water before it is reintroduced into the CIC circuit. Attritional and fines loss from the process are offset by introducing new carbon into an attrition tank.



Refining

Pregnant solution from the elution column after cooling and pressure reduction, will flow via gravity to a secure gold room, where it is processed in two electrowinning cells. Gold will be deposited onto knitted-mesh steel wool cathodes in the electrowinning cells. Loaded cathodes will be power washed to collect gold-bearing sludge. This material is filtered to reduce moisture and retorted for mercury removal. Retort residue is combined with fluxes—borax, silica, and soda ash—and smelted in an induction furnace to produce doré bullion and slag. Doré will be transported off-site for further refining, while slag is processed to recover prills for re-smelting. Finished gold bars are securely stored onsite prior to shipment by aircraft to an off-site facility.

17.2.6 Reagents

Main reagents for the Project include sodium cyanide (NaCN), sodium hydroxide (NaOH), HCl, antiscalant and lime.

Sodium cyanide briquettes in secure containers will be delivered to site for preparation of cyanide solution. The cyanide solution will be added to the barren tank for heap leaching extraction and to the elution circuit for gold stripping from loaded carbon. The principles and standards of practice for the transport to site and handling of cyanide on site will be in accordance with the guidelines set out in the International Cyanide Management Code (ICMC).

Sodium hydroxide will be delivered to site in concentrated solution form in 1 tonne totes. The caustic will be mixed with water for dilution and stored for distribution to the acid wash and strip circuits.

Hydrochloric acid solution will be supplied to site in liquid form in 1 tonne totes. The solutions will be used for the acid washing circuit to remove inorganic impurities deposited on the carbon.

Antiscalant will be also supplied to site in liquid form in 1 tonne totes. The solution will be metered directly from the totes for distribution in the plant.

Hydrated lime will be delivered to the site as raw material and stored in a 200 tonne lime silo. The lime will be added to the heap leach feed conveyor prior to stacking on the heap leach pad.

17.2.7 Laboratory

An assay and metallurgical laboratory will be equipped to perform sample preparation and assays by atomic absorption, fire assay, and CN soluble analyses.

17.2.8 Services and Utilities

17.2.8.1 Plant / Instrument Air

Compressed air for both instrumentation and plant operations will be produced by compressors to meet the processing facility demands. Instrument air will undergo filtration and drying processes to achieve the required quality standards. The compressed air will be distributed via air receivers throughout the plant.

17.2.8.2 Water

Raw water for the site will be sourced from a well field adjacent to the Yukon River, utilizing multiple wells to accommodate seasonal variations in demand. The facilities will feature a modular system for potable water provision. Raw water designated for process usage will primarily serve reagent preparation and



make-up for leaching solutions, while firewater requirements will be supplied directly from raw water sources.

17.2.8.3 Plant Power Demand

Process power will be provided by four on-site generators configured in an N+2 (2+2) arrangement. The estimated peak summer demand for the crushing, handling, conveying, leaching, and processing plant is approximately 9,500 KVA.

17.3 QP Comments

The processing design for the Project has undergone several studies in the past. The current process configuration is now sufficiently developed to meet or exceed the PEA requirements, with production capacity aligned to the mine plan. The selected combination of gyratory and cone crushers is expected to provide a reliable solution as compared to mineral sizers for the required crushing operations.

WSP recommends advancing the Project and its process design to the next phase of project development..



18.0 PROJECT INFRASTRUCTURE

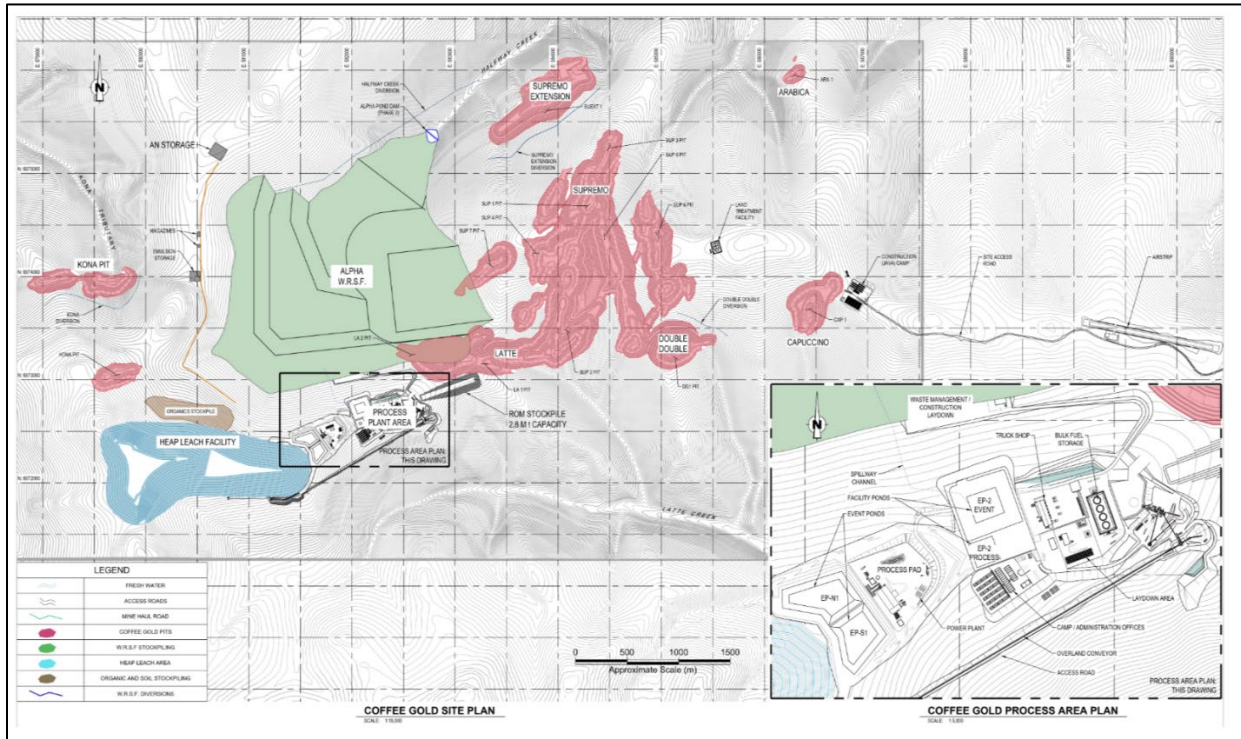
18.1 General Site Arrangement

The preliminary Coffee Project site layout and process plant general arrangement is shown in Figure 18.1. Infrastructure is concentrated near the process plant to maximize operational efficiency during northern Canadian winters. The major infrastructure at the Coffee site includes:

- Access and site roads;
- HLF and event ponds (described in Section 17.0);
- Airstrip;
- Power plant and power distribution;
- Fuel storage;
- Process plant and related facilities;
- Truck shop, warehouse, and storage areas;
- Mine dry and office complex;
- Camps (construction and operations);
- Waste management facilities;
- Explosives storage;
- Water supply; and
- Site water management facilities.



Figure 18.1
Preliminary Site General Layout Arrangement



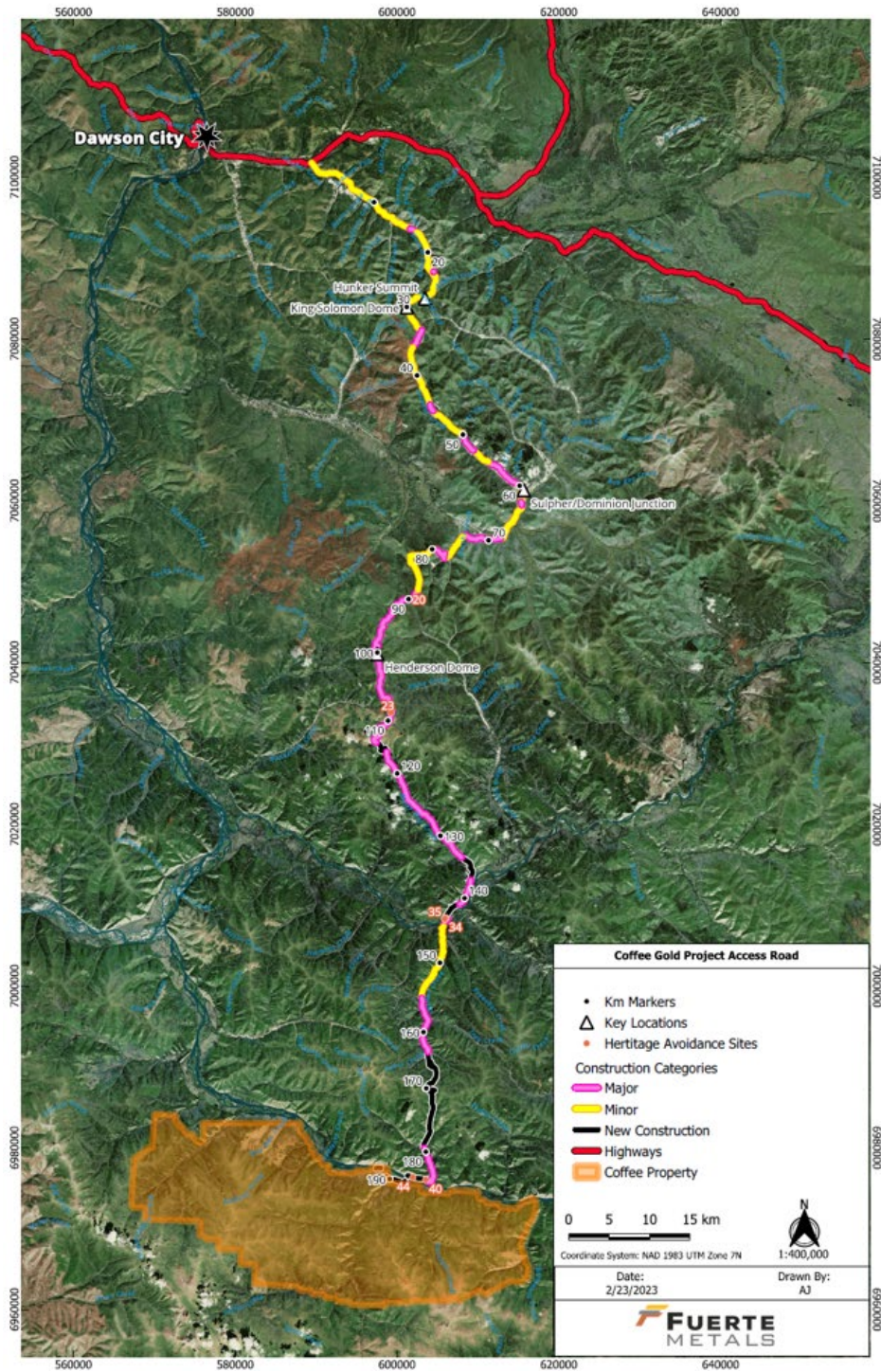
18.2 Site Access Roads

As a critical component of the project infrastructure, Kaminak evaluated route options in 2015, and the Dawson City route was selected due to its lower initial capital cost and lower impact on environmentally sensitive areas and First Nation's traditional territory. The proposed NAR crosses the Stewart and Yukon Rivers. It will be operated throughout the year, except for periods when the Stewart River and Yukon River are either freezing up in the fall, through early winter, or breaking up in the spring.

Figure 18.2 illustrates the locations of NAR on Google Maps.



Figure 18.2
NAR on Google Maps



Source: Fuerte Metals (2023)



18.2.1 Northern Access Route

The all-weather access road between Dawson and the mine site will be the main site access road and form a critical component of the Project infrastructure. The NAR will be used to provide equipment, fuel, and other supplies during construction and operations.

The road (5 m in width) has been designed for single lane, two-way traffic with pullouts. In the valley bottoms, where higher standards are more easily attained, the road design speed is 50 km/h. In mountainous terrain the road design speed is 30 km/h.

The proposed NAR will be about 214 km long, starting roughly 13 km east of Dawson City, YT, and following Yukon Government roads for the first 58 km until the Sulphur-Dominion Junction. From there, it extends another 130 km to connect with the Java exploration access road at the Coffee property. The Java Road links the planned mine site to a barge landing on the Yukon River's south bank, is about 20 km in length, and will need upgrades for future mine use.

From the Sulphur-Dominion junction to Java Road, the NAR uses about 100 km of upgraded placer mine roads and adds 30 km of new construction, all built to the same standard. The single-lane road with pullouts will allow two-way traffic at 30 to 50 km/h and ensure year-round access.

In addition to road work, barge crossings on the Stewart and Yukon River are required to access the Project site. Four barge landings will be required on either side of the Stewart and Yukon Rivers. Each barge landing will consist of a concrete ramp for docking the upstream-facing barge. During the open-water season, barges will be used to transport supply trucks across the rivers. During winter months, ice bridges will be constructed across the rivers. Storage facilities for all major consumables have been sized to allow for 15 weeks of road access interruptions during spring break-up and fall freeze-up.

Two 75 tonne capacity barges will be operated, one on the Stewart River and the other on the Yukon River. They will not operate during the freeze-up, prior to ice road use, or when the ice is breaking up in the spring.

Ice roads will be established annually to allow for winter river crossings. In conjunction with the ice road on the Yukon River, a 3.6-km-long winter road located south of the Yukon River will be used annually. It will minimize the distance traveled on ice roads over the Yukon River.

The proposed route crosses the Stewart and Yukon Rivers. It will be operated on a year-round basis except for periods when the Stewart River and Yukon River are either freezing up in the fall through early winter or breaking up in the spring.

18.3 Site Roads

Site roads will consist of haul roads and service roads. In general, site roads will be constructed with embankment fills. The embankment material will be rock material sourced from infrastructure earthwork activities or from open pit waste material.

Appropriate thicknesses of roadbed material will be utilized according to the existing ground conditions.

18.4 Airstrip

Air transport is the main method used for moving mine staff and incidental freight to and from site. It will feature a non-instrument Visual Flight Rules (VFR) approach with 500 feet minimum decision altitude (MDA).



18.5 Camps

Camp facilities will be prefabricated, skid-mounted, and hard-walled for year-round use in Yukon's cold climate, built to National Building Code standards. A Yukon permit and engineering approval are required before occupancy.

18.5.1 Construction Camp

The construction camp (Java comp) will be located between the airstrip and Supremo Pit, and away from the main infrastructure area during construction. It will accommodate up to 200 people in single rooms with shared washrooms across five dorm wings, plus kitchen, diner, and recreation hall.

18.5.2 Operations Camp

The operations camp will be located close to the processing facilities. It will have room for 228 people in single rooms with private washrooms. The camp will contain a kitchen, diner and recreation hall connected by arctic corridors. The camp will have both wastewater and potable water treatment plants.

18.6 Buildings and Support Structure

Main buildings for the Coffee Gold Project site will include the following:

- **ADR Process Plant Building:** A steel framed, insulated, fabric covered building. The ADR building will have a dimension of approximately 38 m wide x 90 m long x 17 m tall.
- **Assay Laboratory:** Located adjacent to the process plant. This facility will serve the plant assay, environmental, and metallurgical requirements. The laboratory consists of pre-fabricated modules and ancillary equipment, such as drying ovens, dust and fume control, and heating equipment.
- **Truck Shop and Wash Bay:** Steel framed, fabric covered building.
- **Warehouse and Laydown Area.**
- **Mine Dry:** The mine dry complexes will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions.
- **Site Office Facility:** The site office facility will be constructed of modular units manufactured off site.
- **Construction and Permanent Camps.**

18.7 Explosive Storage

Explosives storage facilities will accommodate the peak storage requirements dictated the winter freeze-up, when the NAR is not fully operational. The explosive storage facilities will consider ammonium nitrate storage silos, an emulsion plant, powder magazines and cap magazines.



18.8 Utilities

18.8.1 Power

A combination of power plant configurations will be utilized to fulfill the on-site electrical generation requirements. The base case involves establishing a primary power plant at the plant site pad, which will supply power to crushing and handling, conveying, the ADR plant, heap leach operations, and auxiliary facilities such as the administration building, plant dry, and mine maintenance facility. The generators are designed for liquid natural gas (LNG) substitution and will comprise four diesel-fired reciprocating engine generator sets (gensets) arranged in an N+2 (2+2) configuration. Two generators will operate during summer months, while one will suffice in winter. Efficient building heating will be achieved through a glycol loop system, with heat recovery from the power plant implemented when feasible. Each generator will have a prime rating of 5.6 MW, operating at 720 rpm and producing electricity at 4,160 V. The maximum gross power output will total 22 MW, based on four gensets running at 100% capacity (4 x 5.6 MW). Estimated electrical loads at the plant site are detailed in Table 18.1.

Table 18.1
Coffee Site Estimated Power Load

Description	Peak Summer (kVA)	Peak Winter (kVA)	Energy Annual (kWh)
Crushing and handling	731	336	3.0 M
Heap Leach and Conveying	2,370	527	10.5 M
Process Plant	6,425	3,258	24.0 M
Onsite Infrastructure	2,284	2,644	12.3 M
Grand Total	11,810	6,765	50.7 M

Source: Hatch (2024)

The power plant will be constructed on a concrete pad and enclosed in a steel framed, fabric covered building.

18.8.2 Fuel

18.8.2.1 Diesel Fuel

The diesel storage is sized for 15 weeks of diesel use at full production, supporting mining, equipment, and power needs. This ensures supply during periods when road deliveries are blocked by river freeze-up or thaw. Fuel dispensing equipment for mining, plant services, and freight vehicles will be located adjacent to the fuel tank bund and the fueling area will drain into the bund. A fuel transfer module will provide fuel to the power plant day tank and diesel consumers in the process plant.

18.8.2.2 Liquid Natural Gas (LNG)

The LNG storage facility, located near the power plant on the ADR plant pod, will be sized for four days of onsite storage as part of the LNG supply contract. It will comply with CSA Z276, requiring a 23 m setback from buildings and property lines, and 1.5 m between containers.



18.8.3 Water

18.8.3.1 Raw Water

Site water will be drawn from a well field at the Yukon River, with several wells needed to meet seasonal demand. A pump station at the wells and a booster station along the 20 km pipeline, following the access road, will deliver water to the Java construction camp, operations camp, and process plant fresh water tank.

18.8.3.2 Potable Water

A modular potable water system will be supplied to serve the facilities. The system will be set up and operated throughout construction. It will be based at the operations camp, and water will be delivered either by pipes or by truck to more distant facilities.

18.8.3.3 Fire Water

Fire water will be sourced directly from raw water. The site's fire protection will rely on a firewater main, hydrants, and standpipe system, all supplied by a firewater tank and a modular pump unit. This arrangement includes a main electric pump, a jockey electric pump, and a backup diesel pump. The entire firewater pumping system will be housed inside a modular building.

18.9 Communications

Field equipment, such as two-way radios and cell phones, have been deployed to facilitate exploration. A microwave radio link to Dawson City will also be set up. Currently, there are two communications towers installed offsite at King Solomon Dome and Henderson Dome, along with one tower located at the main site.

18.10 Sewage System

Sewage will be processed by a wastewater treatment plant. All domestic wastewater will be directed to the adjacent treatment plant. Treated effluent will be discharged to the surface for soil exfiltration, and dewatered sludge will be incinerated.

18.11 Water Management

18.11.1 Water Strategy

The water management strategy for the Coffee Gold Project is summarized as follows:

- Ensure adequate flexibility in water management to meet the operational needs of the Project while remaining protective of the receiving environments downstream of the Project.
- Maintain the physical integrity and stability of the Project through controlled water management.
- Manage water that could potentially be affected by the Project (e.g., contact water, excess HLF water) in accordance with industry best management practices.
- Ensure that the quantity and quality of water discharged from the mine complies with applicable regulatory requirements.



- Minimize alteration to the pre-development drainage network, and the volume of contact water to be managed.
- Maintain the physical integrity and stability of slopes and watercourses downstream of the Project.
- Minimize any potential effects to surface water and groundwater quantity and quality in the receiving environment by minimizing transport of parameters of potential concern from the Project

18.11.2 Climate and Hydrology

Two weather stations have been installed at the site, adjacent to the mine access road, between the camp and the proposed HLF (at elevation 975 masl) and a second automated weather station was installed at an elevation of 1,340 masl and adjacent the proposed HLF footprint (i.e., ~400 m to the northwest of the HLF). These weather stations used to assist with the climate and some of the hydrology inputs (for example to assist with precipitation estimates). In addition to these weather stations there are 15 local flow monitoring stations that provide either continuous streamflow gauging or instantaneous field observations.

With this data an initial hydrologic model has been developed, using rainfall, snowpack, temperature and streamflow records collected onsite to estimate the runoff response in Halfway Creek and other regions of the Project and to assist with the initial water management infrastructure design concepts. This model represents soil, climate and shallow groundwater conditions for natural catchments and provided peak flow and storm volumes to support initial engineering designs.

18.11.3 Water Infrastructure

Precipitation that comes into contact with the mine infrastructure (such as the plant pad, overburden stockpile and waste rock facility) will be collected and retained in ponds to minimize the concentration of total suspended solids (TSS) in contact water before it is discharged to the receiving environment. The primary pond will be the Alpha Pond, located below the Alpha WRSF in Halfway Creek. The Facility Pond will collect runoff from the plant site area and discharge towards Alpha Pond.

The Alpha Pond will be constructed downstream of the Alpha WRSF with the primary purpose to control sediment but will also have the secondary benefit of allowing a set location for water quality to be checked downstream of the WRSF. The Alpha Pond is expected to have the largest potential for increased sediment, due to the upstream Alpha WRSF, and will require the largest dam structure with the largest associated construction costs. In comparison to the Alpha Pond, the Facility Pond is small and more focused on sediment control at the source (i.e. at the top of the catchment around the plant pad).

The Environmental Design Flood (EDF) selected for both ponds was the 1 in 10-year 24-hour rainfall event, based on their purpose of controlling sediment and following the BC MOE technical guidance for sediment ponds (BC MOE, 2015). Emergency spillways were sized for an Inflow Design Flood (IDF), based on the expected dam consequence classification. The IDF assigned to Facility Pond was the 1 in 200-year flood event, based on an expected dam consequence classification of Low. The Alpha Pond dam was assigned a significant dam consequence classification, and multiple IDF events were evaluated, including one 200-year flood, a 1,000-year flood, and a maximum snowmelt-dominated flood.

TSS will be removed by gravity settling and/or addition of flocculant, if required. Flocculant addition is a contingency measure and has not been included in the design. Sediment will accumulate within the pond and will be removed periodically so that the ponds maintain full design capacity for effective treatment. Each pond geometry design includes consideration of sediment accumulation in the calculations for pond performance and includes maintenance access for regular sediment removal.



Ponds will discharge via an engineered spillway and a pumping system. The Facility Pond will also be equipped with a pumping system to route runoff to the processing plant for makeup water, as needed.

Four diversion channels will be constructed to divert non-contact water around disturbed areas. The Double Double diversion, Kona Pit diversion and Supremo Extension diversion will divert non-contact water around the associated pits. The Halfway Creek Diversion will divert water around the Alpha WRSF. All channels will be in-place throughout operations and decommissioned in closure and are designed to convey the 1 in 10-year return period flood event, based on the design criteria applied to downstream sediment ponds. More extreme events may result in reduction of available freeboard or partial overtopping of diversion berms, which have been accounted for in the evaluation of downstream infrastructure and water balance modelling.

The open pit dewatering system is designed to store up to the 10-year storm event and pump water to either the Facility Pond or other inactive mining areas. Pumping infrastructure were sized on a per-square-kilometer basis based on the extent of the active mining area. The most critical period for pit dewatering identified from the mine plan was Year 2, at which time mining in the Latte Pit, Supremo Pit and the Kona Pit will all be active. The quantity of pumps was determined based on the storm volume and the total head required to pump water to Facility Pond.

The Alpha WRSF will be built through Halfway Creek and will include an engineered rock drain along the natural channel bottom to convey upstream water through the dump towards Alpha Pond. The rock drain was sized to convey the 1 in 200-year flood event, as it will remain operational through the post-closure period. The design assumes the upgradient Halfway Creek Diversion has been breached (not included to reduce flows), resulting in a more conservative (larger) peak inflow.

The Heap Leach Facility will include three event ponds and a rainwater pond.

18.12 Heap Leach Facility Water Treatment

The QP for water treatment has reviewed the vendor WTP design work (WesTech, 2025) based on the 2024 Internal Engineering Report and considers it relevant and current. The following discussion summarizes content from that report.

The water treatment strategy for the HLF considers the following:

- HLF generates a positive water balance; excess solution removal is required to maintain Event Pond storage capacity.
- HLF solution contains parameters of concern; active water treatment is required prior to release to environment.
- WTP treats excess HLF solution starting in year-2 of operations – pit dewatering may also be diverted into the WTP for treatment when possible
- WTP effluent gravity flows to Alpha WRSF rock drain to the Alpha Pond and ultimately to Halfway Creek.

The mine camp water treatment and contact water requiring TSS removal are not considered in the WTP design.

The HLF WTP will be decommissioned once no longer required. In the Post Closure phase, HLF drain down may be routed to a semi-passive treatment facility post-active rinsing and closure of the HLF, if required.



Treated effluent from the WTP flows to the Alpha WRSF rock drain to the Alpha Pond, and ultimately to Halfway Creek where compliance point monitoring is required. The WTP reduces the concentrations of cyanide species, ammonia/nitrogen species, dissolved metals/metalloids, sulphate and Total Dissolved Solids (TDS) to achieve the water quality objectives established via project water management and water quality modelling (Goldcorp, 2025, Lorax, 2023).

The WTP will be sized to accommodate the peak projected solution removal rates. The facility will require operational flexibility to accommodate seasonal variability in influent flow and chemistry and to support shutdown during lower-flow periods.

The WTP design comprises the following process areas:

1. **Cyanide Destruct** – Reactor tanks and chemical dosing systems for Weak Acid Dissociable (WAD) cyanide destruction via copper-catalyzed hydrogen peroxide oxidation.
2. **Ammonia Oxidation** – Reaction tankage and dosing systems for breakpoint chlorination using sodium hypochlorite.
3. **Metals / Metalloids Precipitation** – Reaction tanks, clarifier, and filtration equipment for metals and metalloids removal via hydroxide precipitation and ferric hydroxide co-precipitation and/or organosulphide precipitation.
4. **Nitrate Removal**– An electro-biochemical reactor (EBR) system with power supply, controls, and carbon dosing/handling to enable biological denitrification under variable loads and seasonal conditions.
5. **Membrane Treatment** – Ultrafiltration (UF) pretreatment and reverse osmosis (RO) skid(s), including feed pumps, clean-in-place (CIP) systems, and associated controls. Waste streams from this system are currently planned for disposal to an approved facility.
6. **RO Brine Management** – Brine treatment tanks, lime addition, a dedicated clarifier for gypsum precipitation/settling, and piping to return desaturated brine to the HLF. This system provides a sulfate sink via gypsum removal to manage sulfate accumulation in the recycle circuit.
7. **Chemical Storage and Distribution** – Bulk chemical storage and metering systems are required for key reagents including hydrogen peroxide and catalyst addition (copper sulphate) for the Cyanide Destruction Area, sodium hypochlorite for the Ammonia Treatment Area, lime, coagulant and polymer for the Metals/Metalloids Precipitation as well as Brine Management, organic carbon and nutrient for the EBR, and acid for pH adjustment.

The WTP is powered by the site power plant. The access road is designed for bulk treatment reagent deliveries. Winterization includes allowances such as building enclosure, heat tracing, insulation, and freeze protection, where required, to support year-round operation.

WTP dewatered solids from metals/metalloids precipitation and filtration, and gypsum solids from RO brine treatment, requires solids handling infrastructure. This includes sludge collection, pumping infrastructure, dewatering with filter press or similar, and temporary storage prior to disposal. Final disposal location(s) and waste classification requirements will be confirmed during subsequent study phases and will need to be aligned with the project waste management and permitting framework.



18.13 Waste Rock Management

During operations, the primary locations for waste rock storage and placement will be within a central WRSF, the Alpha WRSF, and within the mined-out pits. The cumulative waste rock volumes over the projected 13 years of active mining is 688 Mt. Annual waste rock volumes are variable (between 23 and 78 Mt/a) based on mining schedule and the number of concurrent pits being mined. This leads to active waste management plans that evolve with the mine plans with higher waste tonnages generated in Years 3 and Years 7 to 9.

The Alpha WRSF will be constructed primarily by end-dumping waste rock material in the Halfway Creek valley. The initial portion of the WRSF will be required to be built bottom up to allow for the rock drain construction as well as to help improve the overall facility stability. Following this initial development more conventional waste rock placement (load, haul, dump) approaches will be utilized. Expansion of the Alpha WRSF will be carried out by first constructing the base rock drain area, followed by placement of additional waste volumes.

The design of the Alpha WRSF considered the following:

- Waste rock type and its influence on crushing and breaking for preliminary rock drains
- Terrain and extent of colluvium and ice-rich material for final geometries
- Seasonal saturation of near surface colluvial soils and the presence of underlying permafrost for overall WRSF arrangements.

Pit backfilling was done based on the annual mine plan and available waste rock storage. Haul distance estimates were used to determine placement of the waste rock between the mined-out pit storage and the waste rock dump. Mined out pits used a 20% reduction in stored tonnage to account for the swelling of fractured and mined material.



19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The gold market is mature globally with numerous reputable refiners readily available. As such, a market study for the saleability of gold products or for the expected price of gold was not undertaken.

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.

The selling costs related to the refining of the doré to produce saleable gold and to the doré shipping cost estimates (see Table 19.1) used in this analysis were benchmarked based on recently completed projects of a similar nature.

Table 19.1
Selling Costs for Gold Doré

Description	Unit	Doré
Refining Cost (Gold)	US\$/oz	5.0
Gold Payable	%	99.995
Doré Shipping Charge	US\$/oz	15.0

19.2 Commodity Price

The QP (William Richard McBride, P. Eng.) has adopted the price projection for the PEA economic analysis model's base case as presented in Table 19.2

The pricing is representative of average value of the consensus opinions of several financial institutions in the December 2025 to January 2026 time period as documented in the following Figure 19.1 provided February 11, 2026 by Fuerte Metals.



Figure 19.1
Image of Consensus Gold Price Forecasts Table

Street Consensus Estimates						
Gold (US\$/oz)						
Gold (US\$/oz)	2026	2027	2028	2029	Long-term	
Average	\$4,662	\$4,626	\$4,316	\$4,110	\$3,620	
Median	\$4,569	\$4,550	\$4,450	\$4,000	\$3,300	
High	\$6,000	\$6,550	\$6,000	\$5,500	\$4,909	
Low	\$4,000	\$3,750	\$3,500	\$3,250	\$2,871	
Broker	2026	2027	2028	2029	Long-term	Report Date
BMO	\$4,550	\$4,075	\$3,675	\$3,425	\$3,000	15-Dec-25
Beacon	\$4,000	\$3,750	\$3,500	\$3,250	\$3,250	13-Jan-26
BofAML	\$4,538	n/a	n/a	n/a	n/a	Press (Jan 2026)
Canaccord	\$4,401	\$4,576	\$4,728	\$4,909	\$4,909	23-Jan-26
CIBC	\$6,000	\$6,500	\$6,000	\$5,500	\$4,000	04-Feb-26
Cormark	\$4,500	\$4,500	\$4,500	\$4,500	\$4,500	30-Jan-26
Desjardins	\$4,358	\$4,547	\$4,705	\$4,705	\$4,705	10-Dec-25
Deutsche	\$4,450	\$4,552	n/a	n/a	\$4,450	07-Jan-26
GMP Stifel	\$4,000	\$4,200	n/a	n/a	\$3,000	12-Jan-26
Goldman	\$5,400	n/a	n/a	n/a	n/a	Press (Jan 2026)
Haywood	\$4,813	\$5,000	\$4,500	\$4,500	\$4,500	27-Jan-26
HSBC	\$4,588	\$4,625	\$4,700	\$4,775	n/a	18-Jan-26
Jefferies	\$4,200	\$4,000	\$3,800	\$3,600	\$3,000	24-Jan-26
JPMorgan	\$5,708	\$6,550	n/a	n/a	\$3,850	01-Feb-26
Macquarie	\$4,225	\$3,800	\$3,700	\$3,500	\$3,000	12-Dec-25
Morgan Stanley	\$4,600	\$4,075	\$3,650	\$3,500	\$2,871	15-Dec-25
National Bank	\$5,200	\$5,200	\$4,500	\$4,000	\$3,200	03-Feb-26
PI Financial	n/a	n/a	n/a	n/a	n/a	No deck
Raymond James	\$4,500	\$4,000	n/a	n/a	\$3,300	14-Jan-26
RBC	\$4,600	\$5,100	\$4,500	\$4,000	\$3,000	10-Dec-25
Scotia	\$4,600	\$4,600	\$4,400	\$4,200	\$3,400	26-Jan-26
TD	\$4,650	\$4,600	\$4,200	\$3,800	\$3,600	22-Jan-26
UBS	\$4,675	\$4,275	\$4,000	\$3,600	\$3,250	12-Dec-25

Source: Available equity research



The consensus price used was an average against recovered gold of US\$4,110/oz Au for Y1 of production and US\$3,620/oz Au long term for the balance of the LOM. The selected gold prices are viewed as conservative compared to the February 18 EOB spot price of near US\$5,000/oz.

**Table 19.2
Commodity Pricing**

Element	Unit of Measure	Financial Model	
	Year	Y1	LT
Au	US\$/oz	4,110	3,620

With the current gold market, three year trailing average pricing for establishing a fixed gold price for the economic analysis was considered as not applicable to the Coffee Gold project.

19.3 Contracts

Contract terms and refining charges are yet to be established with refining companies. It is expected that any future contract terms will be typical. Metal prices in the sales contracts are expected to reference prices similar to those published in the London Metal Exchange (LME).



20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section provides an overview of environmental considerations for the proposed Coffee Gold Project, including aspects related to site access, land tenure, baseline environmental conditions, and anticipated environmental approval requirements. There are currently no identified environmental aspects which are anticipated to preclude development of the Coffee Gold Project. The Coffee Gold Project has completed the regulatory requirement for an Environmental and Socio-Economic Assessment. The information provided below is based on the Yukon Water Board- Type A Quartz Mining Water Use Licence and the Quartz Mining Licence Application for the Coffee Gold Mine (Goldcorp Kaminak Ltd. 2023). The Project as described in this PEA differs from the project assessed under YESAA 2017-0211. Accordingly, permitting is not assumed to proceed unchanged. A permitting strategy will be developed to assess Project changes against YESAA 2017-0211 and existing licensing applications, and to determine whether amendments or additional regulatory processes may be required.

20.2 Environmental Baseline Studies

Baseline studies across disciplines have been undertaken since 2010 and continue to be ongoing to support the Water Licence and Quartz Mining Licence applications and to meet the Decision Document for the Coffee Gold Mine Project (YESAB File No. 2017-0211). Table 20.1 provides a summary of environmental baseline and site characterization studies completed at the time of the Water Licence and Quartz Mining Licence Application submissions.

Table 20.1
Summary of Baseline Studies

Environmental Component	Data Range (Compiled to date)
Climate and Hydrology	2010 to 2021
Surface Water Quality	2010 to 2020
Physical Hydrogeology and Groundwater Quality	2013 to 2019
Fish and Aquatic Resources	2017 to 2021
Vegetation and Wildlife	2017 to 2021
Surficial Geology, Permafrost, Terrain, Stability and Soils	2012 to 2016
Geochemistry	2014-2019

20.3 Closure, Decommissioning and Reclamation

The Project has prepared a Reclamation and Closure Plan (RCP) as a requirement for a Type A Water Use License and a Quartz Mining Licence. This RCP was developed using an objectives-based approach that aligns with Reclamation and Closure Planning Guidelines for Quartz Mining (Yukon Water Board and Yukon Department of Energy, Mines and Resources, 2013) as well as the International Council on Mining & Metals (ICMM) Integrated Mine Closure Good Practice Guide (ICCM, 2019) and the Reclamation and Closure Planning for Quartz Mining Projects: Plan Requirements and Closure Costing Guidance (Yukon Department of Energy, Mines and Resources and Yukon Water Board, 2013).

20.3.1 Regulatory Requirements

A Reclamation and Closure Plan (RCP (Newmont Corporation, 2025)) was prepared for the then-defined Project as a requirement for a Type A Water Use License for Quartz Mining Projects. This RCP is cited as



being developed using an objectives-based approach that aligns with Reclamation and Closure Planning Guidelines for Quartz Mining (Yukon Water Board and Yukon Department of Energy, Mines and Resources, 2013) as well as the ICMM Integrated Mine Closure Good Practice Guide (ICCM, 2019) and the Reclamation and Closure Planning for Quartz Mining Projects: Plan Requirements and Closure Costing Guidance (Yukon Department of Energy, Mines and Resources and Yukon Water Board, 2013)).

Financial security is collected and held in the Yukon by the Minister of Energy, Mines and Resources, pursuant to the Quartz Mining Act, Waters Act, and associated regulations. The Coffee Gold Project holds financial assurance for existing exploration disturbance (refer to Section 4.2.3). As the project progresses through approvals and the site is subject to additional disturbance, additional financial assurance relevant to the approved RCP will be required. The financial assurance will be reviewed every two years (Government of Yukon, 2024).

20.3.2 Closure Scope and Activities

The RCP (Newmont Corporation, 2025) cites the development of a post-mine vision for the Project that includes the following key elements, which are supported by related closure principles and objectives.

- The site has returned to pre-mining conditions to the extent practicable, with enhancements for future generations.
- By planning for future generations, First Nation rights, titles, and interests within their Traditional Territory have been protected and respected.

To meet the vision, principles and objectives, the primary closure activities for each closure domain included in the RCP, and as amended to reflect the PEA mine design, are summarized below:

- **Open pits:** Kona and Double Double Pits will be backfilled during operations. In the absence of backfill design, it has been assumed 20% of the waste rock generated from the added open pits (Capuccino/Arabica and Supremo extension) is assumed to be backfilled in-pit. Rock closure berms will be constructed around open pits that are not backfilled to surface and surrounding disturbed areas will be reclaimed. Pre-existing closure costs have been scaled by 240% for the PEA to reflect the increased open pit footprint.
- **Alpha WRSF:** Resloping of the facility will be conducted during operations, to maintain an overall slope of 3:1, intended to achieve long-term stability criteria. Progressive capping will also be undertaken using 1m of low-arsenic material (at or below 60 mg/kg). While the RCP identifies the Alpha WRSF will be left to naturally revegetate, the PFS closure cost estimate includes soil placement and seeding and assumes adequate material, which have been retained given the uncertainty inherent in the natural revegetation strategy. Alpha WRSF closure costs were scaled up by 185% for the PEA to reflect the increased volume of waste material that could potentially be deposited in the WRSF as opposed to backfilled into open pits, noting the Alpha WRSF redesign was not available for the development of the PEA closure cost estimate.
- **HLF:** Following cessation of operations, the HLF will be rinsed, with contained water actively treated prior to release to the environment. An unchanged allowance has been included for the construction and maintenance of a passive water system post rinsing. The HLF will be regraded progressively as part of operations and post-rinsing, a store and release engineered cover, including a geosynthetic clay liner (GCL), will be progressively installed. The HLF ponds will be backfilled and remediated, unless they can be repurposed for the passive water treatment system, should it be required. The size of the HLF and associated closure costs (other than post operational rinsing) have been scaled by 140% to reflect the expanded PEA design, with rinsing duration extended proportionally to the previous RCP assumption of 200 days of rinsing per HLF section.



Active water treatment rinsing flow rates and operational unit costs were updated to those provided by Ensero Solutions (2026) for this PEA.

- **Infrastructure** (crusher, processing plant, camps, bulk explosives area, etc.): Infrastructure and equipment, except for what can be salvaged or recycled¹, will be dismantled and disposed of in the landfill (or otherwise as approved in the RCP). Concrete foundations will be demolished to just below ground level and concrete pads will be broken up to allow water infiltration but otherwise left in place. Demolition rubble will be disposed of in the landfill, if approved.
- Buried pipelines and electrics will be removed to 1m depth and underground fuel and reagent lines will be flushed and capped. Closure costs assume disposal of materials in the on-site landfill. Remaining disturbance will be regraded or scarified, as applicable, with growth media applied where required and areas seeded to promote revegetation.
- It is noted the existing exploration disturbance and infrastructure used to support exploration activities (for example, the Java and Coffee exploration camps) is permitted under Quartz Mining Land Use Permit # LQ00512 and Type B Water Use Licence MN16-034-2. As such, the requirements for reclamation of any disturbances and decommissioning of any buildings and infrastructure are addressed under these regulatory instruments and the liabilities for closure and reclamation of exploration-related infrastructure and disturbed areas are not incorporated as part of this PEA closure cost update.
- **Soil stockpiles, laydown yards, other areas of disturbance** – Scarified as necessary and seeded. These areas have been scaled to match the conceptual increase in disturbance footprint and operational life.
- **Roads** – Haul roads and internal roads will be scarified, regraded as per landscape designs, and seeded. For costing purposes, the haul road width was increased to 30m and the area scaled to reflect the increased open pit activity. Assumptions around the size and closure of the NAR were retained, including the assumed closure method of removing culverts and bridges to re-establish pre-construction surface drainage patterns, removing the barge landings, scarifying the road surfaces to promote water infiltration and vegetation re-establishment, contouring of the road surfaces and cut and fill slopes in accordance with landscape designs and revegetation of disturbed areas. No ice roads have been allowed during the rinsing or closure period.
- **Post-closure monitoring** – Allowances for monitoring aspects such as water quality and hydrology, hydrogeology, wildlife and vegetation, fish and fish habitat, benthic invertebrate/periphyton /sediment and physical stability monitoring of landforms is included. Helicopter access has been included following the reclamation of the NAR. The prior assumption of 9 years post-closure monitoring has been retained for the PEA.
- **General** – A number of uncertainties exist around cover designs, passive water treatment requirements and revegetation strategies and their long-term success. Refining the closure engineering designs and progressing execution readiness throughout the life of the asset has been provided for in the 5% Engineering, Design and Construction Plan allowance. Modest allowances have been included for future reclamation research, as committed to in the RCP, including on-site trials and monitoring. In addition, a closure study allowance has been provided to support the progression of good practice closure planning over the life of the asset, beyond minimum regulatory closure planning requirements. This allowance is intended to facilitate refinement of closure

¹ Salvage and recycle value has not been included in the closure cost estimate.



objectives, risks and opportunities, scope and post-closure land and asset transition prior to detailed engineering. It is also intended to support additional reclamation research beyond the minimal allowances noted above as well as stakeholder engagement and closure permitting activities throughout the life of the asset and as closure approaches.

20.3.3 Primary Closure Uncertainties

Primary uncertainties and pre-existing closure planning and cost assumptions that have been carried forward and expanded into the PEA design are listed below. As a result of these uncertainties, the closure cost estimate could materially change as the closure strategy is developed, uncertainties addressed and assumptions updated.

- **Operational material movement:** Backfilling of the nominated open pits is assumed to occur during operations and therefore does not incur a closure cost. Progressive regrading of the HLF and Alpha WRSF, placement of the 1m low-arsenic surface cover on the open pit backfill and Alpha WRSF, and the progressive placement of ice-rich material contained within the frozen soil stockpile area onto the Alpha WRSF are also assumed to occur during operations. Operational backfilling is also assumed to prioritize covering pit wall exposures that could contribute elevated arsenic concentrations to surface water runoff. Pit spillways are assumed to be constructed as part of operations for those pits predicted to overflow post-closure.
- **Long term facility design and operations to design:** It has been assumed all facilities (HLF, open pits, Alpha WRSF) and their associated water management are being designed as permanent landforms, in consideration of long-term climate change predictions without the need for retrofit in the closure phase.
- **Cover systems for the Alpha WRSF and HLF:** No engineered cover has been allowed for the Alpha WRSF, while a GCL cover has been included for the HLF. Further cover system engineering and operational field trials will be required to test and demonstrate the ability for the design to meet performance objectives. Similarly, once cover designs and reclamation research is completed, a reclamation materials inventory will be required to enable identification of available resources and their location, plus any treatment (e.g., crushing) necessary to make them suitable for their purpose.
- **Active water treatment volumes and duration:** No active water treatment is assumed to be required from the open pit passive discharge. These assumptions have not been revised for the PEA design. Active water treatment has only been allowed for the duration of the HLF rinsing and does not currently include any other contact water treatment. The requirement for increased volumes and / or ongoing active water treatment requires further facility design and subsequent water quality modelling, as well as input of the results of cover trials.
- **Passive water treatment type, duration and maintenance requirements:** The allowance to implement and maintain, if necessary, passive treatment using permeable reactive barriers technology for “polishing” of heap solutions within event ponds has been retained. There remains significant uncertainty regarding the need for this system, as well as its design, implementation and long-term maintenance requirements. Engineering design of the passive water treatment system is considered to be included within the 5% Engineering, Design and Construction Plan allowance and provision has also been made for an on-site pilot of the design.
- **Open pit closure safety requirements:** Specific safety requirements for open pit closure will be determined once the pits have been constructed. Open pit closure measures will fulfill the requirements set out in the Workers’ Safety and Compensation Act and Regulations. No allowance for addressing post-closure settlement of open pit backfill has been provided.



- **Demolition costs:** Due to the remoteness of the site and the associated transport logistics, demolition costs calculated in the Standardized Reclamation Cost Estimator (SRCE) closure cost calculator may not fully represent the logistics required for a remote site to meet the RCP commitments of salvaging and recycling materials where possible. Similarly, assumptions around retention of underground piping and services may not be approved.
- **Waste management:** The proportion of demolition waste suitable for off-site recycling versus on-site disposal remains uncertain and will be dependent on demolition planning, materials characterization, market availability for recyclable materials, transportation logistics, and regulatory requirements at the time of closure.
- **Closure phase logistics:** There is insufficient detail at this stage regarding closure phase logistics such as the requirement for the ice road during the rinsing or active closure phases, vehicle and staff requirements, closure vs operational equipment mobilisation and demobilisation.
- **Closure schedule:** Given the closure logistics, the schedule for closure could require adjustment. Any adjustment to schedule may impact the duration of costs such as maintenance of water management structures and roads as well as General and Administration costs. Additionally, the duration of post-closure monitoring and maintenance requirements and active water treatment are currently not well defined and are unlikely to be until closer to the time of closure.
- **Workforce costs:** No workforce termination or relocation costs are included in this cost estimate, nor are retention bonuses to retain site knowledge as the operation transitions to closure.
- **Socio-economic transition contributions:** No socio-economic transition costs are included in the closure plan or associated cost estimate. The RCP notes the Stakeholder Engagement Management Plan has been developed to align with the interests of affected First Nations and will be updated as the project planning progresses. Commitments made during consultation and within subsequent benefit agreements with affected First Nations will be used to inform social closure planning in future iterations of the RCP, which will provide for associated closure cost inclusions.
- **Reclamation requirements:** The RCP identifies a number of areas for future reclamation research which will further define reclamation requirements, as will the future agreement of specific end land use requirements. These research results and land use agreements could materially impact closure activity requirements, hence closure liability.
- **Financial assurance payments:** As identified in Section 20.4.1, financial assurance will be required by the Government of Yukon to cover reclamation of the site in the event of an operator default. At this stage, there is insufficient detailed planning to reasonably estimate the amount of that financial assurance and form of the assurance, such as cash or surety bonds, hence any associated costs that would be incurred. As a result, the closure cost update in Section 20.4.5 does not include any related expenditure.

20.3.4 Closure Schedule

The closure schedule for the PEA is provided in Table 20.2 and reflects the closure phases identified in the RCP, amended for the PEA mine design and closure assumptions.



**Table 20.2
PEA Closure Schedule**

Mine Life Phase	Primary Activities	Years	Duration (years)
Operational phase	Mining, processing, progressive closure	1 to 13	13 years
Closure phase: Post-mining Closure Stage	HLF rinsing and active water treatment	14 to 21	8 years
Closure phase: Active Closure Stage	Demolition and reclamation	22 to 26	5 years
Post-closure Phase	Monitoring, maintenance and reporting, passive water treatment as required	27 to 50 Monitoring and maintenance nominally provided for 10 years: 27 to 37	24 years (10 years post-closure monitoring)

Closure costs have been scheduled for a total period of 50 years, noting this represents a nominal time period as originally scheduled in the provided cost estimate, rather than achievement of an end to potential closure liability.

20.3.5 PEA Level Closure Cost Update

The closure cost update for the PEA used the Coffee PFS SRCE 2024 v7 (JDS, 2024) as its foundational basis, which was calculated using the SRCE 2.0 Model, Beta Version 01p. Key assumptions from the 2024 PFS closure cost model were carried forward into the PEA estimate, unless adjusted as part of the following primary adjustments.

- Scaled costs for the increased PEA design of primary landforms and disturbance areas such as open pits, Alpha WRSF, four HLF ponds, soil stockpiles, and the HLF. In-pit backfilling volumes and locations were not available for the updated PEA cost estimate, hence assumptions were applied based on the previous backfill assumptions.
- As a result of the increased HLF, post-operational rinsing was calculated to require an additional two years.
- HLF rinsing active water treatment flow rate and unit costs were updated to those provided by Ensero Solutions (2026).
- Amended dimensions of haul roads from 10 m widths to 30 m widths to match that cited in the RCP and usual for mining operations.
- Amended rates to third party contractors using the Canada BC Road Builder and Heavy Construction Association - 2025-2026 Equipment Rental Rate Guide, with other rates escalated to 2026 dollars. This may be contrary to the RCP which states operational fleet will be used.
- Additional closure planning costs were added as identified in Section 20.4.2 for reclamation research as identified in the RCP, including site trials, closure planning and reporting throughout the life of the asset, and closure studies and projects, beyond engineering and execution readiness, required to define an achievable and executable closure for the site.
- Schedule amendments:



- As a result of the increased heap leach facility, post-operational rinsing was calculated to require an additional 2 years.

Given the level of uncertainty associated with the closure methods, and as a result, the PEA level closure cost update (refer to Section 20.4.3), a contingency has not been applied.

The resultant PEA level closure cost estimate is provided in Table 20.3. Closure cashflow has been scheduled based on the assumed timing of closure activities over a 37-year period beyond the 13-year production plan to equate to a total of 50 years of closure costs – a timing assumption carried forward from the PFS (refer to Section 22.4.8).

Table 20.3
PEA Level Closure Cost Estimate Update

Closure Domain	Cost (CA\$ million)
Waste WRSF (Alpha WRSF)	\$ 1.6
HLF	\$ 3.0
Solution Management (including HLF rinsing, passive and active water treatment)	\$ 77.3
Open Pits	\$ 0.3
Roads (including constructed NAR sections and barge crossings)	\$ 20.7
Process Ponds	\$ 2.3
Miscellaneous disturbed areas (yards, infrastructure footprints, landfill, etc)	\$ 1.8
Demolition and Equipment Removal	\$ 11.1
Sediment & Drainage Control	\$ 0.5
Monitoring – including post-closure monitoring	\$ 10.6
Closure Planning and permitting (including reclamation research, field trials and operational closure plan and cost estimate updates)	\$ 5.8
General and Administrative (includes Construction Management, camp operating costs, barge and access road maintenance during the closure period, management and administration, sites services and the like and the SRCE mobilisation and demobilisation classified as “miscellaneous costs”)	\$ 25.1
Directs Subtotal	\$ 160.0
Engineering, Design and Construction Plan (5%)	\$8.0
Contractor Overhead and Profit (5%)	\$8.0
Contract Administration (1%)	\$1.6
Contingency	Not included
TOTAL	\$ 177.6

20.4 Environmental Assessment and Environmental Permitting

20.4.1 Environmental Assessment Process

The Project was subject to an assessment under the *Yukon Environmental and Socio-Economic Assessment Act*, administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB), which was completed in March 2022. The assessment documents the potential environmental and socio-economic effects of the Project by evaluating baseline information, the proposed mine plan, required mitigations and commitments, and assessment of project effects. The Decision Document for the



Coffee Gold Mine Project was issued on behalf of the Decision Bodies (Natural Resources Canada, Transport Canada, and Government of Yukon) that accept the recommendation that the Project be allowed to proceed subject to terms and conditions specified in the Referral Report (YESAB, 2022). The YESAB file number is 2017-0211. In addition, the Project has received approval for the renewal and amendment of an active Class 4 quartz exploration permit (MLU LQ00552) (YESAB file number 2020-0074). The Class 4 approval was approved with an effective approval date of May 28, 2021, with all work completed by May 27, 2031.

Most terms and conditions which the Project (under YESAB file number 2017-0211) is subject to are environment, socio-economic, water quality and monitoring related. Those related to design are as follows:

- Term and Condition 12 – Material shall not remain on the ROM stockpile at the end of operations, during scheduled or unscheduled closure, without prior confirmation of long-term attenuation capacity of the schist pad for the key COPs, arsenic and uranium that has been reviewed and accepted by the relevant regulator.
- Term and Condition 17 – The proponent must define acceptable performance conditions for the Alpha Pond in terms of water quantity and quality for seepage from the Pond as part of the adaptive management for the Alpha Pond.
- Term and Condition 18 – The proponent shall provide contingency plans contemplating scenarios where the management of surface effluent discharged from Alpha Pond fails to meet water quality objectives in Halfway Creek.
- Term and Condition 24 – The proponent shall only landfill inert substances at the project site. The Proponent shall work with affected First Nations and regulatory agencies to develop an Approved Landfill Materials list and mechanism for establishing suitably low leachability risk for proposed landfill materials.

20.4.2 Territorial Approvals

The primary remaining approvals required for mine construction and operation are a Water Use Licence and Quartz Mining Licence. The Project has submitted Type A Water Use Licence application for a Quartz Mining Undertaking and an application for a Quartz Mining Licence in November 2023. These applications are currently under review by the respective regulators. Other authorizations, including those required for all-season access road construction, are also being reviewed by regulators. The following additional territorial regulations and approvals that may be applicable to the Project are provided in Table 20.4.

Table 20.4
Regulatory Authorizations and Approvals Applicable to the Coffee Gold Mine Project - Territorial

Act and Regulation	Regulation	Regulatory Approval	Project Activities	Responsible Agency
<i>Boiler and Pressure Vessels Act</i>		Registration	Installation of power boilers over 10 kW, heating boilers over 20 kW, pressure vessels and piping systems	Community Services, Building Safety
<i>Building Standards Act</i>		Building Permit	Construction of buildings outside of a municipality	Community Services, Building Safety
		Plumbing Permit	Installation of plumbing outside of Whitehorse	



Act and Regulation	Regulation	Regulatory Approval	Project Activities	Responsible Agency
<i>Dangerous Goods Transportation Act</i> Dangerous Goods Regulations	Dangerous Goods Regulations	Certificate and/or permit for transport of dangerous goods	Transportation of dangerous goods within Yukon on highways	Highways and Public Works, Transport Services Branch
<i>Electrical Protection Act</i>	Canadian Electrical Code (23 rd edition)	Electrical Permit	Electrical work	Community Services, Building Safety
<i>Environmental Act</i>	Air Emission Regulations	Air Emissions Permit	<ul style="list-style-type: none"> • Operation of fuel burning equipment greater than 5 million British thermal units per hour (Mbtu/hr) • Release of air pollutants • Operation of solid waste incinerator • Excavation of a quarry that is greater than 4 ha 	Environment, Environmental Programs
	Contaminated Sites Regulation	Land Treatment Facility Permit	Operation of a land treatment facility for contaminated soils	
		Relocation Permit	Relocation of contaminated material	
	Ozone Depleting Substances and Other Halocarbon Regulation	Ozone Depleting Substances and Other Halocarbons Permit	<ul style="list-style-type: none"> • Servicing or installation of equipment containing ozone depleting substances • Purchasing, handling, and services in/of ozone depleting substances equipment • Use of ozone depleting substances and equipment 	
	Solid Waste Regulations	Waste Management Permit	<ul style="list-style-type: none"> • Solid waste disposal facility • Operation of solid waste incinerator 	
	Special Waste Regulations	Special Waste Permit Waste Manifest	<ul style="list-style-type: none"> • Handling, disposal, generation, or 	



Act and Regulation	Regulation	Regulatory Approval	Project Activities	Responsible Agency
			storage of special (hazardous) waste <ul style="list-style-type: none"> • Transportation of dangerous goods/waste 	
	Storage Tank Regulations	Application for Operation, Closure, Abandonment, or Renovations to Storage Tanks Above-Ground Storage Tank Permit	<ul style="list-style-type: none"> • Storage and handling of petroleum products • Use of storage tanks containing petroleum and allied petroleum products 	Community Services, Fire Marshal's Office
<i>Forest Protection Act</i>	Forest Protection Regulation	Burning Permit	Burning of refuse (wood)	Community Services, Protective Services
<i>Forest Resources Act</i>	Forest Resources Regulation	Forest Resources Permit	<ul style="list-style-type: none"> • Clearing of forest resources incidental to other activity 	Energy, Mines and Resources, Forest Management Branch
<i>Gas Burning Devices Act</i>	Gas Regulations	Gas Installation Permit	Gas installation or modification	Community Services, Building Safety
<i>Highways Act</i>	Bulk Commodity Haul Regulations	Bulk Commodity Agreement	Bulk commodity hauling	Highways and Public Works, Transport Services Branch
	Highways Regulations	Over-dimensional or Overweight Vehicle Permits	Oversize trucking	
		Permit under Highways Act	Construction of new road access	Highways and Public Works, Transportation Maintenance Branch
	Access Permit	Construction of road access on a highway right-of-way		
<i>Historic Resources Act</i>	Archaeological Sites Regulation	Archaeological Site Regulation Permit	Search for and research at archaeological and paleontological sites	Tourism and Culture, Heritage Resources
<i>Lands Act</i>	Lands Regulations	Land Use Permit	Use of Yukon lands, as defined in the <i>Act</i> for the following: <ul style="list-style-type: none"> • Construction of a solid waste disposal facility • Storage and handling of 	Energy, Mines and Resources, Land Branch



Act and Regulation	Regulation	Regulatory Approval	Project Activities	Responsible Agency
			petroleum products <ul style="list-style-type: none"> • Conduct of geotechnical studies • Construction of new road access • Construction of a new bridge crossing 	
	Quarry Regulations	Quarry Permit, Quarry Lease	Removal of gravel/sand from a quarry on Yukon lands	
<i>Land Titles Act</i>	Land Titles Plans Regulation Land Titles Office Regulation	Registration of interest in land	Title to land	Justice, Land Titles
<i>Yukon Workers' Safety and Compensation Act</i>	Workplace Health and Safety Regulations	Notification to Director OHS	Trenching excavation in excess of 6 m	Yukon Workers' Safety and Compensation Board
		Underground and Surface Authorization to Conduct Blasting in Yukon	Blasting – underground or surface	
		Temporary Blaster's Permit	Temporary blasting	
		Explosives Magazine Storage and Use Permit for Mining	Explosives storage	Yukon Workers' Safety and Compensation Board
<i>Public Health and Safety Act</i>	Regulations Respecting the Sanitation of the Camps in the Yukon Territory	During operation of the camp, provision of information to the Health Officer in respect of the Regulations, as required.	Maintenance of a camp in a sanitary condition	Health and Social Services, Environmental Health Services
	Drinking Water Regulation	<ul style="list-style-type: none"> • Approval to construct a large public drinking water system • Permit to operate a large public drinking water system 	Construction, installation, modification, and operation of a drinking water system	



Act and Regulation	Regulation	Regulatory Approval	Project Activities	Responsible Agency
	Sewage Disposal Systems Regulation (applicable to camps less than 50 people)	Permit to install a sewage disposal system	Installation of an on-site sewage disposal system	
	Eating and Drinking Places Regulation	Permit to Operate a Food Premise	Operation of a food premises	
<i>Quartz Mining Act</i>	Quartz Mining Land Use Regulation	Quartz Mining License	Quartz mining operation	Energy, Mines and Resources, Mineral Resources Branch
<i>Territorial Lands (Yukon) Act</i>	Land Use Regulation	Land Use Permit	Use of territorial land, as defined in the <i>Act</i> for the following: <ul style="list-style-type: none"> • Construction of a solid waste disposal facility • Storage and handling or petroleum products • Establishment and use of fuel caches of more than 4,000 L of any single container of more than 2,000 L on Commissioner's Land • Use of more than 50 kg of explosives on Commissioner's Land in any 30-day period • Temporary use of occupation of Commissioner's Land • Conduct of geotechnical studies • Construction of new road access • Construction of a new bridge crossing 	Energy, Mines and Resources, Lands Branch
	Territorial Lands Regulation	Application for land lease or purchase	Tenure for land lease or agreement of sale	



Act and Regulation	Regulation	Regulatory Approval	Project Activities	Responsible Agency
	Resource Roads Regulation	Primary Permit/Access Permit	Access road	Energy, Mines and Resources
<i>Waters Act</i>	Waters Regulation	Water Licence Use	<ul style="list-style-type: none"> • Direct use of water • Deposit of waste • Construction of watercourse crossing • Watercourse training 	Yukon Water Board
<i>Yukon Environmental and Socio-economic Assessment Act</i>	Assessable Activities, Exceptions and Executive Committee Projects Regulations	Decision Document	<ul style="list-style-type: none"> • All Project activities 	Decision Bodies: Government of Yukon, Natural Resources Canada, Transport Canada

20.4.3 Federal Environmental Approvals

The following federal regulatory authorizations and approvals are applicable to the Coffee Gold Project are outlined in Table 20.5.



**Table 20.5
Regulatory Authorizations and Approvals Applicable to the Coffee Gold Mine Project – Federal**

Act	Regulation	Regulatory Approval	Project Activities	Responsible Agency
Explosives Act	Explosives Regulations	ANFO Permit	Manufacture of ANFO	Natural Resources Canada, Minerals and Metals Sector
Fisheries Act		Section 35(4) Authorization	To carry a proposed work, undertaking, or activity likely to result in the death of fish or the harmful alteration, disruption or destruction of fish habitat.	Fisheries and Oceans Canada
	Metal and Diamond Mining Effluent Regulations	Authorization to deposit an effluent that contains a deleterious substance	Applies to mines that exceed an effluent flow rate of 50 m ³ per day, based on effluent deposited from all the final discharge points of the mine, and that deposit a deleterious substance in any water frequented by fish or in any place under any conditions where the deleterious substance or any other deleterious substance that results from the deposit of the deleterious substance may enter any such water	

20.5 Consultation and Engagement

Consultation and engagement have been ongoing since 2012 with the communities and First Nations affected by the Project. Substantial consultation was required in advance of and during the environmental and socio-economic process (YESAB file number: 2017-0211) and in advance of filing licence applications for the Coffee Gold Project. Engagement will continue throughout the LOM to inform project decision making. Any permitting required to include mine plan adjustments resulting from this PEA will require additional engagement and consultation steps. Fuerte is committed to ongoing and meaningful engagement with affected First Nations and communities in accordance with regulatory requirements, company policies and industry best practices.

20.6 Hydrogeology

Multiple hydrogeological, geotechnical and geocryological drilling programs have been undertaken at the Coffee Gold Project between 2013 and 2017, informing a robust baseline hydrogeological characterization of the site. The well-developed groundwater monitoring network includes seven Westbay systems, 11 conventional monitoring wells, 12 thermistor/vibrating wire combination installations, plus numerous standalone thermistor strings and vibrating wire piezometers. The majority of the drilling programs have included a component of bedrock hydraulic testing. Groundwater monitoring campaigns initiated in 2014 have collected several years of nearly continuous water level data with groundwater quality data collected



from over 20 monitoring events. The data are most recently summarized in a 2019 hydrogeology baseline report (Lorax, 2021) submitted as a part of the 2023 Type A Quartz Mining Water Use Licence Application (Kaminak Goldcorp 2023).

The Coffee Gold Project is located in an area classified as extensive discontinuous permafrost. The site data indicate permafrost thickness ranges from largely absent on south facing slopes to over 160 m thick on north facing slopes, with permafrost thickness decreasing with elevation. Groundwater levels in ridge areas, coincident with mine infrastructure, range from 220 m below ground surface to occasional artesian conditions where faulting occurs. Groundwater level hydrographs are observed to fluctuate by up to 30 m at some locations. Hydraulic conductivity data range over several orders of magnitude and is typical of a fractured bedrock system. Mineralized structures and creek traces represent areas of enhanced hydraulic conductivity.

A 3D numerical groundwater model was developed for the Coffee Gold Project in support of the 2017 Project Proposal submission (Lorax (2017) in Goldcorp (2017)) for an assessment under the Yukon Environmental and Socio-Economic Assessment Act. The groundwater model was calibrated to creek baseflows and over 40 water level targets. The model incorporated permafrost occurrence based on field mapping and vertical extents determined from thermistors and other observations. Pits positioned in unfrozen ground south of the ridge top were found to intersect the water table, while those excavated on north-facing slopes were found to be perched in permafrost. During operations, pit lakes were expected to form through dominantly meteoric inputs. Ultimately, groundwater fluxes into the pits, and seepage losses from the pit lakes, were found to be relatively small - less than 2 L/s per facility advanced on south facing slopes.

The PEA mine plan proposes to expand and deepen pits previously simulated by the Project Proposal groundwater model and develop new pits that may interact with the groundwater system. It is recommended that the PEA mine plan be incorporated into the groundwater model to account for heightened interaction between the mine and the groundwater system. Additional hydrogeological data collection and instrumentation is recommended in new pits where groundwater interactions are expected, and this information used to validate and/or update the groundwater model in due course.



21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

The project CAPEX, SUSEX, and OPEX were compiled by WSP; and are based on the Project and ongoing scope of work presented in this Report. Unless indicated otherwise, all CAPEX and OPEX costs are expressed in Canadian Dollars (CA\$ or \$) and are based on Q1 2026 pricing.

WSP developed the CAPEX, SUSEX, and OPEX for mining based on the mining physicals and equipment fleet provided by Open Contour.

WSP developed the CAPEX and OPEX for the On-site development, Crushing and Handling, Process Plant, On-site infrastructure, and Off-Site Infrastructure. WSP provided the closure costs.

SRK developed the CAPEX for Water Management with OPEX tabulated by WSP based on SRK's pump requirements.

Forte Dynamics provided the CAPEX for the Heap Leach Facility while all associated OPEX was included in the overall processing OPEX tabulated by WSP.

During compilation of costs, WSP applied necessary indirect costs and contingencies as appropriate.

All contributors to the CAPEX, SUSEX, and OPEX are outlined in Table 21.1.

Table 21.1
List of Cost Estimation by Contributor Firm

Description	
Open Pit Mining	WSP, Open Contour
On-site development (airstrip)	WSP
Crushing and Handling	WSP
Heap Leach	Forte Dynamics
Process Plant	WSP
On-Site Infrastructure	WSP
Off-Site Infrastructure	WSP
Water Management	SRK, WSP
Water Treatment Plant	Ensero
Indirect	All
Contingency	All
Reclamation and Closure	WSP

The CAPEX was developed to deliver an overall accuracy range of -30% to +50%. Ranges could exceed those shown, if there are unusual risks.



For this PEA, the CAPEX reflects an engineering procurement construction management (EPCM) type execution model. Although some individual elements of the CAPEX may not achieve the target level of accuracy, the overall CAPEX fall within the parameters of the intended accuracy.

21.2 Capital Cost Estimate

21.2.1 Capital Cost Summary

The CAPEX consists of direct and indirect capital costs as well as contingency. Provisions for SUSEX consider mining, water management, and electrics related to the mining. Amounts for mine closure and rehabilitation of the site are shown in Section 20.4.5 of the Report.

Table 21.2 presents the CAPEX, SUSEX, and total capital. Shown separately in the Table are Owner's costs and EPCM, that are included as indirects, and the overall contingency cost.

The project capital cost totals \$998.2 million and the sustaining capital cost totals \$692.7 million. The capital expenditure by area forecast for the entire mine life is provided in Table 21.2.

Table 21.2
Capital Cost Estimate by Area

Description	CAPEX (CA\$ million)	SUSEX (CA\$ million)	Total Capital (CA\$ million)
Open Pit Mining	85.3	252.3	337.6
On-site development	15.9	24.1	40.0
Crushing and Handling	65.2	7.6	72.7
Heap Leach	87.1	159.7	246.8
Process Plant	76.7		76.7
On-Site Infrastructure	174.8		174.8
Off-Site Infrastructure	71.3		71.3
Water Management	31.9	25.4	57.3
Water Treatment Plant	26.2	26.2	52.3
Indirects	188.0	52.0	240.0
Contingency	175.9	145.4	321.3
Total Major Area Capital*	998.2	692.7	1,690.9

*Numbers may not sum precisely due to rounding.

The exchange rate used for items quoted in US\$ is shown in the Table 21.3.



**Table 21.3
Currency Exchange**

Foreign Exchange	CA\$
CA\$	1.39
US\$	1.00

21.3 Project Capital

The following sections outline the project capital costs required by each area, including the breakdown between direct and in-direct costs.

21.3.1 Open Pit Mining

Open Contour developed the LOM plan and established the mining equipment and operator requirements over time (see Section 16). WSP determined the mine CAPEX based on the equipment requirements inclusive of an operational readiness training determination for the operators of the equipment.

An internal costing database tabulated for a recent (December 2025), similar sized Canadian gold project and Costmine's Cost Estimation Service were used for equipment costs.

The WSP operator training cost determination is based on the number of operators identified for first year open pit equipment operation based on that year's overall mine excavation requirements. The annual cost of each operator is used to establish a monthly cost for the training that is then applied to the operator count to establish the training cost.

Table 21.4 shows the mine CAPEX that includes the capitalized mine equipment purchases based on the equipment requirements for the three years of pre-production. The equipment costs shown reflect a procurement plan where 70% of each item's cost is applied in the year the machine is ordered and 30% of the item's cost is applied upon delivery.

For mine indirects, the G&A group in Year -1 includes an owner team to procure the mine equipment. The EPCM contractor is not foreseen involved in the mine capital. The equipment costs include freight and logistics.



**Table 21.4
Pre-Production Mining CAPEX Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Primary Mining Equipment	62.4	-	15.6	78.1
Auxiliary Mining Equipment	22.6	-	5.6	28.2
Miscellaneous Equipment	0.3	-	0.1	0.4
Transfer from OPEX**	0.0	1.1	0.3	1.4
Total OP Mining	85.3	1.1	21.6	108.1

*Contingency of 25% applied to total of direct and indirect costs

**Transfer from OPEX represents operator training aligned with equipment requirements.

**Numbers may not sum precisely due to rounding.

21.3.2 On-site Development

On-site development shown in Table 21.5 represents the cost allowance for an airstrip, and infrastructure site earthworks.

**Table 21.5
CAPEX – On-Site Development Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Airstrip	1.8	0.6	0.2	2.7
Infrastructure Site Earthworks	14.1	4.8	4.7	23.7
Total On-Site Development**	15.9	5.4	5.0	26.3

* Contingency applied by WSP varies between 10% to 25% of total direct and indirect costs dependant on item.

**Numbers may not sum precisely due to rounding.

21.3.3 Crushing and Handling

Table 21.6 depicts the Crushing and Handling facility CAPEX requirements. The estimated cost is based on the Internal Engineering Report (JDS, 2024) cost estimate with escalation to the current date and includes expenses for primary and secondary crushing, adding lime, transporting and storing mineralized material, as well as overland conveying and stockpiling. Indirect costs are estimated at 34% of direct costs based on industry standard benchmarking. Included are allowances for EPCM, Owner's Costs, Spares, Vendor Assistance, Freight, First Fills, Operational Readiness, and Construction Allowances.



Table 21.6: CAPEX –Crushing and Handling Detail

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Primary & Secondary Crushing Station	50.8	17.3	16.3	84.4
Fine Material Conveying and Stockpiling	2.8	0.9	0.9	4.6
Overland Conveyor	10.7	3.6	3.6	17.8
ROM Stockpile	1.0	0.3	0.3	1.6
Total Crushing and Handling**	65.2	22.2	21.1	108.5

*Contingency applied by WSP varies between 10% to 25% of total direct and indirect costs dependant on item.

**Numbers may not sum precisely due to rounding.

21.3.4 Heap Leach

Table 21.7 depicts the Stage 1 Project Capital Costs for the HLF CAPEX requirements. Capital cost estimates for the heap leach pad, solution distribution, collection system, and the ponds are proportionally adjusted from previous project-specific experience and industry standard benchmarking.

**Table 21.7
Stage 1 CAPEX – Heap Leach Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Leach Pads	18.4	0.0	4.6	23.0
Solution Distribution & Collection System	26.7	0.0	6.7	33.3
Ponds	39.0	0.0	9.7	48.7
On-Heap Conveying	3.1	0.0	0.8	3.9
EPCM	0.0	13.8	3.4	17.2
Total Heap Leach**	87.1	13.8	25.2	126.1

*Contingency of 25% applied to total of direct and indirect costs

**Numbers may not sum precisely due to rounding.

21.3.5 Process Plant

Table 21.8 depicts the process plant CAPEX requirements. The estimated CAPEX covers expenditures for areas such as process plant building, carbon adsorption (CIC circuits), acid wash and elution, carbon regeneration, electrowinning and gold room, as well as processing utilities.

Indirect costs are estimated at 34% of direct costs based on industry standard benchmarking. Included are allowances for EPCM, Owner's costs, spares, vendor assistance, freight, first fills, operational readiness, and construction allowances.



**Table 21.8
CAPEX – Process Plant Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Process Plant Building	21.5	7.3	7.2	36.0
Carbon Adsorption	3.2	1.1	1.0	5.3
Acid Wash & Elution	20.5	7.0	6.6	34.2
Carbon Regeneration	0.1	0.0	0.0	0.2
Electrowinning & Refining	0.2	0.1	0.1	0.3
Reagent Systems	2.0	0.7	0.7	3.3
Process Utilities	29.1	9.9	9.7	48.8
Total Process Plant**	76.7	26.1	25.4	128.1

*Contingency of 25% applied to total of direct and indirect costs

**Numbers may not sum precisely due to rounding.

21.3.6 On-Site Infrastructure

Table 21.9 depicts the estimated On-Site Infrastructure CAPEX requirements. Indirect costs were estimated at 34% of direct costs based on industry standard benchmarking before inclusion of the Transfer of OPEX costs. These indirect costs for On-Site Infrastructure then account for project wide costs related to Community Relations, Off-site Offices and IT Expenses, Environment, Engineering, and HSSE Costs. Included in the 34% indirects are allowances for EPCM, Owner's Costs, Spares, Vendor Assistance, Freight, First Fills, Operational Readiness, and Construction Allowances. Contingency is estimated at 10% for budgeted or quoted items such as the Java Camp and 25% for remaining items.



**Table 21.9
CAPEX – On-Site Infrastructure Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Electrical Supply & Distribution	44.2	15.0	13.6	72.9
Water Supply & Distribution	2.7	0.9	0.9	4.5
Assay / Metallurgical Laboratory	3.7	1.3	1.1	6.1
Java Camp (construction)	20.0	6.8	2.7	29.5
Plant Site Accommodations	10.7	3.6	3.6	18.0
Ancillary Facilities	53.7	18.2	17.3	89.3
Bulk Fuel Storage & Distribution	9.1	3.1	3.1	15.3
IT & Communications	1.7	0.6	0.6	2.8
Site Mobile Fleet	29.0	9.9	9.7	48.5
Transfer from Opex**	0.0	17.9	4.5	22.3
Total On-Site Infrastructure***	174.8	77.3	57.1	309.2

*Contingency applied varies between 10% to 25% of total direct and indirect costs dependant on item.

** Transfer from OPEX represents Community Relations, Off-site Offices and IT Expenses, Environment, Engineering, and HSSE Costs

***Numbers may not sum precisely due to rounding.

21.3.7 Off-Site Infrastructure

Table 21.10 depicts the estimated off-site infrastructure CAPEX requirements. Indirect costs are estimated at 34% of direct costs based on industry standard benchmarking. Included are allowances for EPCM, Owner's Costs, spares, vendor assistance, freight, first fills, operational readiness, and construction allowances. Contingency is estimated at 10% representing the site access road, which is a budgeted item based on quotes received.

**Table 21.10
CAPEX – Off-Site Infrastructure Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Site Access Road	68.0	23.1	9.1	100.2
Aggregate Quarrying	3.3	1.1	0.4	4.8
Total Off-Site Infrastructure	71.3	24.2	9.5	105.0

*Contingency of 10% applied to total of direct and indirect costs as site access road based on quotes.

21.3.8 Water Management

Water management initial capital costs shown in Table 21.11 were developed based on quantities modelled for each component. In some instances, quantities were scaled based on changes in length or catchment



area relative to previous designs (completed in 2022 and 2024). Unit costs were developed using estimates previously developed for the project, adjusted for inflation. Pit dewatering costs assume the use of Godwin HL 225M pumps, with 12-inch diameter (approx. 323.8 mm / DN300) high density polyethylene pipe (DR17 or similar), with a pump assigned for every 1 km² of open mining area and every 100 m of static head from the pit floor to the Facility Pond, during the most critical mining year (identified as Year 2). All costs were inflated to Q1 2026 using the Industrial Product Price Index (IPPI) based on Primary Metal Manufacturing costs. A 30% factor was used to calculate indirect costs (from the direct costs) based on experience related to similar projects.

**Table 21.11
Water Management CAPEX Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Pit Dewatering	3.6	1.1	1.2	5.8
Halfway Diversion (WRSF Opt 1)	3.6	1.1	1.2	5.8
WRSF Drainage (Rock Drain)	4.3	1.3	1.4	7.0
Facility Pond – Northwest	2.5	0.8	0.8	4.1
Facility Pond – South	2.9	0.9	1.0	4.8
Alpha Pond	15.0	4.5	4.9	24.3
Total Water Management**	31.9	9.6	10.4	51.8

*Contingency of 25% applied to total of direct and indirect costs

**Numbers may not sum precisely due to rounding.

21.3.9 Water Treatment

Table 21.12 depicts the estimated initial CAPEX requirements for water treatment for a design flowrate of 35 L/s. Direct costs were estimated using a combination of current and historical budgetary quotations, parametric estimating, and analogous estimating techniques. Where applicable, an internal costing database was utilized, including historical pricing from similar projects. Indirect costs were estimated to be 32% of total direct costs and include EPCM. Where historical pricing was used, costs were escalated to 2025 dollars. No contingency is included for the initial CAPEX but a contingency of 25% is applied as Sustaining Capital (Section 21.4.9).

**Table 21.12
Water Treatment Plant CAPEX Detail**

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Water Treatment Plant	26.2	8.4	0	34.5
Total Water Treatment Plant **	26.2	8.4	0	34.5

*Contingency of 25% applied as Sustaining Capital (final year of construction occurs during Sustaining period)

**Numbers may not sum precisely due to rounding.



21.4 Sustaining Capital

The SUSEX begins in Year 4 in alignment with the timing of recovery of doré from the heap leach process which commences in the second half of that year. The following sections outline the SUSEX required by each area, including the breakdown between direct and indirect costs. The SUSEX by area is depicted in Table 21.2.

21.4.1 Open Pit Mining

The Mine SUSEX is based on the LOM plan applying unit cost to the physical inputs. An internal costing database tabulated for a recent (December 2025), similar sized Canadian gold project and Costmine's Cost Estimation Service were used for equipment costs.

Table 21.13 presents the mine SUSEX required in the operating years for open pit mining, split by type of equipment and reflect a procurement plan where 70% of each item's cost is applied in the year the machine is ordered and 30% of the item's cost is applied upon delivery.

For mining indirects the mine management and mines technical services will work with the procurement group to order the requisite equipment. Freight and logistics are included in the equipment costs shown.

Table 21.13
Mining Sustaining Capital Detail

Description	Direct SUSEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Primary Mining Equipment	177.1	0.0	44.3	221.4
Auxiliary Mining Equipment	70.1	0.0	17.5	87.6
Miscellaneous Equipment	5.0	0.0	1.3	6.3
Total OP Mining**	252.3	0.0	63.1	315.4

*Contingency of 25% applied to total of direct and indirect costs

**Numbers may not sum precisely due to rounding.

21.4.2 On-site Development

Table 21.14 depicts on-site development SUSEX requirements. A cost allowance has been considered for cover additional on-site development such as road expansions and extensions.

Table 21.14
On-site Development Sustaining Capital Detail

Description	Direct SUSEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Roads and Other Development	24.1	8.2	8.1	40.4
Total On-Site Development	24.1	8.2	8.1	40.4

*Contingency of 25% applied to total of direct and indirect costs



21.4.3 Crushing and Handling

Table 21.15 depicts the crushing and handling facility SUSEX requirements. The SUSEX is considered as an allowance for extending the overland conveyor.

Table 21.15
Crushing and Handling Sustaining Capital Detail

Description	Direct SUSEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Overland Conveyor	7.6	2.6	2.5	12.7
Total Crushing and Handling	7.6	2.6	2.5	12.7

*Contingency of 25% applied to total of direct and indirect costs

21.4.4 Heap Leach

Table 21.16 depicts the four future stages, Stage 2 to Stage 5, as the SUSEX expenditure requirements for the HLF. Similar to the CAPEX estimations, SUSEX costs are based on previous project-specific experience and industry standard benchmarking.

Indirect costs consisting of EPCM were estimated at 15.8% of direct costs.

Table 21.16
Heap Leach Sustaining Capital Detail

Description	Direct SUSEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Leach Pads	102.4	0.0	25.6	127.9
Solution Distribution & Collection System	40.0	0.0	10.0	50.0
On-Heap Conveying	17.3	0.0	4.3	21.6
EPCM	0.0	25.2	6.3	31.6
Total Heap Leach	159.7	25.2	46.2	231.1

*Contingency of 25% applied by WSP to total of direct and indirect costs

21.4.5 Process Plant

There is no SUSEX allowance considered for the Process Plant.

21.4.6 On-Site Infrastructure

There is no SUSEX allowance considered for the On-site Infrastructure.

21.4.7 Off-Site Infrastructure

There is no SUSEX allowance considered for Off-Site Infrastructure.



21.4.8 Water Management

Water management SUSEX shown in Table 21.17 were developed based on quantities modelled for each component. The WRSF drainage costs (rock drain construction) were scaled based on the previous 2024 design work and then scaled based on the diversion length to accommodate the increased WRSF footprint. The Double Double, Kona, and Supremo Extension diversions are new, and costs were developed from length-based quantities. Additional pit dewatering costs were included as active mining pits increased to match requirements based on ear 2 of the mining plan. All costs were inflated to Q1 2026 using the Industrial Product Price Index (IPPI) based on Primary Metal Manufacturing costs. A 30% factor was used to calculate indirect costs (from the direct costs) based on experience related to similar projects.



Table 21.17
Water Management Sustaining Capital Detail

Description	Direct SUSEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Pit Dewatering	3.6	1.1	1.2	5.8
Double Double Diversion	1.1	0.3	0.3	1.8
Kona Pit Diversion	1.9	0.6	0.6	3.1
Supremo Extension Diversion	2.2	0.7	0.7	3.6
Waste Dump Drainage (Rock Drain)	8.6	2.6	2.8	13.9
Alpha Pond	8.0	2.4	2.6	13
Total Water Management**	25.4	7.6	8.2	41.2

*Contingency of 25% applied to total of direct and indirect costs

**Numbers may not sum precisely due to rounding.

21.4.9 Water Treatment

Table 21.18 depicts the estimated SUSEX requirements for water treatment. Direct and indirect costs were calculated in the same manner as the CAPEX. Where historical pricing was used, costs were escalated to 2025 dollars.

Table 21.18
Water Treatment Plant Sustaining Capital Detail

Description	Direct CAPEX	Indirect	Contingency *	Total
	(CA\$ million)	(CA\$ million)	(CA\$ million)	(CA\$ million)
Water Treatment Plant	26.2	8.4	17.3	51.8
Total Water Treatment Plant **	26.2	8.4	17.3	51.8

*Contingency of 25% applied to total of direct and indirect costs of both the project and sustaining capital

**Numbers may not sum precisely due to rounding.

21.5 Rehabilitation and Closure Costs

At the end of the mine life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. Closure costs provide no asset value and are treated differently from equipment acquisitions in the tax assessment. The costs are detailed in Section 20.3.5.

21.6 Operating Costs

Table 21.19 summarizes the estimated LOM operating cost per tonne of material processed along with the US\$ cost per ounce gold produced for sale.

The unit costs discussed in this are averages over the LOM plan and there can be fluctuations in either annual costs or annual gold ounces sold that can affect a particular year's \$US/oz sold result for the overall operation.



Table 21.19
Site Operating Unit Costs

Description	CA\$/t processed	US\$/oz
Open Pit Mining	30.30	699
Site Services	2.81	65
Processing	6.94	160
General & Administrative	5.41	125
Water Treatment Plant	1.95	45
Total Site OPEX*	47.41	1,094

*Numbers may not sum precisely due to rounding.

21.6.1 Open Pit Mining

Open pit mining costs shown in Table 21.20 have been estimated based primarily on the mine plan which included physicals, equipment fleet and labour requirements. Utilization and performance data for equipment were used to arrive at the operating hours per machine type. Labour salaries were scaled from the 2024 Internal Engineering Report (JDS, 2024) at a 5% annual inflation rate. Fuel costs used were CA\$1.1/L and CA\$1.34/L, respectively for clear and coloured fuel. The remaining operating costs were benchmarked from a recently completed PEA for a gold project with a similar production rate, Q4 2025 vendor quotes and Costmine's Cost Estimation Service.

Drilling costs include items related to production, wall control and probe drilling. Blasting costs include items related to explosives loading and blasting. Loading and hauling costs include items related to material movement. Overhead costs include items related to mine maintenance administration, technical services (mine engineering and geology) and mine administration. Dewatering was the cost of in-pit dewatering identified through use of SRK's pit pump requirements. Other costs include various items such as road maintenance, dump maintenance, grade control and stockpile rehandle.

The average mine operating cost is US\$699/oz Au sold or CA\$30.30/t processed providing a total cost over the LOM of CA\$2,743.4 million.



**Table 21.20
Mining Operating Costs**

Activity	LOM Total (CA\$ million)	CA\$/t processed	CA\$/t mined
Overhead (Labour)	269.8	2.98	0.35
Drilling	239.2	2.64	0.31
Blasting	378.9	4.18	0.49
Loading	165.8	1.83	0.21
Hauling	846.4	9.35	1.09
Dewatering	119.8	1.32	0.15
Other	723.6	7.99	0.93
Total Mine OPEX	2,743.4	30.30	3.52

21.6.2 Process

Table 21.21 shows the estimated process operating costs, categorized in process labour, laboratory, maintenance and operating supplies, reagents and consumables, and power costs. The costs were scaled from the Internal Engineering Report (JDS, 2024) at a 5% annual inflation rate. Fuel costs used were updated to CA\$1.34/L representing Q1 2026.

**Table 21.21
Process Operating Costs**

Activity	LOM Total (CA\$ million)	CA\$/t processed
Process Labour	177.9	1.97
Laboratory	4.4	0.05
Maintenance & Operating Supplies	111.4	1.23
Reagent & Consumables Cost	152.0	1.68
Power Cost	182.8	2.02
Total Process OPEX*	628.5	6.94

*Numbers may not sum precisely due to rounding.

The average process operating cost is US\$160/oz Au sold or CA\$6.94/t processed providing a total cost over the LOM of CA\$628.5 million.

21.6.3 Site Services

Table 21.22 shows the estimated site services operating costs. The costs were scaled up from the Internal Engineering Report (JDS, 2024) at a 5% annual inflation rate, with the exception fuel/power. Fuel costs used were updated to CA\$1.34/L for the Q1 2026.



The average site services operating cost is US\$65/oz Au sold or CA\$2.81/t processed providing a total cost over the LOM of CA\$254.6 million.

Table 21.22
Site Services Operating Costs

Activity	LOM Total (CA\$ million)	CA\$/t processed
Equipment	49.0	0.54
Labour	107.4	1.19
Infrastructure Maintenance	3.7	0.04
Power	94.4	1.04
Total Site Services OPEX*	254.6	2.81

*Numbers may not sum precisely due to rounding.

21.6.4 General & Administrative

Table 21.23 shows the estimated G&A operating costs. The costs were scaled up from the Internal Engineering Report (JDS, 2024) at a 5% annual inflation rate. Fuel costs used were updated to CA\$1.34/L for the Q1 2026.

The average G&A operating cost is US\$125/oz Au sold or CA\$5.41/t processed providing a total cost over the LOM of CA\$489.4 million.

Table 21.23
G&A Operating Costs

Activity	LOM Total (CA\$ million)	CA\$/t processed
Labour	145.2	1.60
Operational Charges	343.3	3.79
Construction Charges	0.9	0.01
Total G&A OPEX*	489.4	5.41

*Numbers may not sum precisely due to rounding.

21.6.5 Water Treatment

Table 21.24 summarizes the estimated OPEX requirements for the WTP. Reagent consumption was estimated based on the annualized average influent flow rate to the WTP (~16 L/s) and the predicted water quality for the early phase (Years 2 to 13). Operating labour costs were developed based on the estimated full-time equivalent (FTE) staffing required to operate a facility of comparable size and complexity. The Operation and Maintenance (O&M) costs were estimated as a percentage of the capital cost of major process equipment. The OPEX for the WTP from Year 14 onward are accounted for in the closure costing outlined in Section 20.3.5.



Table 21.24
Water Treatment Plant Operating Costs

Activity	Annual Total (CA\$ million)	LOM Total (CA\$ million)	CA\$/t processed
O&M	1.2	14.7	0.16
Labour Charges	1.6	19.1	0.21
Treatment Chemical/Reagents	10.4	124.8	1.38
Utilities	1.5	18.0	0.2
Total WTP OPEX*	14.7	176.5	1.95

*Numbers may not sum precisely due to rounding.



22.0 ECONOMIC ANALYSIS

22.1 Resultant Summary

The Coffee Gold Project financial results in an after-tax NPV @ 5% of CA\$ 3,004.5 million with a 43.5% IRR and a payback of 1.97 years.

22.2 General

This report presents a conceptual study of the potential economic viability of the Coffee mineral resources. There are no mineral reserves as part of this evaluation. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The PEA is preliminary in nature and the production plan for use in the economic analysis includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. To advance the study beyond the PEA level, a Mineral Reserve estimate would have to be made, and Inferred Resources would need to be upgraded or removed from the production plan.

The economic analysis is based on the DCF method on a pre-tax and after-tax basis. Current Yukon tax regulations were used to assess corporate tax liabilities. The key metrics determined in the analysis are the NPV at a discount rate of 5%, the IRR and the payback period. A sensitivity analysis was carried out to assess the impact of variations in gold grade, recoveries, gold price, CAPEX, and OPEX on the financial metrics.

For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis.

The production schedule used in this analysis is based on the LOM and process plan detailed in Sections 16.0 and 17.0. The economic analysis is developed in terms of calendar years.

The CAPEX and OPEX used for the evaluation are the same as those in the estimates detailed in Section 21.0.

All costs and pricing are in Q1 2026 Canadian Dollars. No provision is made for the effects of inflation in this analysis.

22.3 Forward Looking Information

This section contains forward-looking information under Canadian securities law. The results of the analysis rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ materially from those presented here. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item:

- Projected commodities markets and prices of gold.
- Currency exchange rate fluctuations.
- Amount of mineralized material and material grade.
- Proposed mine production plan.



- Mining dilution and mining recovery.
- Geotechnical or hydrogeological considerations during mining.
- Process plant production plan.
- Heap Leach production plan.
- Recovery rates of gold.
- Ability of plant, equipment, processes to operate as anticipated.
- Capital costs of equipment and installations.
- SUSEX and OPEX.
- Closure costs and unforeseen reclamation expenses.
- Environmental, social, and licensing risks.
- Taxation policy and tax rate.
- Royalty agreements.
- Cost inflation.
- Project schedule and approvals timing.
- Availability of funding.
- Ability to maintain social license to operate.
- Unrecognized environmental risks.

22.4 Economic Criteria

22.4.1 Production Decision

The pre-production period is shown in the PEA as having a three-year duration with mining then starting in year four (Year 1). There is no mining shown during the three-year pre-production period.

22.4.2 Metal Price

Project revenues consist of doré sales. No other product is considered as part of the economic analysis. As outlined in Section 19.0, the economic analysis utilizes a long-term consensus commodity pricing forecast accepted by the WSP QP that is representative of the average of several financial institutions. The consensus gold price forecast provides a variable price deck of US\$4,110/oz Au in Year 1 and US\$3,620/oz Au long term applied to the payable gold ounces produced by the LOM from Year 2 onward.



22.4.3 Production Schedule

The production schedule used in this analysis is based on the LOM production plan outlined in Sections 16.

22.4.4 Process Recoveries

Heap leach recovery values as outlined in Section 13 illustrate the recovery values varying by rock type and pit. Table 22.1 shows the resultant overall annual recovery rates.

**Table 22.1
Overall Annual Recovery**

Year	Recovery %
2029	84%
2030	77%
2031	79%
2032	66%
2033	81%
2034	77%
2035	74%
2036	81%
2037	81%
2038	74%
2039	78%
2040	82%
2041	81%

Additionally, a three-month delay was applied to the ultimate recovery of the payable metal from the heap leach.

22.4.5 Commercial Terms and Freight

The commercial terms for the sale of gold and transportation of doré used in this analysis were established by the WSP QP based on benchmarked industry standard rates. These commercial terms are as shown in Table 22.2.

**Table 22.2
Commercial Terms for the Sale of Gold Doré**

Description	Unit	Doré
Refining Cost	US\$/oz	5.00
Gold Payable	%	99.995
Doré Transportation Cost	US\$/oz	15.00



22.4.6 Mineral Royalties

As indicated in Section 4.3, the following royalties that were applied to the economic analysis:

- A 3% NSR royalty payable to Newmont in Year 1 of production. For the remaining years this royalty is removed due to the buyback of this royalty for US\$100 million, a one-time charge occurring in Year 1 of production.
- Other NSRs with a total sliding scale effective rate of 2.2%-2.8%, net of a one-time buyback for CA\$2 million, exercised ahead of the commencement of production. The effective rate increases over time, reaching a maximum in year ten.

22.4.7 Capital and Operating Costs

The CAPEX used in this analysis are taken from the estimates detailed in Section 21.

The initial Project Capital is shown against the three pre-production years, with the bulk of the costs occurring in Year -1. Capital costs in Year 1 and beyond are shown as SUSEX expenditures, aligned with the schedule for mining provided from the mine plan outlined in Section 16. The schedule for the remaining SUSEX expenditures was determined using timelines provided by the QPs for those items.

The OPEX used in this analysis are taken from the estimates detailed in Section 21.

22.4.8 Other Costs

Other costs are the Closure Cost, the Working Capital, and the Salvage Value of equipment.

The reclamation and closure costs are detailed in Section 20 and total CA\$177.6 million. A progressive decommissioning schedule was identified for the cash flow that starts in Year 3. The postproduction closure amounts total CA\$171.5 million over a 37-year period beyond the 13-year production plan to equate to a total of 50 years of closure costs – a timing assumption carried forward from the PFS.

The yearly working capital amounts are based on one month of the ensuing year's operating costs with a net over the life of mine of zero.

The salvage value of equipment used in the economic analysis totals CA\$22.4 million in the later stages of the mine life. A salvage value for the mining equipment was determined based on the operating lifespan of the mining equipment against its usage, for a total of CA\$16.4 million. A 10% allowance for fixed equipment salvage against the processing mechanical equipment procurement cost, resulted in a fixed equipment salvage value of CA\$6.0 million.

22.4.9 Taxes

The following taxes are applicable:

22.4.9.1 Mining Tax

The mining tax rate applicable to projects in Yukon was applied against the annual Earnings Before Interest, Taxes, Depreciation, and Amortization, (EBITDA) adjusted for non-deductible royalties, cash reclamations expenses, and allowances for depreciation, development, and exploration, to arrive at a taxable income from Operations in any given year. For years where there is an operating loss generated as part of the course of business activities, then no mining tax is applicable that year. The Yukon mining tax is a sliding



rate up to 12% and averages 11.8% for the positive cashflow operating years of the Project and equalled \$1,004.9 million over the LOM.

22.4.9.2 Corporate Income Tax

The combined federal and provincial income tax rate for corporations in Yukon is 27% and this rate was applied to calculate the overall corporate income taxes and equalled \$1,848.2 million over the LOM. Taxable income is determined based on operating income adjusted for capital cost allowances and Canadian exploration and development expenses and other taxable deductions. All deductions and rates are based on currently enacted legislation and can be subject to change in the future.

Operating losses generated as part of the course of business activities can be carried forward and used to offset taxable income.

22.4.9.3 Tax Summary

The total tax payable in the economic analysis for the PEA equalled \$2,852.7 million.

22.5 Base Case Cash Flow Analysis and Economic Results

Based on the assumptions provided in this Report and at an assumed weighted average gold price of US\$ 3,656/oz based on gold ounces recovered over the Project life, the financial results indicate a positive pre-tax NPV of CA\$4,916.3 million, at a discount rate of 5%. As shown in Table 22.3, the after-tax NPV at a discount rate of 5% is CA\$3,004.5 million. This creates an after-tax IRR of 43.5% and an after-tax payback of 1.97 years.

**Table 22.3
Economic Results Summary**

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	CA\$ million	4,916.3	3,004.5
IRR	%	59.8	43.5
Payback Period	Years	1.52	1.97

To arrive at these results, a LOM discounted cash flow model was developed using the production schedule produced by Open Contour. The key technical and cost inputs are summarised in Table 22.4.



Table 22.4
Coffee Gold Project PEA: Key Technical Assumptions and Cost Inputs

Description	Unit	Value
Gold Price (Average LOM Base Case)	US\$/oz Au	3,656
Exchange Rate	US\$/CA\$	1.39
Discount Rate	%	5
Project Parameters		
LOM	years	13
Total Potential Mill Feed Tonnage	Mt	90.5
Average Mined Grade	g/t Au	1.25
Average Gold Recovery	%	77.5
Contained Gold	koz	3,644
Recovered Gold	koz	2,824
Average Annual Gold Production	koz/a	217
Average Mining Rate – All material per calendar day	t/d	177,687
Strip Ratio	-	7.6:1
Revenue Related Costs		
Selling Costs	CA\$ million	78.5
Royalty NSR	CA\$ million	394.9
LOM Operating Costs		
Open Pit Mining	CA\$ million	2,743.4
Site Services	CA\$ million	254.6
Processing	CA\$ million	628.5
General and Administrative (G&A)	CA\$ million	489.4
Water Treatment Plant	CA\$ million	176.5
Total Operating Cost¹	CA\$ million	4,292.4
Sustaining Cost Estimate		
Direct Cost	CA\$ million	495.2
Indirect Cost	CA\$ million	52.0
Contingency	CA\$ million	145.4
Total Sustaining Capital Cost	CA\$ million	692.7
Project Capital Estimate		
Direct Cost	CA\$ million	634.3
Indirect Cost	CA\$ million	188.0
Contingency	CA\$ million	175.9
Total Project Capital Cost	CA\$ million	998.2
Other Costs		
Sustaining Working Capital	CA\$ million	-16.0
Project Capital Working Capital	CA\$ million	16.0
Closure	CA\$ million	177.6
Royalty Buybacks	CA\$ million	141.0
Salvage Value	CA\$ million	-22.4
LOM Operating Unit Costs		
Open Pit Mining	\$/t processed	30.30
Site Services	\$/t processed	2.81
Processing	\$/t processed	6.94
General and Administrative (G&A)	\$/t processed	5.41
Water Treatment Plant	\$/t processed	1.95
Total Operating Unit Cost	\$/t processed	47.41
Total Cash Cost ²	US\$/oz Au	1,214
AISC ³ (LOM)	US\$/oz Au	1,386

¹ Total operating cost is after transfer of year -1 to capital and refers to onsite charges that cover open pit mining, processing and material handling, and onsite general and administrative costs during production period.

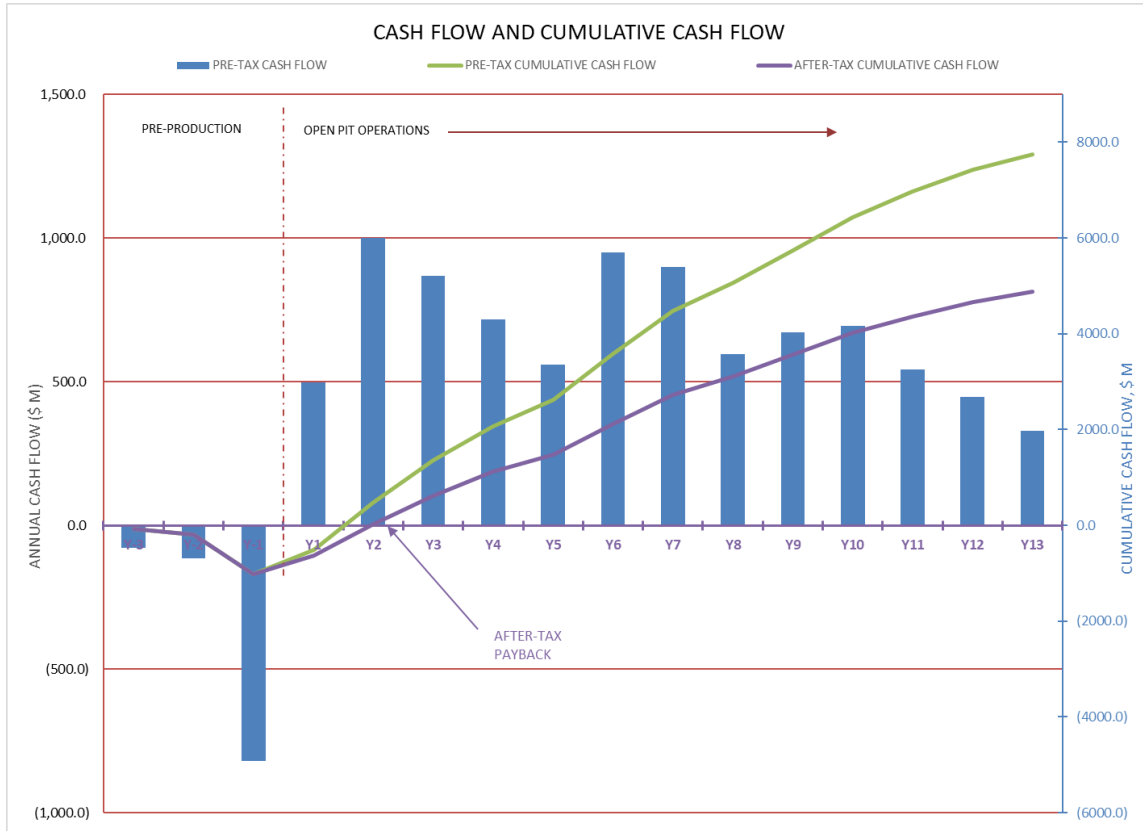
² Total cash costs are a non-GAAP measure and are calculated as the sum of doré transportation, treatment and refining charges, royalties, and total operating costs, divided by the quantity of ounces sold.



³ AISC is a non-GAAP measure and is calculated as the sum of doré transportation, treatment and refining charges, royalties, total operating costs, and sustaining capital costs, divided by the quantity of ounces sold.

The annual and cumulative cash flows of the project are presented by Figure 22.1.

Figure 22.1
Annual and Cumulative Cash Flows



Source: WSP, 2026

The estimated annual cash flows for the Project is provided in Table 22.5.

The resulting pre-tax cash flow estimated for the Project over the LOM was \$7,596M, and the after-tax cash flow was estimated at \$4,743 M.



Table 22.5
Summary of Production Schedule and Cash Flows - Base Case by year

Cash Flow Model	Total	Units	YR-3	YR-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Beyond	
Mill feed production tonnage	90,532	k t				7000	7400	7400	7400	7400	7400	7400	7400	7400	7400	7400	7400	7400	2131	
Mill feed head grades, Au	1.25	g/t				1.46	1.66	1.23	1.34	1.07	1.64	1.48	1.01	1.15	1.25	1.02	0.81	0.94		
Ounce Production	2,823.7	koz				207.0	298.3	249.5	215.6	206.9	275.3	268.9	211.3	215.4	220.5	198.0	165.4	91.7		
Commodity price, Au	3,656	US\$/oz				4,110	3,620	3,620	3,620	3,620	3,620	3,620	3,620	3,620	3,620	3,620	3,620	3,620	3,620	
Gross Revenue - US\$	10,323	US\$ million				851	1080	903	780	749	997	973	765	780	798	717	599	332		
Exchange Rate	1.39	CAN\$:US\$				1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	1.39	
Gross Revenue - CAN\$	14,349	CA\$ million				1182	1501	1255	1085	1041	1385	1353	1063	1084	1109	996	832	461		
Selling Costs	(78)	CA\$ million				(6)	(8)	(7)	(6)	(6)	(8)	(7)	(6)	(6)	(6)	(6)	(5)	(3)		
Net Smelter Return	14,270	CA\$ million				1177	1493	1248	1079	1035	1378	1345	1057	1078	1103	991	828	459		
Royalites	(395)	CA\$ million				(61)	(33)	(27)	(24)	(27)	(36)	(35)	(27)	(30)	(31)	(28)	(23)	(13)		
Net Revenue	13,875	CA\$ million				1115	1460	1221	1055	1008	1342	1310	1030	1048	1072	963	805	446		
Operating Costs	(4,292)	CA\$ million				(195)	(311)	(343)	(333)	(342)	(382)	(403)	(367)	(376)	(356)	(374)	(376)	(134)		
Sustaining Direct Capital Costs	(495)	CA\$ million				(190)	(109)	(9)	(3)	(72)	(6)	(8)	(47)	(1)	(17)	(32)	(0)	(0)		
Sustaining Indirect Capital Costs	(52)	CA\$ million				(20)	(9)	0	(1)	(11)	0	(1)	(7)	0	0	(4)	0	0		
Sustaining Contingency Costs	(145)	CA\$ million				(61)	(29)	(2)	(1)	(21)	(2)	(2)	(13)	(0)	(4)	(9)	(0)	(0)		
Sustaining Working Capital	16	CA\$ million				(10)	(3)	1	(1)	(3)	(2)	3	(1)	2	(1)	(0)	20	11		
Project Capital Direct Costs	(634)	CA\$ million	(57)	(73)	(504)															
Project Capital Indirect Costs	(188)	CA\$ million	(14)	(28)	(146)															
Project Capital Contingency Costs	(176)	CA\$ million	(7)	(15)	(154)															
Project Capital Working Capital	(16)	CA\$ million	0	0	(16)															
Closure Costs	(178)	CA\$ million				0	0	(0)	0	(0)	(0)	(0)	(0)	(0)	(0)	(1)	(1)	(2)	(171)	
Royalty Buybacks	(141)	CA\$ million	0	0	0	(141)														
Salvage Value	22	CA\$ million				0	0	0	0	0	0	0	0	0	1	0	0	8	13	



Cash Flow Model	Total	Units	YR-3	YR-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Beyond
Taxes	(2,853)	CA\$ million				(122)	(338)	(270)	(222)	(200)	(301)	(297)	(216)	(216)	(237)	(195)	(145)	(114)	21.4
Cash flow results																			
Pre-tax cash flow	7,596	CA\$ million	(78)	(116)	(820)	499	999	867	717	560	949	899	595	672	694	542	446	328	(158)
Cumulative pre-tax cash flow	—	CA\$ million	(78)	(194)	(1,014)	(516)	483	1,351	2,067	2,627	3,577	4,476	5,071	5,742	6,437	6,979	7,425	7,754	7,596
After-tax cash flow	4,743	CA\$ million	(78)	(116)	(820)	377	661	598	494	360	649	602	379	456	457	347	301	215	(137)
Cumulative after-tax Cash Flow	—	CA\$ million	(78)	(194)	(1,014)	(638)	23	620	1,115	1,475	2,123	2,725	3,104	3,560	4,017	4,364	4,665	4,880	4,743



22.6 Sensitivity Analysis

Table 22.6 depicts the results of a sensitivity analysis that was conducted using the Section 22.5 economic analysis as the base case to assess the impact of changes in the price of gold on the Project's NPV at a 5% discount rate, Payback Period and IRR.

Table 22.6
Economic Metrics Sensitivity to Variations in the Gold Price

Gold Price (US\$/oz)	2500	3300	PEA 3656 ¹	4500	Near Spot ²	5500
Pre-tax NPV (5%) (CA\$ million)	2,008	3,980	4,916	7,201	8,340	9,798
After-tax NPV (5%) (CA\$ million)	1,138	2,403	3,004	4,470	5,200	6,134
After-tax IRR (%)	22.3%	36.0%	43.5%	57.7%	62.1%	72.4%
Payback (years)	3.51	2.39	1.97	1.51	1.46	1.22

¹Analyst consensus prices as at February 11, 2026: US\$4,110/oz Au in 2029 and US\$3,620/oz Au in 2030 and beyond
²Near Spot Price of US\$5,000/oz on February 18, 2026, reflecting LBMA PM closing price of US\$5,003/oz Au

The sensitivity analysis also assessed the impact of changes in gold grade, process recovery, gold price, CAPEX, SUSEX and OPEX on the Project's NPV at 5%, the IRR, the payback in years, and the percent variation in the NPV at 5%. The impact of each variable is examined individually with an interval of $\pm 20\%$ and increments of 10% applied. It is to be noted that the margin of error for cost estimates at the PEA study level is typically -30% to +50%.

The after-tax results of the sensitivity analysis are shown in Table 22.7 to Table 22.11 and Figure 22.2 to Figure 22.3. The NPV of the Project is most sensitive to variations in the gold grade, gold price and gold recovery projections, followed by OPEX and initial CAPEX. Although the intervals and increments assessed are different, the spider graphs show the NPV appears to be equally sensitive to variations in grade, recovery and price. Overall, within the limits of accuracy of the cost estimates in this study, the Project's potential after-tax viability does not seem significantly vulnerable to the under-estimation of either capital costs or operating costs, when taken individually.



Table 22.7
Economic Metrics Sensitivity to Variations in the Total Operating Cost

Description	Units	After Tax Net Present Value @ 5%				
		-20%	-10%	Base	10%	20%
Percent Variation	%	-20%	-10%	Base	10%	20%
Total Operating Cost	CA\$ million	3,434	3,863	4,292	4,722	5,151
NPV @5.0%	CA\$ million	3,369	3,186	3,004	2,823	2,641
IRR	%	46.6%	45.0%	43.5%	41.8%	40.2%
Payback	Years	1.86	1.91	1.97	2.02	2.09
Percent Variation in NPV	%	12.1%	6.1%	0	-6.1%	-12.1%

Table 22.8
Economic Metrics Sensitivity to Variations in the Grade/Recovery

Description	Units	After Tax Net Present Value @ 5%				
		-20%	-10%	Base	10%	20%
Percent Variation	%	-20%	-10%	Base	10%	20%
Recovery	%	62.0%	69.7%	77.5%	85.2%	93.00%
NPV @5.0%	CA\$ million	1,804	2,405	3,004	3,604	4,203
IRR	%	30.4%	37.2%	43.5%	49.4%	55.2%
Payback	Years	2.71	2.28	1.97	1.75	1.58
Percent Variation in NPV	%	-39.9%	-20.0%	0	20.0%	39.9%



**Table 22.9
Economic Metrics Sensitivity to Variations in the Gold Price**

Description	Units	After Tax Net Present Value @ 5%				
		-20%	-10%	Base	10%	20%
Percent Variation	%	-20%	-10%	Base	10%	20%
Gold price	US \$/oz	2,896	3,258	3,620	3,982	4,344
NPV @5.0%	CA\$ million	1,798	2,402	3,004	3,607	4,210
IRR	%	30.3%	37.1%	43.5%	49.5%	55.3%
Payback	Years	2.72	2.28	1.97	1.75	1.57
Percent Variation in NPV	%	-40.2%	-20.1%	0	20.1%	40.1%

**Table 22.10
Economic Metrics Sensitivity to Variations in the Project Capital Cost**

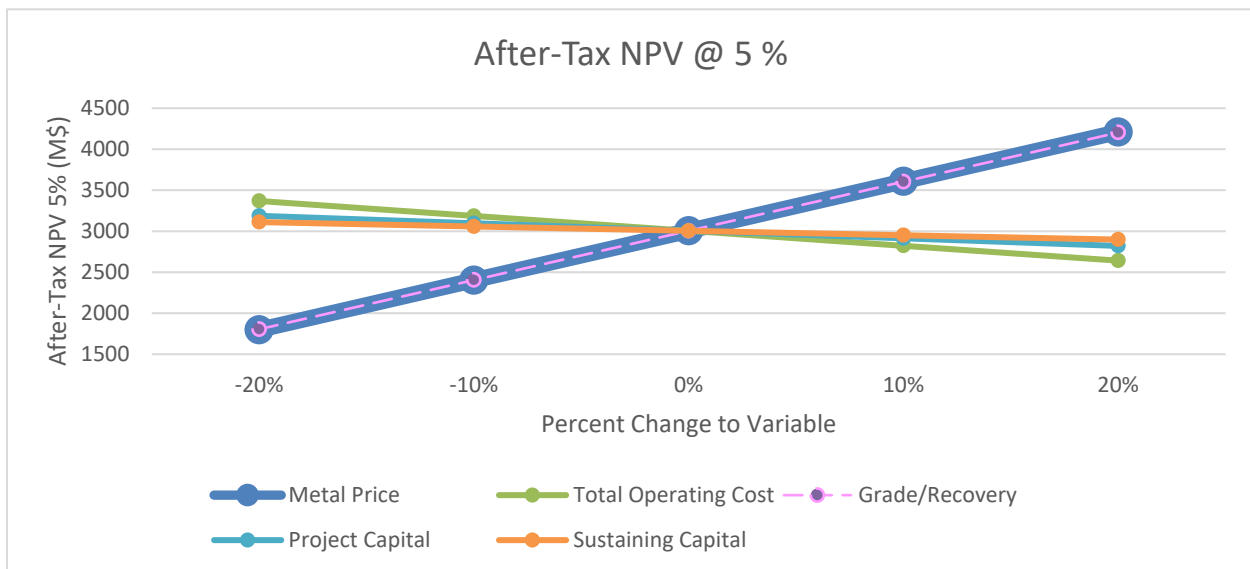
Description	Units	After Tax Net Present Value @ 5%				
		-20%	-10%	Base	10%	20%
Percent Variation	%	-20%	-10%	Base	10%	20%
Project Capital	CA\$ million	799	898	998	1098	1198
NPV @5.0%	CA\$ million	3,188	3,096	3,004	2,913	2,821
IRR	%	52.8%	47.7%	43.5%	39.8%	36.7%
Payback	Years	1.66	1.81	1.97	2.13	2.30
Percent Variation in NPV	%	6.1%	3.1%	0	-3.1%	-6.1%



Table 22.11
Economic Metrics Sensitivity to Variations in the Sustaining Capital Cost

Description	Units	After Tax Net Present Value @ 5%				
		-20%	-10%	Base	10%	20%
Percent Variation	%	-20%	-10%	Base	10%	20%
Sustaining Capital	CA\$ million	554	623	693	762	831
NPV @5.0%	CA\$ million	3,111	3,058	3,004	2,951	2,898
IRR	%	45.5%	44.5%	43.5%	42.5%	41.5%
Payback	Years	1.85	1.90	1.97	2.03	2.10
Percent Variation NPV in	%	3.5%	1.8%	0	-1.8%	-3.5%

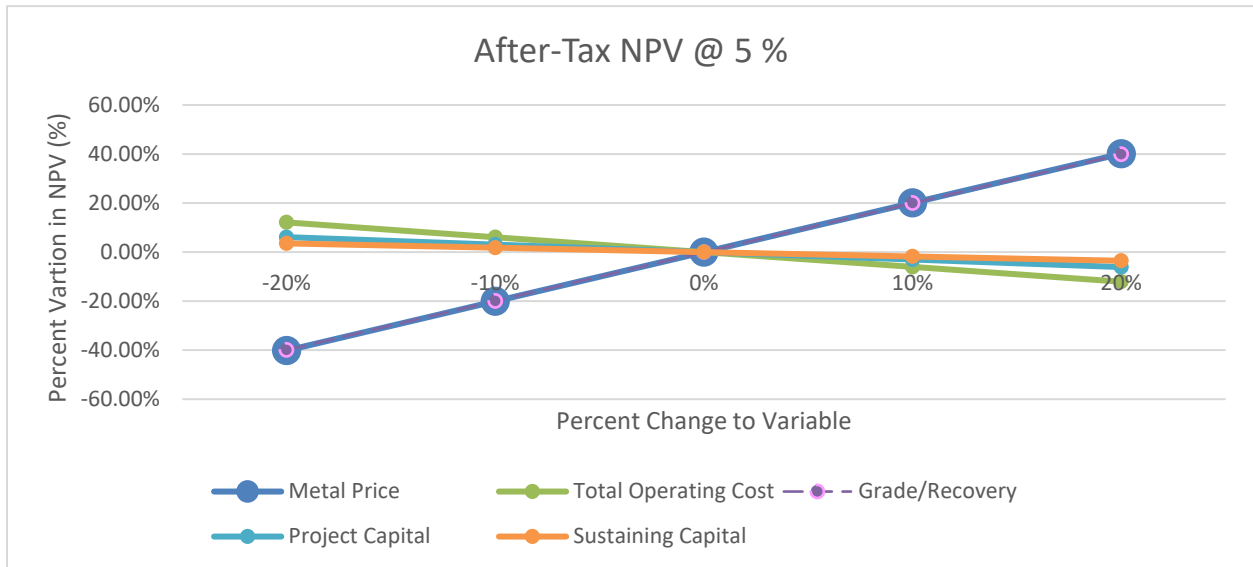
Figure 22.2
After-Tax NPV 5%: Sensitivity to Metal Price, Grade/Recovery, Project Capital, Sustaining Capital and Total Operating Cost



Source: WSP, 2026



Figure 22.3
After-Tax Percent Variation in NPV 5%: Sensitivity to Metal Price, Grade/Recovery, Project Capital, Sustaining Capital and Total Operating Cost



Source: WSP, 2026

Table 22.12 shows the NPV sensitivity of the Project to alternative discount rates of 0%, 5%, 8%, 10%, and 12%. The economic indicators, both Pre-Tax and After-Tax, for the various discount rates are provided.



Table 22.12
Economic Indicators with variations on Discount Rate

Discounting rate 5%	Units	Pre-Tax	After-Tax
Payback period	years	1.52	1.97
IRR	%	59.8%	43.5%
Discount Cash Flow	CA\$ million	4,916	3,004
Discount Rate Sensitivity			
NPV @ 0%	CA\$ million	7,596	4,743
NPV @ 5%	CA\$ million	4,916	3,004
NPV @ 8%	CA\$ million	3,837	2,301
NPV @ 10%	CA\$ million	3,268	1,931
NPV @ 12%	CA\$ million	2,794	1,622



23.0 ADJACENT PROPERTIES

The Micon QPs note that there are three known Mineral Projects adjacent to the Coffee Gold Project. The Casino Copper-Gold Porphyry Deposit owned by Western Copper and Gold Corporation is located approximately 30 km southeast of the main drilled area, the White Gold Corporation mineral claims are located immediately to the northwest of the Coffee Gold Project, and the Boulevard Project owned by Independence Gold Corporation is located immediately to the southwest



24.0 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information pertaining to the Coffee Gold Project have been disclosed under the relevant sections of this Report. No additional information or explanation is necessary to make this Technical Report understandable and not misleading.



25.0 INTERPRETATION AND CONCLUSIONS

Based on the exploration and drilling completed to date, the following facts are pertinent to the progression of the Coffee Gold Project.

25.1 Geology and Mineral Resource Estimation

25.1.1 Exploration/Drilling

The gold mineralization discovered at the Coffee Gold Project to date is hydrothermal in origin and both structurally and lithologically controlled. The current interpretation leans more towards linear shear hosting structures; however, a closer look at the “epicentre” of the mineralization suggests that the existence of a mega IRG system could be possible. This requires a systematic investigation.

Reconnaissance exploration/drilling elsewhere on the property has been successful in discovering the presence of silver mineralization of potentially economic grades associated with gold in some areas of the deposits across the property.

25.1.2 Mineral Resources Status

The growth potential for the mineral resource is favourable as the main Supremo deposit(s) remain open for expansion along strike in the north-northeast direction and down dip. In addition, there is potential for growing the resource via new discoveries, based on the fact that several known mineral occurrences within the greater Project area remain to be test drilled for resource evaluation. Thus, in the Micon QP’s opinion, the deposit/mineral resource is poised for growth on three fronts as follows:

- Additions from the already discovered deposits not included in the current resource (i.e., sparsely drilled parts of Americano, Espresso, Cappuccino, Dolce, French Press and Sugar).
- Additional exploration in the greater Project area.
- Deeper drilling to explore the likely potential for sulphides at depth.

Furthermore, the addition of silver into the MRE will increase the value of the resource; hence, the need to assess the silver enriched areas. This can be achieved initially by mineralogical work to determine whether the silver enrichment is due to localized phenomena.

25.1.3 Uncertainty and Risks

All mineral resource estimates have a degree of uncertainty or risk associated with them, due to technical, environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors, among others. All mineral resource estimates also present their own opportunities.

Factors that may affect the MRE include fluctuations in the price of metals, in particular gold and changes in the metallurgical recoveries and bulk density assignments. In addition, it is the Micon QP’s opinion that the factors set out below could affect the mineral resource estimate.

- The geological interpretations and assumptions used to generate the estimation domain.
- The confidence assumptions and methods used in the mineral resource classification.
- Economic assumptions used in the cut-off grade determination.



- Input and design parameter assumptions that pertain to the open pit mining constraints.
- Assumptions as to the continued ability to access the Project site, retain mineral and surface rights titles, maintain the operation within environmental and other regulatory permits, and maintain the social license to operate.

To mitigate risks related to metallurgy/bulk density, additional detailed investigations are recommended prior to undertaking feasibility studies. Risks associated with fluctuations in the price of metals are uncontrollable; however, a modest long-term forecast gold metal price has already been considered in determining the economic factors for the MRE.

The exploration work completed, and the results obtained to date, are satisfactory to justify further work to move the Coffee Gold Project to the next level towards building a mining venture.

The systematic progression of the Coffee Gold Project towards becoming a mining project will be guided by clearly defined short-term and medium-/long-term objectives whilst not losing sight of the fact that the overall resource size within the Project area remains the topmost factor for a sustainable future and the associated long term investment decisions.

25.2 Mining

Mining risks to the PEA include defined mill feed quantities, metal grades, associated waste rock quantities and the estimated mine costs.

Changes in metal prices may increase the economic cut-off grade or reduce the size of the selected open pit limits. These changes will reduce the size of the pits and resource base to include into the mine plan.

The ability of the mining operation to meet the annual production rate and anticipated grade control standards and recoveries.

Reduced selectivity with the mining fleet chosen may reduced mining or milling recoveries. Increased mining dilution would result in an increased cost of achieving the planned PEA metal head grades and production.

25.3 Mineral Processing & Metallurgy

There have been a number of test programs conducted on Coffee material by reputable laboratories. The preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. Good metal accountability across a range of testwork results were observed and added confidence to the data review.

Mineralized material demonstrated amenability to the heap leaching process. Generally oxide material had high recoveries, transition material showed moderate results, while the sulphide sample had poor recovery, indicating refractory behaviour. Temperature impacts on leaching recovery has been tested and the results indicated that minimal variation in leaching kinetics is to be expected between a cold temperature of 4°C and warm ambient temperature of 22°C. Based on the agglomeration and compacted permeability tests, cement agglomeration is considered not necessary.

In the WSP QP's opinion, the existing samples and testing overall are adequate for a PEA study and align with processing plans.



25.4 Recovery Methods

25.4.1 Crushing and ADR

The processing design for the Project has undergone several studies. The current process configuration is sufficiently developed to meet the PEA requirements, with production capacity aligned to the mine plan. The selected combination of gyratory and cone crushers is expected to provide a reliable solution as compared to mineral sizers for the required crushing operations.

Opportunities exist to investigate the feasibility of reducing the final crushed product size (e.g., to 37.5 mm) to enhance recovery outcomes. Another opportunity is to evaluate the economics of gold recovery through flotation route for the sulphide material.

25.4.2 Heap Leach

The heap leach pad has been designed to meet the leach capacity requirements for this PEA. Material properties have been assumed at this time, with additional site investigations and material characterization work including testing mineralized material, as included in the recommendations. Current analyses indicate the heap leach pad will contain the design tonnage. Cost estimates made for the CAPEX and SUSEX associated with the heap leach pad were developed using pricing based on project-specific experience and available site investigation information.

25.5 Infrastructure

25.5.1 General

The infrastructure key consideration will focus on ensure year-round access where possible, with contingency plans for seasonal river-crossing limitations, and emphasizes self-sufficient utilities, water, and waste management.

Several comprehensive studies have been conducted for the Coffee project, resulting in well-planned infrastructure for a PEA that enables the Project to advance to its next phase of development.

25.5.2 Water and Waste Management

The water and waste management components have been developed following regional best practice and incorporating available site-specific information. Due to the past and ongoing permitting and licensing work for the Coffee Gold Project, much of the water management information has been looked at in closer detail since 2018. Specifically, the water management infrastructure components would be expected to be at levels beyond what is typically required for a PEA.

25.5.3 Heap Leach Facility Water Treatment

The proposed design for the WTP was developed for the 2024 Internal Engineering Report (JDS, 2024). As a result, there are uncertainties related to key design assumptions in terms of influent water quality, flow rates, and treatment objectives. While the WTP design was developed based on the results from multiple treatability testing, there are uncertainties associated with the testing, and additional testing may be required as outlined in Section 26.5.2. The WTP was designed to handle the discharge of peak flowrates of excess solution from the HLF and the associated water quality for the then-defined Project in 2023 to achieve treatment targets for the same. Cost estimates made for the CAPEX and SUSEX associated with the HLF WTP were developed based on the predicted flowrates and assumed construction duration over two years.



25.6 Environmental, Social and Permitting Considerations

25.6.1 Environmental, Social and Permitting Considerations

The environmental, social and permitting material contained in this section has been drawn from the Yukon Environmental and Socio-Economic Assessments for both the Coffee Gold Mine Project (File Number: 2017-0211) and the Advanced Quartz Exploration-Coffee Gold (File Number: 2020-0074), the Water Licence and Quartz Mining Licence applications and territorial and federal regulations. The Project as described in this PEA differs from the project assessed under YESAA 2017-0211. Accordingly, permitting is not assumed to proceed unchanged. A permitting strategy will be developed to assess Project changes against YESAA 2017-0211 and existing licensing applications, and to determine whether amendments or additional regulatory processes may be required.

25.6.2 Closure, Decommissioning and Rehabilitation

The closure cost estimate includes significant uncertainty and may change materially. Potential changes in post closure duration, regulatory requirements, societal expectations, or the realization of closure risks during the mine life, could materially affect the closure cost estimate. Although discounted closure costs may not materially affect project economics, the associated risk remains relevant and should be considered.

25.7 Costs

The costs as shown are representative of a PEA study with a -30% to +50% level of accuracy.

25.8 Economics

The after-tax financial results show Coffee Gold as a viable Project with a NPV@5% of CA\$3,004.5 million, an IRR of 43.5%, and a Payback in Years of 1.97.



26.0 RECOMMENDATIONS

26.1 Geology and Mineral Resources

Coffee is a project with significant potential that warrants additional work. A considerable effort has been made to delineate the deposit and test metallurgical methods and recoveries as well as mining and associated infrastructure to develop the Project. All activities completed to date have yielded positive results encouraging Fuerte to continue further studies. Accordingly, infill and step-out drilling are recommended to upgrade and expand the resource. The approximate budget for this work is CA\$ 20 M.

26.2 Mining

26.2.1 Mining

The following recommendations and opportunities are made related to the mining:

- Autonomous mining may reduce mining costs and personnel count for the project. The project would benefit from a 'Technology roadmap' study. A technology 'roadmap' should be written in collaboration with other vendors in the space.
- Mine and grade control strategy with respect to mining the type of mineralized vertical vein structures should be detailed before fleets are chosen. Operational improvements to reduce mine dilution will optimize head grade and cost per ounce produced.
- Finalize fleet size and type with updated quotes and availability.
- Permit constrains should be understood and implemented within the mine plan detail such as permafrost mining and dump building.
- Haul road plans should be detailed given the newest pit designs.
- Inpit backfilling plans should be detailed given the newest pit designs. Finalize and detail in pit dumping given the changes in pit geometry and potential for deeper pits in the future due to higher metal prices.
- Determine how the Alpha WRSF is filled. Low level lifts will create long downhill loaded hauls. Efforts should be made to reduce these hauls as they will be costly.
- Leach pad construction (earthworks fill material) should be included in the mine planning detail.

26.2.2 Geotechnical Data Collection & Future Studies

26.2.2.1 *Recommendations for Future Data Collection*

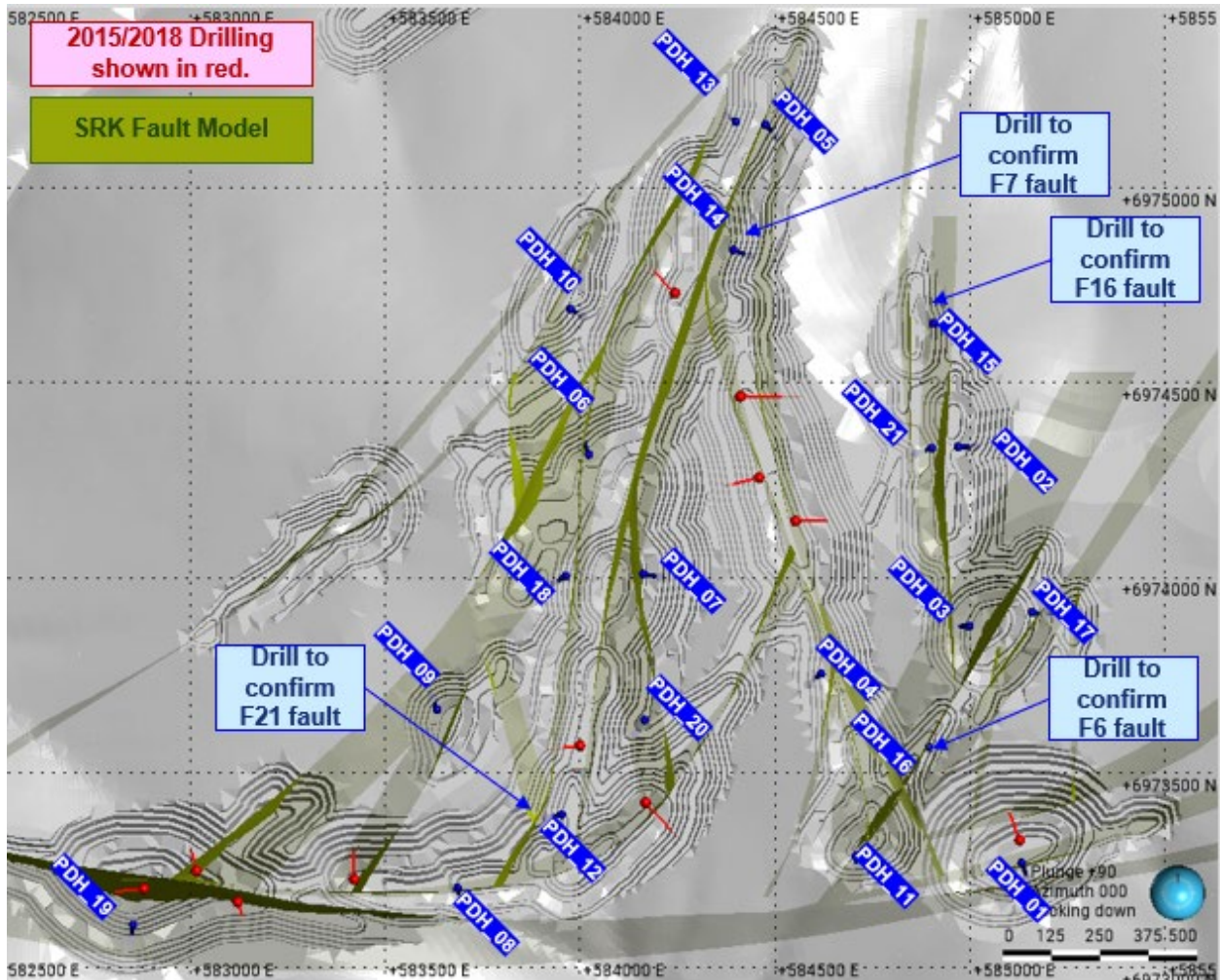
To advance the geotechnical model to PFS-level confidence, additional geotechnical drilling at a density of approximately one hole per 250 m of slope length is recommended (Fillion 2018), with priority given to untested structural domains (e.g., Double-Double, South, and Latte South). Figure 26.1 summarizes approximate locations of recommended drillholes, comprising 21 boreholes totaling approximately 5,000 meters.

The drilling program is designed to ensure adequate spatial coverage and targeted geotechnical data gaps. At least one geotechnical drillhole is planned for each significant pit wall orientation to adequately define



rock mass conditions for a PFS-level study. Additional targeted drilling is incorporated to characterize critical areas such as haul ramps and zones of potentially adverse structural features or known faults, where localized ground conditions may control stability.

Figure 26.1
Planned Boreholes in Plan View on Updated 2026 Design



26.2.2.2 Recommendations for Future Studies

Recommendations for future studies are included to improve confidence in geotechnical characterization and elevate the Coffee Gold project to the next phases of study. These recommendations include the following:

- Validation of strength parameter inconsistencies (e.g., fresh schist UCS, GSI assignments, cohesion on foliation) through supplemental laboratory testing
- Revision of structural model conducted after geotechnical data collection programs to confirm characterization of previously identified structures as well as identify structures not previously modelled.



- Targeted review in areas where major structures intersect pit wall benches, potentially forming planar, wedge, or bi-linear failure mechanisms.
- Data collection and characterization of groundwater conditions to support future pore-pressure-sensitive analyses.
- Identification of pseudo-static considerations and assessments for later study stage stability analyses.
- Characterization of discontinuity persistence (through methods like wall mapping) as well as as-built bench conformance and backbreak assessment to reconcile inputs and assumptions in slope stability analyses.
- Pit design refinement, including removal or flattening of structural “noses”.

26.3 Mineral Processing & Metallurgical Testing

In the next phase of project development, it is recommended to evaluate the impact of blending on extraction performance and flowsheet selection. Preliminary flotation testing on sulphide samples has yielded promising results. It is recommended to conduct trade-off studies to assess the feasibility of a flotation processing route for sulphide material. Should the economic analysis prove favourable, additional flotation testwork is suggested to enhance confidence in the process.

To build on these recommendations, the evaluation of blending should include detailed modelling of how different material types interact during processing. The effect of blending ratios on metallurgical recovery, concentrate quality, and reagent consumption can be studied through laboratory and pilot-scale tests. By optimizing blends, the Project can maximize resource utilization while minimizing operational risks.

26.4 Recovery Methods

26.4.1 Crushing and ADR

The following are recommended for the process plant design in the next phase of the Project:

- Evaluate secondary crushing in an open-circuit versus a closed-circuit configuration;
- Optimize final crushed product size and crushing circuit general arrangement;
- Perform detailed thermal analysis for solution management under cold weather conditions;
- Develop a detailed operations and equipment maintenance plan during the during the coldest period each year, which is assumed to be from January to March

26.4.2 Heap Leach Pad

A detailed geotechnical field investigation is under development to further define subsurface conditions beneath and surrounding the heap leach pad, process plant areas, and ponds, to include detailing permafrost extents.

The geotechnical program will include strength testing for material delivered to the heap leach pad in the first six months, Years 1-2, Years 3-4 and beyond and include any other material type changes that may occur periodically during mining that may contain varying strength parameters. Testing will include shear



strength testing, decrepitation testing, and compacted permeability at a minimum. A comprehensive site investigation plan for material collection, testing, and characterization is being developed for these activities.

Detailed thermal modelling should be performed once additional strength data, updated climate data, and other design parameters are obtained or refined. These ongoing advancements to the project will be incorporated into project plans during subsequent phases of the project.

26.5 Infrastructure

26.5.1 Infrastructure General

Infrastructure development for remote heap leach operations should be guided by a self-sufficient design approach tailored to cold climates, with an emphasis on constructability, reliability, and long-term operational performance. It is essential to address construction sequencing and logistics during the early planning stages, given the limited summer construction period and reliance on seasonal transportation.

Fuel storage and handling facilities should be appropriately sized to accommodate extended resupply intervals. These facilities will include secondary containment, winterized transfer systems, and spill prevention measure.

26.5.2 Water & Water Management

Additional geotechnical investigations will be required to support the next stage of engineering, specifically along the footprint of the proposed Alpha WRSF, Alpha Pond, and new diversion alignments around the pits. This additional investigation (test pitting and drilling) would be done to improve foundation characterization and construction material availability and to give additional information and parameters for additional stability and deformation calculations. Similarly, the next stage of engineering will include more advanced hydrologic modelling to evaluate ranges of long-term snowmelt volumes to inform pond sizing. Past investigation performance on site would indicate that around three to four test pits could be completed each day (with approximately 60 total test pits targeted to be completed). For the drilling the focus would be on slower drilling with better recovery of overburden and frequent sample collection so ice contents and characterizations can be noted. This would be expected to include around 510m of drilling with drill hole depths typical being short (less than 50m in length).

For the water and waste management infrastructure, three weeks of test pitting and a month or more of additional drilling (diamond drilling using chilled brine or sonic drilling using brine or dry methods) would be required to be completed to advance designs and to get additional suitable foundation (specifically permafrost and overburden) material characterization data. This drilling would be done within the WRSF footprints, and in the water management (diversions and pond) footprints and alignments. Exact locations would be expected to be adjusted in the field based on the foundation conditions encountered in the initial test pitting and initial water management drillholes.

Several risks and uncertainties have been identified that will be looked at in more detail as part of the next stages of design:

- Confirm borrow sources for material to construct the ponds and dams, to avoid higher capital costs to source material and / or to manufacture (crush) materials to required technical specifications.
- Timing of diversion channels excavations to avoid additional cladding and geotextiles if the work is completed in warmer months when cuts may expose areas with higher percentages of permafrost.



- Quantity of waste rock in the early mining years as low quantity of competent waste rock for the Alpha Rock Drain could lead to delays with the waste rock storage facility construction or constrain initial waste management operations.
- Alpha Pond dam foundation changes and upgrades may be required as foundation seepage is expected until more information becomes available.
- Timing of Alpha Pond construction as the planned winter construction avoids increase construction costs from temporary water diversion systems, including coffer dams and pumping systems to control flows and keep the foundation area dry.
- Additional site investigation into the end of the Halfway Diversion to assess the potential for more advanced PFS designs having additional or more robust energy dissipation requirements to avoid the current steep grades (beyond 10% grades) that will require energy dissipation measures.

Collection of additional streamflow data is available in Halfway Creek during spring freshets to calibrate the hydrologic model

26.5.3 Heap Leach Facility Water Treatment

The following are recommended to confirm and optimize the HLF WTP design:

- Reassess the WTP design basis to reflect changes associated with the HLF design advancement and the potential treatment of the pit dewatering stream.
- Conduct additional treatability testing using leached water samples using mineralized material to reassess treatment requirements and operating conditions.
- Update and optimize the WTP design based on the revised design basis and the treatability test results.
- Complete process waste classification for dewatered solids and clean-in-place wastes and evaluate appropriate management and disposal strategies.
- Update the cost estimate to reflect the optimization.

26.6 Environmental, Social & Permitting Considerations

26.6.1 Closure Planning Recommendations

Closure planning and associated closure costs carry a level of uncertainty into operations that can only be addressed once the mine is operating. As such, the recommendations for the following actions are focussed on those activities that can address the key uncertainties relevant to this stage of the project.

- **Improve progressive closure inclusion in operational planning:** Progress the LOM engineering designs and operational material movement schedules to a higher level of project definition, to increase confidence that progressive closure commitments, currently unaccounted for in the closure cost estimate, are included in the project design and operational plan.
- **Improve closure cost estimation for the project:** Collation of a project-specific closure cost model built up from first principles, to provide for a more logical calculation of closure liability and to include a more detailed consideration of closure logistics and schedule. The cost model should also incorporate risk and uncertainty for better understanding of closure cost ranges, sensitivities and the potential impact on the project economics. An updated cost model will also allow for enhanced understanding of financial assurance requirements and inclusion into cost modelling. The cost model should be prepared in a manner that allows for scenario planning and can be taken all the way through project planning to operations to closure without significant redevelopment. A robust basis of estimate document should be developed to accompany the cost model.



- **Progress closure design to reflect the updated PEA mine design:** Updating the erosion, geochemistry and water management modelling, accounting for climate change predictions, and associated closure requirements will provide for a more robust closure cost estimate and understanding of areas of greatest uncertainty.
- **Stakeholder engagement:** Progressing early and meaningful stakeholder and rightsholder engagement for improved understanding of closure requirements and socio-economic transition requirements.



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28.0 DATE AND SIGNATURE PAGE

"Charley Murahwi" {signed and sealed}

Charley Murahwi, M.Sc., P.Geo., FAusIMM

Micon International Limited

Signing Date: April 7, 2026

Effective Date: March 27, 2026

"Hannah Chiew" {signed and sealed}

Hannah Chiew, P.Eng.,

Ensero Solutions

Signing Date: April 7, 2026

Effective Date: March 27, 2026

"Samantha Barnes" {signed and sealed}

Samantha Barnes, P.Eng.

SRK Consulting Limited

Signing Date: April 7, 2026

Effective Date: March 27, 2026

"John Kurylo" {signed and sealed}

John Kurylo, P.Eng.

SRK Consulting Limited

Signing Date: April 7, 2026

Effective Date: March 27, 2026

"Barry Carlson" {signed and sealed}

Barry Carlson, P.Eng., SME-RM

Forte Dynamics

Signing Date: April 7, 2026

Effective Date: March 27, 2026

"Russell Downer" {signed and sealed}

Russell Downer, P.Eng.

Opencountour

Signing Date: April 7, 2026

Effective Date: March 27, 2026



“Rachel Wyles” {signed and sealed}

Rachel Wyles, P.Eng.

WSP Canada Inc.

Signing Date: April 7, 2026

Effective Date: March 27, 2026

“Marc Rougier” {signed and sealed}

March Rougier, P.Eng.

WSP Canada Inc.

Signing Date: April 7, 2026

Effective Date: March 27, 2026

“Kim Ferguson” {signed and sealed}

Kim Ferguson, FAusIMM

WSP Canada Inc.

Signing Date: April 7, 2026

Effective Date: March 27, 2026

“Dachun (Davind) Jin” {signed and sealed}

Dachun (Davind) Jin, P.Eng.

WSP Canada Inc.

Signing Date: April 7, 2026

Effective Date: March 27, 2026

“William Richard McBride” {signed and sealed}

William Richard McBride, P.Eng.

WSP Canada Inc.

Signing Date: April 7, 2026

Effective Date: March 27, 2026

“Laura-Lee Findlater” {signed and sealed}

Laura-Lee Findlater, P.Geo.

Lorax Environmental Services Ltd.

Signing Date: April 7, 2026

Effective Date: March 27, 2026



CERTIFICATE OF QUALIFIED PERSON Charley Murahwi

I, Charley Murahwi, state that:

- (a) I am a Senior Economic Geologist at: Micon International Limited, 212 King St. W., Suite 501, Toronto, Ontario, M5H 1K5, Canada.
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of: 27 March 2026 (the “Technical Report”).
- (c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of the University of Rhodesia (1979) in Geology, [BSc.] and Rhodes University (1996) in Economic Geology [MSc.]. I am a certified professional geologist, registered with the Professional Geoscientists Ontario [PGO] reg. # 1618 and with PEGNL reg. #05662. I am also a FAusIMM (membership number 300395). My relevant experience after graduation and over for the purpose of the Technical Report is that I have practiced my profession continuously since 1980, with a focus on exploration, production and the evaluations of precious metals, porphyry systems and base metal deposits.
- (d) My most recent personal inspection of each property described in the Technical Report occurred on July 28, 2025, and was for a duration of 1 day.
- (e) I am responsible for Item(s) 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.18, 1.19, 1.20.1, 1.21.1, part of 3, 4 -11, part of 12.1, 12.2, 12.5, 14, 23, 24, 25.1, 26.1 & 27.2 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) My prior involvement with the property that is the subject of the Technical Report is as follows. I am responsible for the Technical Report titled “NI 43-101 Technical Report for the 2025 Mineral Resource Estimate Update on the Coffee Gold Project, Yukon, Canada” with an effective date of August 21, 2025.
- (h) I have read NI 43-101 and The Technical Report OR the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report OR the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Toronto, Ontario this 8th of April, Year.

“Charley Murahwi” {signed and sealed} Charley Murahwi, P. Geo.,

FAusIMM, Micon International Limited.

CERTIFICATE OF QUALIFIED PERSON Hannah Chiew

I, Hannah Chiew, state that:

- (a) I am a Professional Engineer at:
Ensero Solutions
#1480 - 1090 W Georgia St,
Vancouver, BC V6E 3V7
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of 27, March, 2026 and a report date of 8 April 2026 (the "Technical Report").
- (c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of University of British Columbia with B.A.Sc. in Chemical Engineering in 1997, a M.A.Sc. in Civil Engineering in 2011, and a P.Eng. in BC, Yukon, Ontario, Newfoundland and Labrador. My relevant experience after graduation and over 25 years for the purpose of the Technical Report includes water treatment projects in all phases of development and execution from conceptual, Front-End Engineering Design (FEED), prefeasibility study (PFS), feasibility study (FS), detailed design, construction, commissioning, and start-up.
- (d) I did not visit the property that is the subject of the Technical Report.
- (e) I am responsible for Item(s) 1.13.4, 1.21.6, 18.12, 21.3.9, 21.4.9, 21.6.5, 25.5.3 and 26.5.3 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) I have not had prior involvement with the property that is the subject of the Technical Report.
- (h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Port Moody, British Columbia this 7th of April, 2026.



2026-04-07

Hannah Chiew, P.Eng. Engineers Yukon #3259



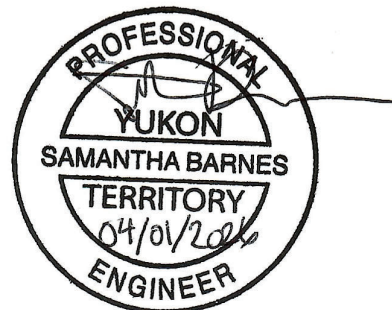
CERTIFICATE OF QUALIFIED PERSON **Samantha Barnes**

I, Samantha Barnes, state that:

- (a) I am a Principal Consultant and Water Resources Engineer at:
SRK Consulting (Canada) Inc.
2600-320 Granville Street
Vancouver, BC, V6C1S9, Canada
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of: March 27 2026 and a report date of April 8 2026 (the "Technical Report").
- (c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of Western Ontario and from University of British Columbia with a BSc in Civil and Environmental Engineering, Bachelor of Science, and a MEng in Civil Hydrotechnical Engineering. I am a registered Engineer in Yukon, BC, Saskatchewan and Ontario. My relevant experience after graduation and over the last 14 years for the purpose of the Technical Report includes a focus on mine water management specifically for mining projects located in North America, Europe, South America, and West Africa, with a significant focus of work completed in northern Canada and Yukon Territory.
- (d) The requirement for a site visit is not applicable to me.
- (e) I am responsible for Item(s) 1.13.2, 1.20.5.2, 1.21.5.2, 18.11, 21.3.8, 21.4.8, 25.5.2, 26.5.2 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) My prior involvement with the property that is the subject of the Technical Report is as follows. I have been involved in past water management infrastructure designs, past dam break analysis, and permitting support (from approximately 2018 to 2025) .
- (h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Whitehorse Yukon Canada this 1st of April, 2026.

Samantha Barnes, MEng, P.Eng





CERTIFICATE OF QUALIFIED PERSON John Kurylo

I, John Kurylo, state that:

- (a) I am a Principal Consultant and Geotechnical Engineer at:
SRK Consulting (Canada) Inc.
2600-320 Granville Street
Vancouver, BC, V6C1S9, Canada
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of: March 27 2026 and a report date of April 8 2026 (the "Technical Report").
- (c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of British Columbia and from Imperial College London with a BSc in Geological Engineering and a MSc in Civil Engineering, with a specialization in soil mechanics and engineering seismology. My relevant experience after graduation and over the last 15 year, for the purpose of the Technical Report includes, a focus on mine waste and water management [specifically for mining and civil infrastructure projects located in the United Kingdom, Europe, Central Asia, South America, Scandinavia, Africa, and North America (primarily in Canada)]
- (d) My most recent personal inspection of each property described in the Technical Report occurred on March 24 2026 and was for a duration of one day.
- (e) I am responsible for Item(s) 1.13.2, 1.13.3, 1.20.5.2, 1.21.5.2, 2.4, 18.11 (18.11.1, 18.11.2, 18.11.3), 18.13, 21.3.8, 21.4.8, 25.5.2, 26.5.2) of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) My prior involvement with the property that is the subject of the Technical Report is as follows. I have been involved in past water management infrastructure designs, past dam break analysis, and permitting support (from approximately 2018 to 2025) and assisted with some site investigations (linked to water management infrastructure) in 2018
- (h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Vancouver, British Columbia, Canada this 1st of April 2026.

John Kurylo, MSc, P.Eng



V1.2021



CERTIFICATE OF QUALIFIED PERSON Barry Carlson

I, Barry Carlson, state that:

- (a) I am a Principal Process/Metallurgical Engineer at:
Forte Dynamics (Part of SLR Consulting)
1730 S College Ave, Suite 205
Fort Collins, Colorado, 80525
- (b) This certificate applies to the technical report titled **NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada** with an effective date of: March 27, 2026 (the “Technical Report”).
- (c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Colorado State University with a B.S. Engineering, 1988, and also a registered member of SME, and member of ASCE. My relevant experience after graduation and over 38 years for the purpose of the Technical Report includes design and analyses of heap leach facilities including cost estimation.
- (d) My most recent personal inspection of each property described in the Technical Report occurred on August 2018 and was for a 2-day site visit. There has been no activity on site or material change to the site in regards to heap leach design since my visit.
- (e) I am responsible for Item(s) 1.12.2, 1.20.4.2, 17.2.4, 21.3.4, 21.4.4, 25.4.2, and 26.4.2 of the Technical Report.
- (f) I am an independent of the issuer as described in section 1.5 of NI 43-101.
- (g) My prior involvement with the property that is the subject of the Technical Report is as follows. Performed heap leach design and cost estimation for the facility.
- (h) I have read NI 43-101 and The Technical Report or the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, The Technical Report or the part of the Technical Report for which I am responsible , contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.



Dated at Fort Collins, Colorado this 01 of April, 2026.

A handwritten signature in black ink, appearing to read 'Barry Carlson', written over a horizontal line.

Signature of Qualified Person

Barry Carlson, P.E., P.Eng, QP, SME-RM

A handwritten signature in black ink, appearing to read 'Barry Carlson', written above a circular professional seal.



04-01-2026



CERTIFICATE OF QUALIFIED PERSON
Russell Downer, P. Eng.

I, Russell Downer, P Eng., certify that:

1. I am a Mine Engineer at:
Opencontour LLC
1195 E Princeton Ave
Salt Lake City, Utah, USA, 84105
2. This certificate applies to the technical report titled **"NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada"** with an effective date of: March 27, 2026 (the "Technical Report").
3. I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Queen's University with a Bachelor of Science in Mine Engineering, 1995. I am a member of the self-regulating association Professional Engineer of Ontario (No. 90544180). My relevant experience after graduation and over 30 years for the purpose of the Technical Report includes work on numerous open pit projects, within project engineering studies and within Mine Operations, on Mineral Reserve estimates, Mine Planning, and Mine cost estimates.
4. My most recent personal inspection of each property described in the Technical Report occurred on August 28, 2018 and was there for a duration of one day.
5. I am responsible for Items 1.11, 1.20.2, 1.21.2, 16, 25.2, and 26.2 of the Technical Report.
6. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
7. I have had prior involvement with all recent press releases related to this Technical report.
8. I have read NI 43-101 and the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
9. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Salt Lake City, Utah this 2nd of April, 2026

Russell Downer (Ontario PEO No. 90544180).





CERTIFICATE OF QUALIFIED PERSON Rachel Wyles

I, Rachel Wyles, state that:

- (a) I am an Environmental Engineer at: NI 43-101 Qp Certificate W
WSP
840 Howe
Street, 10th
Floor
BC, Canada
V6Z 2M1

- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of: March 27 2026 and a report date of April 8 2026 (the "Technical Report").

I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of Nottingham UK with a M.Eng. in Environmental Engineering and Resource Management. I am a registered Engineer in Yukon and BC. My relevant experience after graduation and over 20 years for the purpose of the Technical Report includes a focus on environmental assessment and environmental permitting for industrial and mining facilities.

- (d) The requirement for a site visit is not applicable to me.
- (e) I am responsible for Item(s) 1.15.1, 1.15.3, 20.1, 20.2, 20.4, 20.5, 25.6 and 27.4 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) I have not had prior involvement with the property that is the subject of the Technical Report.
- (h) I have read NI 43-101 and The Technical Report OR the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report OR the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Vancouver BC Canada this 7th of April, 2006.

Rachel Wyles, MEng, PEng





CERTIFICATE OF QUALIFIED PERSON Marc Rougier

I, Marc Rougier, state that:

- (a) I am a Senior Technical Director in the Mine Stability Group at:
WSP Canada Inc.
6950 Century
Mississauga, Ontario, Canada
- (b) This certificate applies to the technical report titled “NI 43-101 Technical Report for the 2025 Mineral Resource Estimate Update on Coffee Gold Project, Yukon Canada” with an effective date of: March 27, 2026 and a report date of 8 April 2026 (the “Technical Report”).
- (c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Queen’s University with a BSc in Geological Engineering, 1991 and a P.Eng in Ontario and NWT with reinstatement in Yukon pending. My relevant experience after graduation and over thirty-five years for the purpose of the Technical Report includes engineering geology and open pit slope design from scoping through operations to closure.
- (d) The requirement for a site visit is not applicable to me.
- (e) I am responsible for Item(s) 1.11.1, 16.1 and 26.2.2 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) I have not had prior involvement with the property that is the subject of the Technical Report.
- (h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the technical report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Oakville, Ontario this 2nd of April, 2026.

Marc Rougier, P.Eng (Ontario) Signed and Sealed.

Marc Rougier --
P.Eng. - PEO

Digitally signed by Marc Rougier --
P.Eng. - PEO
DN: cn=Marc Rougier -- P.Eng. -
PEO, c=CA, o=Certifico Pro,
ou=Professional Engineers Ontario
Date: 2026.04.07 18:40:25 -04'00'



CERTIFICATE OF QUALIFIED PERSON KIM FERGUSON

I, Kim Ferguson, state that:

- (a) I am a Director Mine Closure at:
WSP Canada Ltd
#1000 - 840 Howe Street (Robson
Court)
Vancouver, British Columbia, V6Z 2M1
- (b) This certificate applies to the technical report titled *Fuerte Announces A Positive Preliminary Economic Assessment For The Coffee Gold Project (22 February 2026)*, *Fuerte Files Technical Report For The Coffee Gold Project (8 April 2026)* and *NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of 27 March 2026 and a report date of 8 April 2026 (the "Technical Report")*].
- (c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Murdoch University with a Bachelor of Environmental Science (1998) and I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM # 338483). My relevant experience after graduation and over the past 26 years for the purpose of the Technical Report includes over 25 years of experience in the operational, corporate and consulting aspects of resources sustainability, approvals and governance, environmental and social impact assessment and management and closure across a broad geographic, commodity and operational status and scale spectrum. Since 2012, I have had a primary focus on mine closure, including strategic and tactical planning and execution, closure cost estimation, governance and multi-disciplinary integration. I have prepared or managed numerous strategic, regulatory and tactical closure plans and associated cost estimations across multiple jurisdictions and mine life phases, including in execution. I was also twice the Chair of the International Council of Mining and Metals Closure Working Group, aimed at the delivery of guidance for improved awareness and delivery of integrated mine closure across the global industry.
- (d) The requirement for a site visit is not applicable to me, as there was no relevant operational disturbance or landforms existing on the site pertaining to my area of responsibility.
- (e) I am responsible for Items 1.15.2, 1.20.6.1, 1.21.7.1, 20.3, 25.6.2, 26.6.1 (relevant to Closure, Decommissioning and Reclamation) of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) I have not had prior involvement with the property that is the subject of the Technical Report.
- (h) I have read NI 43-101 and Items 1.15.2, 1.20.6.1, 1.21.7.1, 20.3, 25.6.2, 26.6.1 (relevant to Closure, Decommissioning and Reclamation) of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Items 1.15.2, 1.20.6.1, 1.21.7.1, 20.3, 25.6.2, 26.6.1 (relevant to Closure, Decommissioning and Reclamation) of the Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Vancouver, British Columbia this 2 April 2026

[Signature of Qualified Person]

Kim
Ferguson

Digitally signed by Kim Ferguson
DN: cn=Kim Ferguson, c=CA,
o=WSP, ou=B. Env. Sc. FAusIMM
338483,
email=kim.ferguson@wsp.com
Date: 2026.04.02 11:23:19 -07'00'

Kim Ferguson - FAusIMM # 338483



CERTIFICATE OF QUALIFIED PERSON – Dachun (David) Jin

I, Dachun (David) Jin , state that:

- (a) I am a Team Leader, Metallurgy at:
WSP Canada Inc.
2 International Blvd
Toronto, Ontario M9W 1A2
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of 27 March 2026 and a report date of 8 April 2026 (the “Technical Report”).
- (c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Tianjin University with a bachelor’s degree in Chemical Engineering graduated in 1996. I am a registered member of Professional Engineers of Ontario (PEO), License #100145629. For the purpose of the Technical Report includes experience in metallurgical and mineral processing projects for various commodities, including gold, silver and base metals, encompassing all phases from conceptual design through to detailed engineering.
- (d) I did not visit the property that is the subject of the Technical Report.
- (e) I am responsible for Item(s) Item 11.9, 1.12 1.13.1, 1.20.1.3, 1.20.3, 1.20.4.1, 1.20.5.1, 1.21.3, 1.21.4, 1.21.5.1, 13, 17.1, 17.2.2.1, 17.2.3, 17.2.5- 17.2.8, 17.3, 18.1-18.10, 21.3.2, 21.3.3, 21.3.5-21.3.7, 21.4.2, 21.4.3, 21.4.5-21.4.7, 21.6.2-21.6.4, 25.3, 25.4.1, 25.5.1, 26.3, 26.4.1, 26.5.1, 27.1 and 27.3 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) I have not had prior involvement with the property that is the subject of the Technical Report.
- (h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Oakville, Ontario this 8th of April, 2026.

Dachun (David) Jin, P.Eng. PEO#100145629





CERTIFICATE OF QUALIFIED PERSON (William Richard McBride)

I, William Richard McBride, state that:

- (a) I am a Senior Principal Mining Engineer at:
WSP Canada Inc.
33 Mackenzie Street
Sudbury, ON, Canada, P3C 4Y1
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of: 27 March 2026 and a report date of 8 April 2026 (the “Technical Report”).
- (c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Queen’s University (Kingston) with a Bachelor of Science degree in Mining Engineering granted in 1973. I am a Registered Member of the Professional Engineers of Ontario (PEO), License Number 29888013. My relevant experience after graduation for the purpose of the Technical Report includes over 50 years of working as a mining engineer and consultant working on projects involving multiple commodities such as copper, gold, and nickel and projects involving public disclosure reporting.
- (d) I, William Richard McBride a WSP QP responsible for mining costs, market studies, and economic analysis, did not make a site visit as there is no pertinent information to examine on the site pertaining to his area of responsibility.
- (e) I am responsible for Items 1.1, 1.14, 1.16, 1.17, 1.20.7, 1.20.8, 2, parts of 3, 19, 21.1, 21.2.1, 21.3.1, 21.4.1, 21.6.1, 22, 25.7, 25.8 of the Technical Report.
- (f) I am independent of the issuer, Fuerte Metals Corporation, as described in section 1.5 of NI 43-101.
- (g) I have not had prior involvement with the property, that is the subject of the Technical Report.
- (h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury / Ontario this 8th day of April 2026.

William Richard McBride, P. Eng. PEO # 29888013





CERTIFICATE OF QUALIFIED PERSON LAURA-LEE FINDLATER

I, Laura-Lee Findlater, state that:

- (a) I am a Senior Hydrogeologist at:
Lorax Environmental Services Ltd.
305-1770 West 7th Avenue
Vancouver, BC V6J 4Y6
- (b) This certificate applies to the technical report titled NI 43-101 Technical Report for the 2026 Preliminary Economic Assessment on the Coffee Gold Project, Yukon, Canada with an effective date of 27 March 2026 and a report date of 8 April 2026 (the “Technical Report”).
- (c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of UBC with Bachelor of Science in Geological Sciences, 2001, and am a registered Professional Geoscientist with Engineers and Geoscientists BC and a Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists. My relevant experience after graduation and over 20 years of professional practice for the purpose of the Technical Report includes design, execution and reporting of groundwater baseline programs and directing numerical groundwater modeling studies in support of mine licensing and permitting.
- (d) My most recent personal inspection of each property described in the Technical Report occurred on May 13th, 2015 and was for a duration of 28 days.
- (e) I am responsible for Item(s) 20.6 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) My prior involvement with the property that is the subject of the Technical Report is as follows. I designed and supervised hydrogeological drilling programs at the property to establish the baseline groundwater monitoring network. I directed numerical groundwater modeling studies to simulate impacts of mining on the groundwater system. I have authored technical reports and plans related to groundwater submitted in support of the 2017 YESAB Project Proposal Submission and the 2023 Type A Quartz Mining Water Use Licence Application.

Dated at Vancouver, BC this 8th day of April, 2026.



Laura-Lee Findlater, P.Ge., EGBG Registration # 38298