

National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA

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LIST OF APPENDICES

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LIST OF ACRONYMS

AAL	American Assay Laboratory	ISO	International Standards Organization
A.R.S.	Aggregate Mine Land Reclamation Act	MDA	Mine Development Associates
ADEQ	Arizona Department of Environmental Quality	MIP	Maximum intensity projections
ADWR	Arizona Department of Water Resources	MPO	Mine Plan of Operations
Ag	Silver	MRDI	Mineral Resources Development Inc
Ai	Abrasion index	MSGP	Multi-Sector General Permit
AISC	All-In-Sustaining-Cost	MSO	Minable Stope Optimizer
ALS	ALS Laboratories	NaCN	Sodium Cyanide
AMEC	Association of Mining and Exploration Companies	NBM	Nevada Bureau of Mines
APP	Aquifer Protection Permit	NEPA	National Environmental Protection Act
ARD	Absolute relative difference	NI 43-101	National Instrument 43-101 Technical Report
ATF	Alcohol, Tobacco and Firearms	NN	Nearest neighbor
Au	Gold	NOI	Notice of Intent
AWQS	Aquifer Water Quality Standards	NPV	Net Present Value
AZG	Arizona Gold Corporation	NSPS	National Emission Standards for Hazardous Air Pollutants
BADCT	Best available control technology	OK	Ordinary Krige
BDL	Below detection	PAX	Potassium amyl xanthate
BLM	Bureau of Land Management	PbO	Lead Oxide
Bmsl	Below mean sea level	PEA	Preliminary Economic Assessment
BWi	Bond Work Index	PFS	Preliminary Feasibility Study
CAPEX	Capital Expense	PG	Professional Geologist
CCD	Counter-current decantation	ppm	Parts per million
CDH	Copperstone Drillhole	PRJ	Pearson, deRidder and Johnson, Inc.
CHF	Cemented Hydraulic Fill	QAQC	Quality Assurance/Quality Control
CHF	Cemented Hydraulic Fill	QP	Qualified Person
CIL	Carbon-in-leach	RBF	Radial Basis Function
CIM	Canadian Institute of Mining, Metallurgy and Petroleum	RC	Reverse Circulation
CRF	Cemented rock fill	RDI	Resource Development Inc.
CRIRSCO	Committee of Mineral Reserves International Reporting Standards	RF	Non-cemented rockfill
Cu	Copper	RMR	Rock mass rating
CV	Coefficient of variation	RQD	Rock Quality Data
DCU	Drillhole Copperstone	SGLD	Sabre Gold Mines Corporation
DOR	Decision of Record	SME-RM	Society of Mining and Metallurgy and Exploration – Registered Member
FONSI	Finding of No Significant Impact	SRM	Standard Reference Material
FW	Footwall	SWS	Schlumberger Water Services
g/L	Grams per litre	TOMCL	Trans Oceanic Mineral Company Ltd.
g/mt	Grams per metric tonne	stpd	Short Tons per day
GCMP	Ground Control Management Plan	stph	Short Tons per hour
HEA	Hanlon Engineering and Associates	TSF	Tailings storage facility
HRC	Hard Rock Consulting	TSX	Toronto Stock Exchange
HW	Hanging wall	UPFX	Upper Fracture
ICMC	International Cyanide Management Code	USGS	United States Geological Survey
ID	Inverse distance	VLF	Very low frequency
IDS	International Directional Services	WELNAV	Wellbore Navigation Inc.
IP	Induced Polarization	WOL	Whole Ore Leach
IRR	Internal Rate of Return		

1 EXECUTIVE SUMMARY

1.1 Introduction

Mining Americas Inc. (“MAI”, the Owner) (TSX-V: MAI) is a company focused on acquiring, exploring, and developing base and precious metals projects in North America. Mining Americas Inc. acquired the Copperstone property under the terms of the business combination with Sabre Gold Mines Corporation (“SGLD”) that closed on January 27, 2025. SGLD was formed in August of 2021 with the merger of Arizona Gold Corp. (“AZG”) and Golden Predator Mining Corp. (“Golden Predator”). AZG was previously known as Kerr Mines, Inc. (“Kerr”) and completed the name change to AZG in December of 2020. Kerr acquired the Copperstone Project (the “Project”), a historically productive high-grade gold mine located in La Paz County, Arizona in 2014. The mine is fully permitted with significant mining infrastructure, mineral resources and processing infrastructure in place.

The Copperstone Project encompasses approximately 15.8 square miles of surface area and mineral rights in La Paz County, Arizona, roughly 19 miles north of the town of Quartzsite. The Project area covers all or portions of Sections 17 through 23 and 28 through 32, T6NR19W; Sections 1, 2, 10 through 14, 22 through 27, and 36, T6NR20W; and Section 24, 25, and 36, T7NR20W, Gila and Salt River Meridian. The approximate geographic center of the Project area lies at 33°52’6”N latitude, 114°17’42”W longitude. Map coverage of the Project area is provided by the 1:24,000-scale, Moon Mountain SE and Moon Mountain NE, U.S.G.S. 7.5-minute topographic quadrangles.

This Pre-Feasibility study (PFS) outlines a restart plan for the Copperstone Mine starting at 600 stpd in year 1, increasing to 1000 stpd from year 2 to year 6 and producing approximately 291,000 oz Au over that period. This Pre-Feasibility is based on a Proven and Probable Mineral Reserve of 2.13 million tons at 0.14 oz/ton (4.87 g/t). In addition to the reserves, Copperstone has Inferred resources of 441,694 tons at 0.12 oz/ton (4.04 g/t).

This report was prepared in accordance with the requirements and guidelines set forth in NI 43-101 Companion Policy 43-101CP and Form 43-101F1 (June 2011), and the mineral resources presented herein are classified according to Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Definition Standards - For Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on November 19, 2019. The mineral resource statement reported herein is based on all available technical data and information as of December 1, 2025. The effective date of this report in full is May 27, 2026.

1.2 Key Outcomes

The NI 43-101 highlights are as follows:

- The plan uses Proven and Probable Reserves of 2,132,000 tons at 0.14 oz/ton (4.87 g/t) – 303,000 oz Au
- Capital expenditure of 57.2M
- Mine life (LOM) of 6.5 years
- LOM payable production of 292,000 oz Au
- LOM Operating costs (C1) of \$873/ oz Au, \$1,338/ oz Au
- \$341.5 M NPV 5 After Tax, IRR 110.5%, Payback 1.13 years (\$US 3,500/oz Au)

1.3 Reliance on other Experts

The Qualified Persons (QP) opinions contained herein are based on information provided by Mining Americas Inc. and by others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for it.

Non-QP specialists relied upon for specific advice includes:

- Dorsey Whitney LLP (Dorsey)
- Stantec Consulting Services Ltd.

1.4 Property Description and Ownership

The Copperstone Project is located in La Paz County, Arizona, approximately 31 km (19 miles) north of Quartzsite. The Project encompasses approximately 15.8 square miles (about 10,137 acres or 4,102 hectares) of mineral rights and surface area centered at approximately 33°52'06"N latitude and 114°17'42"W longitude. Figure 1-1 shows the location of the project.

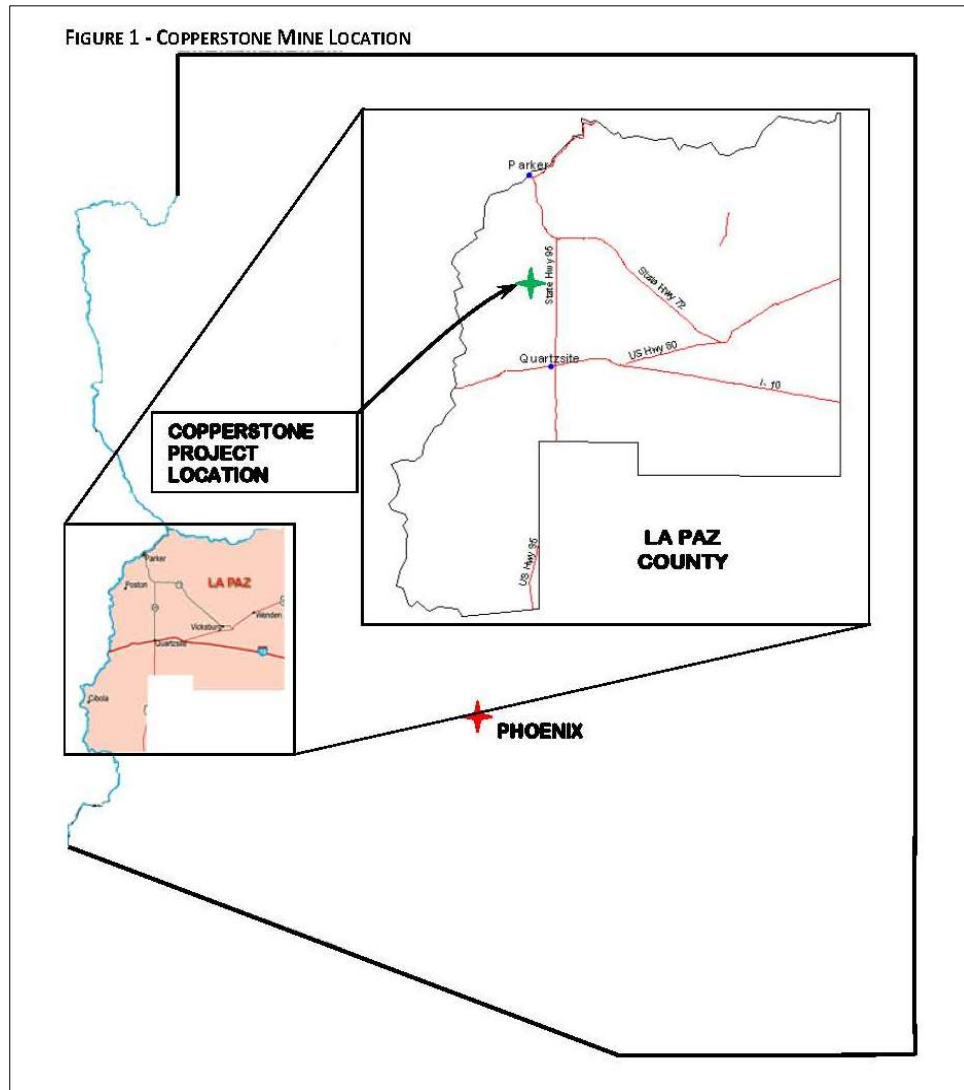


Figure 1-1 Copperstone Property Location

The Project consists of 546 federal unpatented mining claims controlled by Mining Americas Inc. (MAI) through its wholly owned subsidiary, American Bonanza Gold Corp. Following completion of the acquisition of Sabre Gold Mines Corp. in January 2025, MAI became the sole owner of the Copperstone Project.

MAI holds a 100% leasehold interest in the Project under a long-term lease agreement with the Patch Living Trust. The lease extends to June 12, 2026 and is renewable at MAI's option for successive ten-year terms under the existing terms and conditions. The lease requires annual rental payments of US\$40,000 and a 1.5% gross production royalty.

The Project is subject to a precious metals stream agreement with Star Royalties, under which Star is entitled to receive 4% of annual gold production at a purchase price equal to 25% of the prevailing spot gold price. The agreement remains in effect following the acquisition by Mining Americas.

The existing disturbance area is subdivided into six operational zones based on geological, structural, and mineralization characteristics that are referenced throughout this report.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.5.1 Accessibility

The Copperstone Project is located in western Arizona, approximately 31 km (19 miles) north of Quartzsite and is accessible year-round via U.S. Highway 95 and Cyprus Mine Road, a well-maintained gravel road extending approximately 9 km (5.5 miles) from the highway to the Project site.

1.5.2 Climate

The Project is situated within the Basin and Range physiographic province on the relatively flat terrain of the La Posa Plain adjacent to the Dome Rock Mountains. Elevations range from approximately 650 to 825 feet (198 to 251 m) above sea level. The area is characterized by sparse desert vegetation, low relief, and arid climatic conditions.

The regional climate is typical of the Sonoran Desert, with hot summers, mild winters, and low annual precipitation averaging approximately 90 mm (3.5 inches). Climatic conditions permit year-round exploration, construction, mining, and processing activities.

1.5.3 Infrastructure and Local Resource

The Project benefits from proximity to established regional communities and infrastructure. Quartzsite, located 31 km south of the Project, provides basic services and accommodations, while Parker, the La Paz County seat, is located approximately 32 km to the north. Major supply centers, transportation infrastructure, and skilled labour are available in Phoenix, approximately 200 km (125 miles) east of Quartzsite, and Yuma, approximately 130 km (80 miles) south.

Existing site infrastructure includes administrative offices, a crushing circuit, a mill facility (with mechanical equipment removed), warehouse facilities, maintenance shops, assay laboratory buildings, change house facilities, core storage areas, septic systems, and employee accommodation hook-ups. The site is serviced by a commercial 69 kV power line and on-site electrical substation.

Water for Project activities is supplied from three permitted groundwater wells and stored in a 375,000-gallon tank. Groundwater extraction rights are authorized by the Arizona Department of Water Resources. Existing surface rights, rights-of-way, and utility infrastructure are considered adequate to support the proposed mining, processing, tailings storage, and waste management facilities.

1.6 History

The Copperstone property has been explored and operated by a succession of companies since the first recorded interest as a copper prospect in 1968. Cyprus (as Cyprus-Amoco) identified Copperstone as a gold target in 1980 and, between 1980 and 1986, completed extensive diamond and RC drilling that led to construction of a 2,500 stpd carbon-in-pulp mill and open-pit mining from 1987 to 1993. Cyprus reported historic production of 514,000 oz gold from 5.6 Mt grading 0.089 oz/ton. Following Cyprus's departure, the property was leased and explored successively by Santa Fe Pacific Gold (1993), Royal Oak Mines (1995–1997), and Asia Minerals Inc. (1998–2001), which later became American Bonanza Gold Mining Corp. American Bonanza advanced the project through feasibility studies and constructed a 450 stpd flotation mill, mining the D zone from two declines from 2012 to 2013 and producing approximately 16,900 oz gold from 163,000 tons grading 0.104 oz/ton. Arizona Gold Corp. (formerly Kerr Mines) acquired the project in 2014 and conducted further surface and underground drilling between 2015 and 2021. Sabre Gold Mines Corp. ("SGLD") was formed in 2021 through the merger of Arizona Gold Corp. and Golden Predator Mining Corp. and completed RC infill drilling later that year. Mining Americas Inc. ("MAI") acquired the Copperstone property through its business combination with SGLD, which closed on January 27, 2025.

Historical exploration at Copperstone includes multiple generations of ground geophysical surveys (magnetics, electromagnetics, IP/resistivity, gravity, and seismic) conducted between 1981 and 2007, surface and underground geological mapping, and channel and core sampling programs. In 2021, sampling and metallurgical testing of the Copperstone tailings was completed by Ore Pro, indicating an average grade of 0.81 g/t Au and an estimated cyanide leach recovery of 68%.

Historic drilling at Copperstone totals 1,486 holes for 634,018 ft (475 core holes, 831 RC holes, 18 RC/core holes, and 162 holes of unknown type drilled for production), completed by ten operators between 1984 and 2021. Drilling methods, core handling, sample preparation, and assay laboratories varied by operator and era; available documentation on sampling, security, and QA/QC protocols is correspondingly variable, ranging from minimal records for early Cyprus, Santa Fe, and Royal Oak drilling to comprehensive, modern QA/QC programs (duplicates, blanks, and certified reference materials) for the more recent Asia Minerals, American Bonanza, Arizona Gold Corp., and Sabre Gold Mines drilling campaigns. Independent reviews (AMEC, 2006; Telesto, 2011; and the current QP) have assessed the reliability of historic assay data, generally finding acceptable accuracy and precision for the more recent and better-documented programs, with lower confidence assigned to certain early Royal Oak assays subsequently addressed through check-assay programs. A formal drillhole database for the project was established by SGLD using MxDeposit® software in 2024.

All previously reported historic mineral resource and reserve estimates predate current NI 43-101 and CIM reporting standards and are superseded by the current Mineral Resource Estimate (Section 14) and maiden Mineral Reserve (Section 15) presented in this report.

1.7 Geology and Mineralization

The Copperstone Project is situated at the northern tip of the Moon Mountains in west-central Arizona, regionally within the Basin and Range geo-physiographic province, and within the westernmost extent of the Whipple-Buckskin-Rawhide detachment system. The Whipple-Buckskin-Rawhide detachment system is centrally located within the Maria fold and thrust belt (Reynolds et al., 1986), which extends from southeastern California to central Arizona. Mid-Tertiary low-angle normal faults (detachment faults) are recognized as significant regional structures in this portion of the Basin and Range, where major detachment faults locally overprint earlier Cretaceous thrust faults. In general, mylonitic foliations are low-dipping and contain well-developed northeast-plunging mineral lineations. Upper plate rocks as young as mid-Tertiary dip moderately to the southwest and are cut by northeast-dipping normal faults.

In the vicinity of the Project area, the Moon Mountain detachment fault carries sedimentary and volcanic rocks of Paleozoic, Mesozoic, and Tertiary age over a ductilely deformed footwall consisting primarily of granitic intrusive rocks. The top of the granitic lower plate rocks are marked by the brecciated Copper Peak granite, which is exposed over an area of roughly 2 km² surrounding and to the south of Copper Peak, in the northeastern part of the Moon Mountains. The northern margin of this unit is truncated by the Moon Mountain detachment fault. A weakly to strongly developed tectonic fabric is present over much of the exposed extent of the granite and is characterized by flattened and stretched quartz grains and deformed potassium feldspar.

Gold mineralization at Copperstone is largely restricted to the Copperstone fault (also referred to as the Copperstone shear or the Copperstone structure), a shallow angle, northeast-dipping, semi-planar zone of shear that accommodated both Cretaceous compressional (thrust), and later mid-Tertiary extensional (detachment) deformation. -The bulk of the gold historically produced from the Copperstone mine is hosted by the Copperstone structure. The structure strikes about N30° to 60°W and dip from 20° to 50° to the northeast. The associated fault zone ranges from 45 ft to 185 ft in width with characteristic fault gouge, multi-phase breccia textures, shear fabric, and intense fracture sets across this width. Dr. Richard Sillitoe (2025) states that gold mineralization was likely introduced during the late Cretaceous (Laramide) compressional stage of deformation along the Copperstone fault.

1.8 Deposit Types

The Copperstone deposit is currently interpreted as a structurally controlled, multistage, low-angle fault-hosted gold deposit associated with Late Cretaceous compressional deformation and subsequent mid-Tertiary extensional reactivation.

Previous workers proposed alternative genetic models. Salem (1993) interpreted Copperstone as a detachment fault-related epithermal gold system, suggesting that mineralization formed from ascending saline hydrothermal fluids circulating along regional detachment-related structures. In this model, gold deposition was triggered by fluid boiling and mixing within structurally prepared zones along the Copperstone fault and associated high-angle faults. However, subsequent reviews by Sillitoe (2025), and later Segerstrom (this report), concluded that available geological evidence does not conclusively support

classification of the deposit as a special detachment fault-related epithermal deposit type. Sillitoe further suggested that gold mineralization is more likely related to Cretaceous thrust-related structural development, with later modification during extensional tectonism.

Hydrothermal alteration associated with mineralization is characterized by chloritization, silicification, iron-oxide development (specularite and hematite), and localized potassic alteration. Gold mineralization occurs principally within brecciated and sheared zones along the Copperstone fault system and associated secondary structures.

Paragenetic studies indicate multiple stages of hydrothermal activity. Early mineralization consists of quartz-chlorite-specularite-hematite assemblages associated with gold deposition, followed by later quartz-adularia-carbonate-fluorite assemblages containing variable copper oxide minerals and minor sulfides. A late-stage barren quartz-fluorite-barite assemblage represents the final hydrothermal event.

Gold occurs primarily as native gold within quartz veins and breccia matrices and locally as inclusions within quartz. Fluid inclusion studies reported by Salem (1993) indicate mineralizing fluids with moderate to high salinities and estimated deposition temperatures averaging approximately 290°C, decreasing during later stages of the hydrothermal system. These data are consistent with hydrothermal fluid flow along major fault structures, although the precise genetic classification of the deposit remains a subject of ongoing interpretation.

1.9 Status of Exploration

The first recorded commercial interest in the Copperstone property was as a copper prospect in 1968. Charles Ellis of the Southwest Silver Company (“Southwest Silver”) controlled the Continental Silver claim group from 1968-1980. Newmont Gold Company (“Newmont”) leased the property in 1975. A geophysical survey was conducted and one drillhole completed in an attempt to verify porphyry copper mineralization. The attempt was unsuccessful.

In 1980, Southwest Silver drilled six rotary holes with unknown results and then dropped the claims. Cyprus identified the Copperstone property as a gold target and undertook a drilling campaign from 1980 to 1986. Cyprus began baseline, financial and metallurgical studies that led to mine design, initial construction and a partially completed decline in 1986.

In 1987, Cyprus commissioned construction of a 2,500 ton/day carbon-in-pulp mill and started open-pit mining. The mine was designed, constructed, and operated as a zero-discharge facility (Miller et al., 1994). Mining continued until 1993 when the pit neared the groundwater table, which was the limit of the original mining permits. Ackerman (1998) reported production by Cyprus at Copperstone of 514,000 oz of gold from 5,600,000 metric tonnes of mill feed grading 0.089 oz/ton Au.

Santa Fe Pacific Gold Corporation (“Santa Fe”) leased the property in 1993 and completed 12,500 ft (3,810 m) of RC drilling on seven exploration targets. Gold mineralization was encountered in one hole in the footwall of the Copperstone fault.

Royal Oak Mines (“Royal Oak”) drilled a total of 28,413.5 ft (8,660 m) in 34 holes between 1995 and 1997. Several high-grade gold intercepts to the north and east of the open pit showed potential for underground mining.

Asia Minerals drilled 15 holes (A98-1 to 15) in November 1998 for a total of about 10,979 ft (3,346 m). Each hole was drilled with RC methods from the surface to a predetermined depth and then core drilled through the target interval. The drilling program was designed to explore the C and D zones (MRDI, 1999). Golder Associates and MRDI Canada completed a scoping level study after the 1998 drilling program was completed.

Asia Minerals drilled 11 more holes in early 2000. Total footage was 8,609 ft (2,624 m). Holes were designed to test the strike length of the D zone, with the best intercept in hole A00-10 which assayed 0.943 oz/ton Au over 10.5 ft (3.2 m).

On March 4, 2002, American Bonanza gained control of a 100% equity interest in Copperstone subject only to the royalty schedule payable to the Patch Living Trust. American Bonanza announced on May 5, 2003, that significant high-grade gold mineralization was sampled in the decline in the D zone. In June 2003, an underground drill station was completed. Drilling began in July, and by May 17, 2004, American Bonanza had drilled 32 underground core holes in the D zone for a total of 9,208 ft (2,807 m).

American Bonanza continued drilling in 2004, including underground drilling from a drill bay in the exploration decline. Additional drilling was completed in 2006 and 2007. A variety of studies and reports were commissioned by American Bonanza between 2007 and 2010, culminating in a feasibility study, including an updated mineral resource estimate, completed in 2010. American Bonanza maintained control of the Copperstone Project until AZG’s (as Kerr Mines, Inc.) acquisition in June of 2014.

In 2015, AZG (as Kerr Mines, Inc.) completed 4 core drillholes targeting the Footwall zone totaling 3,045 ft (928 m). In 2017, AZG (as Kerr Mines, Inc.) drilled 72 core holes totaling 19,380 ft (5,907 m) and 11 RC holes totaling 7,360 ft (2,243 m). The 2017 drilling targeted the Footwall zone along strike and the A/B zone down dip from surface. The underground drilling confirmed historic drilling results and extended gold mineralization up and down dip in the D zone. In 2019, one hundred RC drillholes totaling 17,020 ft (5,188 m) were drilled by AZG (as Kerr Mines, Inc.). From September 2020 through January 2021, AZG completed 21 drillholes totaling 16,625 ft (5,067 m) from surface to define the Footwall zone, step out from gold mineralization in the C zone and A/B zone, and collect metallurgical samples in the A/B zone. Concurrently, between November 2020 through April 2021, AZG completed 31 core drillholes totaling 8,556 ft (2,608 m) from underground drilling stations. Another 13 diamond core drillholes totaling 1,093 ft (333 m) were completed by AZG in April 2021 targeting expected gold mineralization in order to support and guide follow up reverse circulation drilling on close-spaced centers.

SGLD continued the infill drilling program by completing 85 RC drillholes between October and December 2021 totaling 9,855 feet (3,004 meters). The goals of the infill drilling program were to gain an understanding of short-range variability in gold mineralization, test and confirm grade control procedures, develop production modeling methods, and to gain information to support stope design for trial mining purposes.

In March of 2021, Ore Pro was engaged to perform the sampling and metallurgical testing of the Copperstone tailings.

“The average of the 38 tailings samples by fire assay is 0.81 g/t Au with a range of 1.81 to 0.55 g/t Au. The average 24-hour agitated cyanide soluble copper, silver, and copper values are 0.57 g/t Au, 0.15 g/t Ag, and 598 g/t Cu. Presuming the gold fire assay result for each sample is approximately the value of the agitated cyanide head sample then the cyanide soluble gold value gives an estimation of the percent gold recovery of each sample under these leaching conditions. The cyanide leach recovery is estimated as 68% Au recovery. The cyanide soluble gold recovery ranges from 93-percent to 1-percent.” “Cyanide leach tests were performed on six of the tailings samples to determine the gold recovery in a 24-hour leach cycle.” “Reviewing these results show two obvious trends. First the calculated gold recoveries are similar from tailings samples taken from different parts of the impoundment. Gold recoveries range from 81.8 % for the lowest grade sample (0.48 g/t Au) to 92 % for the highest-grade sample (1.02 g/t Au). The other important result is that all the samples leached had about a 0.11 g/t Au leached tailings assay even though the calculated head grade of the samples ranged from 0.48 g/t Au to 1.02 g/t Au (Ore Pro, 2001).”

In 2017, AZG (as Kerr) commenced logging new drillhole lithology, alteration, mineralization, and structural information into a simplified and standardized format. SGLD continued the use of this format in its RC infill drilling program. Efforts have been made to bring historical drilling information into the same format. SGLD established a formal drillhole database for the Project using MxDeposit® software in 2024.

1.10 Drilling

No drilling has been completed at the Copperstone Project on behalf of Mining Americas Inc. All geological interpretations, mineral resource estimates, and technical assessments presented in this report are based on historical drilling programs completed by previous owners and operators. Historical exploration activities are described in detail in Section 6, with drillhole collar locations and drilling information provided in Figures 10-1 through 10-3 and Appendix B - Section 9 Drill Hole Collar Locations.

1.11 Sample Preparation, Analysis and Security

No additional sampling has been conducted at the Copperstone Project by or on behalf of Mining Americas Inc.; all sampling data are derived from historical programs completed by previous owners and operators and are described in Section 6. Although documentation for sample preparation, QA/QC, and security is incomplete for some legacy drilling programs, verification through re-assaying and duplicate sampling supports the reliability of the Cyprus, Santa Fe, and Royal Oak datasets for mineral resource estimation. Sampling completed by Asia Minerals, American Bonanza (excluding 2012–2013 production drilling), and post-2015 operators is supported by adequate QA/QC documentation and is considered suitable for inclusion in the mineral resource estimate.

1.12 Data Verification

A three-day site inspection was completed by the Qualified Person (QP) in January 2025 to review surface and underground geology, verify drillhole collar locations, inspect core and RC sample storage facilities, and evaluate the geological model. Field observations confirmed the reported lithologies, alteration, and mineralization styles, and verified that drillhole collar locations are sufficiently accurate for mineral resource estimation. The QP interprets the principal mineralized structure as a low-angle fault or shear zone, rather than a detachment fault, based on underground observations.

An independent audit of the historical assay database included verification of 4,871 sample intervals (5.35% of the database) against original laboratory certificates and drill records. The review confirmed that historical sampling, analytical procedures, and data management were generally consistent with industry standards. Isolated transcription and database export errors were identified, quantified, and remediated prior to mineral resource estimation. The QP considers the validated assay database to be sufficiently accurate and reliable for mineral resource estimation, while recommending a comprehensive review of the historical database to recover a limited number of high-grade assays omitted through historical transcription and database conversion errors.

1.13 Mineral Processing and Metallurgical Testwork

The Copperstone Project is supported by an extensive metallurgical database developed through multiple testing programs conducted between 1986 and 2022 by several independent laboratories and metallurgical consultants. These programs have evaluated a range of processing alternatives, including whole-ore cyanidation, flotation, gravity concentration, concentrate leaching, comminution, thickening, filtration, and cyanide destruction. The testwork has been completed on representative samples from the principal mineralized zones and provides a robust basis for process design and recovery estimation at the Pre-Feasibility Study level.

Results from the most recent metallurgical programs confirm that the Copperstone mineralization is highly amenable to conventional cyanide leaching following crushing and grinding. Gold extractions consistently exceeded 95% under optimized conditions, with recoveries approaching 98% for representative composite samples. Testwork also demonstrated that cyanide consumption is primarily influenced by the presence of cyanide-soluble copper, which varies between mineralized zones but can be effectively managed within the proposed process design.

Comminution testing indicates that the mineralized material exhibits medium hardness characteristics and is suitable for conventional grinding technologies. Additional testwork on thickening, filtration, carbon adsorption, carbon stripping, and cyanide destruction has provided the engineering design criteria required to support process plant design, reagent consumption estimates, and tailings management strategies.

The metallurgical investigations completed to date provide a high degree of confidence in the selected processing flowsheet and support the recovery assumptions and operating parameters adopted for this Pre-Feasibility Study.

1.14 Mineral Resource Estimate

A preliminary Mineral Resource Estimate ("MRE") for the Copperstone Project was prepared and verified by Scott Zelligan, P.Geo., with an effective date of December 1, 2025. The estimate incorporates drilling completed through December 1, 2025 and is based on a validated database containing assay, geological, collar, and survey information from 1,219 drill holes.

1.14.1 Resource Estimation Methodology

Mineralized domains were defined using indicator-based modelling rather than manually interpreted wireframes to better represent the variable geometry of the structurally controlled gold mineralization. Gold grade thresholds of 0.1, 0.2, 1.0, and 2.0 g/t Au were used to establish mineralized envelopes suitable for both open-pit and underground mining evaluations.

Drill hole assay data were composited to 5-foot intervals and evaluated using domain-specific statistical analyses. Grade capping was applied on a domain-by-domain basis to limit the influence of high-grade outliers and improve estimation stability. Capped gold grades were estimated using inverse distance squared (ID²) interpolation within hard-boundary mineralized domains.

A rotated block model was constructed using parent block dimensions of 20 ft × 20 ft × 5 ft. Density values were assigned by lithology using historical specific gravity data. Previously mined open-pit and underground voids were modelled and excluded from the resource estimate.

1.14.2 Resource Classification

Mineral Resources were classified in accordance with the CIM Definition Standards (2014) and CIM Best Practice Guidelines (2019), incorporated by reference into NI 43-101. Classification was based primarily on drill spacing, geological continuity, data quality, and confidence in the estimation.

Classification criteria included:

- Measured Resources: drill spacing generally not exceeding 82 ft × 82 ft;
- Indicated Resources: drill spacing generally not exceeding 164 ft × 164 ft; and
- Inferred Resources: drill spacing generally not exceeding 200 ft × 200 ft.

Model validation included visual inspection, statistical comparison of block estimates and informing composites, histogram analysis, and sectional reviews, all of which indicated that the block model adequately reflects the distribution and continuity of mineralization.

1.14.3 Mineral Resource Statement

The Mineral Resource Estimate (Table 1-1) is reported using optimized underground mining shapes generated with Deswik Stope Optimizer and a cut-off grade of 2.2 g/t Au (0.064 oz/ton Au). Resource reporting assumptions include a gold price of US\$3,000/oz, underground mining costs of US\$102.87/ton, processing costs of US\$42.64/ton, G&A costs of US\$16.39/ton, metallurgical recovery of 95%, refining costs of US\$12/oz, and a minimum mining width of 9 ft.

The Qualified Person considers the reported Mineral Resources to have reasonable prospects for eventual economic extraction. Previously mined areas have been excluded from the estimate. Mineral Resources are inclusive of Mineral Reserves where applicable and are reported in accordance with NI 43-101 disclosure requirements.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate may be materially affected by mining, metallurgical, economic, environmental, permitting, legal, social, and other modifying factors. Continued drilling has the potential to upgrade portions of the Inferred Mineral Resource category to higher-confidence classifications.

Table 1-1 The Copperstone Resource Estimate

Classification	Tons (tons)*	Tonnes (tonnes)*	Au Grade (oz/ton)	Au Grade (g/t)	Contained Au (troy ounces)*
Measured*	2,423,000	2,198,000	0.15	5.08	359,000
Indicated*	2,046,000	1,856,000	0.13	4.54	271,000
M&I*	4,469,000	4,054,000	0.14	4.83	630,000
Inferred*	442,000	401,000	0.12	4.04	52,000

*Notes:

1. The effective date of the Mineral Resource Estimate is December 1, 2025
2. The Mineral Resource Estimate has been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) incorporated by reference into NI 43-101, and CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing or other relevant issues.
3. The mineral resource is reported at an underground mining cut-off of 0.064 oz/ton (2.2 g/t Au) within estimated blocks which meet the criteria of a minable shape (minimum mining width of 9ft). The cut-off is based on the following assumptions: a gold price of \$3,000/oz; assumed mining cost of \$102.87/ton (\$113.16/tonne), process costs of \$42.64/ton (\$46.90/tonne), general and administrative and property/severance tax costs of \$16.39/ton (\$18.03/tonne), refining and shipping costs of \$12.00/oz, a metallurgical recovery for gold of 95%, and a 4.5% gross royalty.
4. Resources are reported using at a minimum overall gold cut-off grade of 0.064 oz/ton Au (2.2 g/t Au). Internal dilution is incorporated into the MSO shape volumes as part of the estimated quantities and grades.
5. Previously mined areas were modelled based on all available records and were excluded from reporting of this mineral resource.
6. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Reserves are estimated from Measured and Indicated Mineral Resources. Inferred Mineral Resources are not included in the Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and may be materially affected by modifying factors including but not restricted to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

8. There has been insufficient exploration to define the Inferred Resources tabulated above as an Indicated or Measured Mineral Resource, however, it is reasonably expected that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. There is no certainty that any part of the Mineral Resources estimated will be converted into Mineral Reserves.
9. Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.
10. Scott Zelligan, P. Geo. is responsible for reviewing and approving the Copperstone mine underground Mineral Resource Estimate. Mr. Zelligan is a Qualified Person ("QP") as set out in NI 43-101

1.15 Mineral Reserve Estimate

The Mineral Reserve Estimate for the Copperstone Project was prepared in accordance with the CIM Definition Standards (2014) incorporated by reference into NI 43-101. Measured Mineral Resources were converted to Proven Mineral Reserves and Indicated Mineral Resources were converted to Probable Mineral Reserves. Inferred Mineral Resources were excluded from reserve estimation and treated as waste.

1.15.1 Reserve Conversion Methodology

Mineral Reserves were estimated using optimized underground mining shapes generated with Deswik Stope Optimizer (Deswik SO) based on a mechanized cut-and-fill mining method. Optimized stope envelopes were subsequently reviewed and refined to create practical mining shapes consistent with the proposed mining method and geotechnical constraints.

The reserve estimate assumes nominal stope dimensions of approximately 10 ft mining height, 14 ft average mining width, and a minimum mining width of 9 ft. Vertical wall orientations were utilized in the stope design process.

1.15.2 Modifying Factors

The reserve estimate incorporates modifying factors consistent with industry practice for mechanized cut-and-fill mining in similar geotechnical conditions. External dilution of 10% at zero grade was applied to stope shapes, while a mining recovery factor of 95% was applied to contained metal. Internal dilution associated with sub-cut-off material is inherently included within the optimized stope shapes.

The reserve cut-off grade was derived using the following key assumptions:

- Gold price: US\$3,000/oz
- Mining cost: US\$102.87/ton mined
- Processing cost: US\$42.64/ton milled
- G&A cost: US\$16.39/ton milled
- Refining and selling cost: US\$12.00/oz
- Metallurgical recovery: 95%
- Royalty burden: 3% of gross revenue
- External dilution: 10%

Based on these assumptions, a reserve cut-off grade of 0.0642 oz/ton Au (2.2 g/t Au) was applied.

1.15.3 Mineral Reserve Statement

The Mineral Reserve Estimate has an effective date of December 1, 2025 and is summarized in Table 1-2.

Table 1-2 The Copperstone Mineral Reserve Estimate

Classification	Tons (tons)	Tonnes (tonnes)	Au Grade (oz/ton)	Au Grade (g/t)	Contained Au (troy ounce)
Proven	1,159,000	1,052,000	0.15	5.08	172,000
Probable	972,000	882,000	0.14	4.61	131,000
Total (Proven & Probable)	2,132,000	1,934,000	0.14	4.87	303,000

Notes to Mineral Reserve estimate:

1. The effective date for the Mineral Reserve is December 1, 2025
2. The Mineral Reserve Estimate has been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) incorporated by reference into NI 43-101.
3. Costs were based on the operating costs ranges as outlined in Table 15-3.
4. Mineral Reserves were estimated using a gold price of \$3,000/oz, metallurgical recovery of 95%, and a cut-off grade of 0.0642 oz/ton Au (2.2 g/t Au). Mining recovery of 95% and external dilution of 10% at zero grade applied.
5. Inferred mineral resources are not mineral reserves and do not have demonstrated economic viability. Inferred resources are excluded from this reserve estimate and treated as waste.
6. Numbers are rounded to the nearest ton or ounce; rounding may cause minor discrepancies in totals.
7. Reserves are contained within the measured and indicated mineral resources. No additional material has been added.
8. Peter Szkilnyk, P.Eng., is responsible for reviewing and approving the Copperstone mine underground Mineral Reserve Estimate. Mr. Szkilnyk is a Qualified Person ("QP") as set out in the NI 43-101

1.15.4 Factors Affecting Mineral Reserves

The Mineral Reserve Estimate is sensitive to changes in key technical and economic assumptions, including gold price, operating costs, metallurgical recovery, mining recovery, dilution, and geotechnical conditions. A sustained reduction in gold price or increases in mining and processing costs could reduce economically recoverable reserves.

Reserve estimates also depend on the achievement of the assumed metallurgical recovery, mining recovery, and dilution factors during operations. Variations in ground conditions may affect stope dimensions, dilution levels, and ground support requirements. In addition, future changes to permitting requirements, environmental regulations, or operating conditions could impact the economic viability of the reserves.

Based on the information available at the time of this study, the Qualified Person is not aware of any material legal, environmental, permitting, social, or political issues that would reasonably be expected to adversely affect the extraction of the reported Mineral Reserves.

1.16 Mining Methods

The Pre-Feasibility study underground mine plan for the Copperstone Project includes approximately 2,132,000 tons of mineralized mill feed to be extracted by underground mining in 6.5 years. The underground mine designs and schedule utilize measured and indicated mineral resources as part of the analysis. Mineral resources that are mineral reserves have demonstrated economic viability. The mine production schedule calls to produce 600 stpd for first 12 months and then ramping up to 1,000 tpd. Mining recoveries of 95% were applied, together with external dilution of 10% at zero grade; internal dilution is inherent in the optimized stope shapes. The mill feed will be placed on a stockpile at the crusher pad and a loader will be employed to feed the crusher at one twelve-hour shifts, seven days per week.

The mine plan for the mineralized material is based on the following criteria.

- Cut and Fill mining method using Rock Fill (“RF”) and Cemented Rock Fill (“CRF”);
- Cut-off grade of 2.2 g/t (0.0642 oz/ton) gold for underground mining;
- Production rate ramped up to 1,000 stpd;
- The underground design applied external dilution of 10% at zero grade and a mining recovery of 95%, with internal dilution inherent in the optimized stope shapes;
- Development drifting and raising of approximately 124,031 ft for the Life of Mine (“LOM”);
- Four operating crews with an average of 21 workers/crew –crews work 10hr shifts, four days on and four days off.

The proposed mining method for the Copperstone Project is an overhand mechanized cut-and-fill approach utilizing cemented rock fill. This method was selected for its flexibility in efficiently extracting ore from zones with low vein dip angles. It also helps minimize dilution through precise geological control and disciplined mining practices.

Underground mining methods were evaluated with the objectives of minimizing dilution, capital and operating costs, and maximizing resource recovery while sustaining the planned mill throughput. The Copperstone mineralization is relatively shallow, with an average dip of approximately 38 degrees. Although certain zones exceed a 45-degree dip and can be mined using gravity-assisted methods, most of the deposit is too flat to be efficiently mined using longhole stoping.

The Copperstone Mine previously operated as an open pit from 1987 to 1993 under Cyprus, producing approximately 514,000 ounces of gold from 6,173,000 tons of mill feed at an average grade of 0.089 oz/ton Au.

In 2012, American Bonanza Gold Corp initiated underground mining via two declines developed from the base of the open pit, focusing primarily on the D Zone located north of the pit. Between January 2012 and July 2013, the operation produced approximately 16,900 ounces of gold from 163,000 tons of mill feed grading 0.104 oz/ton Au (consisting primarily of development material).

As a result of this historic underground mining and an exploration drift completed in 2017, approximately 12,800 feet of underground access development is currently in place. This infrastructure includes two declines extending from the pit and covering roughly 500 feet of strike length, significantly reducing the amount of new development required to bring the mine into full production.

Table 1-3 presents the annual mining schedule. Stopping begins in month ten of Year -1, with development from the current underground ramp to the first mining area beginning month nine of Year -1. Mining of some mineralized material during development is planned for three months, with this material being stockpiled until month one of Year 1, when the process plant will start with a three month ramp up schedule.

Table 1-3 Annual Mining Schedule

	2026	2027	2028	2029	2030	2031	2032	2033	LOM Total
Ore (tons)	47,619	276,505	411,543	419,032	414,340	407,245	137,757	17,586	2,131,627
Grade (oz/ton)	0.1344	0.1938	0.1523	0.1417	0.1232	0.1214	0.1276	0.135	0.1419
Contained Au (oz)	6,401	53,597	62,658	59,382	51,057	49,452	17,584	2,373	302,505
Waste (tons)	112,799	98,645	350,771	362,203	144,569	155,192	40,446	34,467	1,299,093
Development (ft)	8,190	10,480	29,060	32,962	17,350	17,285	4,879	3,826	124,031

1.17 Recovery Methods

The Copperstone Project processing strategy has been developed to maximize gold recovery while minimizing capital and operating costs through the application of proven and commercially established processing technologies. The selected recovery method is based on a review of historical operating data, previous metallurgical investigations and engineering evaluations completed as part of this Preliminary Feasibility Study.

The objective of the process plant is to efficiently recover gold from underground mineralized material while leveraging existing site infrastructure and operational synergies available within Mining America's operating portfolio. The proposed processing facility combines conventional crushing and grinding, flotation concentration, and cyanide leaching of flotation tailings to achieve high overall gold recovery while maintaining operational flexibility to accommodate variations in ore characteristics throughout the life of the mine.

The selected flowsheet incorporates the refurbishment and reuse of existing processing infrastructure where practical and introduces new equipment necessary to achieve the planned production rate and metallurgical performance objectives. Gold-bearing flotation concentrate and loaded carbon produced from the carbon-in-leach (CIL) circuit will be transported off-site for final processing and gold recovery, eliminating the requirement for on-site refining facilities and reducing overall project capital requirements.

1.18 Project Infrastructure

The Copperstone Project benefits from substantial existing infrastructure developed during previous mining operations. The site is in western Arizona on relatively flat desert terrain with good access to transportation,

utilities, water supply, and nearby communities. Existing infrastructure significantly reduces the capital requirements associated with project restart and development.

The Project is accessible via Arizona Highway 95 and a well-maintained 5-mile mine access road. The site is located within reasonable commuting distance of Parker and Quartzsite, Arizona, and Blythe, California, which provide housing, commercial services, schools, and workforce support.

1.18.1 Site Facilities

Existing site infrastructure includes:

- Administration offices;
- Mine maintenance shop;
- Warehouse and storage facilities;
- Analytical laboratory and metallurgical testing facilities;
- Employee change house and associated sanitary facilities;
- Power substation;
- Water supply and storage systems;
- Fuel and lubricant storage facilities;
- Explosives magazines;
- Communications systems; and
- Tailings Storage Facility (TSF)

The administration building, maintenance shop, warehouse, laboratory, and change house remain in place and are suitable for the planned underground mining and processing operations with only minor refurbishment requirements.

1.18.2 Power Supply

Electrical power is supplied by Arizona Public Service (APS) through an existing 69-kV transmission line and on-site substation. The power infrastructure was originally designed for a larger mining operation and has sufficient capacity to support the proposed project. Power will be distributed as 4160 kV, 480 V and 220/110 V throughout the site.

Average site power demand is estimated at approximately 3.9 MW, including mining, processing, water supply, and ancillary facilities. Existing transmission, substation, and site distribution infrastructure are expected to require only limited modifications.

1.18.3 Water Supply

Process and service water will be supplied from a combination of underground mine dewatering and three permitted water wells located near the project. Water is currently delivered to site through existing pipelines and storage facilities.

The projected water balance indicates that water generated from underground operations will exceed operational requirements, resulting in an estimated surplus. Excess water not utilized in processing or dust suppression will be managed through a permitted evaporation pond and/or infiltration facility.

Process water will be recycled wherever practical through reclaim systems associated with the process plant and tailings storage facility. Potable water will continue to be supplied by licensed third-party vendors.

1.18.4 Communications

The Project is served by existing cellular coverage, satellite internet, VOIP communications, and licensed radio systems. Underground communications will be supported by a planned leaky-feeder network. Existing communications infrastructure is considered adequate for the proposed operation.

1.18.5 Fuel, Maintenance, and Explosives Facilities

Existing facilities include fuel storage and dispensing systems, lubricant storage with secondary containment, a vehicle wash bay, maintenance shops, and warehouse facilities. Dedicated explosive storage magazines are located in a secure area away from operational facilities and are designed to meet applicable regulatory requirements.

1.18.6 Site Access, Drainage, and Security

Site access roads, internal haul roads, drainage diversion structures, and perimeter fencing are already established. Existing drainage controls protect site facilities from stormwater runoff and support sediment management. Approximately 45,000 ft of perimeter fencing secures the site and mine facilities.

1.18.7 Tailings Storage Facility

The Project includes a permitted tailings storage facility originally designed for approximately 1.24 million tons of tailings storage capacity. The currently constructed Phase 1A facility has a capacity of approximately 366,000 tons, of which approximately 101,500 tons have previously been utilized.

The remaining capacity of the existing facility is expected to support approximately one year of planned operations. Expansion to the permitted Phase 1B configuration is planned during the first year of operations, followed by a subsequent embankment raise later in the mine life. These staged expansions are expected to provide sufficient storage capacity for the planned operation while maintaining compliance with approved design and regulatory requirements.

1.18.8 Infrastructure Summary

The Copperstone Project is considered well-served by existing infrastructure, including power, water, transportation access, maintenance facilities, communications systems, and tailings storage capacity. Existing infrastructure substantially reduces development risk and capital requirements and is considered adequate to support the proposed underground mining and processing operation with only limited upgrades and expansions.

1.19 Market Studies and Contracts

Market studies indicate that gold products from the Project are expected to be readily marketable through established global refining and smelting networks. Flotation concentrate is planned to be shipped for processing, while loaded carbon from the CIL circuit will be processed at the Pan Mine and sold as doré. The PFS economic analysis is based on a gold price of US\$3,500/oz, well below the US\$4,300/oz 36-month trailing average as of June 2026. No off-take agreements have been executed for the Project; however, a gold streaming agreement is in place whereby Star Royalties is entitled to purchase 4% of annual gold production at 25% of the prevailing spot gold price.

1.20 Environmental Studies, Permitting, and Social or Community Impact

Comprehensive environmental baseline studies have been completed to support ongoing permitting and operation of the Copperstone Project, including assessments of water resources, air quality, biological resources, wildlife, cultural resources, and threatened and endangered species. These studies established baseline environmental conditions and support federal and state regulatory approvals.

Groundwater quality monitoring indicates generally good water quality suitable for process use. Groundwater is naturally slightly alkaline, and all monitored constituents are below applicable Arizona Aquifer Water Quality Standards (AWQS), with the exception of fluoride, which occurs at naturally elevated concentrations throughout the regional groundwater basin. The Arizona Department of Environmental Quality (ADEQ) has approved revised fluoride compliance limits based on regional hydrogeologic studies.

Potential environmental impacts associated with project operations are considered manageable through established mitigation measures. Fugitive dust emissions from crushing, haul roads, stockpiles, and tailings facilities will be controlled through water application, operational controls, and facility design. Noise and vibration impacts are expected to be minimal due to the remote location of the site and the underground mining method. The Project operates as a zero-discharge facility, with all process water contained within engineered systems and managed under existing permit requirements.

Environmental monitoring programs include groundwater quality monitoring, process fluid system inspections, air quality compliance monitoring, and stormwater control inspections. Operational management plans addressing environmental protection, worker safety, emergency response, and regulatory compliance will be maintained throughout the mine life.

The Copperstone Project is an existing brownfield operation with the majority of required environmental permits already in place, including the Mine Plan of Operations (MPO), Aquifer Protection Permit (APP), Air Quality Permit, water rights permits, hazardous waste authorizations, and rights-of-way permits. The current permitting framework supports mining and processing operations at up to 600 tonnes per day. An MPO amendment submitted in August 2025 is under review by the U.S. Bureau of Land Management (BLM) to incorporate process plant modifications, including flotation and cyanide leaching circuits, stabilized tailings management, and water management infrastructure. Minor amendments to the APP and Air Quality Permit are anticipated to accommodate final process plant design changes.

The Project has maintained a strong environmental compliance record, including compliance with groundwater monitoring and air quality reporting requirements. The Qualified Person is not aware of any material environmental liabilities, permitting constraints, or social issues that would materially affect project development.

Closure and reclamation activities will be conducted in accordance with applicable BLM and Arizona regulatory requirements. Reclamation will focus on facilities constructed by the current operator and will include decommissioning of mining and processing infrastructure, closure of underground openings, reclamation of the tailings storage facility, revegetation, erosion control, and long-term environmental monitoring. The current reclamation cost estimate is approximately US\$1.63 million, with an updated estimate to be submitted as part of the ongoing MPO amendment process.

The Copperstone Project has a long operating history and has provided positive economic benefits to the surrounding region. No material social, community, or stakeholder issues have been identified that are expected to adversely affect project development or operations.

1.21 Capital and Operating Costs

Capital and operating cost estimates for the Copperstone Project were prepared to a Pre-Feasibility Study (PFS) level of accuracy and are expressed in constant Q2 2026 US dollars with no allowance for future escalation or currency fluctuations. The estimates were developed using industry-standard estimating methodologies based on the current level of engineering, project definition, vendor budget quotations, contractor pricing, first-principles estimating, quantity take-offs, and benchmarking against comparable mining projects. The overall estimate is considered to have an expected accuracy range of approximately $\pm 25\%$.

The Project contemplates refurbishment and expansion of the existing Copperstone Mine and associated infrastructure, including recommissioning underground mining operations and construction of a new mineral processing facility capable of treating 600 short tons per day during the first year of operations, increasing to 1,000 short tons per day in Year 2. The capital estimate benefits from the use of existing underground development, site infrastructure, buildings, utilities, and selected process facilities remaining from previous operations.

Total life-of-mine capital is estimated at US\$127 million, comprising US\$57.2 million in initial capital and US\$69.8 million in sustaining capital. Initial capital includes expenditures required prior to commercial production, while sustaining capital includes ongoing underground development, equipment purchases associated with the transition from contract mining to owner-operated mining, tailings storage facility expansions, reclamation and closure costs, and other life-of-mine capital requirements. The largest capital components are underground mine infrastructure and development, the mineral processing plant, and associated indirect costs and contingency.

Underground mining is planned to commence using a mining contractor during pre-production and the initial operating period. The Project is designed to transition to owner-operated mining during Month 10 of

Year 1 through the acquisition of an underground mobile equipment fleet, reducing long-term mining costs while minimizing initial capital requirements.

Average life-of-mine operating costs are estimated at US\$145.38 per short ton milled (US\$1,064/oz Au sold), comprising mining costs of US\$88.65/t, processing costs of US\$21.58/t, site general and administrative costs of US\$7.23/t, transportation and refining costs of US\$1.82/t, together with royalties (21.51US\$/t) and production taxes (US\$4.59/t). The resulting all-in sustaining cost (AISC), including sustaining capital, reclamation and sustaining exploration, is estimated at US\$182.82 per short ton milled, equivalent to US\$1,338 per ounce of gold sold.

1.22 Economic Analysis

The economic analysis for the Copperstone Project was completed on a discounted cash flow (DCF) basis using a 5% discount rate, with all costs and revenues expressed in constant Q2 2026 US dollars (no inflation or escalation applied). The Project is evaluated on a 100% equity basis over a total mine life of 7.5 years, comprising 1 year of construction and 6.5 years of operation.

The base case assumes a gold price of US\$3,500/oz and an exchange rate of C\$1.00 to US\$0.72. Under these assumptions, the Project is forecast to process approximately 303,000 oz of mined gold, with 291,161 oz recovered and sold. Initial capital investment, including contingency and working capital, is estimated at US\$57.2 million.

The base case results in a pre-tax NPV (5%) of US\$426 million and a pre-tax IRR of 124.1%, with a payback period of 1.11 years. On a post-tax basis, the Project generates an NPV (5%) of US\$342 million and an IRR of 110.5%, with a payback period of 1.13 years and a payback multiple of 7.87.

Life-of-mine average Cash Operating Cost is estimated at US\$873/oz of gold sold, with a Total Operating Cost (including royalties and production taxes) of approximately US\$1,064/oz, and an All-In Sustaining Cost (AISC) of US\$1,338 /oz.

Based on the results of this economic analysis, the Project demonstrates positive economics at the assumed base case parameters and supports advancing to additional, more detailed engineering and economic studies as presented in Table 1-5.

Table 1-4 Copperstone Project Total Operating Cost per oz Au & per ton Mill Feed

Operating Costs	\$/oz Au	\$/ton ore
Total Mining	-\$649.05	-\$88.65
Total Processing	-\$157.98	-\$21.58
Total Site G & A	-\$52.89	-\$7.23
Transportation and Refining	-\$13.35	-\$1.82
Bi-product credit (silver)	\$0.00	\$0.00
Cash Operating Costs	-\$873.27	-\$119.28
Royalties	-\$157.50	-\$21.51
Production Taxes	-\$33.60	-\$4.59
Total Operating Costs	-\$1,064.37	-\$145.38
Corporate General/Admin	-\$10.30	-\$1.41
Reclamation cost - prorated	-\$17.17	-\$2.35
Exploration costs-sustaining	-\$2.45	-\$0.34
Capital costs - sustaining	-\$244.12	-\$33.34
All-In-Sustaining-Costs	-\$1,338.41	-\$182.82

1.23 Conclusions

The prefeasibility study represents an economically viable, technically credible, and environmentally sound mine development plan for the Copperstone Mine. The project benefits significantly from its proximity to existing sources of labour, existing infrastructure and past production data. The prefeasibility LOM plan demonstrates positive economics. Industry-standard mining and processing methods were used in the study and the QPs are not aware of any fatal flaws that encumber the Copperstone mine restart. It is recommended that the Project proceeds to detailed engineering design and construction.

The most significant potential risks associated with the Project include the design of an effective control system to minimize uncontrolled mining dilution, operating and capital cost escalation, reduced metallurgical recoveries, unforeseen schedule delays and changes in regulatory requirements and metal prices. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

The study indicates that Copperstone, based on calculated Proven and Probable Reserves (Table 1-2) of 2.13 million tons grading 0.142 oz/ton (1.93 Mt of 4.87 g/t) can support six and a half years of mine at a process rate of 600 tons per day for the first year of operations, then 1000 tons per day for the subsequent years.

Ore will be processed at the process plant designed to achieve expected gold recovery of 96.5% through a combination of flotation and CIL. It is expected that over the LOM approximately 291 koz of gold will be produced

Initial capital cost of the project is estimated to be \$57.2 million and \$69.8M of Sustaining Capital. The LOM AISC total is \$1,338/oz. The project NPV (after-tax) is estimated to be \$341.48 million using a 5% discount

rate at a gold price of \$3,500/oz. The IRR (after-tax) is estimated to be 110.5% with a payback period of 1.13 years.

1.24 Recommendations

The Copperstone project is well suited for a restart of mining operations. A summary of the suggested work program to support the restart includes the following components:

- Continue Infill Drilling
- Outline a step out and exploration drill program to test the extensions of the deposit
- It is recommended that the company examine the potential for expansion of the historic open pit operations through a more extensive near surface drill program with the goal of establishing an open pit minable resource
- Conduct mineralogical evaluations on samples from various areas of the deposit to confirm variations in the minerology of the deposit
- Perform additional flotation studies on various zones to examine the optimal reagent schemes.
- Additional filtration and thickening studies
- Optimize optimal grind size and retention times
- Continue to optimize the mine design, including number of access points, stope height and width
- Review the use of a lower cut-off grade in the operational mining plan to take advantage of the high gold price
- Develop grade control procedures
- Complete a comparative analysis of contract and owner-operated mining options to identify the preferred approach
- Recruit experienced underground technical and management personnel as a priority to advance project development
- Optimize the ventilation, water management, and electrical power systems
- Identify and advance suitable buyers and terms for concentrate sales

Estimated costs for the recommendations are shown in Table 1-5.

Table 1-5 Recommended Scope of Work Cost for the Copperstone Project

Recommendation	Estimate
Structural Understanding	\$30,000
Infill and Step Out Drilling	\$3,000,000
Mineral Processing & Recovery Optimization	\$200,000
Mining	
Optimize short term mine design	\$50,000
Review use of lower cut-off grades	\$10,000
Grade Control Program	\$10,000
Contract Mining vs Owner Mining analysis	\$10,000
Key underground mining staff	\$150,000
Optimize Ventilation, Water, and Power systems	\$55,000
Total Mining	\$285,000
Detailed Engineering	\$1,200,000
Near Surface Resource Evaluation	\$50,000
Total Budget	\$ 4,765,000

2 INTRODUCTION

2.1 Introduction

Mining Americas Inc. (“MAI”, the Owner) (TSX-V: MAI) is a company focused on acquiring, exploring, and developing base and precious metals projects in North America. Mining Americas Inc. acquired the Copperstone property under the terms of the business combination with Sabre Gold Mines Corporation (“SGLD”) that closed on January 27, 2025. SGLD was formed in August of 2021 with the merger of Arizona Gold Corp. (“AZG”) and Golden Predator Mining Corp. (“Golden Predator”). AZG was previously known as Kerr Mines, Inc. (“Kerr”) and completed the name change to AZG in December of 2020. Kerr acquired the Copperstone Project (the “Project”), a historically productive high-grade gold mine located in La Paz County, Arizona in 2014. The mine is fully permitted with significant mining infrastructure, mineral resources and processing infrastructure in place.

The Copperstone Project encompasses approximately 15.8 square miles of surface area and mineral rights in La Paz County, Arizona, roughly 19 miles north of the town of Quartzsite. The Project area covers all or portions of Sections 17 through 23 and 28 through 32, T6NR19W; Sections 1, 2, 10 through 14, 22 through 27, and 36, T6NR20W; and Section 24, 25, and 36, T7NR20W, Gila and Salt River Meridian. The approximate geographic center of the Project area lies at 33°52’6”N latitude, 114°17’42”W longitude. Map coverage of the Project area is provided by the 1:24,000-scale, Moon Mountain SE and Moon Mountain NE, U.S.G.S. 7.5-minute topographic quadrangles.

This Pre-Feasibility study outlines a restart plant for the Copperstone Mine starting with a 600 stpd in year 1, increasing to 1000 stpd from year 2 to year 6 producing approximately 291,000 oz Au over that period. This Pre-Feasibility is based on a Proven and Probable Mineral Reserve of 2.13 million tons. In addition to the reserves, Copperstone has Inferred resources of 441,694 tons at 0.12 oz/ton (4.04 g/t).

2.2 Terms of Reference

In 2026 in order to complete this Pre-Feasibility MAI, the Issuer of this report, has engaged a team of independent consultants coordinated by the MAI to contribute to a Pre-Feasibility Study (“PFS”) for the Copperstone Project. The report titled “*National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*” with an effective date of December 1, 2026.

This report presents the mineral reserve statement and documents the results of the PFS and was prepared in accordance with the requirements and guidelines set forth in NI 43-101 Companion Policy 43-101CP and Form 43-101F1 (June 2011), and the mineral resources presented herein are classified according to Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Definition Standards – For Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on November 29, 2019.

GR Engineering Services Ltd. (“GRES”) is responsible for the process plant design, tailing storage; Rockpoint Advisory for the mineral reserve estimate; Zelligan Consulting Inc. for the reserve and data verification; Garth Wilcox P.Eng for metallurgy and processing; Kristal Font Inc. for economic evaluation; Segerstrom Consulting LLC for geology, history and environmental studies.

2.3 Qualified Persons and Personal Inspection

The following individuals, by virtue of their education, experience, and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of the appropriate professional institutions.

The following companies contributed to this technical report and provided Qualified Person (QP) sign-off for their respective sections. All supporting documents cited in this report are referenced in Section 27.

Table 2-1 Pre-Feasibility Study Section and Party Responsibilities

Qualified Persons	Sections Responsible
Scott Zelligan, P.Ge	12, 14 parts of 1, 25, 26
Lawrence Segerstrom, M.Sc. (Geology), CPG	1 through 11, 18, 20 and parts of 12, 25, 26
Garth Wilcox P.Eng	13,17 and parts of 1, 25, 26
Peter Szkilnyk P. Eng	15, 16 and parts of 1, 25, 26
Alex Duggan P. Eng	22, and parts of 25, 26

The QPs preparing this Pre-Feasibility Study are specialists in geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, infrastructure, capital and operating cost estimation, and mineral economics.

None of the QPs or associates employed in the preparation of this report have any beneficial interest in Mining Americas Inc.. The QPs are not insiders, associates, or affiliates of Mining Americas. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Mining Americas and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

2.4 Site Visits and Scope of Personal Inspection

Table 2-2 Site Visits and Scope

Qualified Persons	Dates	Accompanied By	Description of Inspection
Lawrence Segerstrom, M.Sc. (Geology), CPG	January 14 to 16, 2025	Miguel Cardona, VP Exploration	• Examination of surface and underground bedrock exposures • Verification of drillhole collar locations. • Review of core and reverse circulation (RC) chip processing and storage facilities • Discussions with on-site geologists regarding the geology and geologic model • Field observations confirmed previous interpretations of: • Bedrock lithologies • Alteration types

2.5 Sources of Information

This Report has been prepared by the Independent QPs through discussions with Mining Americas employees and by reviewing the existing drill hole database; geologic reports; available maps and cross sections; metallurgical test data; government reports; miscellaneous documentation (reports and scientific papers); and other public and private information as cited throughout the report and referenced in Section 27.

Table 2-3 shows the Mining Americas employees who provided information to the Independent QPs responsible for this Report. The QPs have taken reasonable steps to verify the information provided where possible.

Table 2-3 Non-QP Contributions

Mining Americas or Other	Information Provided for Report Section
Miguel Cardona	Parts of 9 through 11
Kevin Small P. Eng	16, parts of 21
Darren Koningen P. Eng	13,17,21
GR Engineering Services (GRES)	Parts of 17 and 21
Small Mines Development (SMD)	16 parts of 21

The information contained in current report Sections 4 through 11, 13, 14 and 18 was largely previously presented in, and in some cases is excerpted directly from, the technical reports listed above. The Qualified Person has reviewed this material in detail, and finds the information contained herein to be factual and appropriate regarding guidance provided by NI 43-101 and associated Form NI 43-101F1.

2.6 Units of Measure

Unless otherwise stated, all measurements reported herein are U.S. customary, and currencies are expressed in constant 2026 U.S. dollars. Gold grades are presented in troy ounces per short ton (“oz/ton” or “opt”), unless otherwise indicated.

2.7 Important Notice

Each QP that assisted in the preparation of this Pre-Feasibility Study assumes responsibility for those sections or areas of this report that are referenced in Table 2-1. None of the QPs; however, accepts any responsibility or liability for the sections or areas of this report that were prepared by other QPs.

This NI 43-101 was prepared to allow MAI to reach informed decisions respecting the development of the Copperstone Project. Except for the purposes legislated under securities law: (a) any use of this report by any third party is at that party's sole risk, and none of the QPs (nor any of the companies for whom they work) shall have any liability to any third party for any such use for any reason whatsoever, including negligence, and (b) each of the QPs hereby disclaims responsibility for any indirect or consequential loss arising from any use of this report or the information contained herein. This report is intended to be read as a whole, and sections should not be read or relied upon out of context. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report contains estimates, projections, and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP's opinion at the time that they are made, but in most cases involve significant risk and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated, or projected. None of the QPs undertake any obligation to update the forward-looking information.

This report contains the expression of the professional opinion of the QPs based on: (i) information available at the time of preparation, (ii) data supplied by outside sources, (iii) conclusions of other technical specialists named in this report, and (iv) the assumptions, conditions, and qualifications in this report. The quality of the information, conclusions, and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project and are consistent with the intended level of accuracy for a Pre-Feasibility Study.

As permitted by Item 12 of Form 43-101F1, the QPs have, in the preparation of this Report, relied upon certain data provided to the QPs by the Owner and certain other parties. The relevant data and the extent of reliance on such data are described in Section 3 of this Report.

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) opinions contained herein are based on information provided by Mining Americas and by others throughout the course of the study. For information concerning some of the title and permit matters they have relied on other experts as are listed below:

Non-QP specialists relied upon for specific advice includes:

- The Authors of this Technical Report have not independently verified ownership or mineral title with respect to the Copperstone Project's concessions and/or mining claims. The property description presented in this Report (Section 4) is not intended to represent a legal, or any other, opinion as to title. For information concerning the Copperstone mining concessions, the QPs have relied on the July 13, 2026 "Report on Status of Title; Copperstone Mine, La Paz County, Arizona" prepared by Dorsey Whitney LLP (Dorsey) on behalf of Mining Americas. Dorsey obtained information on the title and concessions based on a search of Federal and County title records.
- Stantec Consulting Services Ltd. (Stantec): Stantec provided a permitting readiness assessment for the restarting of the Copperstone Gold Mine April 2024, "Copperstone Gold Mine Project GAP Analysis Permit Review And Operation Requirements Assessment, April 2024". The document outlines what permits, approval updates, and notifications are required before the mine can return to production after being placed on care and maintenance following its last operation.

These items have not been independently reviewed by the Authors and the Authors did not seek an independent legal opinion of these items

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Copperstone Project encompasses approximately 15.8 square miles of surface area and mineral rights in La Paz County, Arizona, roughly 19 miles north of the town of Quartzsite. The Project area covers all or portions of Sections 17 through 23 and 28 through 32, T6NR19W; Sections 1, 2, 10 through 14, 22 through 27, and 36, T6NR20W; and Section 24, 25, and 36, T7NR20W, Gila and Salt River Meridian. The approximate geographic center of the Project area lies at 33°52'6"N latitude, 114°17'42"W longitude. Map coverage of the Project area is provided by the 1:24,000-scale, Moon Mountain SE and Moon Mountain NE, U.S.G.S. 7.5-minute topographic quadrangles. Figure 4-1 shows the general location of the property.

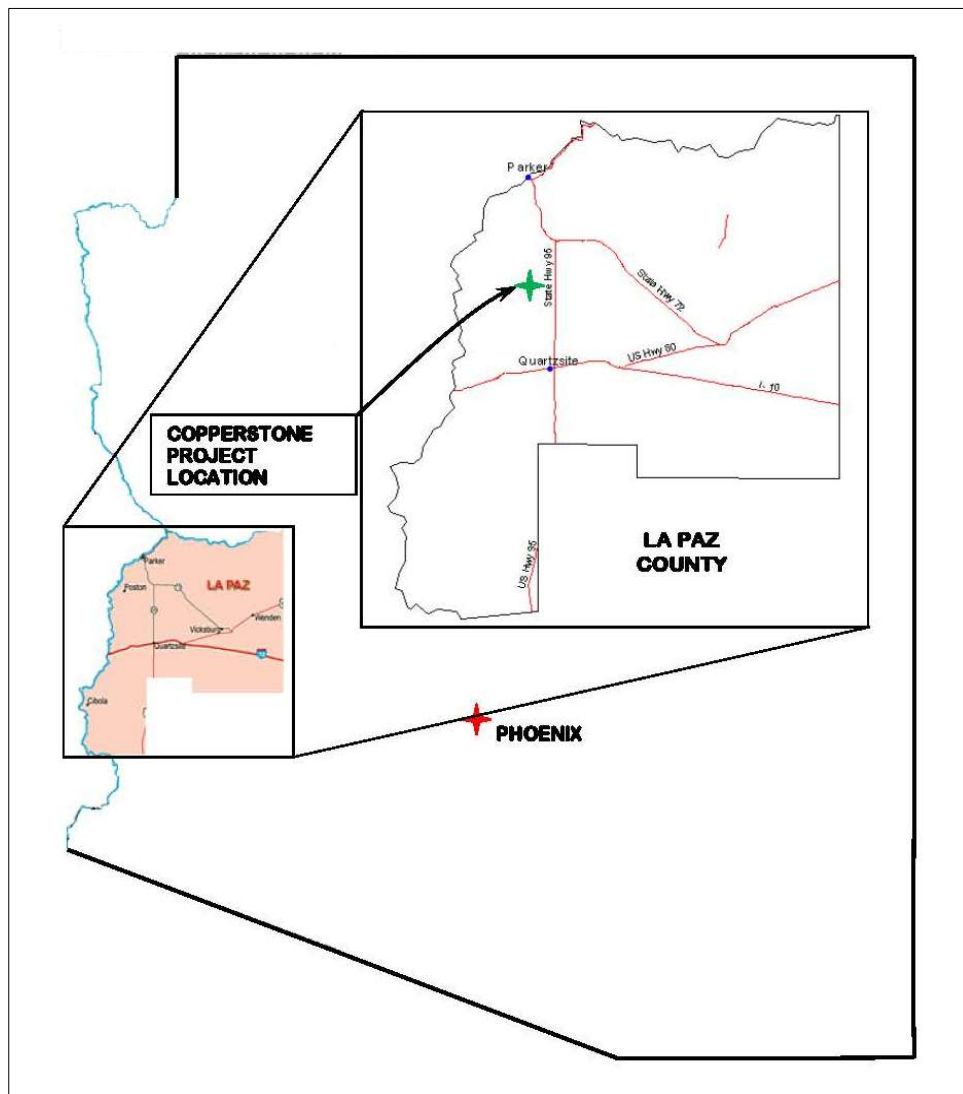
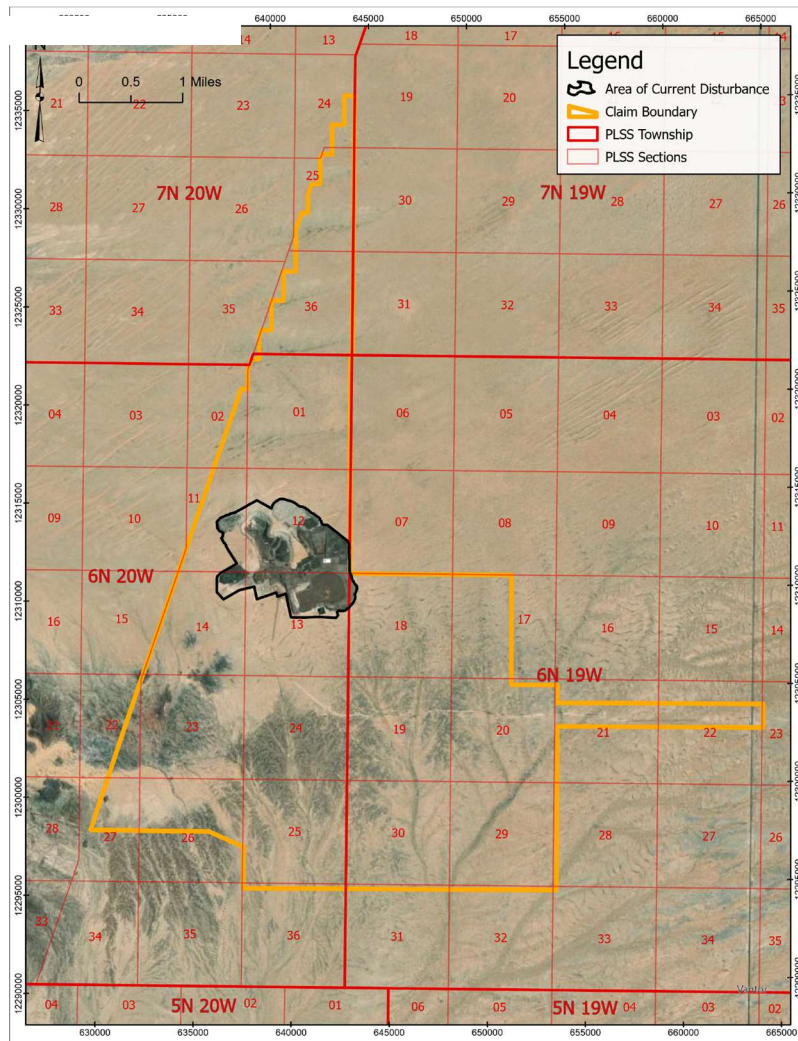


Figure 4-1 Copperstone Project Location

4.2 Mineral Title

MAI controls 546 federal unpatented mining claims which comprise the Copperstone Project area. The federal claims cover approximately 10,137 acres (4,102 hectares). The claim blocks are presented in plan view in Figure 4-2, and a summary list of claim details is presented in Appendix A - Section 4 Mineral Titles.



Note: Arizona State Section locations are not part of the Copperstone Project boundary.

Figure 4-2 Copperstone Project Claim Area

The existing disturbance area shown in Figure 4-2 is further subdivided into six discrete areas or ‘zones’ based on a variety of factors, including proximity to known structures, style of mineralization, and general location within the Project area. These zones are illustrated for reference in Figure 4-3, as they are referred to with some frequency throughout the text of this report.

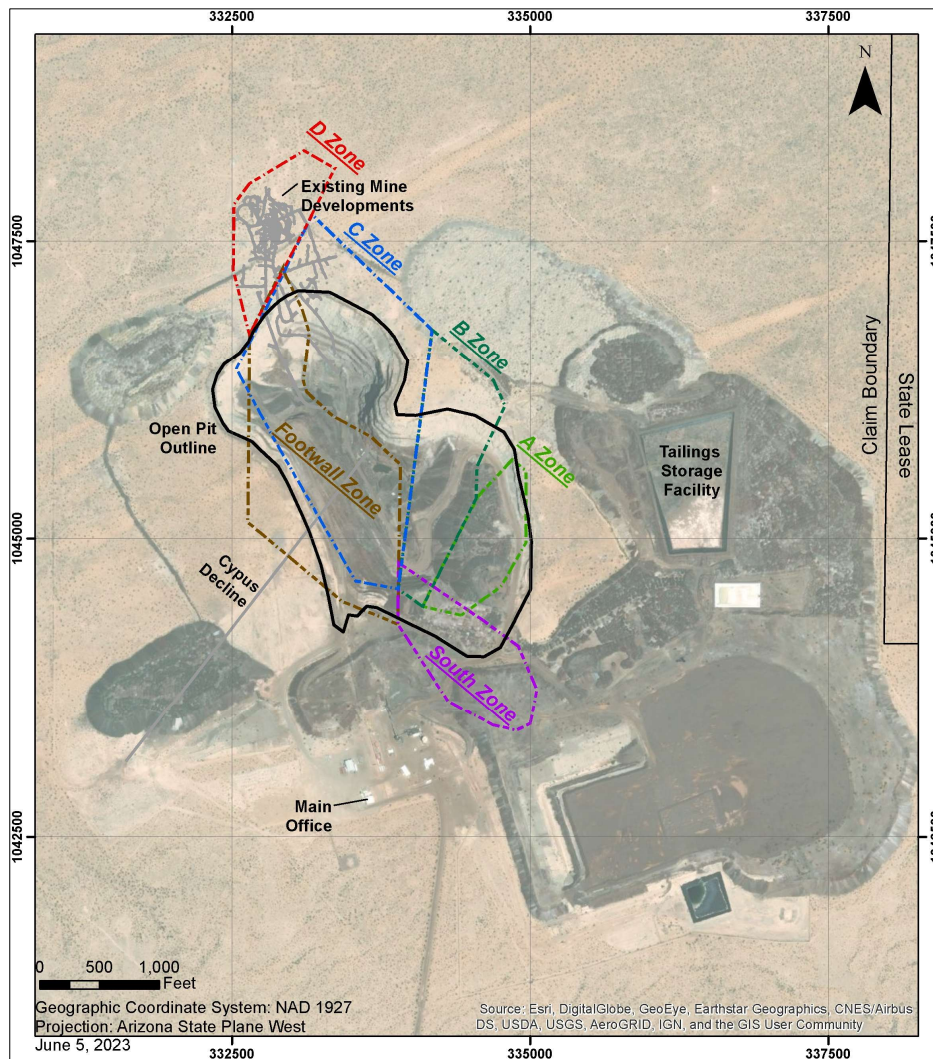


Figure 4-3 Zone Subdivisions within the Existing Disturbance Area

4.3 Property Ownership, Mineral Tenure, Agreements and Encumbrances

On January 27, 2025, Mining Americas Inc. announced finalization of the business combination with SGLD by plan of arrangement under the British Columbia (B.C.) Business Corporations Act. As part of the transaction, SGLD became a wholly owned subsidiary of Mineral Alamos Inc. For more details regarding the terms of the business combination, see Mining Americas (October 28, 2024).

The Copperstone Project is wholly owned by Mining Americas Inc. via Mining Americas 100% ownership of American Bonanza. On October 28, 2024, Mining Americas Inc. announced the acquisition of all issued and outstanding common shares of Sabre Gold Mines Corporation and its subsidiary American Bonanza by way of plan of arrangement under the Business Corporations Act (British Columbia). The arrangement was approved by SGLD shareholders by written consent, by American Bonanza shareholders at a special meeting of shareholder held on January 14, 2025 and by the Supreme Court of British Columbia on January 27, 2025.

MAI holds a 100 percent leasehold interest in the Copperstone Project. The landlord is the Trustee of the Angie Patch Survivor's Trust and the Trustee of Daniel L. Patch Credit Trust "The Patch Living Trust" and the lease was for a 10-year term starting June 12, 1995, was renewed on June 12, 2005 for a 10-year term and renewed on June 12, 2015 for a further 10-year term. The term was extended to June 12, 2026, when the lease was amended and restated on January 4, 2017. The lease is renewable for one or more ten-year terms at the option of MAI under the same terms and conditions. MAI is obligated to pay for all permitting and state lease bonding, insurance, taxes, and to pay the leaseholder a 1.5 percent production gross royalty in addition to an annual rent payment of US\$40,000 per year.

On November 11th, 2024 Trans Oceanic Mineral Company Ltd. (TOMC) has exchanged its 1.5 percent gross production royalty on the project for a 0.5% NSR royalty on the 3 Aces project located in the Yukon, Canada. As a result, the total annual gross production royalty on the Copperstone Project is now 1.5 percent.

On November 12, 2020, the company entered into a US\$18 million precious metals delivery and purchase agreement (the "Purchase Agreement") with Star Royalties to finance the restart of underground operations and gold production at the Copperstone Mine in Arizona. The company has received the first two tranches amounting to \$16,239,600 (US\$12 million). On June 26, 2023, both parties agreed to a stream reduction notice, thus, the final tranche of US\$6 million will no longer be forthcoming. The stream agreement remains in place post the business combination with MAI under the following terms: Star receives 4% of annual gold production at a gold price of 25% of spot.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

General access to the vicinity of the Copperstone Project is provided by Interstate I-10 out of Phoenix, Arizona, approximately 125 miles west of the city of Quartzsite. The primary Project access road, Cyprus Mine Road, is located roughly 19 miles north of Quartzsite on U.S. Highway 95. Cyprus Mine Road is a well maintained, gravel road which terminates 5.5 miles west of the highway at the Project entrance. Access to the Project area is attainable year-round.

5.2 Local Resources and Infrastructure

The community nearest to the Project area is the town of Quartzsite, which hosts a population of roughly 4,000. Parker, the county seat of La Paz County, is located 20 miles to the north. Both Parker and Quartzsite offer standard municipal amenities such as lodging and services, as well as modest supplies of foodstuffs and hardware. Major supply centers and ample skilled and unskilled labor are available in Phoenix, 125 miles to the east of Quartzsite, and in Yuma, roughly 80 miles south of Quartzsite on U.S. Highway 95. Access to the Sante Fe rail line is available in Parker, and international air service and railway access (Union Pacific and BNSF) are both available in Phoenix.

Existing infrastructure at the Copperstone Project includes:

- Office facilities
- Crushing circuit
- Process structure (excluding all mechanical equipment)
- Warehouse
- Equipment maintenance shop
- Assay laboratory buildings
- Change house
- 10 trailer house hook-ups
- Septic system
- Variety of shipping containers which provide secure core storage
- Incoming commercial 69 kV overhead electrical power
- On-site substation.

5.2.1 Availability and Sources of Power

Incoming power is supplied via a commercial 69 kV overhead electrical power line delivered to the main onsite substation that steps down distribution to 4.16 kV for further distribution throughout the site.

5.2.2 Availability and Sources of Water

Water is currently delivered from three permitted water wells to a 375,000-gallon storage tank in the mineral processing area. The right to extract and use groundwater from the aquifer within the La Posa Plain is authorized by the Arizona Department of Water Resources pursuant to A.R.S. Section 45-514. Potable water is delivered by truck. Mine communications are supported by cellular and satellite phone and internet service. Existing surface rights and right of ways are sufficient for all proposed exploration, mining, and processing activities, including tailings and waste storage and disposal areas.

5.3 Climate

The local climate is typical of a hot desert, with mild to warm “winter” weather occurring from November to March, and hot to extreme summer temperatures for the remainder of the year. In the middle of summer, Quartzsite is one of the hottest places in the United States, with recorded temperatures as high as 122 °F (July 1995). Average annual temperatures at Quartzsite range from a low of 59.3°F to a high of 89.5°F. Precipitation averages just ~3.5 inches annually, most of which occurs as rainfall during late summer and early winter months.

5.4 Physiography

The Copperstone Project lies at the southern edge of the Basin and Range geo-physiographic province, which is typified by north-northeast trending mountain ranges separated by broad, flat, alluvium filled valleys. The Project is situated on the flat, sandy desert terrain of the La Posa Plain, at the northeastern end of the Dome Rock Mountains, and is surrounded by a natural desert scrub environment. Vegetation is sparse, and consists primarily of ground hugging shrubs, short woody trees, and cactus. The soils are hyperthermic arid soils of the Superstition-Rositas Association, which is characterized by deep, coarse-textured, nearly level and undulating soils on terraces (Hendricks 1985). Surficial soils in the Project vicinity are classified as “gravelly loamy fine sand” and include aeolian (i.e., wind-blown sand) deposits in hummocks surrounding the many small shrubs. Elevations within the Project area range from 650 to 825 feet above mean sea level.

6 HISTORY

6.1 Historical Ownership and Development

The first recorded commercial interest in the Copperstone property was as a copper prospect in 1968. Charles Ellis of the Southwest Silver Company (“Southwest Silver”) controlled the Continental Silver claim group from 1968-1980. Newmont Gold Company (“Newmont”) leased the property in 1975. A geophysical survey was conducted and one drillhole completed in an attempt to verify porphyry copper mineralization. The attempt was unsuccessful.

In 1980, Southwest Silver drilled six rotary holes with unknown results and then dropped the claims. In late 1980, Dan Patch staked 63 Copperstone claims and leased the property to Cyprus-Amoco. Cyprus then purchased the Iron Reef Claim group from W. Rhea. Additional claims were subsequently added, and the claim block expanded to 284 claims. Cyprus identified the Copperstone property as a gold target and undertook a drilling campaign from 1980 to 1986, which resulted in 73 diamond drillholes and over 496 RC drillholes completed (Pawlowski, 2005). Cyprus began baseline, financial and metallurgical studies that led to mine design, initial construction and a partially completed decline in 1986.

In 1987, Cyprus commissioned construction of a 2,500 ton/day carbon-in-pulp mill and started open-pit mining. The mine was designed, constructed, and operated as a zero-discharge facility (Miller et al., 1994). Mining continued until 1993 when the pit neared the groundwater table, which was the limit of the original mining permits. Cyprus terminated its lease at this time. Ackerman (1998) reported production by Cyprus at Copperstone of 514,000 oz of gold from 5,600,000 metric tonnes of mill feed grading 0.089 oz/ton of gold.

Santa Fe Pacific Gold Corporation (“Santa Fe”) leased the property in 1993, while reclamation activities were underway. Santa Fe completed 12,500 ft (3,810 m) of RC drilling on seven exploration targets. Gold mineralization was encountered in one hole in the footwall of the Copperstone fault.

Royal Oak Mines (“Royal Oak”) leased the property from the Patch Living Trust in 1995. Royal Oak drilled a total of 28,413.5 ft (8,660 m) in 34 holes between 1995 and 1997. Several high-grade gold intercepts to the north and east of the open pit showed potential for underground mining.

Asia Minerals entered into a joint venture with Arctic Precious Metals Inc., a subsidiary of Royal Oak in August 1998. Asia Minerals drilled 15 holes (A98-1 to 15) in November 1998 for a total of about 10,979 ft (3,346 m). Each hole was drilled with RC methods from the surface to a predetermined depth and then core drilled through the target interval. The drilling program was designed to explore the C and D zones (MRDI, 1999). Golder Associates and MRDI Canada completed a scoping level study after the 1998 drilling program was completed.

Asia Minerals drilled 11 more holes in early 2000. Total footage was 8,610 ft (2,624 m). Holes were designed to test the strike length of the D zone, with the best intercept in hole A00-10 which assayed 0.943 oz/ton Au over 10.5 ft (3.2 m). On July 7, 2000, the BLM approved an application from Asia Minerals to construct a 2,000-foot (610 m) decline (Mine Development Associates, 2000). The purpose of the decline was to explore high-grade gold mineralization which had been discovered during surface drilling (AMEC, 2006). On July 26,

2000, the Arizona Department of Environmental Quality approved the proposed underground activity and granted Asia Minerals an exemption from an Aquifer Protection Permit (Mine Development Associates, 2000).

Asia Minerals began a joint venture with Centennial Development Corp. of Salt Lake City in September 2000 (AMEC, 2006). The permitted decline was started from the north end of the pit in a northward direction. It provided a platform for further exploration drilling and allowed for the removal of bulk sample material for metallurgical and milling tests. To that end, a 64-lb high grade sample was sent to McClelland Labs in Sparks, Nevada. It was during this time that Asia Minerals changed its name to American Bonanza Gold Mining Corp. to better reflect the geographic, metal and grade focus of the company.

On March 4, 2002, American Bonanza announced that it had gained control of a 100% equity interest in Copperstone subject only to the royalty schedule payable to the Patch Living Trust. They also announced an agreement with Trilon Securities whereby Trilon would arrange a US\$1.1 million secured credit facility for the company. In November 2002, American Bonanza selected Merritt Construction of Kingman, Arizona to expand the underground development. American Bonanza announced on May 5, 2003 that significant high-grade gold mineralization was sampled in the decline in the D zone. In June 2003, an underground drill station was completed. Drilling began in July, and by May 17, 2004, American Bonanza had drilled 32 underground core holes in the D zone for a total of 9,208 ft (2,807 m).

American Bonanza continued drilling in 2004, including underground drilling from a drill bay in the exploration decline. The company retained certain specialized firms to assist it with collecting environmental, geotechnical, hydrological and metallurgical baseline data in 2004, and in 2005, submitted a Mine Plan of Operations (“MPO”) to the BLM. Additional drilling was completed in 2006 and 2007. A variety of studies and reports were commissioned by American Bonanza between 2007 and 2010, culminating in a feasibility study, including an updated mineral resource estimate, completed in 2010. In 2011 American Bonanza constructed a 450 stpd flotation mill on site and in 2012 started underground mining from two declines that were previously developed in the bottom of the open pit. American Bonanza’s mining focused on the D zone which is to the north of the open pit. From January 2012 to July 2013 American Bonanza produced approximately 16,900 oz of gold from 163,000 tons of mill feed grading 0.104 oz/ton of gold. American Bonanza maintained control of the Copperstone Project until AZG (as Kerr) acquired the Project in June of 2014.

SGLD was formed in August of 2021 with the merger of Arizona Gold Corp. (“AZG”) and Golden Predator Mining Corp. (“Golden Predator”).

MAI acquired the Copperstone property under the terms of the business combination with SGLD that closed on January 27, 2025.

6.2 Historical Exploration

There were early, unsuccessful attempts to use geochemistry for exploration, including a soil gas survey. The post mineral cover, which has been transported to the Copperstone area, coupled with few mobile trace elements associated with the gold mineralization, quickly eliminated geochemical methods available in the 1980's and 1990's as viable exploration tools (Telesto, 2011). The QP has not received or vetted the results of the geochemical analysis but accedes that Telesto's conclusions are reasonable.

Cyprus conducted the following geophysical surveys at Copperstone:

- Ground Magnetism;
- MAXMIN Electromagnetics;
- VLF Electromagnetics;
- Resistivity and Induced Polarization; and
- Gravity.

In September 1981, four ground magnetic test lines were run over gold bearing breccia units previously identified by drilling. These lines utilized 100-ft station spacings which were shortened when anomalous responses spatially correlated with likely subcrop of gold mineralization. Test results were encouraging, and the magnetic grid was expanded to cover the full extent of the claim block at the time, which covered an area of about 3 square miles. All data were taken from lines spaced 300 ft apart with a 100-ft station spacing, which again was shortened when anomalous readings were recognized. The lines, oriented N50E, were run perpendicular to two base lines oriented N40W. The gravity survey was completed and read by February 1982.

Results from this survey emphasized subsurface basalt plugs and indicated potential gold breccia targets. Based on poor correlation between lines and station spacing suspected of being too broad, the entire grid was re-read by pacing in stations at 10-ft station intervals. Additional lines were read in the central drilling area. The main grid was re-read by the end of September 1982, and the additional lines were read during March and February of 1983.

Results from the re-read ground magnetic survey identified gold breccias as magnetic lows, while quartz veining in the southeast gave strong high frequency responses. Basalt plugs and flows were detected near the reservation fence to the northwest and to the southwest. Additionally, several apparent offsets in magnetic trends were interpreted to be related to faulting.

In February-March 1982, Maxmin electromagnetics were used to identify the contrast between clay altered breccia material and surrounding rock types. Test lines were run across the central drilling area using the magnetism grid, and readings were taken from both 400- and 800-ft. separations between transmitter and receiver. Observed responses did not correlate to known geology or gold mineralization. Erratic responses have been postulated to represent fault zones identified by the magnetic survey.

Very low frequency (“VLF”) electromagnetics were run in the central drilling area in the hopes of identifying faults postulated to be responsible for lateral offsets in the magnetics data. Results were not definitive enough to expand the survey.

Resistivity was used in conjunction with Induced Polarization (“IP”) to look directly for mineralized breccia zone subcrops. The gradient electrode array was chosen for rapid coverage, and the transmitter electrode spacings were 6,000 ft with the receiver potential electrode separation at 100 ft. A test survey in the central drilling area was run in December 1982, with the rest of the survey area read during February and March 1983. Gold mineralization responded as IP and resistivity highs, though IP was more useful.

Targets were identified from the electromagnetic surveys by combining low magnetics, and high IP and resistivity responses. Follow up drilling did not identify any new gold mineralized areas.

Mining Geophysical Surveys Inc. conducted a gravity survey to determine the depth to bedrock in the main drilling area to exclude regions too deep for practical mining interests, and regionally identify shallow pediment areas for further exploration. Four east-west traverses were made from the reservation fence on the west to the outcropping range to the east. North-south traverses consisted of a long line along the reservation fence and several short lines near the Copperstone deposit. Field work was completed in December 1982.

The regional survey identified two deep basins, one between Mesquite Mountain and the Plomosa Mountains on the east with a thickness of about 7,000 ft, and another between Plomosa Mountains on the east and the Dome Mountains on the west with a thickness of about 5,400 ft. Overburden thickness between the drilling area and Mesquite Mountain to the north was determined to be between 1,300 and 2,600 ft.

Several results emerged within the central drilling area including a basement fault scarp positioned just east of the southeast basalt plug, overburden depths were found to increase north of the drilling, and thin cover was identified between the drilling area and the Dome Mountains to the southwest (Sandburg, 1984).

The QP is aware of a series of IP and resistivity interpretations dated March 1987, from Mining Geophysical Services Inc., but is not aware of any additional supporting documentation on which to base further discussion.

In 1989, at least one northwest oriented seismic line was run in the main drilling area by Cooksley Geophysics, Inc (Cooksley 1989). The preliminary report does not go into detail on station spacing or data collection, however, the proposal outlines the spacing of stations at 33 ft, using dynamite as the source with shots spaced at 66 ft. Telesto (2011) noted that results from seismic testing were inconclusive. The QP’s review of preliminary interpretations suggests structures with orientations similar to those observed at Copperstone were picked up by the seismic survey.

In October 1990, Cyprus conducted another IP and resistivity survey using Mining Geophysical Services Inc. Ten lines were surveyed using a dipole-dipole electrode configuration with a dipole size of 200 ft. Eight lines were surveyed with a dipole size of 400 ft. The base line was oriented S60E from the Reservation fence. The geophysical lines were run parallel to the Reservation boundary (N18E) at 400-ft intervals. The IP survey

detected anomalous responses, “associated with high resistivity trends believed to reflect sulfide mineralization at depth in a bedrock assemblage” in the northwest half of the property. Resistivity identified apparent effects of a fault contact with granitoid rocks in the north and metasediments to the south. Additionally, the resistivity survey identified high resistivity dike trends south of the fault in the metasediments that may reflect varying degrees of alteration or faulting.

The QP is not aware of any exploration activity, other than drilling, conducted from 1991 to 2000. In July 2000, surface structural mapping was conducted by Asia Minerals personnel within the Copperstone pit. The purpose of the mapping was to ascertain the thickness and configuration of the Copperstone fault, determine RQD characteristics in the footwall, and identify potential weak zones in the open pit high wall. The mapping allowed evaluation of alternative sites for constructing the portal for the underground development work described below (MDA, 2000).

In July 2000, channel sampling in the lower-level pit walls was initiated by Asia Minerals to determine the grade and thickness of the exposed portion of the C zone mineralization, and other structures thought to contain gold mineralization.

American Bonanza started driving an underground exploration decline into the C95-10 area and proximal to the high-grade intercepts of the D zone at the time of the MDA technical report (MDA, 2000). Subsequent mapping and underground sampling were completed.

AMEC reviewed surface and underground mapping by American Bonanza and found them to be conducted in an industry standard manner (AMEC, 2006). The QP did not review the surface and underground sampling work conducted by American Bonanza, and no results were included in the mineral resource model.

In 2004, CloudStreet acquired an ultralight-borne aeromagnetic survey, on behalf of Pearson, deRidder and Johnson, Inc (“PRJ”). The survey area, approximately 2 mi east-west by 3.5 mi north-south, is over and immediately surrounding the Copperstone mine. East-west flight lines were at 50-ft spacings, north-south tie lines were at 100-ft spacings, and elevation did not exceed 150 ft above the surface resulting in approximately 1,254-line miles of coverage. PRJ provided Total Magnetic Intensity (“TMI”) and Reduced to Pole TMI interpretations of the data (PRJ, 2004). A report discussing the results of the survey are not currently in the QP’s possession.

In 2007, Zonge Geosciences, Inc. (“Zonge”) performed a GPS-based ground magnetic survey. Ground magnetics data were acquired on 38 lines oriented north-south, and 9 tie lines oriented east-west for a total distance of 40-line miles of data acquisition (Zonge, 2007). The survey was at a higher resolution than the earlier Cyprus/Amoco survey, in the attempt to detect magnetite associated with gold mineralization. The survey failed to define known areas of magnetite, but did find some smaller magnetic anomalies missed in the earlier surveys (Telesto, 2011).

Zonge staff have interpreted that the horizontal gradient of magnetic potential survey shows a direct correlation of a slow change in magnetic field and known gold mineralization. These anomalies are apparent in the Copperstone pit, and the southwest target. Subsequent drilling of the southwest target confirmed the presence of gold mineralization. Similar anomalies are present on the northeast part of the property,

including the State Claim areas. These targets have not been tested by drilling, and may represent exploration potential.

In 2007, Zonge re-interpreted the Cyprus/Amoco IP data. The strong IP anomaly in the mine area appears associated with the Copperstone fault. A few drillholes have been drilled, along strike, with mixed results suggesting that some exploration potential may exist along strike of the fault to the south.

In March of 2021, Ore Pro was engaged to perform the sampling and metallurgical testing of the Copperstone tailings. Samples were obtained using two methods, described as follows by Ore Pro (2021):

The first sampling method utilized a Ditch Witch FX-30 Vacuum Excavator. Two sections of 6-foot long x 3-inch ID ABS stingers were used to extract samples to a maximum depth of twelve feet. The tailings samples were extracted to the specified depth and into the vacuum tank. Then the samples were dumped out of the tank onto a piece of plywood where they were logged, bagged, labeled, and shipped to Ore Pro's mineral dressing laboratory in Tucson, AZ. The vacuum tank, stinger and hose were cleaned with the onboard pressure washer between each hole to prevent contamination. An Arizona Gold employee was present during the sampling operations, Ore Pro never entered the tailings impoundment without an Arizona Gold employee present.

The second method utilized a 4-inch solid flight auger mounted on a Bobcat T870 tracked skid steer loader. There were 3 each, 4-foot sections of 4-inch auger that were capable of drilling to a maximum depth of 12 feet. The samples were placed on a sheet of plywood and coned and quartered. The samples were logged, bagged, labeled, and shipped to Ore Pro's mineral dressing laboratory in Tucson, AZ.

Individual samples were removed from the sample bags and placed on individual polypropylene tarps for air drying. The samples were turned over by shovel and clumps were broken up until no moisture was visible. After air drying, the individual samples were then rolled six-times from each corner of the tarp for mixing. Larger clumps were broken up until the largest clump was about one half-inch top size. The sample was reduced in size using the "cone and quartering" technique. A sample of approximately four kilograms was removed from each sample bag in this manner. The sub-sample was then passed through a riffle splitter with 3/4 -inch slots. After mixing each sub-sample twice, a final assay sample of about 1500-grams was split out. The remainder of the sub-sample was recombined with the cone and quarter material and placed in the original sample bag. Each 1500-gram assay sample was dried in an oven at 220-degrees F. All clumps were easily broken up after drying. The dried sample was well mixed again and a 150-gram assay sample was then pulverized in a ring and puck vibratory pulverizer to approximately minus-200-mesh. This sample was placed in an assay envelope for shipment to Florin Analytical Services, a department of Kappes, Cassiday & Associates, in Reno, Nevada. The remainder of the 1500-gram assay sample (un-pulverized) was placed into a plastic bag. Six of the tailings samples were then selected for bottle roll testing.

The average of the 38 tailings samples by fire assay is 0.81 g/t Au with a range of 1.81 to 0.55 g/t Au. The average 24-hour agitated cyanide soluble copper, silver, and copper values are 0.57 g/t Au, 0.15 g/t Ag, and 598 g/t Cu. Presuming the gold fire assay result for each sample is

approximately the value of the agitated cyanide head sample then the cyanide soluble gold value gives an estimation of the percent gold recovery of each sample under these leaching conditions. The cyanide leach recovery is estimated as 68% Au recovery. The cyanide soluble gold recovery ranges from 93-percent to 1-percent.

Cyanide leach tests were performed on six of the tailings samples to determine the gold recovery in a 24-hour leach cycle. The bottle roll test procedure is a common method to determine the efficacy of agitated cyanide leaching. The tests run in this set of samples were not designed to optimize the leaching cycle such as leach time, cyanide and lime consumption, percent solids, etc. The leach test criteria were proven by Ore Pro previously as acceptable for producing good soluble gold recovery on other Copperstone tailings samples.

Reviewing these results show two obvious trends. First the calculated gold recoveries are similar from tailings samples taken from different parts of the impoundment. Gold recoveries range from 81.8 % for the lowest grade sample (0.48 g/t Au) to 92 % for the highest-grade sample (1.02 g/t Au). The other important result is that all the samples leached had about 0.11 g/t Au leached tailings assay even though the calculated head grade of the samples ranged from 0.48 g/t Au to 1.02 g/t Au.

6.3 Historical Drilling

Drilling exploration has been carried out at Copperstone in various campaigns, but fairly consistently on an annual basis since 1975. Drilling carried out by previous operators is summarized in Table 6-1. No documentation or other information is available for drilling completed by Newmont and Southwest Silver; the drillholes from these campaigns are not included in the Project database nor are they discussed further here.

Table 6-1 Drilling Carried out by Previous Operators

Company	Year(s)	No. of Holes Drilled	Drilled Footage	Drilling Method
Newmont	1975	1	unknown	unknown
Southwest Silver	1980	6	unknown	RC
Cyprus	1980-1986	578	246,676	mostly RC
Sante Fe	1993	17	12,500	mostly RC
Royal Oak	1995-1997	34	28,414	mostly RC
Asia Minerals	1998-2000	37	20,482	Mostly DDH
Bonanza	2003-2005	262	171,125	Mostly DDH
Bonanza	2006-2008	59	57,641	DDH
Bonanza	2012-2013	162	14,247	-
Arizona Gold Corp. (Kerr)	2015-2021	252	73,080	DDH and RC
Sabre Gold Mines Corp.	2021	85	9,855	RC

Historic drilling carried out by previous operators of the Copperstone Project accounts for all of the drillholes included in the Project database. The earliest drilling for which a reasonable amount of associated information is available was completed in 1984 by Cyprus, with various subsequent drilling campaigns completed prior to the merger between Arizona Gold Corp. and Golden Predator resulting in the formation of Sabre Gold Mines Corp. The historic drilling database contains 475 core holes totaling 251,191.40 ft, 831 RC holes totaling 352,534.8 ft, 18 drillholes with RC pre-collars continued with core drilling ("RC/Core") totaling 16,045.0 ft, and 162 drillholes totaling 14,247.0 ft drilled for production using unknown methods. Most historic drilling was oriented either vertically or angled to the southwest in an effort to perpendicularly intercept the mineralization. Underground drilling by American Bonanza from 2003-2005 is oriented down dip of the mineralization. Limited details are presently available regarding drilling contractors and procedures specific to each campaign. Table 6-2 summarizes historic drilling at the Copperstone Project by operator and by year.

Table 6-2 Summary of Drilling Conducted by Previous Operators by Type and by Year

Operator	Year	Type	Prefix	Count	Total (ft)
Cyprus	1984	Core	CSD-	13	3,406.00
		Core		55	17,209.40
	1985	RC	CS-, CR-, CSR-	504	222,425.00
		RC/Core	CSD-11	1	546
	1986	Core	CSD-	2	1,189.00
	1988	Core	CSD-	3	1,900.30
Santa Fe Gold	1993	RC	DCU-	17	12,500.00
Royal Oak	1995	Core	C95-	2	1,244.50
		RC		11	8,757.00
	1996	Core	C96-	4	4,269.00
		RC		1	958
		RC/Core		1	1,227.00
	1997	Core	C97-	11	8,636.00
		RC		2	1,820.00
		RC/Core		2	1,502.00
Asia Minerals	1998	Core	A98-	15	10,979.20
	2000	Core	A00-	5	3,209.50
		RC		6	5,400.00
	2001	Core	CDH-	11	893
American Bonanza	2003	Core	CRD-03-, CUDH-03-	26	9,808.30
		RC	CRD-03-	2	1,195.00
	2004	Core	CUDH-04-, DU4-, F4-, H4-	113	80,622.10
		RC/Core	CRD-04-	13	11,841.00
	2005	Core	DU5-, H4-, H5-	36	18,109.60
		RC	H5-	71	48,619.80
		RC/Core	H5-124	1	929
	2006	Core	06CS-	27	25,510.00
	2007	Core	07CS-	17	17,983.00
	2008	Core	08CS-	15	14,147.50
	2012	Unknown	DZ-, DZ12-	8	800
Arizona Gold Corp. (Kerr Mining Inc.)	2013	Percussion	520-, 650W-, 654-, 654cc-, 690-, 726-, 730-, 750-, 810-	154	13,447.00
	2015	Core	KER-15-	4	3,045.50
	2017	Core	KER-17S-, KER-17U-	72	19,380.50
		RC	KER-17S-	11	7,360.00
	2019	RC	18-	100	17,020.00
	2020	Core	AZG-20	3	1,063.00
		RC	AZG-20S	20	16,060.00
	2021	Core	AZG-21	28	7,493.00
		RC	AZG-20S-	1	565.00
		Infill Core	AZG-21-PXX-XXC	13	1,093.00
Sabre Gold Mines Corp.	2021	Infill RC	AZG-21-PXX-, 21-PXX-	85	9,855.00
Total Historic Drilling				1,486	634,018.20
Total Historic Core				475	251,191.40
Total Historic RC				831	352,534.80
Total Historic RC/Core				18	16,045.00
Total Historic - Percussion				162	14,247.00

6.3.1 Cyprus 1984 – 1988

Drilling by Cyprus accounts for 39% of total drilling on Copperstone, with 73 core holes totaling 23,705 ft, 504 RC drillholes totaling 222,425 ft, and 1 RC/core drillhole totaling 546 ft. The drillhole database indicates only one drillhole was surveyed down-the-hole. Of the Cyprus drillholes, 547 are angled vertically, 2 are angled to the northwest, 4 are angled to the southeast, 8 are angled to the southwest, 2 are angled west, and

15 do not have survey information in the database and are presumed to be vertical. Drilling by Cyprus defined gold mineralization in the A/B zone and C zone, and provided the first indications of D zone, Footwall zone, and South zone gold mineralization. Exploration drilling to the southwest and northeast did not return significant results.

The primary drill core and chip samples generated by Cyprus were shipped to a Cyprus prep facility in Montana where pulps were made. Pulps were distributed to the various analytical labs from that site (McCartney, 1998).

Cyprus used several laboratories as their primary assay laboratory for exploration drillholes. Before constructing the Copperstone mine laboratory, drill samples were sent to Cone Laboratories in Reno, Nevada, GeoMonitor in Hesperia, California, or CYMET (location not documented) for assay. Cone, GeoMonitor and CYMET all assayed gold and silver by the same methods. Gold and silver were assayed by digesting (amount not documented) sample in aqua regia and reading the gold concentration on an atomic absorption spectrometer. Select mineralized intervals were assayed for gold by standard fire assay on a one assay ton pulp sample with gold concentrations read on an atomic absorption spectrometer following dissolution in acid. Copper was assayed for select drill intervals by aqua regia digestion and read on an atomic absorption spectrometer.

Drill samples assayed at the Copperstone mine lab were assayed by cyanide leach and mineralized intervals (generally Au cyanide greater than 0.1 oz/ton) were assayed by standard fire assay on a one assay ton (29.157 grams) pulp sample with gold concentrations read on an atomic absorption spectrometer following dissolution in acid.

The QP is unaware of any information indicating Cyprus applied any QA/QC measures during their drilling operations. However, steps have been taken by other operators to verify their assay results.

Upon AMEC's recommendation, American Bonanza submitted 232 Cyprus pulps (approximately 7.5% of all Cyprus gold assays) to AAL for check assay to determine the accuracy of the gold assays from the Cyprus drill campaign. The Cyprus drill intervals were selected randomly by AMEC from the assays entered from Cyprus drillholes.

AMEC plotted the results of the check assay program and found the Cyprus assays to be accurate with no significant bias relative to the AAL check assays. Three of the 12 SRMs submitted with the program were outside acceptable limits. AAL re-assayed project samples around the three failed SRMs and found no significant change in gold concentration. AMEC concluded that the accuracy of the Cyprus assays was acceptable based on the results of the check assay program.

The QP is not aware of any documentation pertaining to the security of samples during Cyprus's drilling operations.

6.3.2 Santa Fe Gold 1993

Santa Fe Gold completed 17 RC drillholes totaling 12,500 ft in 1993. No drillholes were surveyed down-the-hole. Ten drillholes, all angled 65 degrees southwest, tested the area southwest of the central portion of the

pit and did not intersect significant mineralization, although some intervals of low-grade mineralization were encountered in DCU-2. One vertical drillhole (DCU-8) within the pit intersected Footwall zone mineralization with 10 ft of 0.920 oz/ton gold. Two vertical drillholes near the pit to the northeast intersected low-grade gold mineralization. Four vertical exploration holes to the northeast did not intersect gold mineralization.

The QP is not aware of any documentation regarding Santa Fe's sampling procedures during their drilling operations. However, only one 10-ft zone of significant mineralization was encountered. The QP reviewed the results of nearby drilling and notes significant intercepts of similar grade within 50 ft of Santa Fe's interval. The QP did not find sufficient reason to disqualify Santa Fe's drilling.

6.3.3 Royal Oak 1995 – 1997

Royal Oak completed 17 core drillholes totaling 14,149.5 ft, 14 RC drillholes totaling 11,535 ft, and 3 RC/core drillholes totaling 2,729 ft. A total of 10 core, 7 RC, and all three RC/core drillholes were surveyed down-the-hole using a single shot camera. The remaining 14 drillholes were not surveyed down-the-hole. Drilling targeted the down-dip extension of the A/B zone, C zone, and D zone gold mineralization to the northeast, with all but 5 oriented vertically. Those 5 angled drillholes were oriented southwest. The programs were successful in demonstrating continued gold mineralization down dip to the northeast. One drillhole, angled southwest on the southwest margin of the pit (C95-02) did not intersect gold mineralization.

The QP is not aware of any documentation pertaining to the preparation of samples during Royal Oak's drilling operations.

Royal Oak, like American Bonanza, employed AAL as their primary assay laboratory. AAL assayed gold by standard fire assay on a 30-gram pulp sample with gold concentrations read on an atomic absorption spectrometer following dissolution in acid (AMEC, 2006).

The QP is unaware of any information indicating Royal Oak applied any QA/QC measures during their drilling operations. However, steps have been taken by other operators to verify their assay results. These efforts have been reviewed by independent parties in 2006 and 2011. The conclusions from both reviews are detailed below. It is the QP's opinion that sufficient work has been done to verify the results for Royal Oaks drilling.

Upon AMEC's recommendation, American Bonanza submitted 30 Royal Oak quarter core samples to AAL for check assay to determine the accuracy of the gold assays from the Royal Oak drill campaign. AAL assayed according to American Bonanza's normal protocol (2AT fire assay with gravimetric finish). The Royal Oak drill intervals were selected by AMEC from available core intervals located at the Copperstone core shed. The highest-grade intercepts were not available for sampling as they had apparently been consumed for check assay and metallurgical purposes. Information on Royal Oak's original QA/QC programs was not provided to AMEC.

AMEC plotted the results of the check assay program and found the AAL gold assays to be significantly different than the Royal Oak assays. There is, in fact, an extremely poor correlation between the paired gold assays. SRM results for the program were found to be acceptable. Given that no original QA/QC data exist to

support the Royal Oak gold assays and that this recent quarter-core check assay program does not support the accuracy of the Royal Oak gold assays, AMEC has a low confidence in the original Royal Oak gold assays. In total, there are 28 Royal Oak gold assays in the assay database that are greater than 0.1 oz/t. AMEC recommends that American Bonanza resample these zones where possible, re-assay them, and replace the original Royal Oak assays with these assays. Where this is not possible, AMEC recommends that American Bonanza re-drill these zones where no supporting assays from Cyprus or American Bonanza core samples are available.

In 2006 AMEC recommended checking the assays from the Royal Oak core holes. Thirty quartered core samples were submitted by AMEC to resample the core with only one sample from within the current resource area. The assay results from AMEC were never provided to American Bonanza and not put in the 2006 Preliminary Assessment Report (Telesto, 2011).

Based on AMEC's 2006 Preliminary Assessment Report, the correlation coefficient performed by AMEC had an R value of 0.71. Although the AMEC R-value was low, the samples collected by AMEC were outside the resource area and have no influence in American Bonanza's resource. Also, every sample collected by AMEC was significantly below the cut-off grade which also has no effect on the gold mineral resource.

To ensure there was no issue with the Royal Oak drillhole data, American Bonanza reviewed the Royal Oak's certified assay sheets from AAL and discovered a total of 18 samples were sampled twice within the resource area. Of these 18 samples 11 of the samples were above the expected cut-off grade of 0.14 oz/ton Au.

American Bonanza performed two correlation coefficient charts for the 18 samples in the resource area that were sampled twice and for the 11 samples that were above the 0.14 ounce/ton Au cut-off. A trend-line was drawn through both correlation coefficient charts. The R-value for all 18 samples that had been assayed twice was 0.9884 and the R-value for the 11 samples above the 0.14 oz/ton Au cut-off was 0.9857. These R-values show very good correlation and give high confidence that the values are correct and that the Royal Oak Mines RC and core sample assays are accurate.

The QP is not aware of any documentation pertaining to the security of samples during Royal Oak's drilling operations.

6.3.4 Asia Minerals 1998 – 2001

Asia Minerals drilled 20 core drillholes totaling 14,188.7 ft, and six RC drillholes totaling 5,400 ft from surface drillhole locations.

The Lang Division of Boart Longyear drilling contractors, based in Salt Lake City, Utah drilled the RC holes. The RVC holes were 6.5 in (16.5 cm) in diameter, and samples were collected at 5-ft. (1.5 m) intervals. HQ size drill core (6.4 cm in diameter) was collected by standard diamond impregnated drill bits, metal alloy core barrel and wire-line methods.

Locations of drillholes were defined on the established Copperstone mine survey grid, using the original Cyprus benchmarks. Drillhole collar locations were surveyed by Bill Lemme, a professional engineer and Arizona registered surveyor. Mr. Lemme was formerly employed as a surveyor for the Cyprus mining

operations and conducted surveys on the property for Royal Oak. Down-the-hole surveys using a standard single shot camera survey tool were completed at regular intervals for all but 8 vertically oriented surface drillholes.

Surface exploration intersected D zone gold mineralization with decreasing returns as drillholes testing the down dip extension of the D zone to the northeast. One drillhole (A00-10) intersected Footwall zone gold mineralization in the same area as DCU-8.

Asia Minerals drilled 11 core holes in 2001 for a total of 893 feet. Given the relatively shallow depth (<120 ft.), none of the underground drillholes were surveyed down-the-hole. The drilling was carried out from a single underground station in the D zone, near an interval of 25 ft. at 2.199 oz/ton gold in C95-10. Drilling was oriented northeast to fan out radially between 35 – 125 degrees azimuth and inclined between 11 degrees below horizontal and 20 degrees above horizontal. Drilling at these orientations is not conducive to intersecting D zone gold mineralization. The drillholes run the risk of following mineralized structures down dip resulting in interval lengths not representative of true thickness or missing the mineralization altogether. Asia Mineral's underground drilling campaign only intersected a few sporadic intercepts of significant grade, indicating that the drillholes missed D zone gold mineralization.

The discussion for Asia Minerals sampling procedures, analysis, and security during their operations relies heavily on the technical report completed by MDA in 2000. The report was written at the time of Asia Minerals operation. The QP reviewed supporting documentation from Asia Minerals and found no discrepancies.

Two geologists conducted the sampling program. The first 2/3 of the 2000 program was under the supervision of Graham Kelsey. The last 1/3 of the 2000 program (including surface sampling), and all of the 1998 program, was under the supervision of Ian McCartney. Mr. Kelsey is a geological consultant to Asia Minerals and was the former Chief Geologist at the Copperstone Mine for Cyprus. He has over 20 years' experience in gold and base metal exploration, much of that in Arizona. Mr. McCartney, presently a geological consultant to Asia Minerals, is a P.Eng. in Canada with over 20 years international experience in gold and base metal exploration.

The contractor's RC drill rigs were well equipped with high volume/pressure compressors and air-cyclone sample collection equipment standard to the industry. The drill column was regularly air-purged by the drill operator during the sampling routine. Standard care was taken to minimize sample contamination from the drill column, collection and field-splitting equipment. The cyclone-splitter unit was periodically washed with a pressurized water hose to remove residual material. Samples were airdried on site prior to shipping to the lab.

The RC samples were collected at 1.5-meter intervals and split at the drill site using a cyclone/rotating cone splitter unit. A member of the contractor's drill crew was trained and assigned the sampling function. When drill chip returns were sufficient, the splitting equipment was adjusted to deliver a minimum of 7.26 kilograms per 1.5 meters to the sample port.

The sample was initially collected in a cleaned 22-liter plastic bucket, then transferred into pre-numbered porous cloth bags with hole ID and from-to footage permanently marked on both the exterior of the bag and a sample card affixed to the bag. The RC sample splits had an average weight of approximately 13 kilograms.

Drill cores were placed into standard waxed cardboard core boxes (3 meters/box) at the drill, and wood footage indicator blocks were placed at the end of each core barrel pull interval. Each core box was permanently labeled with appropriate "Hole ID" and "from/to" interval lengths. A lid was affixed upon filling each box.

Prior to core logging and splitting, geotechnical data was collected for all core holes (logs were completed for all core holes), including RQD data, interval recovery percentage, and other parameters. These measurements were systematically recorded on a standardized geotechnical log form. The core surface was brushed/washed with water to remove drilling mud and photographed prior to logging. Drill core was logged for lithology, mineralization, alteration, and structural features according to a standardized logging template and recorded by Asia Minerals personnel on a standardized geology log form.

Geology codes and observations were marked on the core using permanent yellow or blue wax china markers. Key mineral observations to be preserved for the reference archive portion of the core were circled or X-marked with red china marker.

Core sample intervals were determined by the on-site Asia Minerals geologists on the basis of lithologic, alteration, or mineral abundance contacts where observed. The objective in determining sample boundaries was to characterize the geological association of gold. The maximum core sample interval was 1.8 meters, the minimum interval was 0.6 meter. Sample interval boundaries and interval depth designations were permanently marked on each sample segment with a red china marker prior to sawing.

Core samples and inserted coarse reference samples were assigned sequential series of numbers that are independent of hole number or footage, using standard triplicate lab tag booklets. Duplicate assay tags were inserted in the core boxes at the downhole end of each sample interval. Only the pre-numbered sequential sample ID sequence was indicated on the assay tags. Coarse reference standards were slotted into the sample sequence after suspected highly mineralized intervals using the sequential number system.

The triplicate assay tag retained in the sample booklet was filled out with relevant Hole ID, sample interval and geology code(s). The sample ID, and sample interval was also entered on the corresponding geologic log form.

Drill core was sawed in half utilizing a conventional rock saw and water-cooled diamond impregnated blade. The saw blade and core-carriage were pressure sprayed and cleaned of rock fragments between each sample interval. After sawing a sample interval, one assay ticket was taken from the core box and inserted with one-half of the core in a sample bag. The half core retained for the reference archive was systematically returned to the core box with the duplicate tag positioned at the end of each sample interval. Attempts were made to either retain or re-label prior core markings for the archive cores.

Visible gold, where observed in only one-half of the core pair, was retained with the archive/reference core portion. The outside of each laboratory sample bag was labeled with permanent magic marker to indicate only the sample number. Sample bags were immediately and permanently sealed with lock ties.

In 1998, Asia Minerals utilized Intertek Testing Services (Bondar Clegg) as the principal analytical laboratory for all Copperstone project drill sample analyses. In 2000, Bondar Clegg was also used, but the lab was no longer affiliated with Intertek. Initial sample prep was conducted at the Bondar Clegg sample preparation facility in Reno, NV. Sample pulps were then forwarded directly by the prep facility to the Bondar Clegg analytical lab in Vancouver for analysis. Trace element ICP geochemical analyses were conducted at the Skyline-Actlabs facility in Tucson, Arizona. Both Bondar Clegg and Skyline-Actlabs are reputable and industry-respected commercial analytical service organizations that operate on an international scale. Lab procedures, including any variations to the lab's routine procedures, are summarized below:

- Samples were dried in stainless steel trays (sample tags remained with sample in drying equipment);
- The total sample was crushed to 95% -10 mesh (about 2 mm). Bondar Clegg's normal process was crushing to 75% -10 mesh.
- Crushed material was reduced with a riffle splitter to separate a nominal 1 kilogram for large pulp prep procedure;
- Rejects from step 2 were stored at the lab for future work;
- The 500-gram subset was pulverized to 90% passing -150 mesh using a shatter box type pulverizer);
- A nominal 250-gram subset of pulp was sent for analysis;
- Remaining pulps were stored for future work.
- Re-homogenization of the nominal 250-gram pulp received from the preparation section;
- A two-assay ton (~58.3 gram) charge was weighed from the 250-gram pulp and subjected to standard fire assay with an atomic absorption finish;
- All values over the range of AA reliability (+10 ppm Au) were re-assayed using two-assay ton fire assay with a gravimetric finish.

Metallic screen or "screen fire" analyses are considered one of the most accurate methods of assaying samples containing significant amounts of particulate or coarse gold. Asia Minerals utilized metallic screen analyses for all sample intervals considered significant by the site geologists. Selection of significant intervals for metallic screen analyses was generally determined by the presence of high-grade mineralization in the routine assays. At the geologist's discretion, sample intervals were submitted for screen fire analyses if warranted to assist in interpretation or confirm results from earlier drilling programs. When submitting sample intervals for screen fire analyses, the entire sample sequence was submitted, including internal zones of both high and low-grade material. All metallic screen analyses of Asia Minerals samples were conducted at the Bondar Clegg facilities. In summary, the screen fire procedures employed by Bondar Clegg were as follows:

- A large, nominal 1000-gram, pulp was prepared per normal sample prep protocols;
- A 500-gram pulp was weighed and wet-sieved through a 150-mesh screen with the plus and minus 150 mesh material segregated and dried;
- After homogenization, a two-assay ton split of the minus 150 mesh fraction was fire assayed;
- The entire plus 150 mesh fraction was weighed and fire assayed;
- The two fractional assays were combined by weight-averaging to determine the reported assay.

Thirty-element ICP scans were performed on a subset of the mineralized samples, including all samples within significant intervals. This scan included, among others, the following elements which were considered of greatest interest to Asia Minerals geologists: Cu, Ag, Fe, Mn, As, and Sb.

As a quality assurance measure, BZA reports that Asia Minerals geologists routinely inserted coarse, barren reference material into the drill sample series following drill samples with observed visible gold or following intervals of suspected high-grade mineralization. The barren standards allowed monitoring for possible contamination, and grade smearing from the laboratory's sample prep equipment during processing of high-grade drill samples.

Barren core reference material was prepared on-site by selecting intervals of an existing drill core exhibiting a minimum of 6 m of continuous assays below the lab's gold detection limit of 5 ppb. Reference material was visually inspected by Asia Minerals geologists and prepared from barren, non-fractured, unaltered, sulfide-free, and uncontaminated half-cores from drill intercepts in unmineralized hanging wall geological units. Selected coarse standard material was broken into small pieces and homogenized on a cleaned concrete slab. The coarse, barren material was then placed in clean unmarked sample bags of the same type used to submit drill core samples to the analytical lab.

The coarse, barren standard material was documented and inserted into sample series by Asia Minerals personnel. The coarse standards were inserted at irregular intervals and submitted with the assay lab shipments per the previously described protocol.

Asia Minerals monitored the commercial lab's sample preparation and analytical performance by inserting "blind" coarse blanks, reject duplicates, core shed reference samples and CANMET Standard Reference Material ("SRM") into the sample stream. The QA/QC protocols applied generally conformed to those recommended to Asia Minerals by MRDI (1999). Control samples inserted by Asia Minerals comprised approximately 5% of the sample stream through each of the major components in the primary lab's sample preparation and analyses flow sheet. Appropriate material (coarse pre-crush blanks, duplicate rejects, certified standard pulps, etc.) was utilized for quality assurance throughout the prep and assay procedures. The reference materials were inserted with no unique identifiers that would differentiate them from other materials in the sample prep/analytical stream.

A 120-gram split of every 20th process pulp was prepared and submitted to the qualified umpire laboratory. The primary preparation facility was also responsible for inserting blind pulp blanks and Asia Mineral's SRM's into sample batches submitted for umpire assay. The analytical protocols of the umpire lab were the same as those used at the primary lab.

Asia Mineral's primary lab was the Bondar-Clegg facility in Vancouver, B. C. American Assay Laboratories, located in Reno, Nevada was used as the umpire lab. American Assay is a reputable facility and participates in a full ISO certification program. Asia Minerals instituted the corroboration program recommended by MRDI (1999). The primary lab's preparation facility was responsible for preparing coarse reject duplicates (1 in 20), coarse (pre-crush) blanks (1 in 20) and for inserting client supplied SRM (1 in 25) into the process sample stream. The SRM used was CCRMP gold mineralized material MA-1b with a certified value of 0.497 +/- 0.008 oz/T gold. Following insertion of the quality control samples, a new sequential number sequence was applied to ensure no unique identifiers for QA/QC materials would appear in the submitted sample ID sequence as received by the analytical facility.

The primary analytical facility was requested to report the results of its own internal (non-client) QA/QC sample checks for all batches. These included standard pulp and blank duplicate analyses.

A 120-gram split of every 20th process pulp was submitted to the umpire laboratory. In addition, all samples within significant drill sample intervals were submitted to the umpire lab, including any internal low-grade samples within the interval. All quality assurance samples were prepared and delivered to the umpire lab by the primary lab's Reno preparation facility. The primary preparation facility was also responsible for inserting pulp blanks (1 in 20) and client SRM (1 in 20) into the batches for the umpire lab. The samples were renumbered in a similar fashion to the renumbering employed for the primary analytical facility. The assay procedures used at the umpire lab were the same as those of the primary lab.

Due to budgetary constraints, no check assays were submitted to the umpire lab during the 2000 drill program.

The ability to consistently produce acceptable results in the analytical end of the primary and umpire lab's flow sheet was monitored with certified standards and blanks (SRM). The mean of the SRM assay values during Asia Mineral's program was 0.508 oz/ton Au versus the certified value of 0.497 +/- 0.008 oz/ton Au. The relative difference was +2% during the 1998 drilling program. The SRM analysis reported an acceptable relative difference consistently below 5%. Only one of the standard blank pulps submitted reported a value in excess of twice the lab's lower detection limit of 5 ppb (MRDI, 1999).

Results of the lab's internal monitoring via periodic repeat analyses of pulp duplicates and (non-client) standard blanks were also well within acceptable tolerances.

Coarse reference material and duplicate reject analyses were used to confirm the integrity of the sample preparation protocols and monitor performance of the primary lab. Concern regarding potential contamination of sample prep equipment from processing high-grade Copperstone samples was mitigated by intentionally inserting coarse blanks into the sample stream following suspected high-grade sample intervals. When discrepancies in the analyses of the coarse sample blanks were indicated, they were investigated and resolved to Asia Mineral's satisfaction by the primary laboratory. Given the inherent variance of Asia Mineral's coarse reference material, comparative analyses of coarse blanks and duplicates conducted by the primary lab reported relative variance within acceptable limits.

Sample pulps, standards and blanks submitted for analyses at the umpire facility reported very high correlation (correlation coefficient of 0.99) of primary and umpire lab results. Relative variance for the sample population (92 samples of varied grades) submitted for umpire analyses was well within industry standards.

All samples remained in the custody of Asia Minerals geologists until shipped to the analytical laboratory. All access to drill core logging, sample preparation and storage facilities by other than Asia Minerals personnel was recorded; including the holes and intervals possibly accessed and examined in each instance. The on-site geologist maintained a standard log form for this purpose. All drill core was stored in a sealed and locked shipping container on site. During logging, drill core was returned to the container for overnight storage.

Shipping from the project site to the lab was by USF Bestway, a lab-designated commercial carrier. The samples were picked up from the project site on 1.3 x 1.3 x 1 meter crib pallets, supplied by the analytical lab, and delivered directly to the sample preparation facility. Chain of custody control was documented through standard bills of lading, as well as MRDI-recommended chain of custody forms.

6.3.5 American Bonanza 2003 – 2013

American Bonanza is responsible for 32.9% of the total footage drilled at Copperstone. The following details regarding American Bonanza's drilling efforts are summarized from AMEC (2006) and Telesto (2011).

Drill contractors for the 2003-2005 drilling included Ruen Drilling (diamond) of Clark Fork, Idaho, Layne-Christensen (diamond and RC) of Chandler, Arizona, and Diversified Drilling (RC) of Missoula, Montana. Diamond tools employed included HQ (63.5 mm) diameter tools for surface holes and NQ (47.6 mm) diameter tools for underground holes. American Bonanza drillhole collar locations were surveyed by American Bonanza geologists using a Trimble TSC-GPS system. Underground collar locations were surveyed by transit and chain from control points established by Lemme Engineering Inc., of Phoenix, Arizona.

During the 2003 to 2008 drill programs, American Bonanza RC pre-collar drillholes were surveyed for dip with a single-shot camera within the drill steel at 100-ft intervals. This was done to ensure that drillholes did not droop or rise beyond acceptable limits. RC drillholes deviating more than 3° were terminated and redrilled. Upon completion of the core tail, the RC pipe was removed and the entire drillhole (pre-collar plus core tail) was surveyed by Wellbore Navigation Inc. (WELNAV) of Tustin, California, using a gyroscopic multi-shot tool, which returned azimuth and dip readings at nominal 50-ft intervals. Bonanza underground core holes were surveyed using a single-shot camera at nominal 100-ft intervals.

American Bonanza drill collar locations were surveyed by American Bonanza geologists using a Trimble TSC-GPS system. Cyprus established benchmarks provide survey control. Locations were surveyed in Arizona state plane coordinates, downloaded to computer at the site, and transferred in a spreadsheet to the American Bonanza office in Reno, Nevada for loading to the project database. Underground collar locations were surveyed by transit and chain from control points established by Lemme Engineering Inc., of Phoenix, Arizona.

The following drillholes were not down-hole surveyed:

- Bonanza RC holes CRD-03-01 to 13, nominal 600 ft total depth
- Bonanza underground holes CUDH-03-01 to 15, nominal 300 ft total depth (Telesto, 2011).
- Bonanza underground holes CUDH-04-16 to 25 and 27, nominal 300 ft total depth.
- Bonanza Underground Drillholes DU4-49 (186 ft) and DU4-55 (46 ft).
- Bonanza Surface Drillhole F4-5, 880 ft total depth.
- Thirteen RC Bonanza surface drillholes in 2005, nominal 300 ft total depth.
- Ten Bonanza surface holes from 2006 with total depth ranging from 540 ft to 1,140 ft.
- Six Bonanza surface holes from 2007 with total depth ranging from 440 ft to 1,200 ft.

The following discussion on sample preparation employed by American Bonanza is from AMEC (2006):

Drill core is placed into standard waxed cardboard core boxes by the drill helper at the drill site. Core run intervals are marked on wood blocks and placed at the end of each core run. Core boxes are marked with the drillhole name and drill interval.

Drill core is retrieved from the drill rigs two to three times daily by the project geologists and brought to the core shed. There, core is photographed and logged for lithology and geotechnical information. Lithology log fields for each drillhole include rock type, rock qualifier (grain size, fragment types, iron type, etc.), alteration mineralogy and intensity, structure, and reaction to hydrochloric acid.

Core is then marked for sampling by the geologist on nominal two-foot intervals in visibly mineralized material and on nominal five-foot intervals in visibly unmineralized material. American Bonanza drillholes are sampled in their entirety. Marked intervals are sawn in half by a technician at the core shed. A geologist or technician then bags one-half of the core for assay and the other one-half is retained for further study and third-party review.

Samples for a drillhole are submitted to American Assay & Environmental Laboratories (AAL) in Reno, Nevada as a single batch with four SRMs inserted in the project sample stream. Select mineralized intervals are marked for the measurement of specific gravity, which is also determined by AAL in Reno.

RC holes are drilled with water injection to stabilize the holes. RC samples are collected in five-foot intervals by drill helpers at the drill site. Approximately five pounds of material is collected from a rotary splitter (3 of 12 sections open for ¼ split) installed below the cyclone on the drill rig. Samples are bagged in micro-pore bags, prenumbered according to sequentially numbered sample tickets. Sample bags are then loaded into large plastic mesh bags, sealed with tamper-proof ties, and transported to the core shed.

A small portion of the cuttings are washed and placed in plastic chip trays. Chip trays are labeled with the hole name and the sample interval. RC cuttings are logged for lithology information. Lithology log fields for each drillhole include rock type, rock qualifier, alteration mineralogy and intensity, structure, and reaction to hydrochloric acid.

RC samples from a drillhole are submitted to AAL as a single batch with four SRMs inserted in the project sample stream.

At AAL, samples are prepared as follows:

- Samples are first dried at 100°C until sufficiently dry for further preparation.
- Samples are then crushed to 75% passing a 10-mesh screen.
- The sample is then split until a 300 to 500-gram subsample is generated.
- The sub-sample is then pulverized to 75% passing a 150-mesh screen.

AAL conducts grind tests on 15% of the prepared samples. If a sample fails to meet grind specifications, samples around the failed sample are tested and all samples failing grind specifications are re-pulverized to meet specifications.

American Bonanza employed AAL as their primary assay laboratory. AAL assayed gold and silver by standard fire assay on a 2-assay ton pulp sample with gold concentrations read on an electronic balance (gravimetric finish). Between one and three (mostly two) fire assays were performed for gold and silver for each sample. Copper was assayed by digesting 0.5 grams of sample in aqua regia and determining the assay value by atomic absorption spectrometry.

American Bonanza regularly included four SRM samples with each drillhole laboratory submission to monitor and control assay quality. American Bonanza also submitted select drill intervals to an umpire laboratory for check assay. The four SRM's included two Nevada Bureau of Mines certified SRMs and two SRMs generated in-house by American Bonanza from material at Copperstone. SRM assays represent approximately 4% of the American Bonanza assays in the database. AMEC was not provided with American Bonanza's protocol for evaluating the SRM results.

The two in-house American Bonanza SRM's, 'C-Ore' and 'C-Waste', represent ore-grade and waste-grade material, respectively. C-Ore was generated from material collected from the underground muck bay in the decline at the north end of the Copperstone pit. C-Waste was generated from barren RC cuttings from American Bonanza pre-collar drillholes. The SRM material was stored in five-gallon buckets in the core shed and submitted as nominal five-pound samples in the same bags as the Project samples. The SRM material was submitted to the assay laboratory unprocessed (meaning the material was not crushed, homogenized, or otherwise prepared). The C-Ore material was run-of-mine and resembled a coarse rock-chip sample. The C-Waste material was RC cuttings. No certification program was conducted on these materials to establish their homogeneity or recommended values for gold, silver, and copper.

AMEC plotted control charts for SRMs used in American Bonanza's quality control program. The gold control charts for the two NBM SRMs show that, overall, the AAL gold, silver, and copper assays are accurate and show no significant bias (AAL gold assays for SRM NBM-2b are shown to be biased slightly low). The gold and silver grades of the two SRMs are within the range of expected grades from mineralized project samples (though it could be argued that NBM-3b is too high grade), but the copper grades are significantly lower than the expected grades from copper mineralized material.

AMEC reports that as expected, the control charts for C-Ore and C-Waste show that these materials should not be used to control assay quality. The precision of the gold and copper assays is very poor and there is obvious variation with grade over time where the C-Ore and C-Waste material was likely replenished from

different sources. These variations are not related to laboratory accuracy and cannot be predicted. The control charts show the AAL assays to have poor precision. The control charts show an unacceptable number of assays outside the designated limits for individual drillholes. This indicates that, though AAL assays are accurate on average; they are not precise. Put another way, AAL is able to accurately estimate the true value of a material when all the assay values are averaged, but each individual assay may be far from the true value.

At AMEC's request, AAL re-assayed 10 samples (including and around the failed SRM) from each failed batch in 2005 (based on NBM-2b assays). AMEC reviewed the results of this program and recommends that the new assays replace the old assays in the assay database. The re-assay gold values for NBM-2b for all but one batch were within acceptable limits. AMEC recommended that American Bonanza monitor the quality of AAL gold assays more closely and instruct AAL to re-assay batches which fail to meet quality control standards set for the program.

American Bonanza conducted a program of submitting select drill intervals to Inspectorate in Reno, Nevada to check the accuracy of AAL's gold assays. AMEC was unable to evaluate this check assay program because American Bonanza had reused sample number sequences and was unable to provide AMEC with a key to the original samples to compare the check assay values.

AMEC plotted the absolute relative difference (ARD) for pulp duplicate pairs against the cumulative frequency of the distributions for gold and silver for American Bonanza drill samples. AMEC considers assay precision to be adequate when greater than 90% of the pulp duplicate pairs yield absolute relative differences of less than 10%.

When plotting all duplicate pairs, the AAL precision for gold and silver assays is adequate. Approximately 89% of the gold duplicate pairs and 96% of the silver duplicate pairs yield absolute relative differences of less than 10%. However, when the duplicate pairs, whose average assay is at or below the lower detection limit are removed from the plots, the precision for gold and silver degrades significantly. Approximately 62% of the gold duplicate pairs and only 7% of the silver duplicate pairs yield absolute relative differences of less than 10%. Typically, the assay precision of a group of samples is improved when pairs near the detection limit are removed because assays at the detection limit are, by definition, $\pm 100\%$. In this case, however, the detection limits for gold (0.003 oz/ton) and silver (0.2 oz/ton) are relatively high and so unmineralized samples (of which there are many) consistently return values below the detection limit, thereby producing a high percentage of zero ARD results.

AMEC concluded that the precision of AAL's gold and silver assays was marginal due to coarse gold and silver and the less-than-optimal sample preparation employed. AMEC recommended that American Bonanza improve the quality of their sample preparation protocol.

Drill samples were transported from the drill site to the core shed by American Bonanza geologists and were stored in the secure core shed before being shipped directly from the mine site to AAL via DATS Trucking, Inc at regular intervals. Drill sample bags were closed with tamper-proof ties, and AAL was instructed to report any missing or damaged sample bags upon receipt.

Drilling results from American Bonanza's drilling programs are subdivided into the following discussions:

- Surface Footwall zone drilling,
- Surface down dip extensional drilling,
- Surface exploration,
- Underground D zone drilling 2003 – 2005, and
- Underground D zone drilling 2012 – 2013.

6.3.6 Surface Footwall Zone Drilling

American Bonanza drilled 37 core drillholes totaling 28,986 ft, and 6 RC drillholes totaling 2,505 ft. Twenty-one of the 43 drillholes were surveyed down-the-hole. Twenty-two drillholes were oriented vertically, 15 drillholes were steeply inclined and oriented in various directions, 5 drillholes were oriented to the southwest, and 1 drillhole was oriented to the northwest. Drilling in the Footwall zone centered around previous results in drillholes DCU-8, and Aoo-10, with continued success. Additionally, gold intercepts in H5-141, H5-147, H5-148, H5-163, and o6CS-17 demonstrated the potential for Footwall zone gold mineralization to continue up-dip to the southwest. South zone gold mineralization was also intersected by American Bonanza.

6.3.7 Surface Down Dip Extensional Drilling

Eighty-three core drillholes totaling 77,034 ft, 56 RC drillholes totaling 36,798.8 ft, and 14 RC/Core drillholes totaling 12,770 ft tested D zone gold mineralization as well as the down dip extents of A/B zone, C zone, and D zone gold mineralization. One-hundred thirty-eight drillholes were surveyed down-the-hole, 4 drillholes were surveyed at the top and bottom of the drillholes, and the remaining 11 drillholes were not surveyed down-the-hole. One hundred drillholes have vertical orientations, 47 drillholes are angled to the southwest, and 6 drillholes are steeply inclined to the south. Results generally continued to demonstrate down dip continuity in the A/B, and C zones, however gold grades do tend to decrease down dip. Drilling in D zone mineralization did intersect favorable gold grades, but down dip extensional drilling in the D zone did not intersect significant gold mineralization.

6.3.8 Surface Exploration Drilling

Thirty-seven core drillholes totaling 37,488.5 ft and 11 RC drillholes totaling 10,511 ft constitute American Bonanza's exploration drilling. Thirty-seven drillholes were surveyed down-the-hole. All 48 drillholes were oriented vertically. In general, no significant gold mineralization was intersected by these drillholes. However, nine drillholes approximately 2,500 ft southwest of the Copperstone pit did intersect significant gold mineralization. These drillholes mark the location of the Southwest zone exploration target.

6.3.9 Underground D Zone Drilling 2003 – 2005

American Bonanza completed 77 core drillholes totaling 22,512 ft between 2003 and 2005 from two underground drilling stations, in order to define D zone gold mineralization. Down-the-hole surveys were completed for 49 of the 77 drillholes. Drilling was oriented northeast to fan out radially between 0 – 190 degrees azimuth, with three drillholes oriented at 345 degrees azimuth, and inclined between 27 degrees above horizontal and 22 degrees below horizontal. While results from this drilling show significant lengths of high-grade gold, drilling at these orientations is not conducive to intersecting D zone gold mineralization.

The drillholes run the risk of following mineralized structures down dip resulting in interval lengths not representative of true thickness or missing the gold mineralization altogether.

6.3.10 Underground D Zone Drilling 2012 – 2013

American Bonanza drilled 162 drillholes totaling 14,247 feet for the purpose of production mining. The drillhole types are not presently known, however, all but one drillhole were surveyed at the top and bottom of the drillhole. For the most part, drilling was oriented to intersect mineralization appropriately, resulting in a multitude of bearings and inclination. The limited knowledge about these drillholes precludes them from meaningful discussions about their results and the drillholes were excluded from the mineral resource estimate.

6.3.11 2015-2021 Arizona Gold Corp. (Kerr) Drilling

Arizona Gold Corp. (“AZG”) formally Kerr Mines Inc. (“Kerr”), drilled 252 holes totaling 73,080.00 ft between 2015 and 2021. Drilling consisted of both RC and core (Table 6-3).

Table 6-3 Drilling Totals for AZG

Year	Type	Prefix	Count	Total (ft.)
2015	Core	KER-15-	4	3,045.50
2017	Core	KER-17S-, KER-17U-	72	19,380.50
	RC	KER-17S-	11	7,360.00
2019	RC	18-	100	17,020.00
2020	Core	AZG-20	3	1,063.00
	RC	AZG-20S	20	16,060.00
2021	Core	AZG-21	28	7,493.00
	RC	AZG-20S-	1	565.00
	Infill Core	AZG-21-Pxx	13	1,093.00
Kerr Drilling Totals			252	73,080.00
Total Core			120	32,075.00
Total RC			132	41,005.00

AZG’s sample handling, analysis, and security procedures in 2017 relies heavily on the internal report “2017 QA/QC Procedures and Results, Copperstone Mine”, dated March 13, 2018, and prepared under the direction of AZG’s drilling program supervisor, Mr. R. Michael Smith, SME-RM. The QP has independently verified the results discussed herein. Review of sample handling procedural documents from the 2019 drilling as well as the 2020-2021 drilling are consistent with the following discussion.

Drill core was placed directly into standard waxed cardboard core boxes by the drill helper at the drill site. The core run interval was marked on wood blocks and placed at the end of each core run. Drill core was then received daily by the project geologist and logged at the core shed where it is photographed and logged by site geologists, or technicians under the supervision of geologists. Geotechnical information, as well as lithology, mineral and alteration assemblages, and structural characteristics are captured in the logging process.

Core is then marked for sampling by the geologist on nominal two-foot intervals in visibly mineralized material and on nominal five-foot intervals in visibly un-mineralized material, usually at breaks in mineralization/alteration intensity or lithology. Marked intervals are sawn in half by a technician near the core shed, under supervision of the geologists. A geologist or technician then bags one-half of the core for assay and the other one-half is retained for further use, or future reference and third-party review. The samples are then stacked on a pallet and moved to a loading area where they are loaded into bins, or directly into a pickup truck. All of these activities take place in a fenced-in area at the mine under 24-hour security.

RC samples are collected in five-foot intervals by drill helpers at the drill site, under the direct supervision of an experienced geologist. The geologist made changes as needed regarding utilization of the rotary splitter to assure quality and sample size. A representative sample of the RC chips was taken on every five-foot interval. In holes with thick alluvial overburden, no analytical samples were collected, though a chip-tray record was retained.

RC chip samples were collected in a micro-pore bag. The bag was placed in a five-gallon bucket under the secondary port of the cyclone & rotary splitter in order to direct proper sample delivery. Especially when drilling above the groundwater table, the splitter was adjusted so little overflow occurred on the sample bucket by placing lids or covers over the sample side of the rotary splitter. When large volumes of water were encountered, lids were added to the splitter, reducing the size of the collected sample. Even with the occasional reduction in sample size, the weights were still in the multiple kilogram range. The sampler adjusted the number of lids covering the sample side in order to maintain optimal sample size, but never in the middle of a 5-foot run. This number was then recorded on the sample bag sheet.

Drilling was paused at sampling intervals to ensure all sample from the interval is collected before continuing drilling into the next sample interval. Cyclone and sample splitters were cleared of buildup by spraying with high pressure water between samples. The drillhole was blown clean after each drill rod addition by lifting the rotating drill string off bottom and blowing until there is no return of cuttings.

All drillhole samples are stored at the logging area, under the supervision of the geologists. The area is fenced off from general public entry. Only authorized employees, the supervising geologists, and trained technicians are allowed access.

Samples were delivered directly to AAL by Kerr employees in 2017. In 2019, and 2020-2021 samples were shipped to assay labs directly from site. Chain-of-custody documentation is employed, with the geologist, site manager, delivery employee and lab recipient signing for custody and receipt. The samples are placed onto pallets and wrapped for shipment; the pallets are inspected for integrity throughout shipment to laboratories.

Paper copies of drill logs, geotechnical logs, survey data, shift reports and sample transmittal forms are stored at the geology office and in digital form in the SharePoint files for Copperstone. Images of the core are stored in the SharePoint files. All original data is stored on an external drive for security purposes and backed up on the main geology computer.

6.3.11.1 2015 Surface Drilling

The Copperstone drill program in 2015 was designed to target the Footwall zone, located approximately 300 to 500 feet west and parallel to the main Copperstone zone. The drilling campaign consisted of four diamond drillholes with a total core length of 3,045.5 ft. Kerr contracted West Core Drilling, and Holman Drilling to complete the program. Drillhole diameters included both HQ and NQ. KER-15-01 through KER-15-03 are oriented from a single station, starting vertically, then 65 degrees to the southwest, and finally 45 degrees to the southwest in sequence. These drillholes were surveyed down-the-hole on 50-ft intervals using a single shot camera. KER-15-04 is oriented vertically and is 370 ft northwest from KER-15-01. KER-15-04 was not surveyed down-the-hole. Results from drilling in 2015 are summarized in Table 6-4. The program confirmed a parallel system west of the main Copperstone trend within the latite-phyllite contact. The contact was strongly altered/faulted with hematite and copper oxides present. Highlights from the drill program included 0.405 oz/ton Au over 6 ft and 0.324 oz/ton Au over 5 ft.

Table 6-4 Significant Intercepts for 2015 Drilling by Kerr

Drillhole ID	From (ft.)	To (ft.)	Length (ft.)	Gold (oz/ton)
KER-15-01	567	570	3	0.130
KER-15-02	635	641	6	0.405
KER-15-03	350	355	5	0.324
also	597	602	5	0.114
also	886	900	14	0.076
KER-15-04	No significant intercepts			

6.3.11.2 2017 Surface and Underground Drilling

Kerr contracted Godbe Drilling, LLC (“Godbe”) of Montrose CO, and American Drilling Corp. (“American”) of Spokane Valley WA, to complete 83 drillholes totaling 26,740.5 ft of core and RC drilling. Godbe used a Maxi 10 track mounted drill rig for surface core drilling operations, and a small electric powered skid mounted Versa KMB.8 drill for underground core drilling. Diamond core drilling utilized HQ (2.5 in or 63.5 mm diameter) wireline gauge. Core barrels of different lengths were used depending on recovery and ground conditions. For surface RC drilling, Godbe subcontracted DeLong Construction and Drilling, Inc. (“DeLong”) of Winnemucca, NV for use of a track mounted RC drilling platform. American deployed two skid mount electric powered Atlas Copco U8 drills underground for its underground operations.

Kerr contracted Registered Land Surveyor, 82Bravo LLC of Phoenix, Arizona, to survey all planned and as-built drillhole collars. Cyprus established benchmarks to provide survey control and all surveying produced values in Arizona state-plane NAD27. Most drillholes were set up by front- and back- sight, by a registered surveyor, but a few short holes were added to the program and laid out by Brunton compass. At the end of the program the American Drill started using a Reflex TN14 Rig Alignment instrument, making the process less time-consuming and avoiding the issue of losing paint marks on the ribs.

Underground collar locations were surveyed by Aftermath Engineering, an independent contract engineer & surveyor utilizing the pre-existing underground control points. Kerr utilized a multi-shot camera at 50-ft spacing, supplied to Godbe Drilling by International Directional Services (“IDS”), a drilling service company

out of Chandler, Arizona. RC drillholes were surveyed directly at 50-ft intervals with a gyro instrument by IDS. In October 2017, Kerr began to utilize a Devico gyro instrument, supplied by Minex of Virginia, Minnesota, for most new drillholes. The camera instrument was retained as a back-up and was often used on holes drilled upward from mine workings. Camera survey data was collected at nominal 50-ft intervals as the drill tripped out of the hole after completion. The Devico instrument was utilized when high magnetite contents were detected in D zone core. In this case, readings were collected every 10 ft from bottom to top of the hole.

While the overall core recovery for the 2017 drilling is over 95%, the recovery for zones with >4 ppm Au was 80%. Review of the RC drilling indicated good overall sample recovery.

At first during the 2017 RC drilling, duplicate samples were taken using a “Y”-splitter from the sample port. This method was soon changed into a “Straight Split” by placing the first sample under the sample port and the duplicate under the reject port. This is more appropriate as it tests the efficiency of the cyclone splitting samples.

Figure 6-1 shows the relationship between the two types of duplicates compared to the regular sample. In general, there is a good correlation. However, the presence of two outliers suggests a possible problem in collecting rig duplicates. Unfortunately, there are no samples of higher-grade material greater than 1.6 ppm in this sample set, so it is difficult to extrapolate into high-grade material.

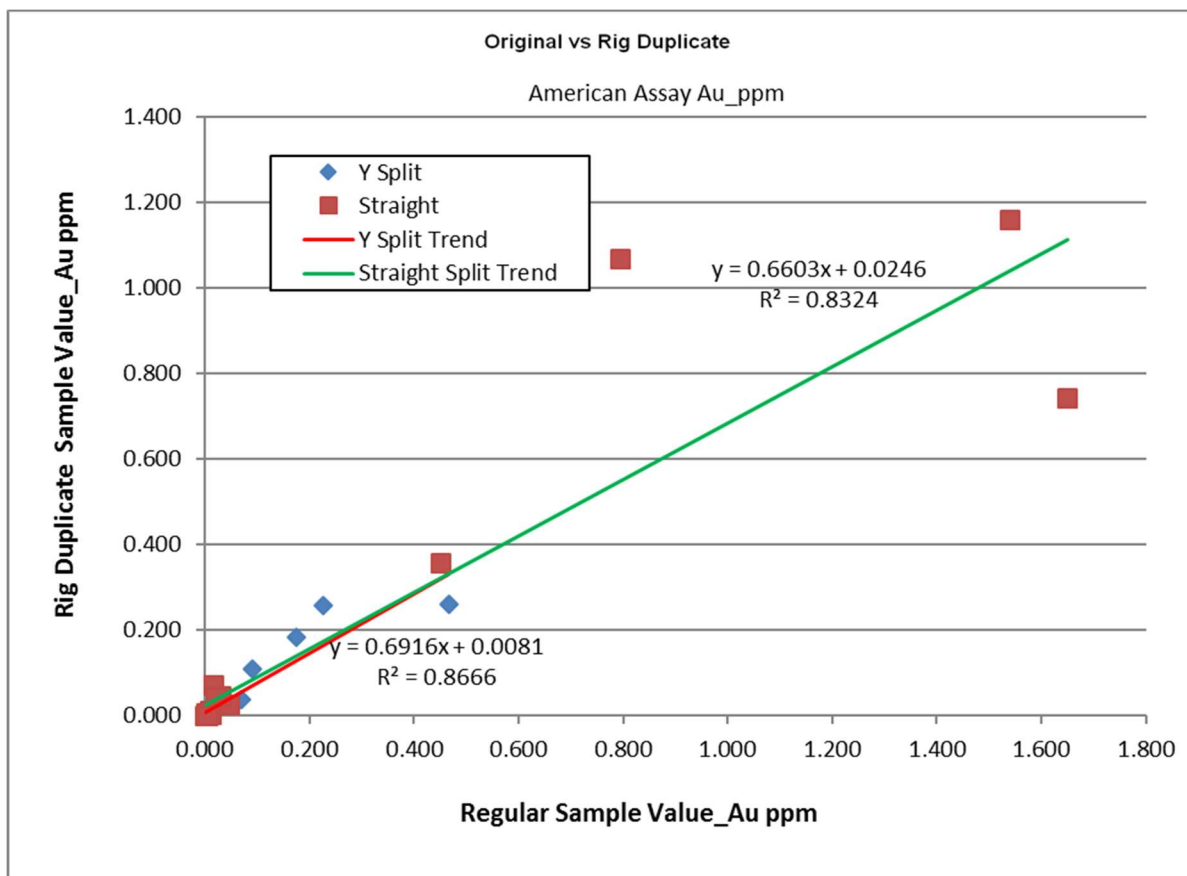


Figure 6-1 Comparison of Y-Split and Straight Split Rig Duplicate Methods

Core and RC samples are submitted to American Assay Laboratory (AAL) in Reno, Nevada, which is ISO Certified. RC chips from the 2019 drilling were submitted to ALS in Tucson, Arizona (ISO Certified). Initially in 2020, RC samples from five drillholes were sent to ALS in Tucson, Arizona. The remaining core and RC samples were submitted to AAL in Reno, Nevada for the 2020, 2021 drilling, and the 2021 infill core drilling. SRM with gold mineralization, purchased from commercial providers, or blank standards are inserted every 20th sample as part of the normal sample number sequence. The geologists supervise the SRM and blank sample insertions to assure uniformity.

At AAL samples were prepared as follows:

- Samples are organized in sequence to check for duplicates and missing samples, a label is generated and applied to each sample, and after crushing and pulverizing rejects receive the sample label designation. Sample preparation is essentially the same for core and reverse circulation (RC) samples.
- Samples are oven dried, as necessary, for 8 to 48 hours.

- Entire samples are passed through a jaw crusher, achieving 70% passing 6 mesh, with weights recorded. River rock, considered a blank, is run through the jaw crusher prior to each job; the "waste" control sample is collected and run through assay procedure with the submitted samples, to check for contamination. The jaw crusher is air blast cleaned before each sample is crushed.
- Samples are then passed through a roll crusher, which reduces the sample 90% passing 10 mesh.
- Samples are then split through a Jones (riffle) Splitter, resulting in about 200 g of sample for pulverization. The splitter and collection pans are cleaned between each sample with air blast.
- Samples are then pulverized using a Ring Mill to obtain 90% passing 150 mesh. Ten percent of the pulverized samples are checked and recorded by screening to confirm results by the lab.
- Metallic Screen sample preparation is as above, but 1000 g. or all of the sample if less than 1 kg. are pulverized. Of that 850 g are split out for screening to +/- 150 mesh separate Fire Assay. Weights are recorded to calculate weighted average grades for the sample and analyze Au distribution. A rotary splitter may be used to select the bulk 1000 g sample for this procedure, with air blast cleaning between samples.

At American Assay Laboratories, samples are fire assayed as follows:

- Fire assaying is the quantitative determination in which precious metals are separated from impurities by fusion. A sample of unknown concentration is combined with flux containing litharge (PbO) and various other compounds. The dense molten lead dissolves the precious metals and separates them from everything else that becomes dissolved, absorbed, or encapsulated into the less dense molten slag that floats on the lead. After the molten material has been poured and cooled, the lead (containing precious metals) can be separated out. Then when re-heated in a cupel, molten lead is absorbed by the cupel leaving behind a prill containing the desired precious metals for analysis.
- Samples are fired using typical procedures, in batches of 72. Each batch therefore has at least 3 QA/QC control samples (1 in 20) inserted into the batch.
- The initial pass is to do a 1 assay ton (about 30 g) fire assay with acid digestion of the prill with ICP analysis for Au. If the resulting value is above 10 ppm Au, a second 2 assay ton fire assay (about 60 g) is done on remaining pulps, with a gravimetric finish. The results of 1 and 2 assay ton Au analyses were graphed, as below in Figure 6-2.

This information would be useful in any discussion regarding the necessity of routine 2AT assays for samples under 100 ppm gold.

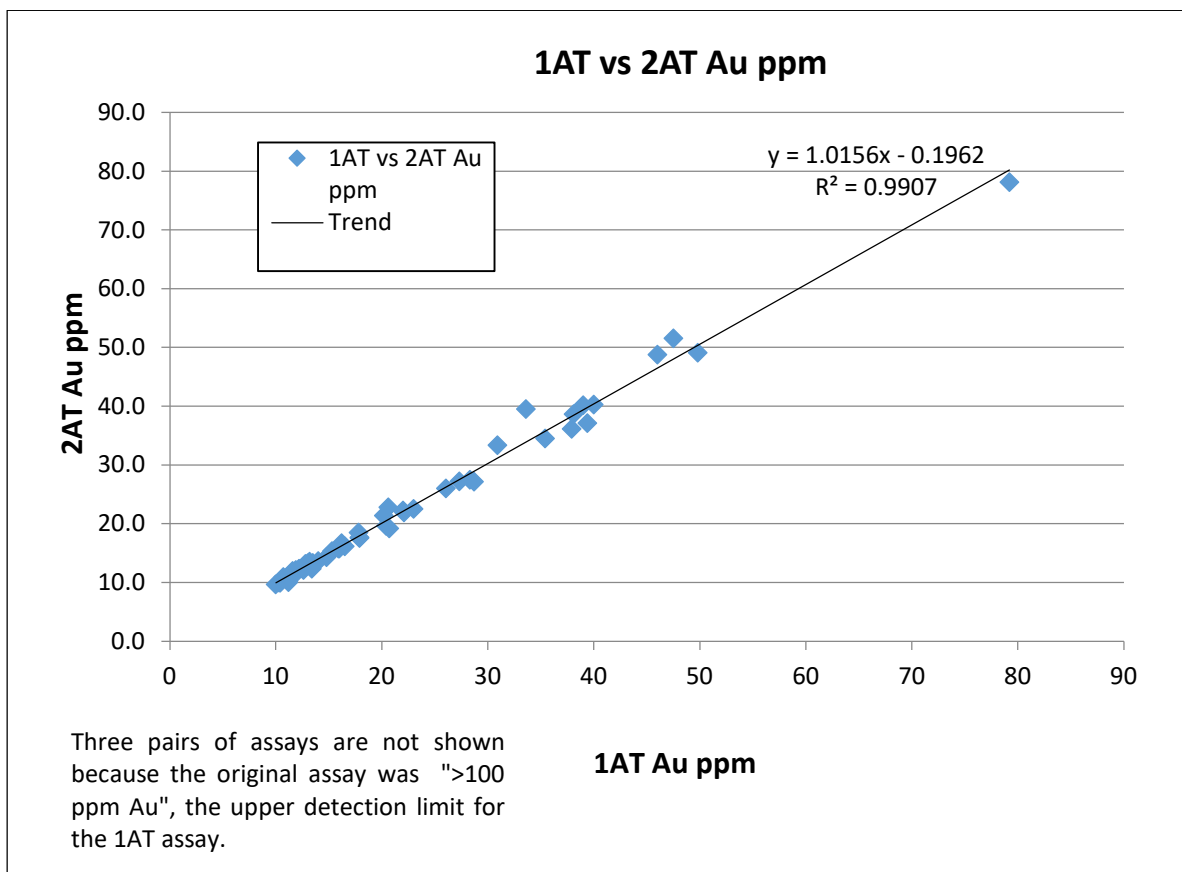


Figure 6-2 Comparison of 1AT with 2AT Charges

Copper and silver were analyzed by AAL using a 0.5 g sample of the pulp, digested in nitric and hydrochloric acid, and heated to complete digestion. The concentration is measured with ICP finish. If the result was over 10,000 ppm Cu or 100 ppm Ag, a 2 g sample was assayed by the same process.

One hundred two (102) American Assay pulps were selected and compared to a second certified assay laboratory, Skyline Laboratories (ISO certified). The samples were mostly random, but it was specified that high-grade samples are somewhat over-represented. Skyline conducted a 1AT/AA assay for gold, followed by a 2AT/GR finish for samples over 10 ppm Au. With the exception of one sample the assays compare favorably. The data show that AAL is in reasonably good agreement with the third-party lab on the initial 1AT assays (Figure 6-3).

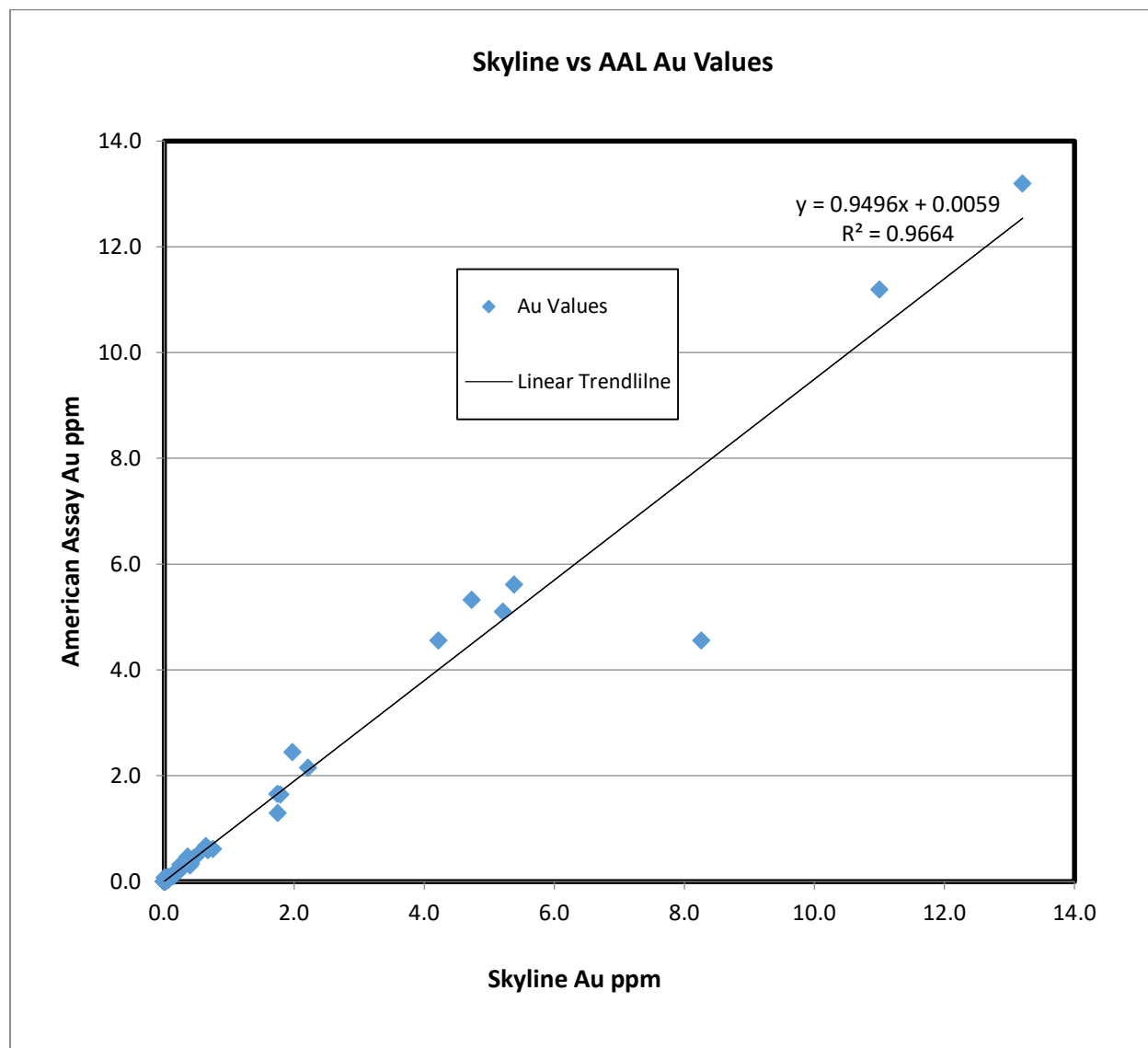


Figure 6-3 Comparison of Skyline with American 1AT Assays, for 2017 Program Pulps

Thirty-eight samples of coarse rejects prepared by American Assay were selected for duplicate analysis at Skyline Laboratories. The entire coarse reject was utilized. An approximate 250 g sample was split utilizing a Jones riffle splitter. The resulting sample was pulped to 85% passing 200 mesh. A 1AT Au fire assay was done as a first pass, with AA finish. Skyline produced both AA and Gravimetric finishes for their work. Results appear to be reasonably comparable (Figure 6-4).

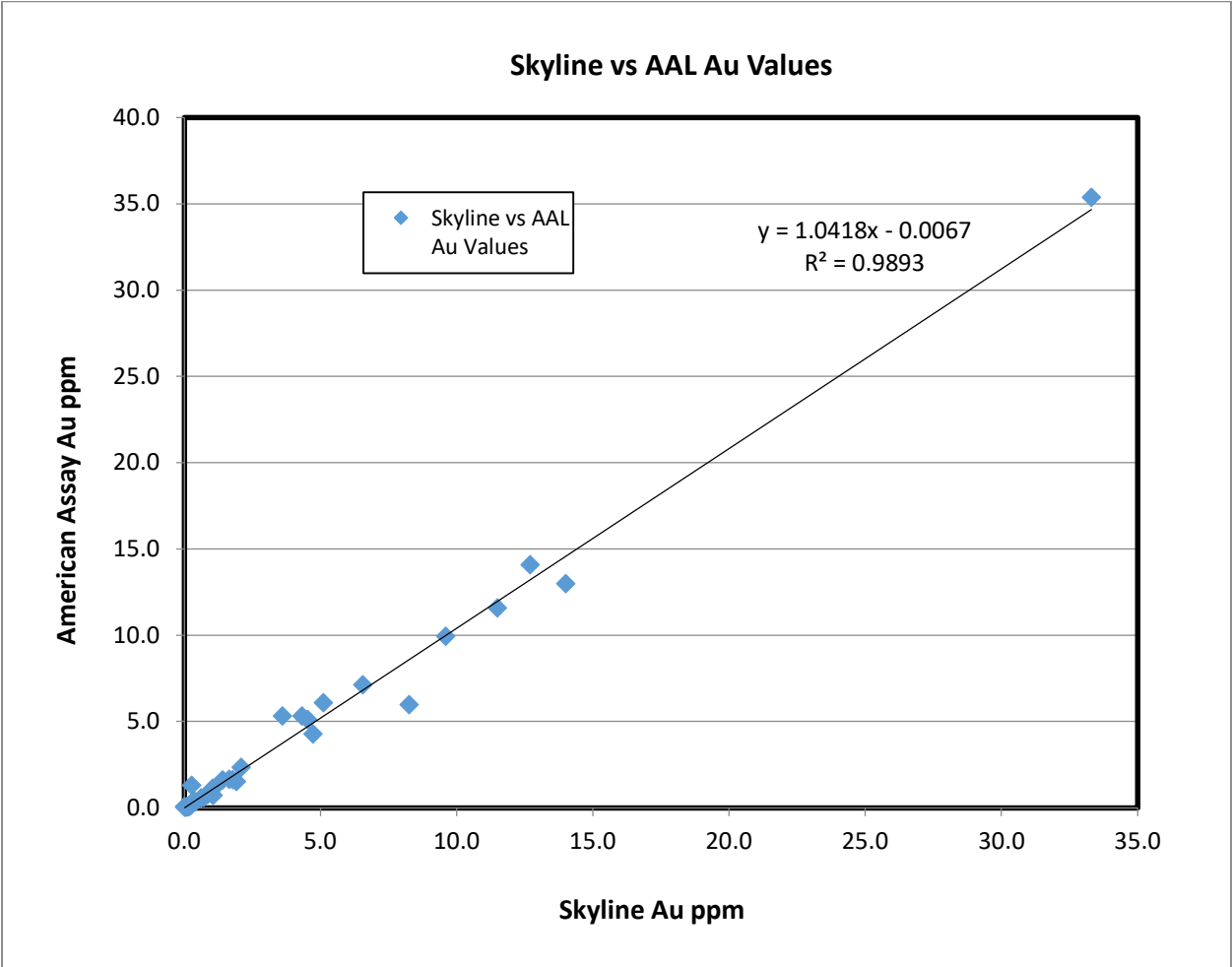


Figure 6-4 Comparison of Original Pulp Analysis by AAL and Skyline on AAL Coarse Rejects

An initial 50 samples were selected for metallic screen assaying, done by AAL. Samples were selected with gold grade ranges from nil to high grade. The weighted average metallic screen assay is plotted against the accepted (average of repeat assays, where applicable) results, to analyze heterogeneity. Results depicted below (Figure 6-5) show that except for two samples, there is excellent correlation.

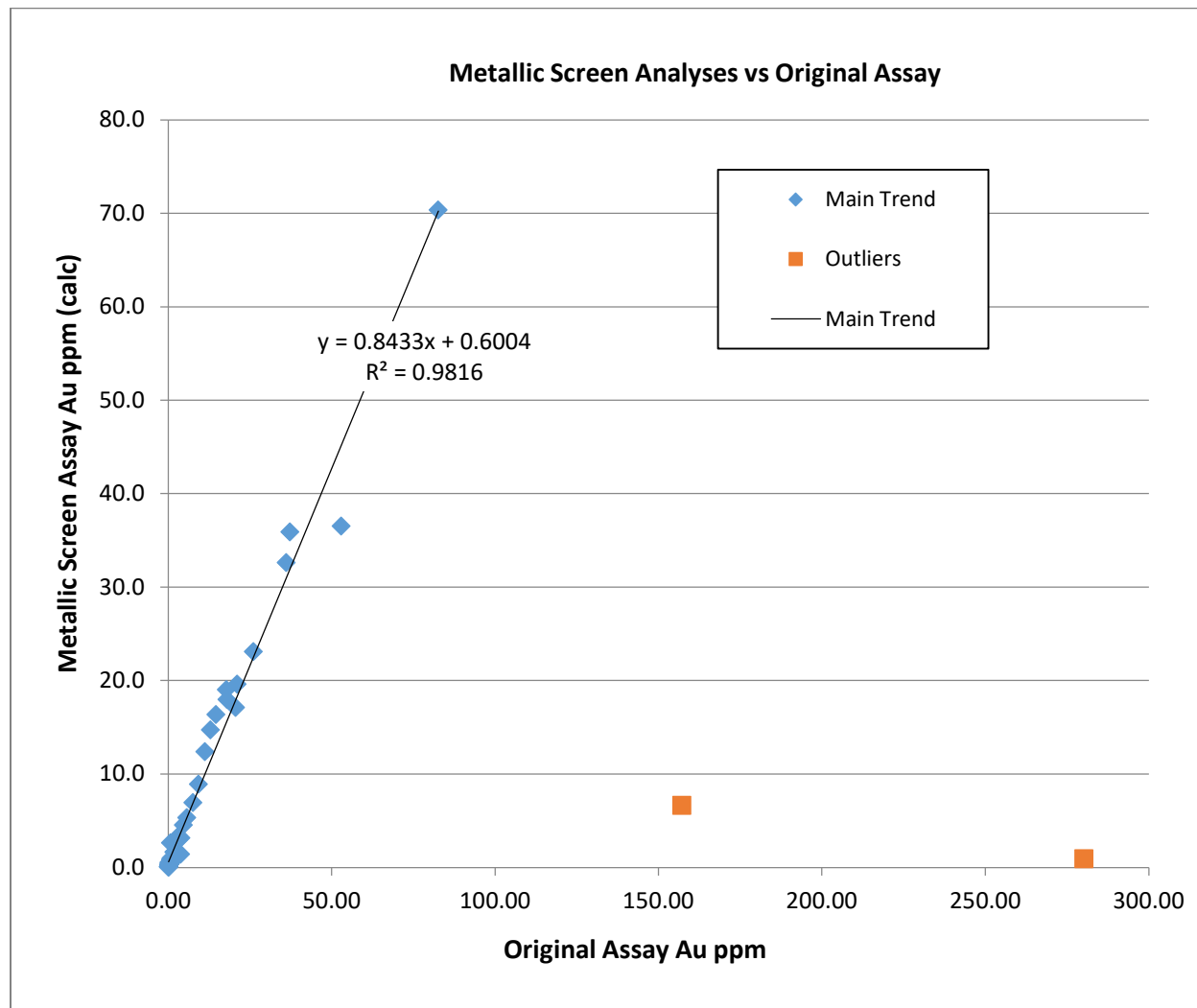


Figure 6-5 Comparison of Original Assay with Metallic Screen Analysis of Coarse Reject, both by AAL

Thirty-two rejects were sent to AAL to produce a bulk pulp (500 g) which was assayed for consistency with the original 250 g pulp. The selected grades covered the entire span of gold grades from 1 ppm to 100 ppm. The results are summarized in Figure 6-6.

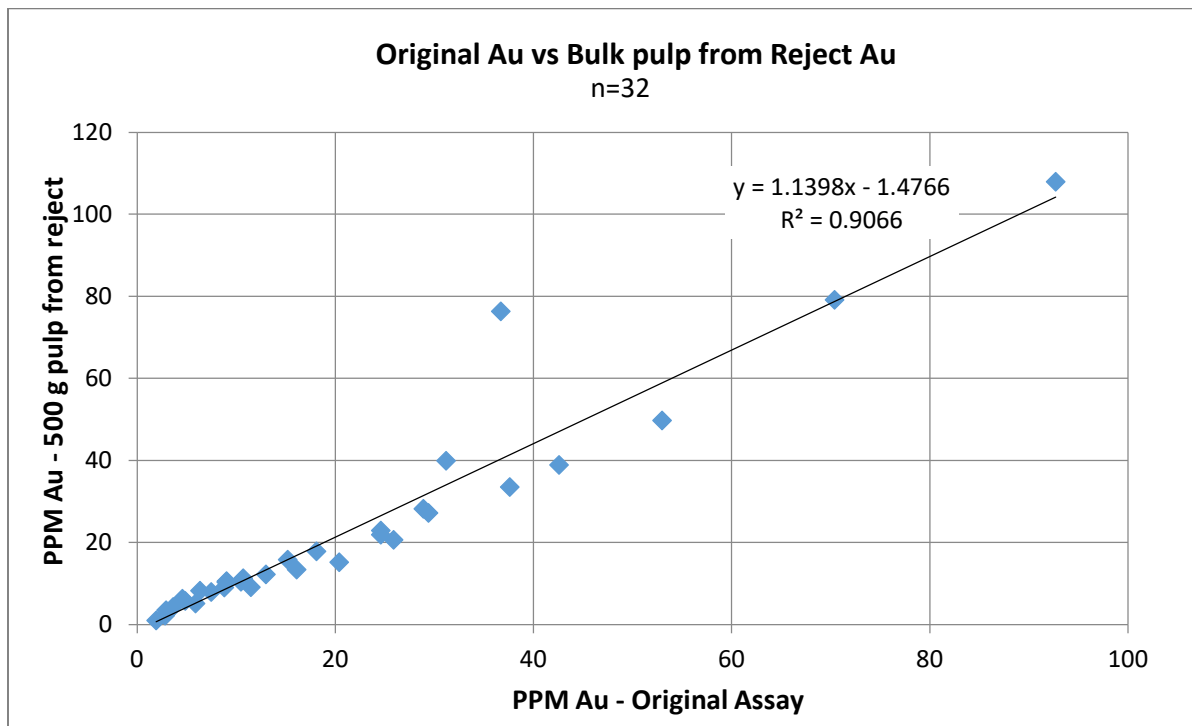


Figure 6-6 Comparison of Original Assay with Assay of 500 g Pulp from Coarse Reject

Graphical analysis of QA/QC Reference Standard results were done on every sample assay batch supplied, as seen below. If a result was outside of two standard deviations, based on values provided by the vendor of the standard, the surrounding 10 samples were re-assayed. The results were reviewed for consistency and potential assay problems.

Reference samples were inserted into the sample stream as every 20th sample. The basic pattern started with a 'waste' sample followed by a randomly selected material (Table 6-5). On occasion, the order was altered, and sometimes an extra sample was inserted, especially on small batches of samples so as to have at least two controls. This pattern results in a minimum 5% of all samples being control samples. Analysis of final reports shows that the actual percentage of controls was 6.1% for the 2017 drilling program. Code names (Table 6-5) were used for ease in designation, but no identifiers were allowed to show on the bags for the control samples except a sample number. In reverse circulation drilling, control samples were inserted at the same frequency, but on a random basis, and more of the W samples were utilized. Results for each of the materials are presented in Figures 6-7 through 6-13.

Table 6-5 Listing of Reference Samples for the 2017 Copperstone Drilling Campaign

Code	Type	Reference ID	Source Company	Expected Au (ppm)	Std Dev	Au Certified
MSL	Pulp	S107009x	MEG	4.7340	0.1940	Yes
MSM	Pulp	S107010x	MEG	6.4050	0.3020	Yes
MS	Pulp	S107011x	MEG	9.2840	0.4340	Yes
MSH	Pulp	S107012x	MEG	16.5030	0.6260	Yes
MH	Pulp	OxP50	Rocklabs	14.8900	0.4930	Yes
L	Pulp	17.02	MEG	0.5110	0.0300	Yes
Blank	Pulp	11.04	MEG	0.0015	-	Yes
W	Rock	Waste	Kerr	0.0050	-	No

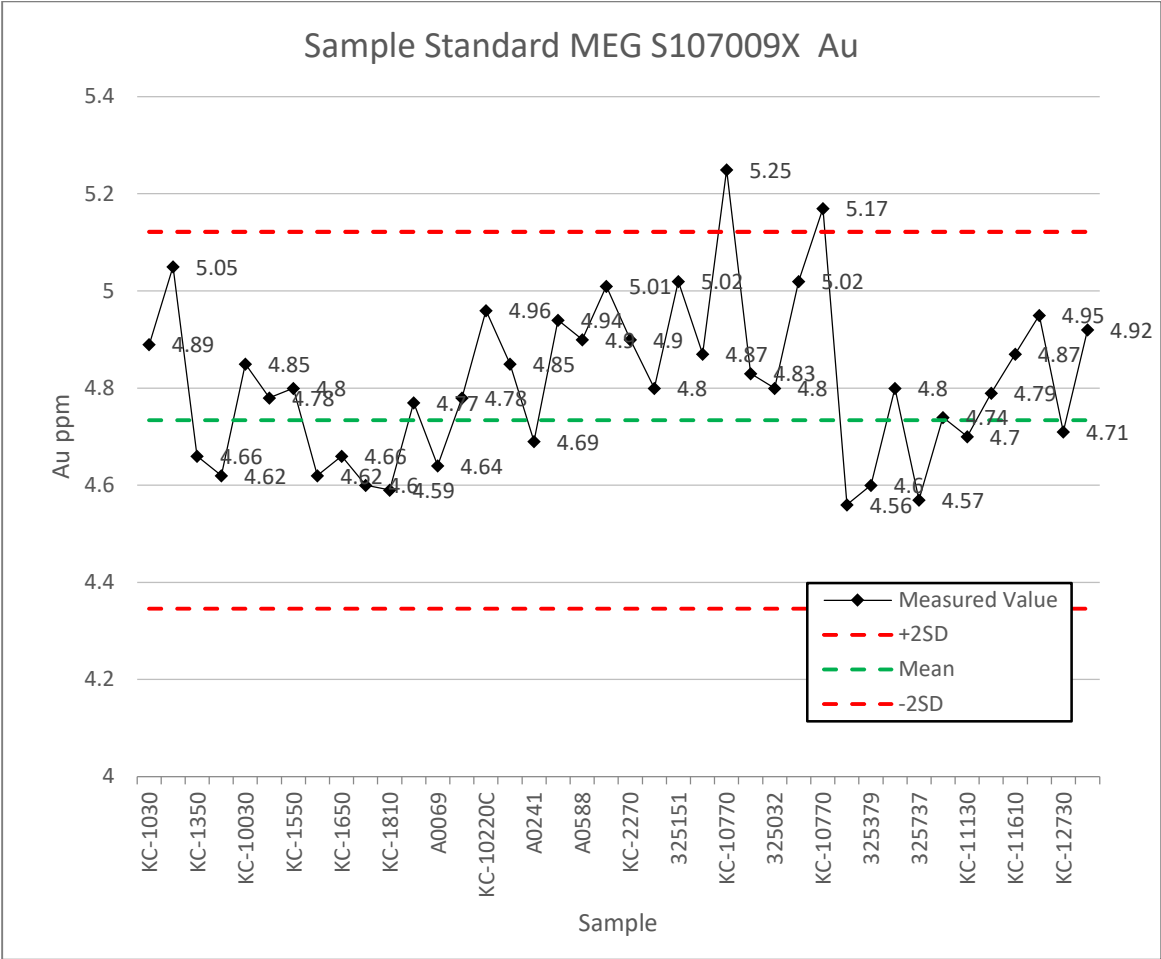


Figure 6-7 Analytical Results for Control Sample MSL, All 2017 Drilling

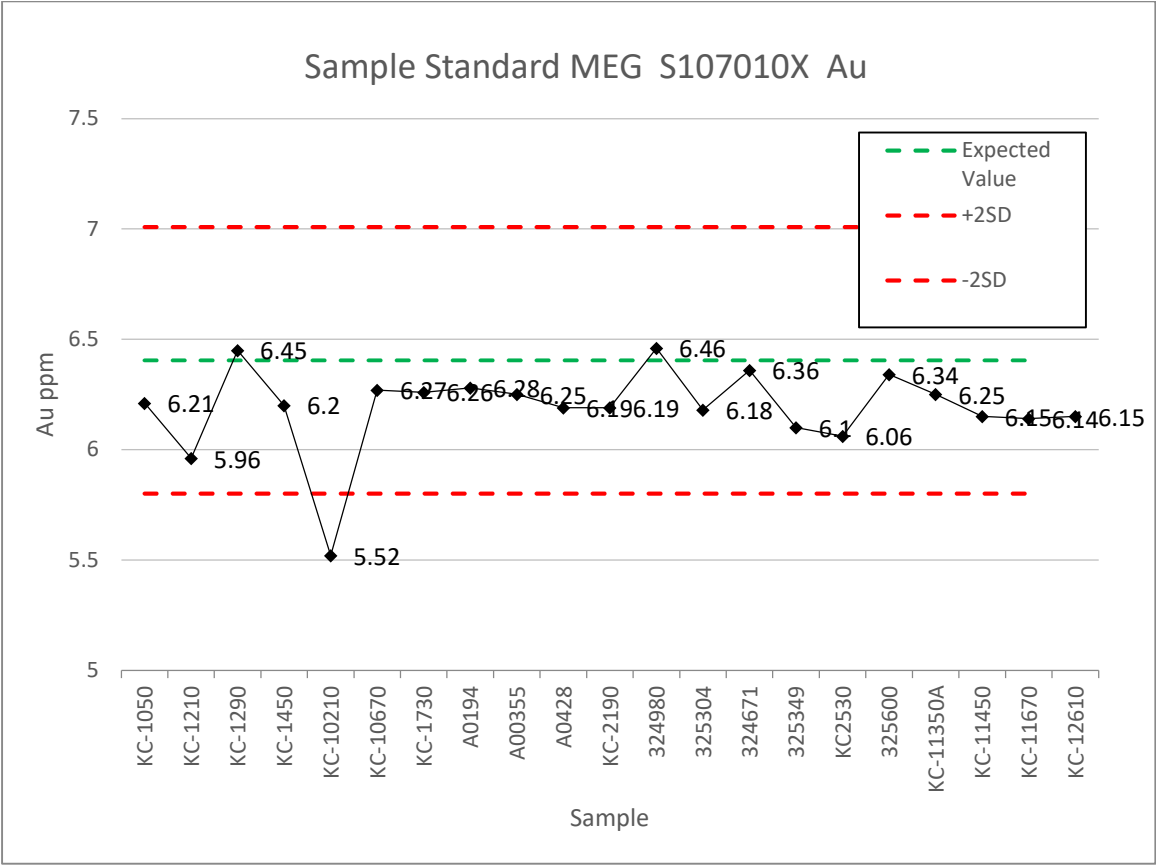


Figure 6-8 Analytical Results for Control Sample MSM, All 2017 Drilling

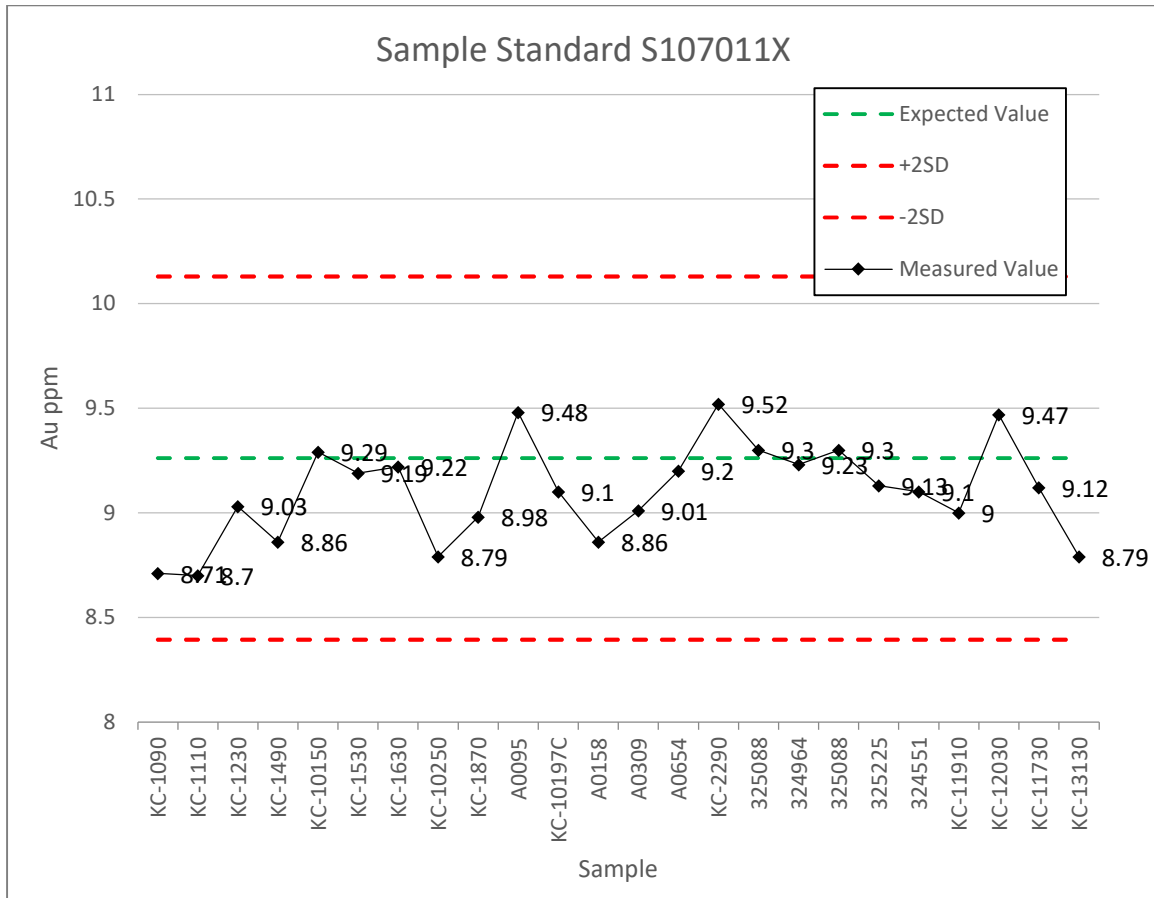


Figure 6-9 Analytical Results for Control Sample MS, All 2017 Drilling

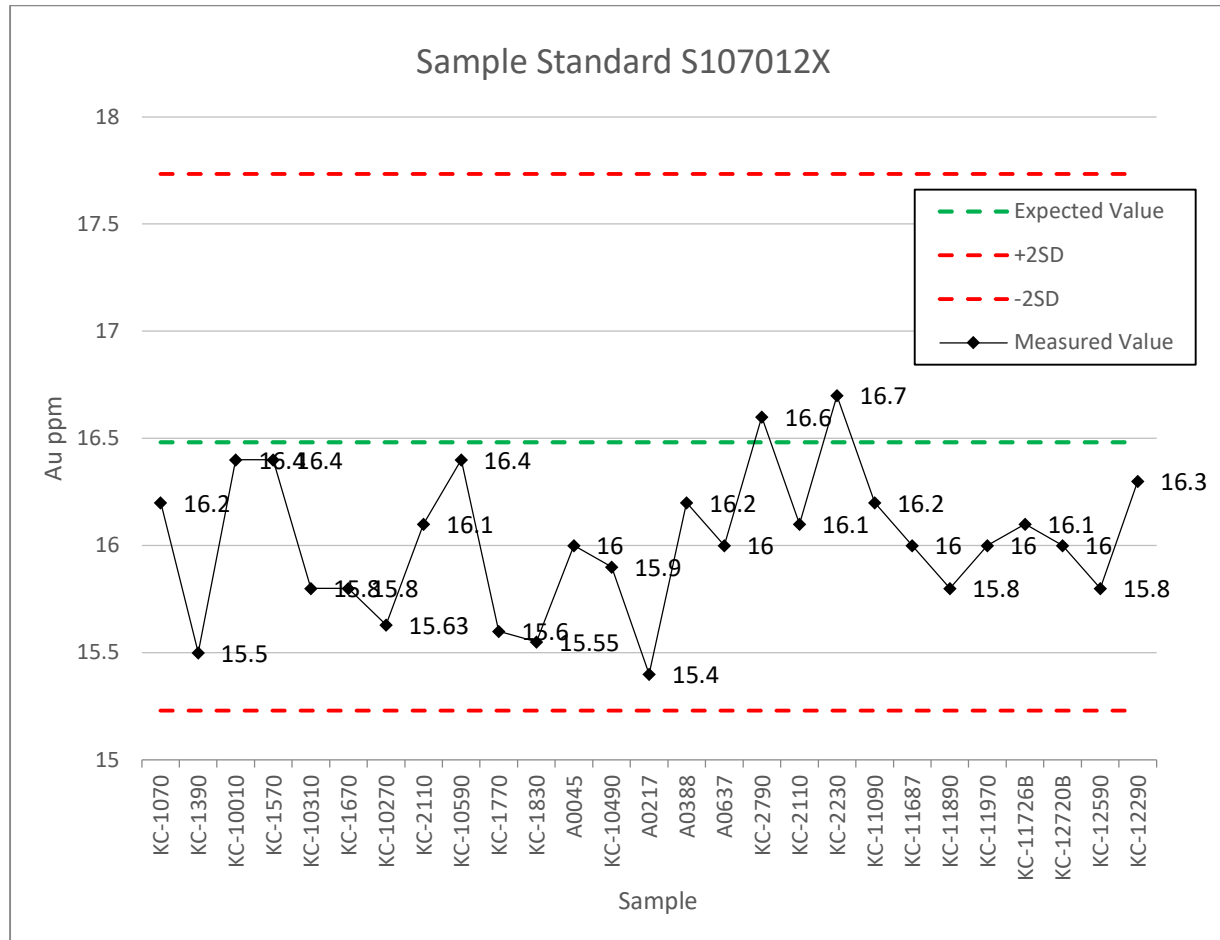


Figure 6-10 Analytical Results for Control Sample MSH, All 2017 Drilling

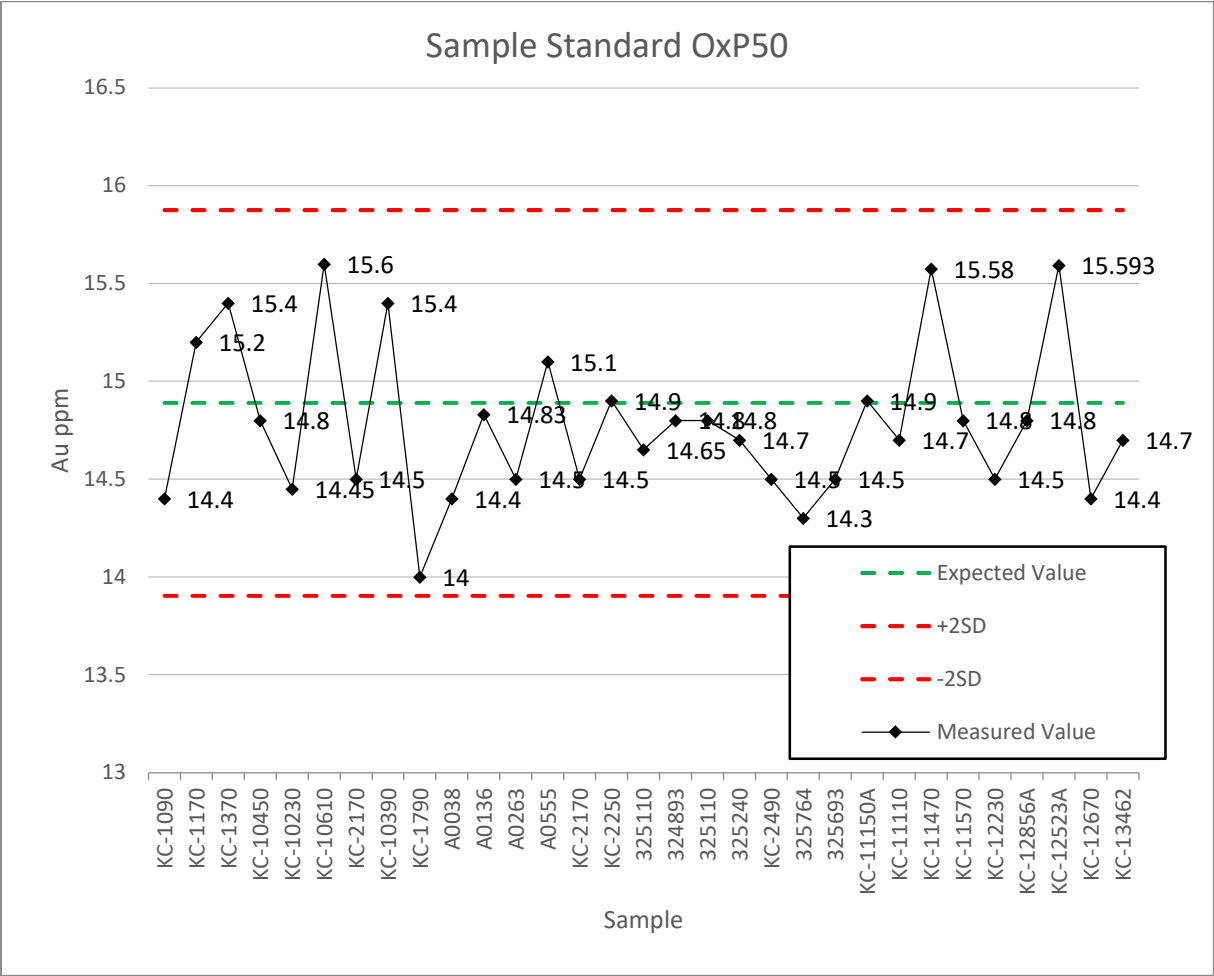


Figure 6-11 Analytical Results for Control Sample MH, All 2017 Drilling

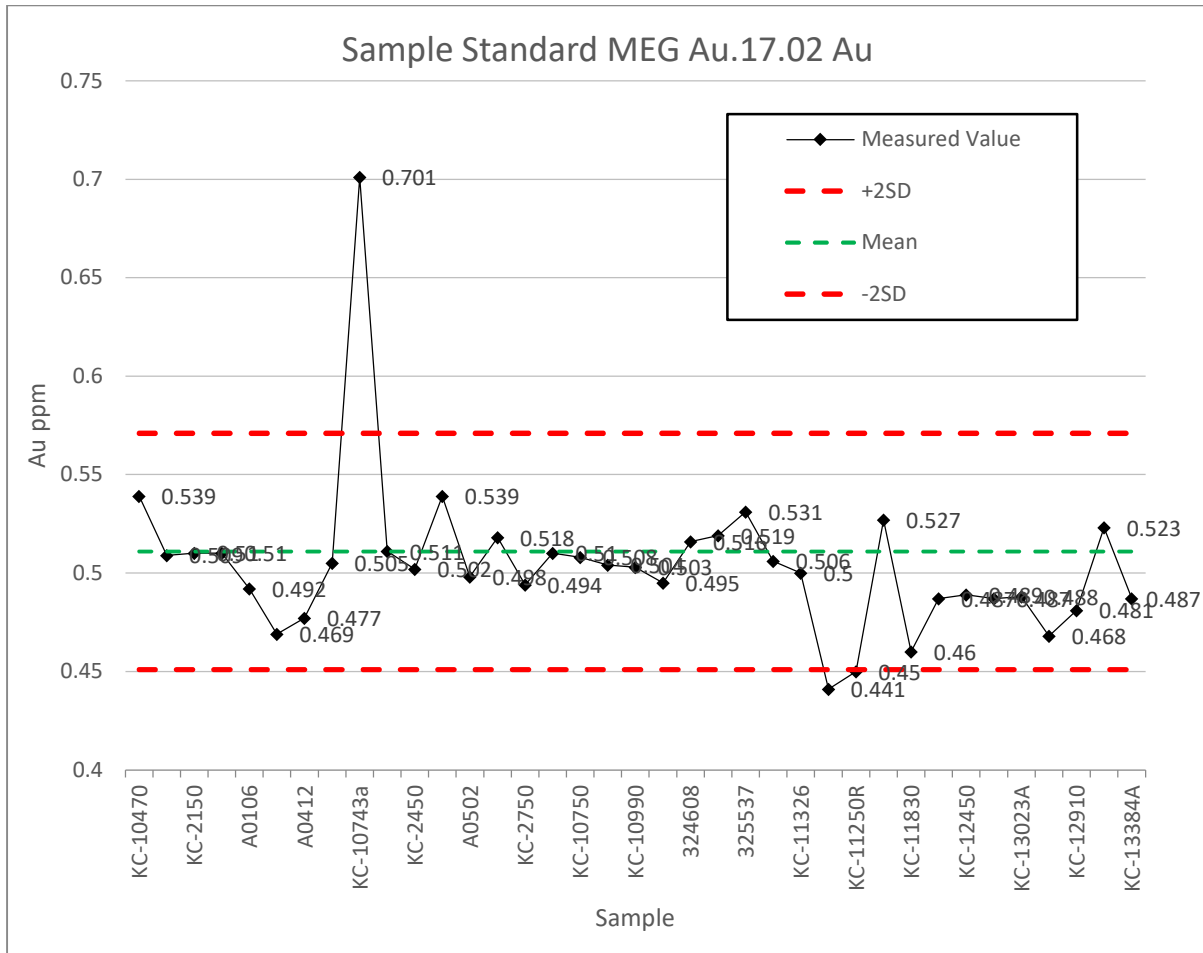


Figure 6-12 Analytical Results for Control Sample L, All 2017 Drilling

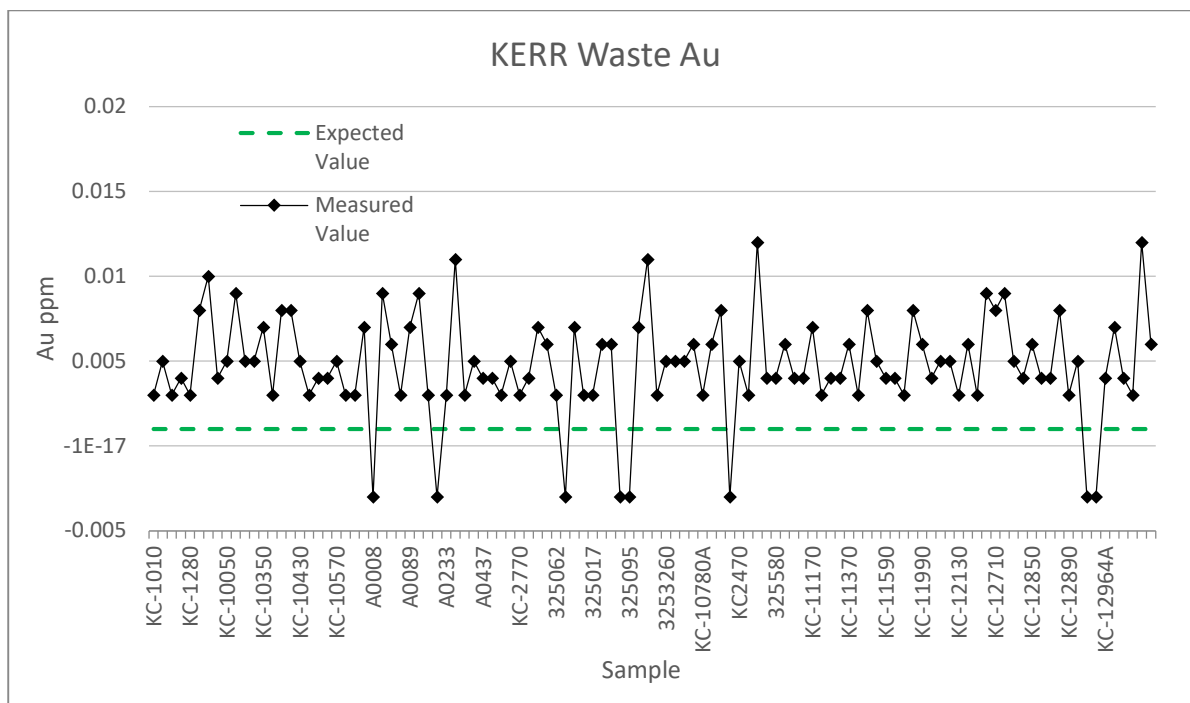


Figure 6-13 Analytical Results for Control Sample W, All 2017 Drilling

All but one assay of MSL, by AAL, have been within ± 2 STDEV of accepted value at 0.525 ppm Au. That sample and 10 surrounding samples were rerun. The core samples returned repeatable values and the control repeated outside of the range again (0.517 ppm Au). This was deemed a bad control and the other results were accepted.

All but one assay of MSM, by AAL, have been within ± 2 STDEV of accepted value and no actions have been taken related to concern about assay results. That sample and included interval were rerun with additional controls added because the original was insufficient for another analysis. All samples were within limits.

There were two incidents of the L standard being out of range. In one case (0.701 ppm Au), a re-assay of the sample (0.539 ppm Au) and surrounding core samples came back within range and included samples also acceptable. In the second case a low value was nearly repeated very near the lower end of the acceptable range. Because adjacent samples checked with the first run, the results were accepted.

The "waste" control sample consists of broken pieces of abrasive reddish concrete bricks; broken pieces give better monitoring of the coarse crushing stage at the laboratory sample prep area, as opposed to utilizing silica sand or pulverized blank standards.

Analytical results for the waste samples from AAL are plotted on Figure 6-13. All assays have been ≤ 0.012 ppm Au with the lowest values at below detection ("BDL") and the average at 0.005 ppm Au. There were no samples that required a review during the 2017 drilling. The data show that contamination during sample preparation is minimal and probably in the range of a few parts per billion.

Drilling from the surface had the primary objective of demonstrating Footwall zone gold mineralization along strike and down dip on approximately 200-foot fences. The drilling results in Table 6-6 confirm Footwall gold mineralization does occur along strike and down dip. The down dip extension of A/B sections of the Copperstone zone gold mineralization was also tested from surface. Two drillholes were successful in extending A/B zone gold mineralization down dip (Table 6-7). One drillhole, KER-17S-23, tested gold mineralization in the south extension of the Footwall zone. This drillhole did not reach the intended target due to structure entrainment and poor sample return.

Table 6-6 Significant Intercepts for 2017 Footwall Zone Drilling from Surface

Drillhole ID	From (ft)	To (ft)	Length (ft)	Au (oz/ton)
KER-17S-02	56.5	60.0	3.5	0.265
also	78.0	80.1	2.1	0.158
KER-17S-03	502.2	520.6	18.4	0.084
KER-17S-04	594.0	606.0	12.0	0.213
includes	603.3	606.0	2.7	0.939
KER-17S-06	415.0	425.0	10.0	0.034
KER-17S-07	494.0	497.0	3.0	0.121
KER-17S-10	776.5	801.0	24.5	0.108
includes	779.0	790.0	11.0	0.232
includes	782.0	790.0	8.0	0.318
includes	776.5	790.0	13.5	0.193
KER-17S-11	730.0	740.0	10.0	0.113
includes	735.0	740.0	5.0	0.148
KER-17S-13	435.0	485.0	50.0	0.134
Includes	440.0	455.0	15.0	0.384
KER-17S-17	567.0	597.5	30.5	0.105
includes	573.0	587.0	14.0	0.200
and	573.0	579.0	6.0	0.226
KER-17S-18	106.0	108.0	2.0	0.053
KER-17S-19	290.0	305.0	15.0	0.100
and	290.0	325.0	35.0	0.083
includes	320.0	330.0	10.0	0.134
KER-17S-21	60.0	70.0	10.0	0.193
and	315.0	435.0	120.0	0.219
includes	320.0	330.0	10.0	0.146
and	400.0	425.0	25.0	0.911
includes	405.0	415.0	10.0	2.186

Table 6-7 Significant Intercepts for 2017 A/B Zone Drilling from Surface

Drillhole ID	From (ft)	To (ft)	Length (ft)	Au (oz/ton)
KER-17S-01	381	398	17	0.140
includes	381	393	12	0.196
also	533	545.8	12.8	0.058
KER-17S-14	375	385	10	0.075
and	520	535	15	0.051

Kerr's underground drilling program had two goals:

- Confirm historic drilling results, and
- Provide a pathway for increasing gold mineral resources with future drilling programs.

These goals were achieved using the following approaches:

- Infill and confirmation drilling in D zone;
- D zone up dip continuity and extension; and
- D zone down dip continuity and extension.

Infill and confirmation drilling in the D zone was successful in confirming grades intersected by previous operators and drilling results are shown in Table 6-8. Drilling was conducted only from the available access in the existing underground workings. While the drillhole orientation does not intersect the D zone perpendicular to gold mineralization, the drillholes were oriented to drill through the footwall and hanging wall of the shallow angle D zone structures. As a result, the interval lengths reported in KER-17U-10, KER-17U-11, KER-17U-12, KER-17U-13, and KER-17U-17 do not reflect the true thickness of D zone gold mineralization.

Drillholes testing the up-dip extension and continuity of the D zone were designed based on previous interpretations. Drilling was oriented to intersect the D zone structures as close to perpendicular as underground access would allow and reported results are considered to approximate true thickness. Only four of the 17 drillholes targeting up-dip extension and continuity in the D zone returned significant grade intercepts (Table 6-9). Based on the current interpretation of the D zone, it is apparent that up-dip mineralization remains open below the current drilling.

Drillholes testing the down dip continuity of D zone gold mineralization were successful with 16 of 30 drillholes returning favorable results (Table 6-10). Drilling was oriented to intersect the D zone structures as close to perpendicular as underground access would allow and reported results are considered to approximate true thickness.

Table 6-8 Significant Intercepts for 2017 D Zone Confirmation and Infill Drilling from Underground

Drillhole ID	From (ft)	To (ft)	Length (ft)	Au (oz/ton)
KER-17U-04	80.0	95.0	15.0	0.150
includes	84.5	95.0	10.5	0.166
KER-17U-05	112.0	122.0	10.0	0.232
includes	114.0	122.0	8.0	0.262
KER-17U-06	83.0	103.5	20.5	0.253
includes	83.0	95.5	12.5	0.378
and	147.5	168.5	21.0	0.091
includes	154.0	168.5	14.5	0.094
KER-17U-11	48.0	61.0	13.0	0.181
includes	48.0	58.5	10.5	0.223
and	197.5	206.5	9.0	0.158
includes	200.0	206.5	6.5	0.209
KER-17U-12	153.0	224.0	71.0	0.673
includes	153.0	213.5	60.5	0.789
includes	163.0	210.8	47.8	0.988
includes	178.0	191.0	13.0	2.626
also includes	185.0	213.5	28.5	1.143
KER-17U-13	44.0	80.0	36.0	0.092
includes	46.5	57.0	10.5	0.258
also	73.0	75.5	2.5	0.179
and	121.5	127.0	5.5	0.097
includes	124.0	127.0	3.0	0.150
KER-17U-14	34.0	81.0	47.0	0.052
includes	75.0	81.0	6.0	0.220
also	155.0	243.0	88.0	0.406
includes	178.0	206.8	28.8	1.127
also	217.0	226.0	9.0	0.164
KER-17U-16	48	153	105	0.116
includes	48	58	10	0.224
also	81.5	90	8.5	0.232
also	104.9	113	8.1	0.308
also	138.5	153	14.5	0.318
KER-17U-17	57.0	62.0	5.0	0.124

Table 6-9 Significant Intercepts for 2017 D Zone Up Dip Continuity and Extension from Underground

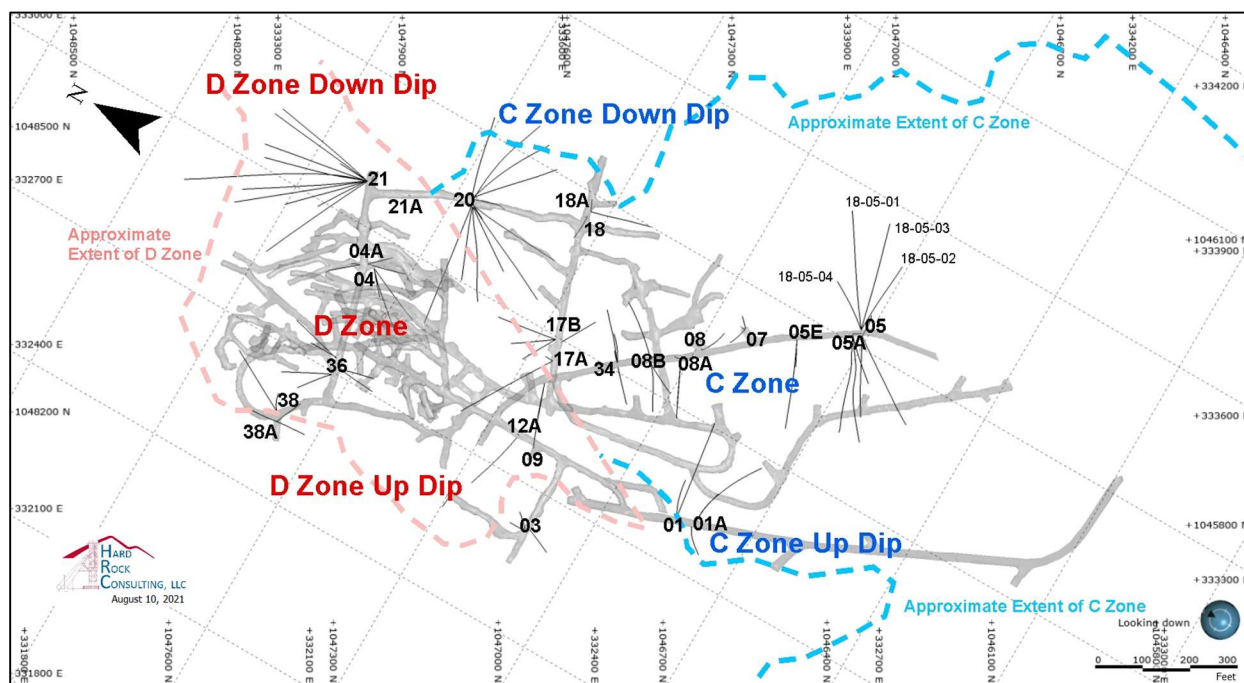
Drillhole ID	From (ft)	To (ft)	Length (ft)	Au (oz/ton)
KER-17U-08	19.0	26.1	7.1	0.141
includes	19.0	23.0	4.0	0.235
KER-17U-09	11.5	12.5	1.0	0.164
KER-17U-21B	42.0	53.5	11.5	0.594
includes	46.0	49.0	3.0	2.257
KER-17U-26	38.1	42.0	3.9	0.102
includes	38.1	40.0	1.9	0.206
and	112.7	118.3	5.6	0.158
includes	112.7	115.4	2.7	0.310

Table 6-10 Significant Intercepts for 2017 D Zone Down Dip Continuity and Extension from Underground

Drillhole ID	From (ft)	To (ft)	Length (ft)	Au (oz/ton)
KER-17U-01	53.0	57.4	4.4	0.131
KER-17U-02	56.0	64.5	8.5	0.095
KER-17U-50	69.0	93.0	24.0	2.998
KER-17U-51	56.0	79.9	23.9	0.167
includes	56.0	72.5	16.5	0.237
KER-17U-52	54.0	70.0	16.0	0.152
includes	54.0	64.5	10.5	0.227
KER-17U-53	52.0	72.0	20.0	0.156
includes	61.0	72.0	11.0	0.277
also	67.8	69.8	2.0	1.082
KER-17U-57	93.3	128.0	34.7	0.136
includes	101.9	111.0	9.1	0.223
KER-17U-59	90.5	111.5	21.0	0.090
includes	90.5	101.7	11.2	0.164
KER-17U-60	45.0	60.3	15.3	0.083
includes	55.0	60.3	5.3	0.169
KER-17U-61	134.0	147.3	13.3	0.096
includes	134.0	139.7	5.7	0.163
KER-17U-65	40.5	54.8	14.3	0.092
includes	43.0	54.8	11.8	0.111
KER-17U-66	59.0	74.0	15.0	0.113
includes	64.0	66.0	2.0	0.815
KER-17U-67	110.9	119.0	8.1	0.115
includes	110.9	113.0	2.1	0.267
KER-17U-68	70.5	82.0	11.5	0.180
KER-17U-72	87.5	97.0	9.5	0.032
includes	87.5	89.0	1.5	0.186
KER-17U-74	54.0	57.0	3.0	0.175
and	112.0	122.0	10.0	0.161
includes	112.0	116.8	4.8	0.325

6.3.11.3 2019 Underground Drilling

From late January through March of 2019, AZG completed 100 drillholes totaling 17,020 ft from multiple underground stations. The purposes of the program were to test the margins of gold mineralization as understood at that time, and converting Inferred mineral resources into Measured and Indicated, in the D zone and portions of the C zone. Major Drilling, out of Salt Lake City, Utah used an underground RC drill to complete the holes. All but four drillholes were surveyed down-the hole at nominal 50-ft intervals. Drillhole locations were limited by access to existing underground developments with acceptable ground conditions and infrastructure. As a result, drilling is oriented in multiple directions with the intent of intersecting gold mineralization as close to true thickness as practical. Review of drillhole logs, sample weights submitted to ALS, and conversations with AZG geologists did not indicate significant difficulties with drilling operations or recovery. Figure 6-14 shows the station locations, drillhole traces, target areas, and existing underground locations for the 2019 drilling program.



Source: Hard Rock Consulting, 2021

Figure 6-14 2019 Underground Drill Station Locations

Samples collected in 2019 were submitted to ALS in Tucson, Arizona. Samples were prepared by first weighing the samples, logging them into the ALS global tracking system and then dried in ovens. Samples are then crushed to 70% passing 2mm, split using a Boyd rotary splitter, and then the split is pulverized to 85% passing 75 microns. Samples were analyzed for gold by fire assay at ALS Reno, Nevada. If the resulting gold assay exceeded 10 ppm, the sample was re-assayed with a gravimetric finish. 144 samples with gold assays greater than 1 ppm, were re-analyzed for gold and copper by cyanide leach at ALS Vancouver, British Columbia.

QA/QC protocol for the 2019 drilling was completed using a combination of duplicates, SRMs and blank material. Two types of duplicate analysis were performed. Crusher duplicates where two pulps are made from the coarse reject, testing for homogenization of coarse reject and pulp duplicates where a second assay is made from the same pulverized sample, testing for homogenization of the pulverized sample. After every 30 drill samples, both of these duplicates are added to the assay stream. Blanks were made from commercially bought paver blocks and inserted as one in every 30 samples. One in every 30 samples is a gold bearing SRM alternating between CDN-GS-7G and CDN-GS-P4G. Analysis of the actual insertion rate of duplicates blanks and SRMs is approximately 1 in 20, 1 in 30, and 1 in 25 respectively. The overall coverage of control samples is 12% or roughly 1 in 8 samples.

An analysis of 82 crusher duplicates compared to the original (Figure 6-15) showed excellent agreement with an R^2 correlation coefficient of 0.99. Assays with values greater than 0.1 g/t Au do not show a significant spread. The apparent spread of assays at lower gold grades are an artifact of a log scale. Analysis of 81 pulp duplicates (Figure 6-16) similarly shows good agreement with an R^2 value of 0.96.

An analysis of 132 blanks (Figure 6-17) shows 123 samples (94%) had gold values below an acceptable limit of 0.01 g/t. CDN-GS-P4G is a low-grade gold standard with a value of 0.468 g/t and a two standard deviations range of ± 0.052 g/t by fire assay. Figure 6-18 shows 62 of 66 results (94%) from that standard were within the defined limits, and no apparent bias is observable. CDN-GS-7G is a high-grade gold standard with a value of 7.19 g/t and a two standard deviations range of ± 0.37 g/t by fire assay. Figure 6-19 shows 58 of 62 results (94%) from that standard were within the defined limits. No bias is observed in the analysis.

While a QA/QC coverage greater than 15% of the total sample data would be more robust, the combined evidence from the duplicates, blanks, and standards indicates the sample results from ALS are reliable.

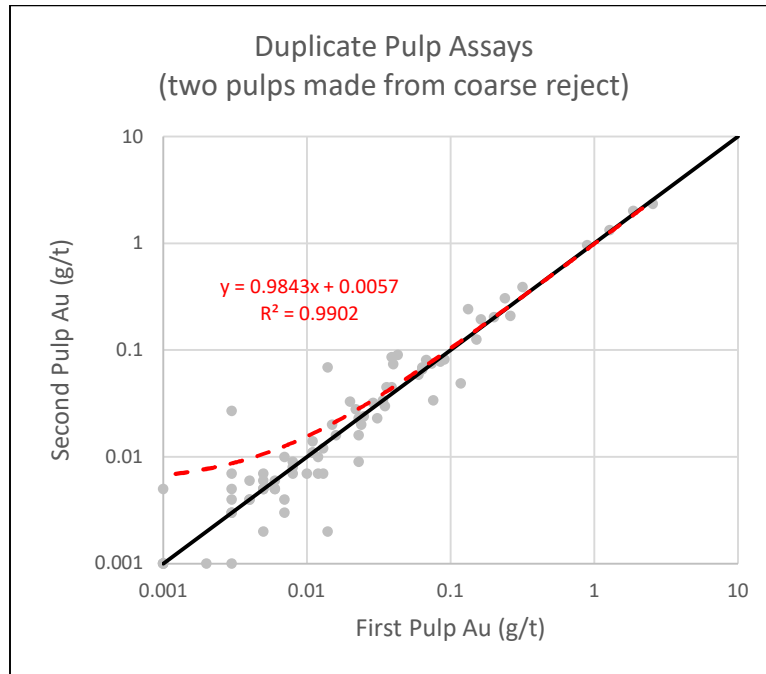


Figure 6-15 Analysis of Crusher Duplicates

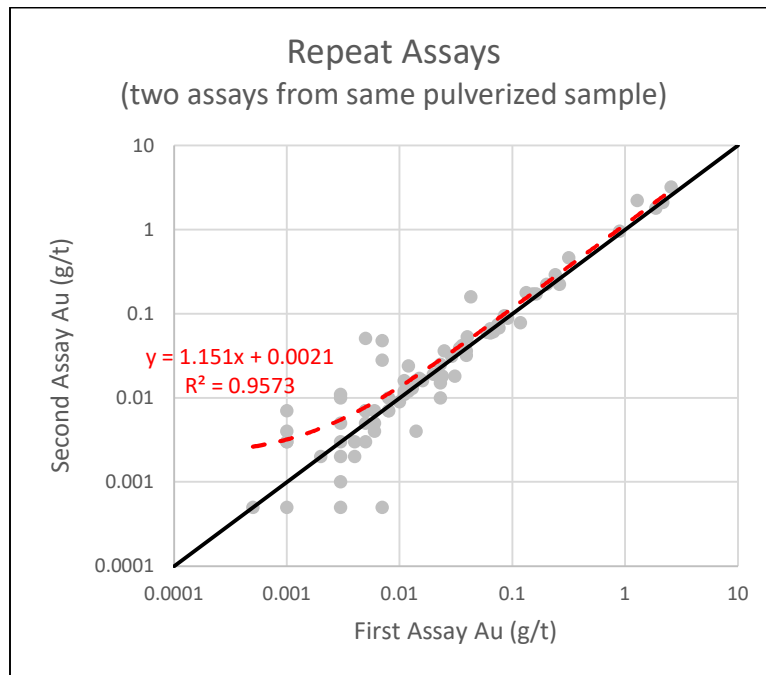


Figure 6-16 Analysis of Pulp Duplicate

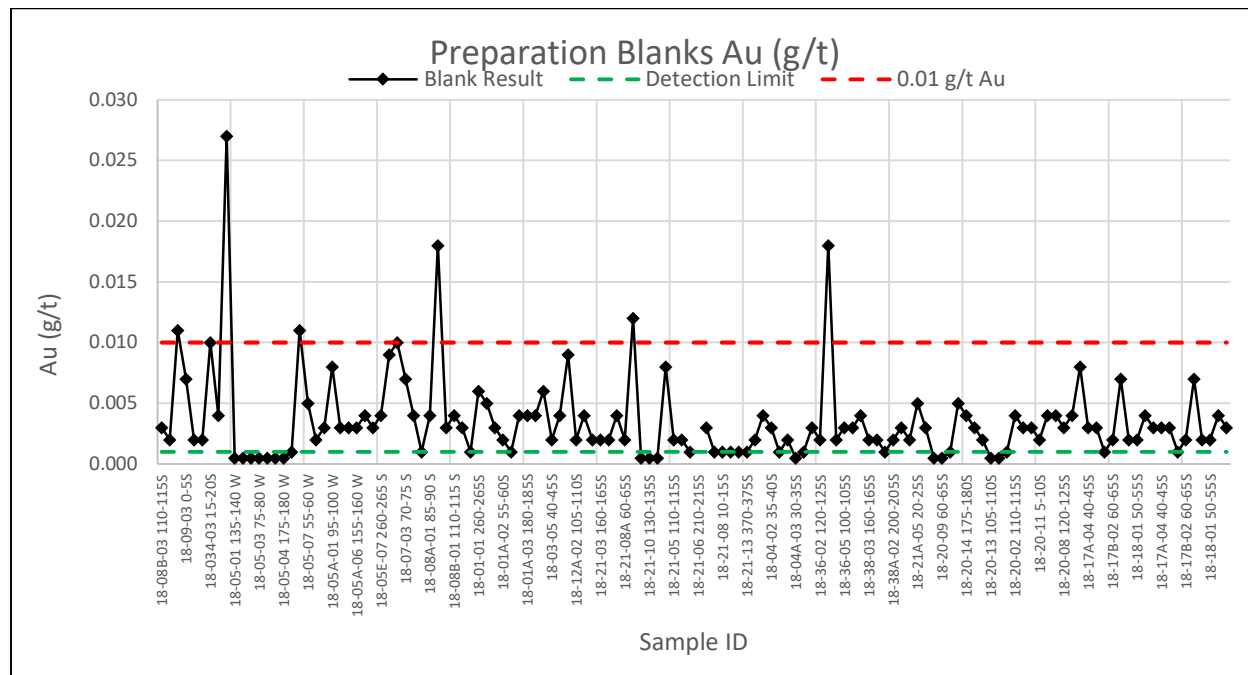


Figure 6-17 Analysis of Blank Material

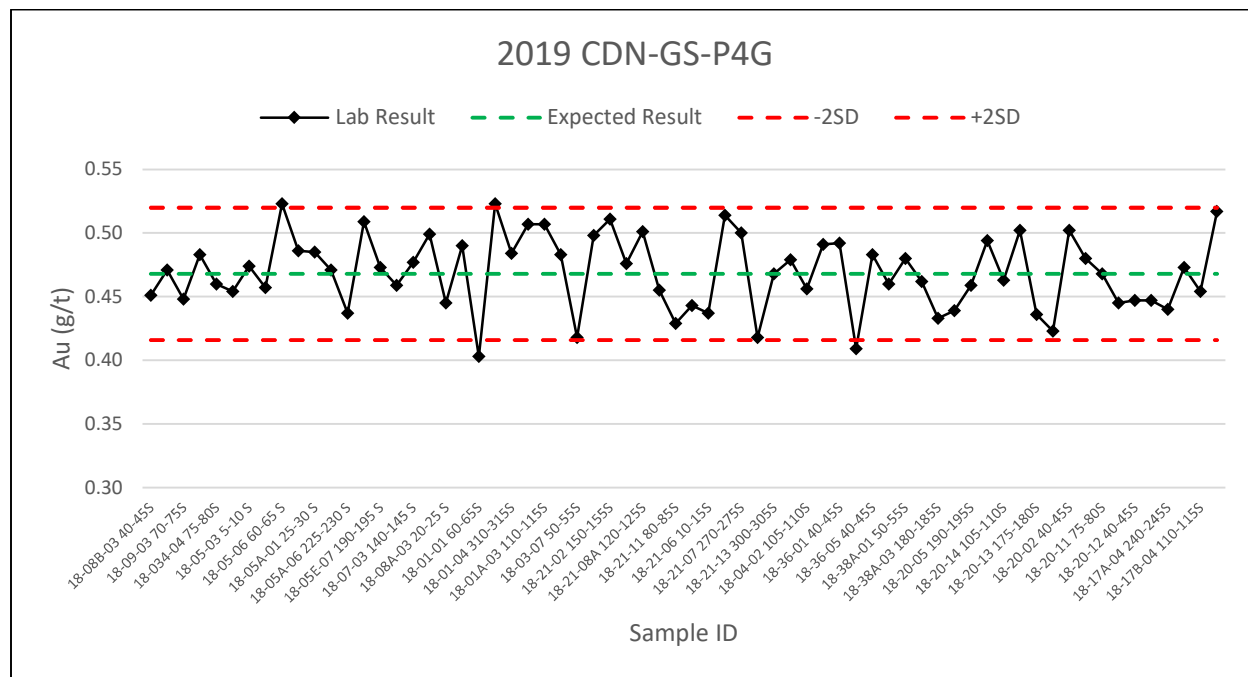


Figure 6-18 Analysis of CDN-GS-P4G

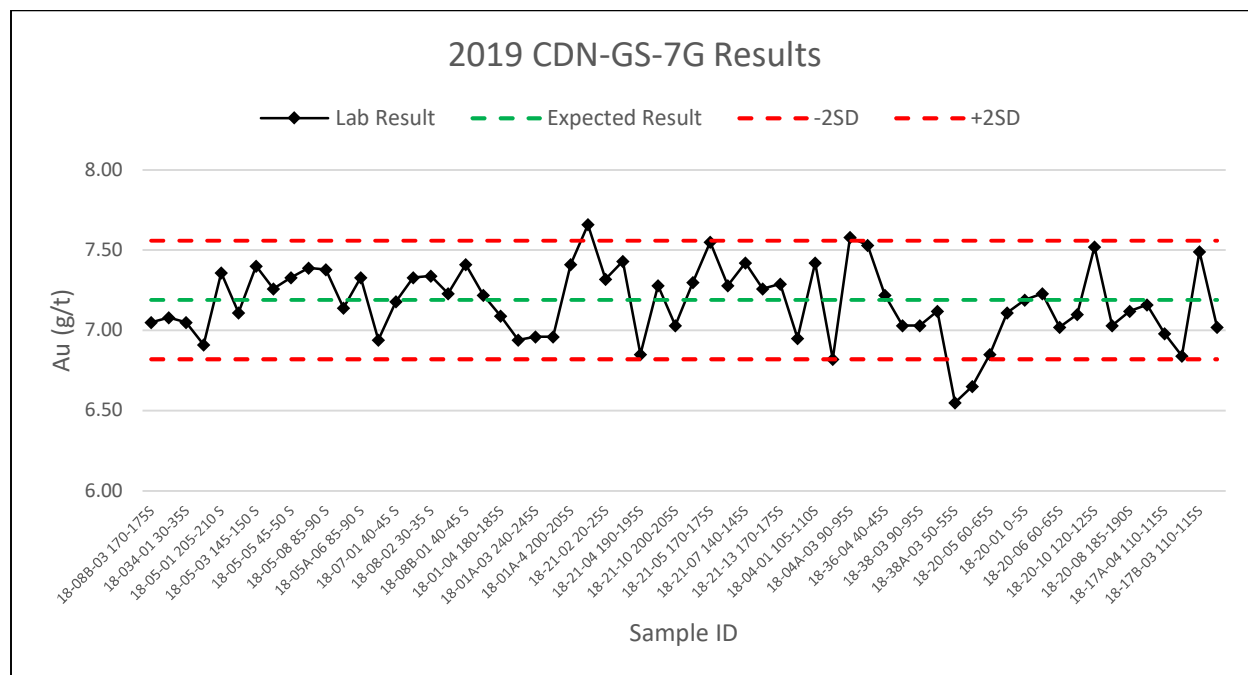


Figure 6-19 Analysis of CDN-GS-7G

Twenty-eight drillholes totaling 4,290 ft were targeted in the C zone. Significant gold intercepts are presented in Table 6-11. Only one drillhole, 18-08B-04, which was angled to explore for mineralization above currently modeled domains and was not successful in intersecting any gold mineralization. The drilling conducted out of stations 05, 05A, 05E, 08, and 08A were successful in intersecting broad zones of significant gold mineralization. While drilling did not intersect significant gold mineralization, drilling from stations 07, 08B, 17A, 17B, and 34 did intersect lower grade gold mineralization where expected, based on modeled domains.

Table 6-11 Significant Drillhole Intercepts Targeting the C Zone

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
18-05-05	No Significant Intercepts							
18-05-06	30	50	20	6.1	0.11	3.71	14	4.3
includes	40	50	10	3.0	0.14	4.80	7	2.2
also	125	135	10	3.0	0.21	7.20	7	2.2
18-05-08	50	65	15	4.6	0.13	4.52	10	3.2
includes	60	65	5	1.5	0.26	8.79	3	1.1
18-05A-01	55	80	25	7.6	0.15	5.15	15	4.6
includes	55	75	20	6.1	0.17	5.77	12	3.7
includes	55	60	5	1.5	0.38	12.85	3	0.9
18-05A-06	45	70	25	7.6	0.11	3.80	15	4.6
includes	45	60	15	4.6	0.15	5.16	9	2.8
and	55	60	5	1.5	0.28	9.54	3	0.9
also	120	145	25	7.6	0.23	7.71	16	4.9
includes	130	145	15	4.6	0.29	9.84	9	2.7
18-05E-01	40	65	25	7.6	0.10	3.27	24	7.4
includes	50	65	15	4.6	0.12	4.15	15	4.5
18-05E-07	65	80	15	4.6	0.28	9.48	15	4.5
includes	65	75	10	3.0	0.41	14.03	10	3.0
18-08-01	40	50	10	3.0	0.12	4.27	8	2.4
18-08A-02	40	80	40	12.2	0.34	11.70	33	10.1
includes	60	80	20	6.1	0.60	20.67	17	5.2
includes	70	80	10	3.0	1.12	38.25	8	2.4
18-08A-03	35	90	55	16.8	0.24	8.25	53	16.2
includes	75	90	15	4.6	0.53	18.25	15	4.6

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

Fourteen drillholes totaling 2,510 ft from stations 18, 18A, and 20 targeted the down dip extension of C zone gold mineralization. Drillholes from each station were successful in intersecting significant zones of gold mineralization. Drillhole 18-18-03A was terminated before target depth was reached. 18-20-01 and 18-20-06 were unsuccessful intersecting any gold mineralization and were targeting the eastern down dip extent of modeled domains. The remaining drillholes were successful in intersecting lower grade gold mineralization. Results of the drilling for this target suggest while gold mineralization is still present, interval thickness is thinning down dip. Table 6-12 shows significant intercepts for down dip extension of the C zone.

Table 6-12 Significant Drillhole Intercepts Targeting the Down Dip Extents of the C Zone

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
18-18-02	40	50	10	3.0	0.80	27.45	7	2.0
includes	40	45	5	1.5	1.53	52.30	3	1.0
18-18A-01	35	45	10	3.0	0.14	4.77	5	1.6
includes	35	40	5	1.5	0.26	8.87	3	0.8
18-20-02	60	70	10	3.0	0.25	8.51	7	2.3
includes	65	70	5	1.5	0.34	11.60	4	1.1
18-20-11	80	90	10	3.0	0.37	12.82	4	1.1
includes	80	85	5	1.5	0.50	17.10	4	1.1

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

The up-dip extension of C zone gold mineralization was tested with five drillholes totaling 1,300 ft from stations 01 and 01A. Only one drillhole, 18-01A-04 was successful in intersecting significant gold mineralization (Table 6-13). The remaining drillholes only intersected low grade gold mineralization suggesting C zone gold mineralization is thinning up dip.

Table 6-13 Significant Drillhole Intercepts Targeting the Up Dip Extents of the C Zone

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
18-01A-04	125	135	10	3.0	0.23	8.04	6	1.9
includes	130	135	5	1.5	0.44	15.20	3	1.0

*True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

Four drillholes totaling 1,000 ft targeted inferred gold mineralization above the C zone. While no significant gold mineralization was intersected by any of the drillholes, 18-05-01, 18-05-02, and 18-05-03 intersected gold mineralization greater than 1 g/t. 18-05-04 only intersected gold mineralization less than 1 g/t.

Twenty-four drillholes totaling 3,710 ft targeted D zone gold mineralization. Drilling from stations 04, 4A, and 36 were located in the center of the target and was successful in intersecting significant and lower grade gold mineralization. Two drillholes reaching west toward D zone from station 20 only intersected low grade gold mineralization. Drillholes from stations 17A and 17B targeting the southeast extension of the D zone were successful in intersecting low grade gold mineralization in the D zone as well as some C zone intercepts. Drillholes from station 38 and 38A targeted the northwest extent of the D zone and intersected isolated low grade gold intercepts suggesting gold mineralization thins out towards the Terminator fault. Drillhole results for the D zone target are shown in Table 6-14.

Table 6-14 Significant Drillhole Intercepts Targeting the D Zone

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
18-04-01	65	85	20	6.1	0.46	15.91	10	3.1
includes	65	80	15	4.6	0.61	21.02	7	2.2
18-04-02	75	85	10	3.0	0.13	4.32	5	1.5
18-04A-03	75	85	10	3.0	0.12	4.20	6	1.7
18-36-03	110	135	25	7.6	0.16	5.50	16	5.0
includes	110	120	10	3.0	0.31	10.65	8	2.6

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

Eleven drillholes totaling 1,025 ft targeted the extent of gold mineralization defined by historic drillhole C95-10 (25 ft grading at 75.4 g/t Au), the underground drillholes completed by Asia minerals, and gold mineralization intersected by KER-17U-21B (3 ft grading at 76.4 g/t Au). None of the drillholes from stations 03, 09, 10, and 12A intersected gold mineralization where anticipated indicating the interpretation of gold mineralization in this area at the time was incorrect. Only 18-09-03 and 18-10-05 intersected low grade D zone gold mineralization.

Fourteen drillholes totaling 3,185 ft from stations 21 and 21A targeted the down dip extension of the D zone. Drilling in this area represents some of the most significant results of the 2019 drilling campaign especially in drillholes 18-21-04, 18-21-06 and 18-21A-05 (Table 6-15). Drillholes 18-21-08, and 18-21-09 did not reach target depth and did not intersect gold mineralization. Drillhole 18-21-13 deflected away from the target and also did not intersect gold mineralization. The remaining drillholes intersected lower grade gold mineralization. The results from this drilling suggest D zone gold mineralization is still open down dip to the north.

Table 6-15 Significant Drillhole Intercepts Targeting the Down Dip Extents of the D Zone

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
18-21-04	210	245	35	10.7	0.51	17.49	17	5.1
includes	220	240	20	6.1	0.86	29.45	10	2.9
18-21-06	190	245	55	16.8	1.17	40.00	25	7.6
includes	210	220	10	3.0	2.87	98.26	5	1.4
and	230	240	10	3.0	0.97	33.19	5	1.4
18-21-11	125	145	20	6.1	0.12	4.26	10	2.9
includes	135	145	10	3.0	0.18	6.15	5	1.4
18-21A-05	45	65	20	6.1	0.44	15.02	18	5.5
includes	45	55	10	3.0	0.65	22.40	9	2.7

*True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains

6.3.11.4 2020 – 2021 Surface RC Drilling

AZG completed 21 drillholes totaling 16,625 ft from surface to define the Footwall zone, step out from gold mineralization in the C zone and A/B zone, and collect metallurgical samples in the A/B zone. Drilling started in September of 2020 and continued into January 2021. Two drilling contractors were used for the program. Alford Drilling, out of Elko Nevada, completed five drillholes totaling 4,025 ft using a Foremost 1500 track mounted RC drill. All holes were surveyed down the hole on nominal 100-ft intervals. National Drilling completed the remaining drillholes using an EDM-90 truck mounted RC rig. All drillholes were surveyed down the hole using a Reflex EZ-GYRO™ multi-shot tool on either 50-ft or 100-ft intervals. Table 6-16 summarizes drillhole IDs by contractor. Drilling was nominally oriented to the southwest, with some drilling angled at different orientations due to limited space within the pit. Collar locations were professionally surveyed by Darling Engineering following completion of the drilling program.

Five RC drillholes from the start of the 2020 drilling had samples submitted to ALS in Tucson, Arizona. Sample preparation and gold assay analysis was the same as described for the 2019 samples. Three drillholes were analyzed for an additional 33 elements using four acid digestion. Table 6-17 summarizes what elements were analyzed by drillhole.

Table 6-16 Surface RC Drillholes by Contractor

Alford Drilling; 1500 Foremost			
BHID	Depth	BHID	Depth
AZG-20S-01	1,295	AZG-20S-08	700
AZG-20S-05	605	AZG-20S-09	800
AZG-20S-06	625		
Total Depth		4,025	
National Drilling; EDM-90			
BHID	Depth	BHID	Depth
AZG-20S-02	1,200	AZG-20S-14	850
AZG-20S-03	750	AZG-20S-15	750
AZG-20S-04	850	AZG-20S-16	1,000
AZG-20S-07	765	AZG-20S-17	700
AZG-20S-10	745	AZG-20S-18	950
AZG-20S-11	750	AZG-20S-21	700
AZG-20S-12	820	AZG-20S-22	480
AZG-20S-13	725	AZG-20S-23	565
Total Depth		12,600	

Table 6-17 ALS Element Analysis by Drillhole

BHID	Analysis
AZG-20S-01	Au, Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn
AZG-20S-05	Au, Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn
AZG-20S-06	Au, Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn
AZG-20S-08	Au
AZG-20S-09	Au

QA/QC protocol for the 2020-2021 drilling incorporated the use of duplicates, several different materials for blanks and four SRMs. Analysis of the QA/QC show insertion rates of approximately 1 in 10 for duplicates, and 1 in 40 for both blanks and SRMs. The overall coverage of QA/QC samples is 16% or roughly 1 in 6 samples. The QP notes that both ALS and AAL show insertion of internal QA/QC samples into the sample stream.

Analysis of duplicates from AAL compared to originals showed excellent agreement with an R^2 value of 0.9994 (Figure 6-20). Gold grades greater than 0.1 g/t did not show any spread and the apparent spread of results less than 0.05 g/t Au is the result of the log scale. Of note, the QP did not find records of duplicates being incorporated into the sample streams for drillholes submitted to ALS.

Blank material consisting of commercial bricks, as well as certified blanks from Rocklabs and Shea Clark Smith were analyzed for both labs. Figure 6-21 shows 116 of 120 blanks (97%) were less than 0.01 g/t Au.

CDN-GS-P4G and CDN-GS-7G SRMs were once again incorporated into the QA/QC protocol. Figure 6-22 shows 42 of the 45 samples (93%) were within two standard deviations for CDN-GS-P4G. 100% of samples

for CDN-GS-7G were within two standard deviations as shown in Figure 6-23. Rocklabs Oxp50 is a high-grade gold standard with an expected value of 14.89 g/t Au and a two standard deviation range of ± 0.2465 g/t. Figure 6-24 shows one sample did not have a sufficient sample size for analysis leaving 15 of 18 samples (83%) within the two standard deviation range. Only two instances of the Rocklab SRM OxQ75 were used for QA/QC. The expected lab result is 50.03 g/t Au and the samples returned values of 48.6 g/t Au and 49.1 g/t.

The combined evidence from the QA/QC protocol for the 2020-2021 drilling indicates the sample results from ALS and AAL are reliable.

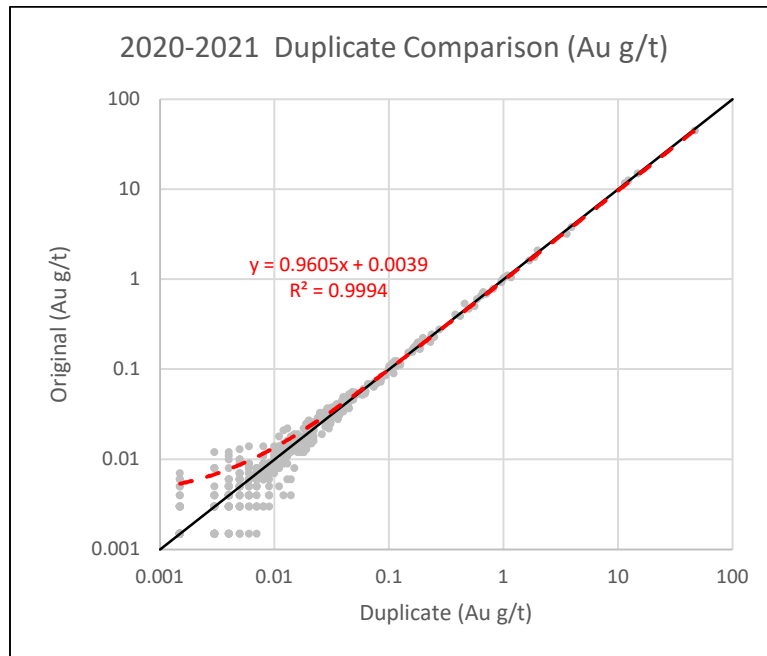


Figure 6-20 Analysis of Duplicates

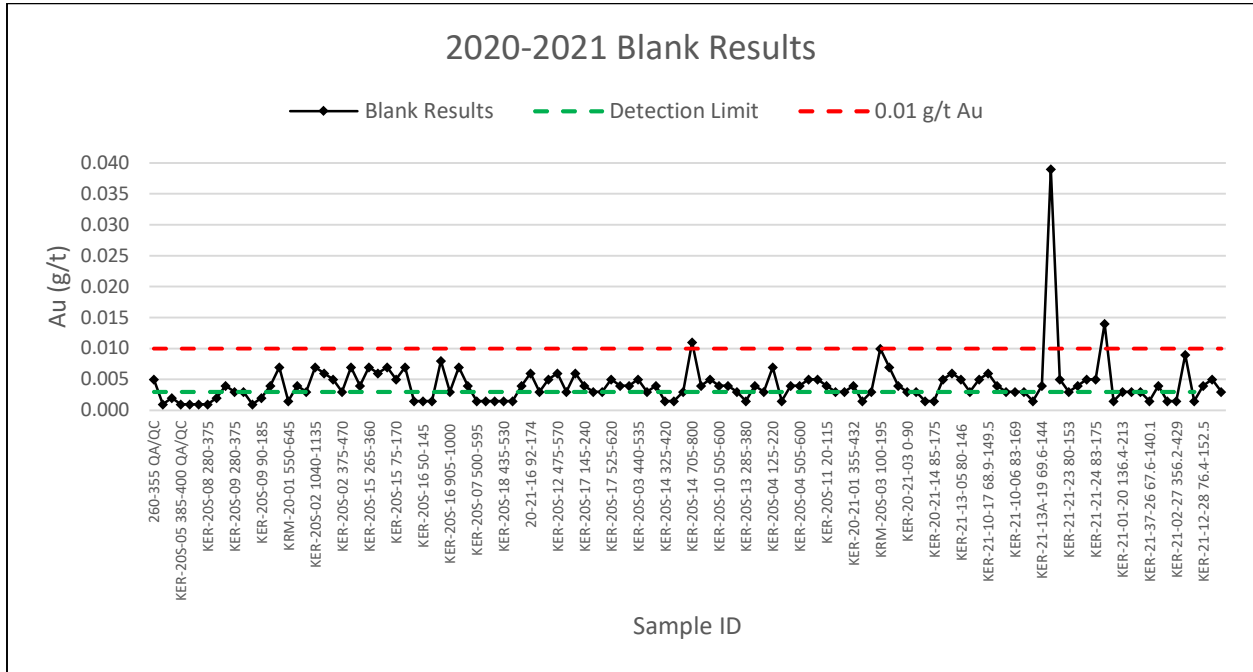


Figure 6-21 Analysis of Blank Results

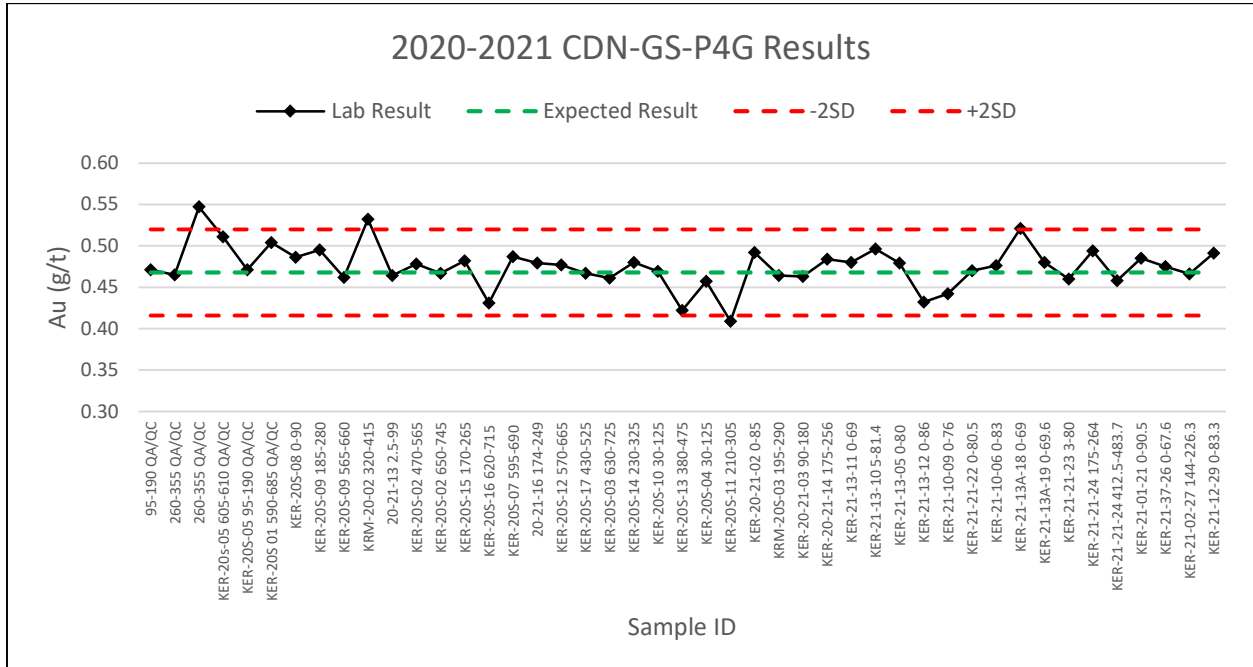


Figure 6-22 Analysis of CDN-GS-P4

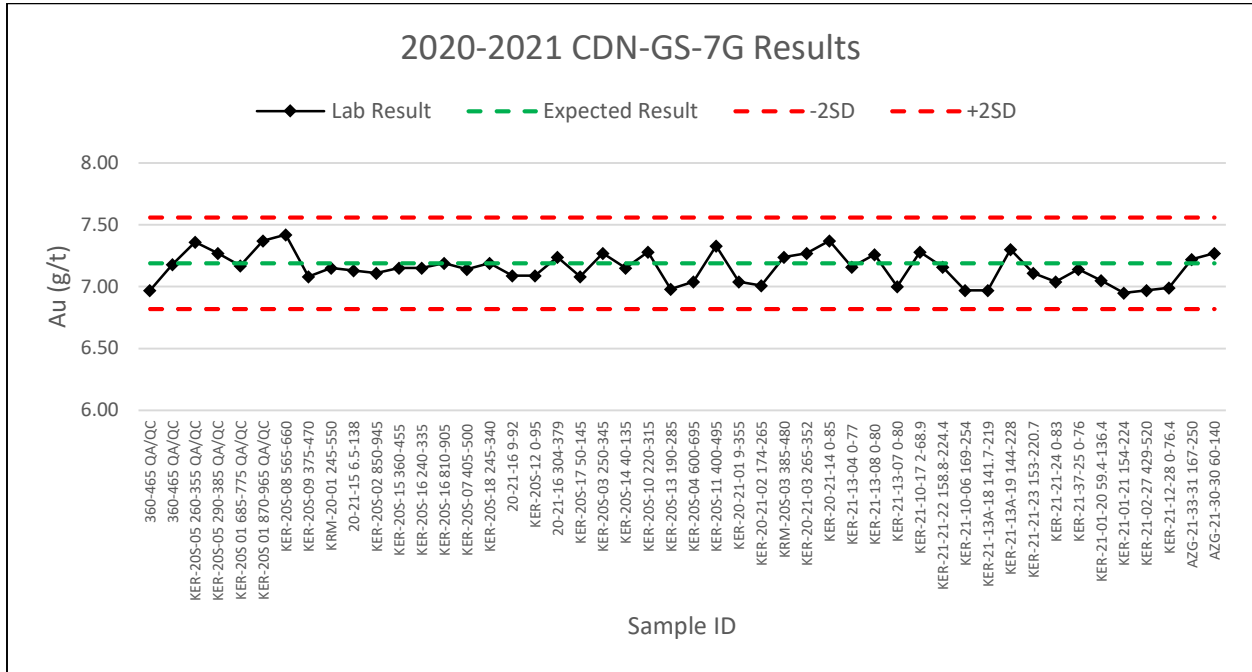


Figure 6-23 Analysis of CDN-GS-7G

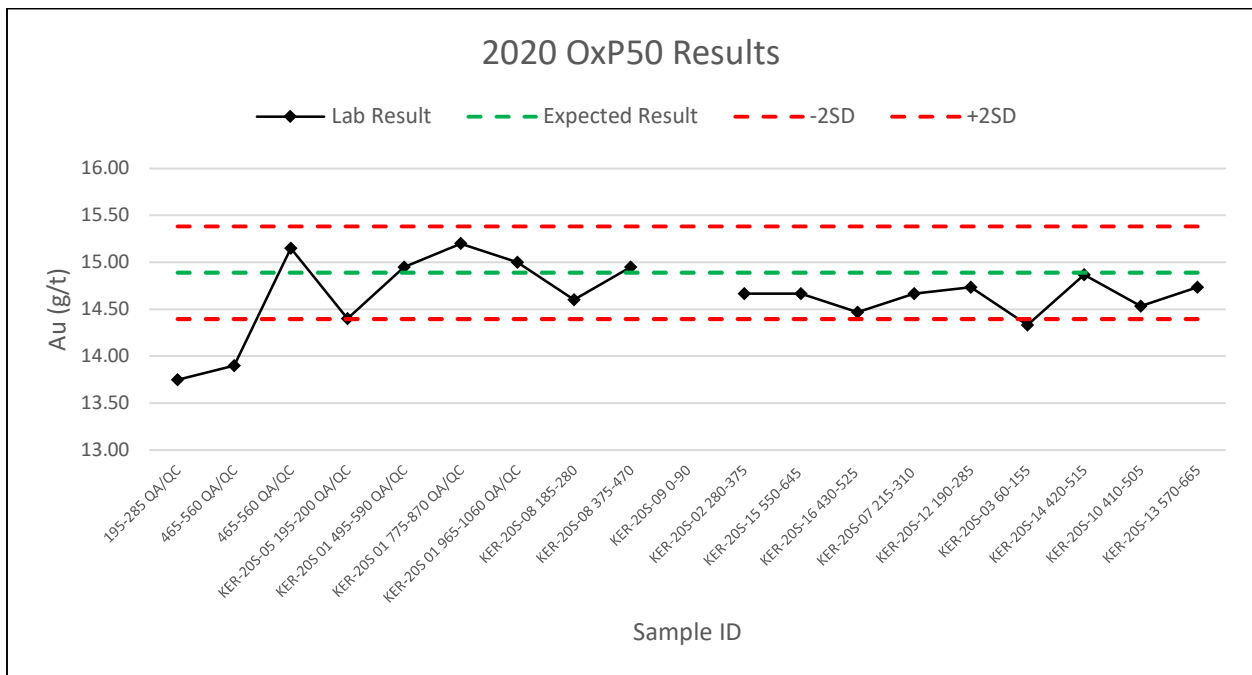


Figure 6-24 Analysis of Oxp50

Drillhole results are summarized in Table 6-18. Four drillholes targeted the C zone. Drillhole AZG-20S-01 successfully intersected the gold mineralization where projected down dip. Drillhole AZG-20S-16 also intersected gold mineralization in an area where limited historic drilling had not shown significant gold mineralization. Additionally, a 5-ft intercept grading 2.7 g/t Au below the C zone indicates the Footwall zone continues down dip. No significant mineralization was intersected in the remaining two drillholes.

Two drillholes targeted the A/B zone. AZG-20S-07 did not intersect gold mineralization southeast of the A zone along strike. AZG-20S-02 targeted between the A and B zones where limited historic drilling had not previously shown significant gold mineralization. The drillhole intersected gold mineralization southeast of the B zone along strike indicating greater continuity between the A and B zones than previous interpretations had shown. Three drillholes, AZG-20S-21, AZG-20S-22, and AZG-20S-23, were drilled for the purpose of collecting metallurgical samples in the B zone and confirmed gold mineralization intersected by historic drilling.

The remaining twelve drillholes all targeted the Footwall zone to determine the extent of gold mineralization along strike, down dip, and at depth. Only drillholes AZG-20S-03 and AZG-20S-11 did not intersect Footwall zone gold mineralization. Both of these drillholes were testing the up-dip extent of the Footwall zone. The remaining drillholes all successfully intersected and further defined the extent and geometry of the Footwall zone.

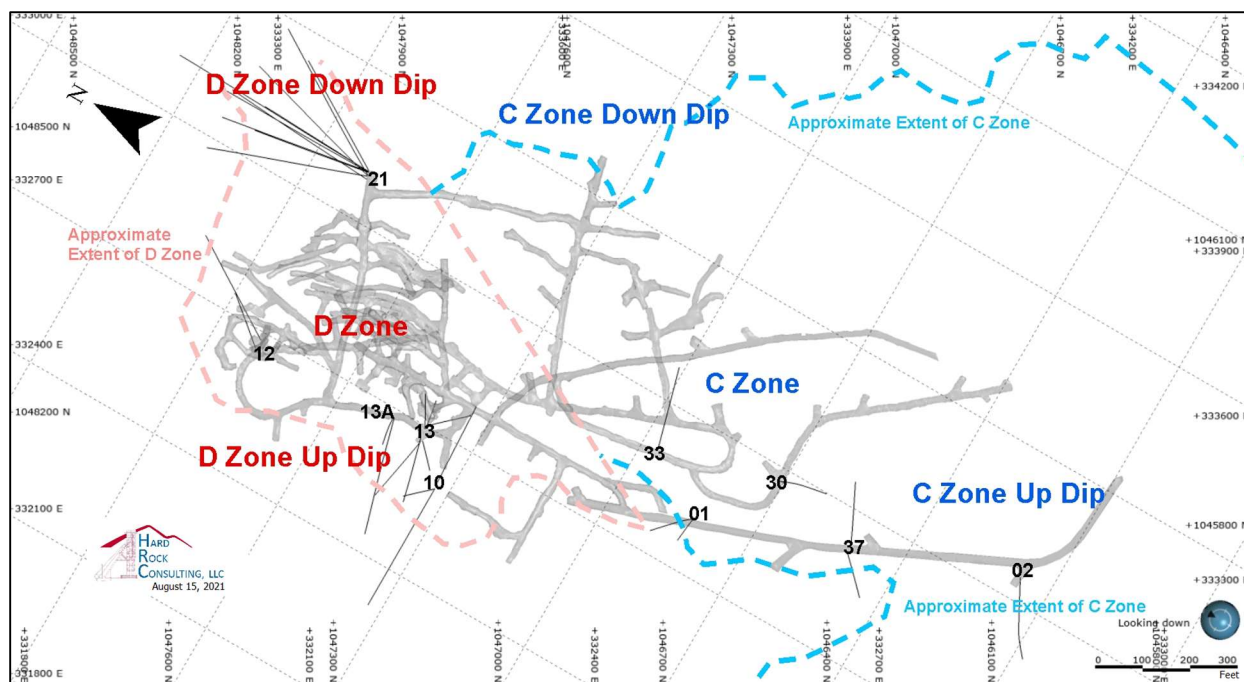
Table 6-18 Significant Results from the 2020, 2021 Surface RC Drilling

Hole ID	Target	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
				ft	m	oz/ton	g/t	ft	m
AZG-20S-01	C zone Down Dip	840	850	10	3.0	0.19	6.4	9	2.8
AZG-20S-16	C zone	730	760	30	9.1	0.06	2.0	27	8.1
includes	C zone	735	745	10	3.0	0.10	3.6	9	2.7
also	C zone	865	870	5	1.5	0.08	2.7	4	1.3
AZG-20S-02	A/B zone	840	850	10	3.0	0.07	2.3	9	2.7
includes	A/B zone	840	845	5	1.5	0.10	3.5	4	1.3
AZG-20S-21	A/B zone	505	525	20	6.1	0.14	4.9	15	4.5
includes	A/B zone	515	520	5	1.5	0.39	13.4	4	1.1
AZG-20S-22	A/B zone	280	310	30	9.1	0.18	6.3	28	8.4
includes	A/B zone	300	310	10	3.0	0.39	13.5	9	2.8
AZG-20S-23	A/B zone	360	410	50	15.2	0.10	3.3	44	13.5
includes	A/B zone	385	405	20	6.1	0.18	6.0	18	5.4
also	A/B zone	435	440	5	1.5	0.04	1.2	4	1.3
AZG-20S-04	FW zone	270	280	10	3.0	0.20	7.0	9	2.9
includes	FW zone	270	275	5	1.5	0.38	12.9	5	1.4
also	FW zone	305	310	5	1.5	0.03	1.1	5	1.4
also	FW zone	565	570	5	1.5	0.06	2.1	5	1.4
AZG-20S-05	FW zone	70	95	25	7.6	0.15	5.0	25	7.6
includes	FW zone	85	95	10	3.0	0.31	10.5	10	3.0
also	FW zone	210	225	15	4.6	0.07	2.3	15	4.6
includes	FW zone	220	225	5	1.5	0.14	4.9	5	1.5
also	FW zone	510	515	5	1.5	0.03	1.2	5	1.5
AZG-20S-06	FW zone	85	90	5	1.5	0.20	6.9	5	1.5
also	FW zone	290	320	30	4.6	0.09	3.0	30	4.6
includes	FW zone	295	300	5	1.5	0.12	4.1	5	1.5
also	FW zone	335	380	45	4.6	0.08	2.8	45	4.6
AZG-20S-09	FW zone	775	780	5	1.5	0.04	1.3	4	1.3
AZG-20S-10	FW zone	185	190	5	1.5	0.03	1.0	5	1.5
also	FW zone	495	500	5	1.5	0.18	6.3	5	1.5
AZG-20S-12	FW zone	445	460	15	4.6	0.10	3.6	13	3.8
includes	FW zone	445	455	10	3.0	0.12	4.3	8	2.6
AZG-20S-14	FW zone	295	300	5	1.5	0.03	1.1	4	1.3
also	FW zone	315	320	5	1.5	0.08	2.6	4	1.3
also	FW zone	330	335	5	1.5	0.06	1.9	4	1.3
also	FW zone	380	385	5	1.5	0.03	1.2	4	1.3
also	FW zone	765	770	5	1.5	0.22	7.5	4	1.3
also	FW zone	795	800	5	1.5	0.09	3.2	4	1.3
AZG-20S-17	FW zone	105	115	10	3.0	0.20	6.7	8	2.5
includes	FW zone	110	115	5	1.5	0.36	12.4	4	1.2
also	FW zone	345	350	5	1.5	0.08	2.8	4	1.2
also	FW zone	585	590	5	1.5	0.03	1.0	4	1.2

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

6.3.11.5 2020 – 2021 Underground Core Drilling

From November 20, 2020, through April 3, 2021 American Drilling Corporation was contracted by AZG and completed 31 NQ size core drillholes totaling 8,556 ft from multiple underground stations using a Hagby 1000 drill rig. The primary purpose of the drilling was to expand gold mineral resources in the D and C zones. All the drillholes were surveyed down-the-hole using a Reflex EZ-GYRO™ continuous survey tool. Drillhole locations were limited by access to existing underground developments with acceptable ground conditions and infrastructure. As a result, drilling is oriented in multiple directions with the intent of intersecting gold mineralization as close to true thickness as practical. A review of approximately 50% of RQD logs show an average recovery of 70%, a median recovery of 80%, with an average RQD of 28%. Drillhole collar locations were professionally surveyed by Darling Engineering. Figure 6-25 shows the locations of the underground drilling stations used in 2020 and 2021.



Source: Hard Rock Consulting, 2021

Figure 6-25 2020, 2021 Underground Drill Station Locations

Twelve drillholes targeted the up-dip portions of the D zone from stations 10, 13, and 13A and results are summarized in Table 6-19. Drillholes AZG-21-10-17, AZG-21-13-07, AZG-21-13A-18, and AZG-21-13A-19 successfully extend known D zone gold mineralization up dip. Additionally, AZG-21-10-17 and AZG-21-13A-18 intersected significant gold grades above known D zone gold mineralization. AZG-21-10-09 did not intersect the D zone up dip. AZG-21-13-12 was angled back towards the D zone and demonstrated high grade continuity between two historic drillhole intercepts. The remaining drillholes explored for mineralization below the D zone and did not intersect significant gold mineralization.

Table 6-19 Significant Results from the 2021 Underground Core Drilling in D Zone Up Dip

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
AZG-21-10-17	41.3	57.0	15.7	4.8	0.10	3.3	12	3.8
includes	41.3	44.3	2.9	0.9	0.21	7.1	2	0.7
includes	53.0	57.0	4.0	1.2	0.16	5.5	3	1.0
also	142.0	144.5	2.5	0.8	0.43	14.8	2	0.6
AZG-21-13-07	9.5	15.5	6.0	1.8	1.28	44.0	3	0.8
also	85.0	90.0	5.0	1.5	0.08	2.6	2	0.7
AZG-21-13-12	42.0	46.8	4.8	1.5	0.64	22.1	5	1.4
also	60.0	64.0	4.0	1.2	0.34	11.7	4	1.2
AZG-21-13A-18	7.2	9.7	2.5	0.8	0.03	1.1	2	0.6
also	149.0	164.5	15.5	4.7	0.07	2.3	12	3.5
includes	161.4	164.5	3.1	0.9	0.16	5.3	2	0.7
AZG-21-13A-19	61.6	69.6	8.0	2.4	0.19	6.4	3	1.0
includes	61.6	65.4	3.8	1.2	0.37	12.7	2	0.5

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

Ten holes were drilled off station 21 to extend the D zone down dip following up significant gold mineralization intersected in 2019. AZG-20-21-13 and AZG-20-21-15 were not oriented properly and missed the D zone. AZG-20-21-01, AZG-20-21-02, AZG-20-21-03, AZG-20-21-14, and AZG-20-21-16 also did not intersect gold mineralization where anticipated. AZG-21-21-22, AZG-20-21-23, and AZG-20-21-24 were angled more steeply and confirmed the 2019 drilling results and successfully intersected D zone gold mineralization at a lower elevation down dip from the 2019 drilling indicating gold mineralization is still present and open to the northeast. Significant intercepts for these drillholes are presented in Table 6-20.

Table 6-20 Significant Results from the 2020, 2021 Underground Core Drilling in D Zone Down Dip

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
AZG-21-21-22	161.0	205.8	44.8	13.7	0.14	4.8	20	6.1
includes	176.0	200.8	24.8	7.6	0.22	7.6	11	3.4
includes	198.5	200.8	2.3	0.7	0.81	27.8	1	0.3
also	264.8	270.2	5.4	1.6	0.14	4.7	2	0.8
AZG-21-21-23	163.0	168.0	5.0	1.5	0.12	4.1	2	0.7
also	189.0	212.6	23.6	7.2	0.10	3.5	10	3.1
includes	189.0	195.0	6.0	1.8	0.19	6.5	3	0.8
includes	207.6	212.6	5.0	1.5	0.21	7.1	2	0.7
also	281.6	293.0	11.4	3.5	0.13	4.5	5	1.5
AZG-21-21-24	215.0	225.0	10.0	3.0	0.55	18.7	4	1.2
includes	215.0	220.0	5.0	1.5	0.79	26.9	2	0.6
also	294.1	300.0	5.9	1.8	0.63	21.7	2	0.7
also	455.0	459.1	4.1	1.2	0.03	1.2	2	0.5
also	493.9	496.7	2.8	0.9	0.37	12.5	1	0.3

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

The nine-remaining core drillholes targeted various zones in the Project. AZG-21-33-31 confirmed C zone gold mineralization. Drillholes intended to expand the C zone up dip did not intersect any significant mineralization, although AZG-21-01-21 did intersect low grade gold. Drillholes targeting the northwest flank of the D zone only intersected low grade gold mineralization. Two drillholes explored for the Footwall zone along strike to the northwest. Although AZG-21-02-27 did not intersect any significant gold mineralization, the intercept in AZG-21-37-36 represents the furthest northwest gold intercept of the Footwall zone greater than 1 g/t. Significant intercepts for these drillholes are presented in Table 6-21.

Table 6-21 Significant Results from the 2021 Underground Core Drilling in Various Targets

Hole ID	Target	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
				ft	m	oz/ton	g/t	ft	m
AZG-21-33-31	C zone	48.6	56.5	7.9	2.4	0.13	4.5	6.2	1.9
includes	C zone	51.8	56.5	4.7	1.4	0.18	6.3	3.6	1.1
AZG-21-37-26	FW zone	152.0	156.7	4.7	1.4	0.03	1.2	5	0.3

* True thickness is calculated using the angle of the drillhole intercept, and the dip and dip direction of the modeled domains.

6.3.11.6 2021 Underground Core Infill Drilling

Thirteen diamond core drillholes totaling 1,093 ft were completed by AZG in April 2021. American Drilling Corporation was contracted to drill the holes using a Hagby 1000 drill rig. The core diameter was NQ size. Eleven of the thirteen drillholes were surveyed down-the-hole using a Reflex EZ-GYRO™ continuous survey tool. Drillhole collar locations were professionally surveyed by Darling Engineering. The holes were drilled from underground into targets of expected gold mineralization to support and guide follow up reverse circulation drilling on close-spaced centers, which will lead to final stope mine planning. The length weighted average recovery for the drilling was 68.5% with an average RQD of 19%. Average recovery and RQD increased to 73% and 27% respectively for intervals with assays greater than 1.5 g/t Au.

Fifty-eight QA/QC samples were inserted by AZG geologist into the sample stream as a check of AAL's accuracy and precision. The QA/QC samples consisted of 42 duplicates from quarter core, six blanks from commercial bricks, as well as certified blanks from Rocklabs and Shea Clark Smith, and 10 SRMs. The overall coverage of QA/QC samples was 23%, an insertion rate of approximately 1 in 4. Duplicates, blanks, and SRM had insertion rates of approximately 1 in 6, 1 in 40, and 1 in 25 respectively.

Figure 6-26 shows the results of duplicate assays compared to their original. The duplicate values are shown to plot close to the normal line and the R^2 correlation coefficient of 0.9962 confirms good agreement between the original and the duplicate results. Only one low grade sample plotted beyond acceptable limits, a success rate of 98%.

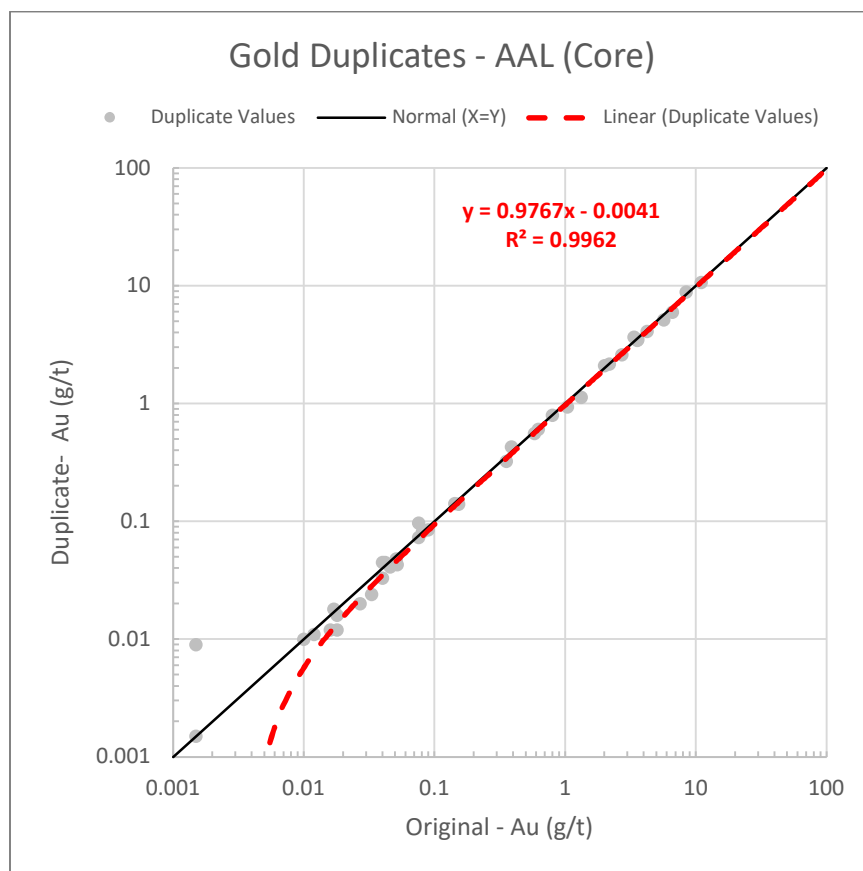


Figure 6-26 Gold Assay Results from Infill Core Duplicates

In addition to the 6 blanks inserted by AZG geologists, AAL personnel inserted an additional 16 blanks, bringing the total up to 22, an insertion rate of approximately 1 in 10. The results of the blanks are plotted in Figure 6-27. Of the 22 blanks, 21 reported values below the acceptable limit of 0.01 g/t Au. Of those, 11 reported values at or below the detection limit 0.003 g/t Au. The success rate of the combined blank analysis was 96%.

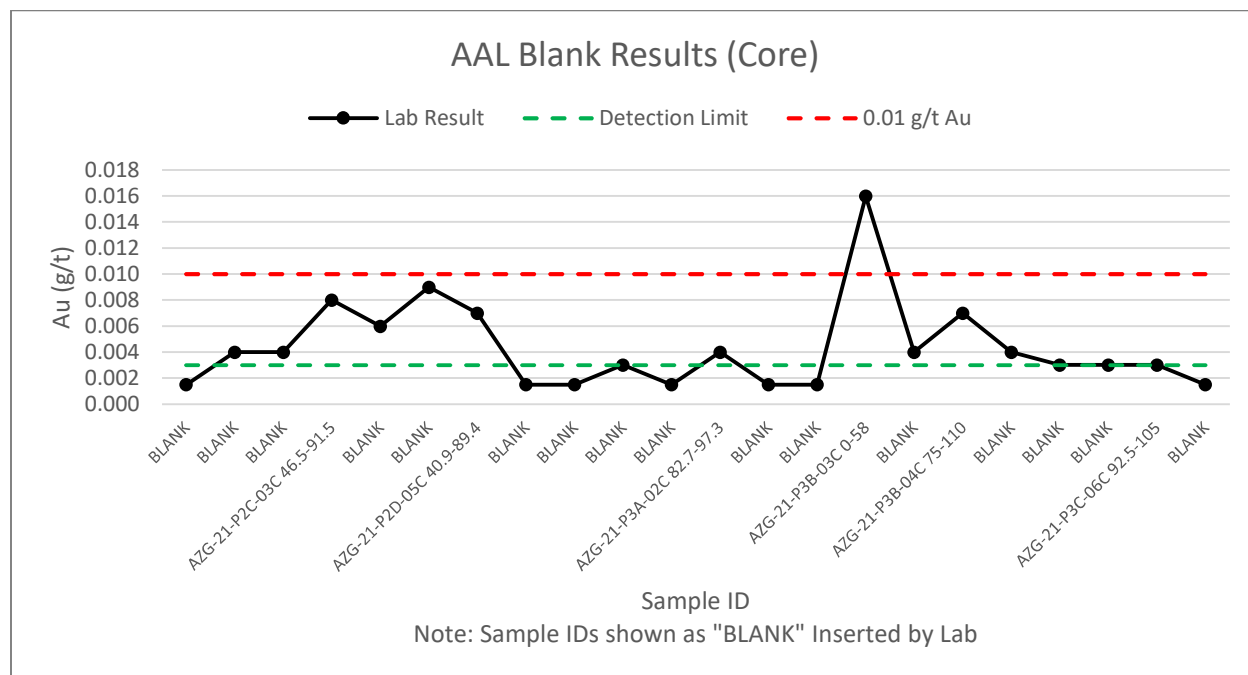


Figure 6-27 Gold Assay Results from Infill Core Blanks

CDN-GS-P4G and CDN-GS-7G SRMs were once again incorporated into the QA/QC protocol. All three results from CDN-GS-P4G (Figure 6-28) and all seven results from CDN-GS-7G (Figure 6-29) reported within acceptable limits.

In addition to the standards inserted by AZG geologists, AAL inserted an additional 16 standards of three different types. CDN-GS-1Z is a mid-grade gold standard from CDN Resource Laboratories Ltd. with a certified value of 1.155 g/t and a two standard deviations range of ± 0.095 g/t by fire assay. All 14 results from that standard were within acceptable limits. OxA147 is a low-grade gold standard from RockLabs with a certified value of 0.082 g/t and a standard deviations range of ± 0.006 g/t by fire assay. All 4 results from that standard were within acceptable limits. SJ111 is a mid-grade gold standard from RockLabs with a certified value of 2.812 g/t and a standard deviations range of ± 0.068 g/t by fire assay. The single result from that standard was within the acceptable limit.

In conclusion, the combined AZG and AAL QA/QC covers 36% of the core samples submitted with an insertion rate of 1 QA/QC sample for every 3 core samples. The combined insertion rate for duplicates was 1 in 6 and 1 in 10 for both blanks and SRM. The success rate of duplicates, blanks, and SRM was 98%, 96%, and 100% respectively. A total success rate of 98%. The combined evidence of the QA/QC analysis confirms the assay results reported by AAL for the core samples are reliable and suitable for reporting and can be included for geologic modeling and mineral resource estimation purposes.

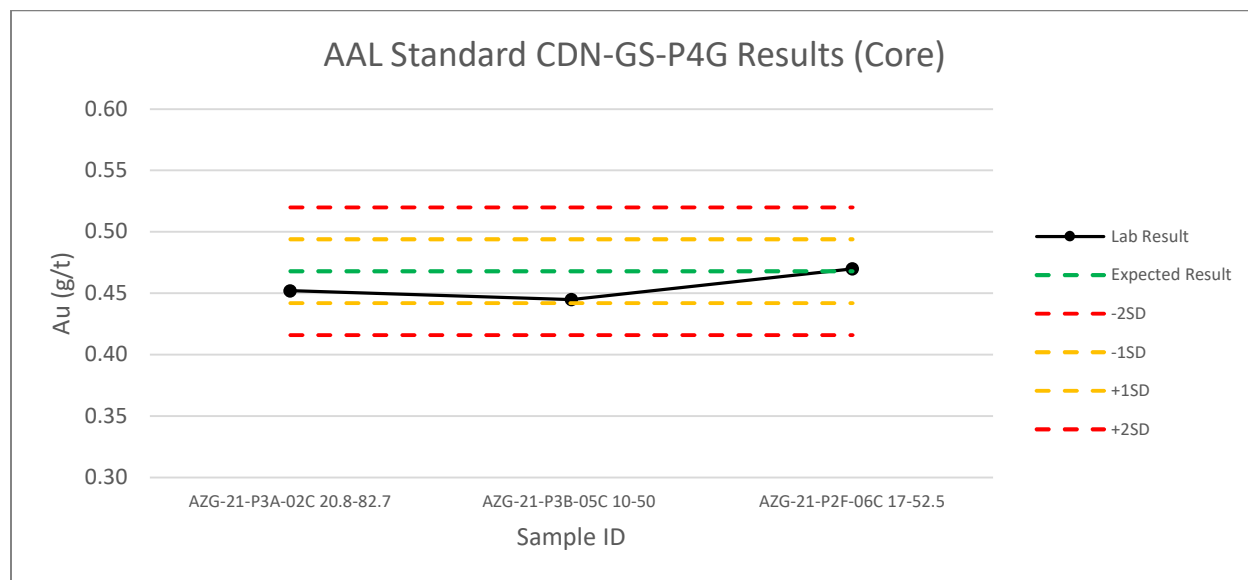


Figure 6-28 Gold Assay Results from CDN-GS-P4G

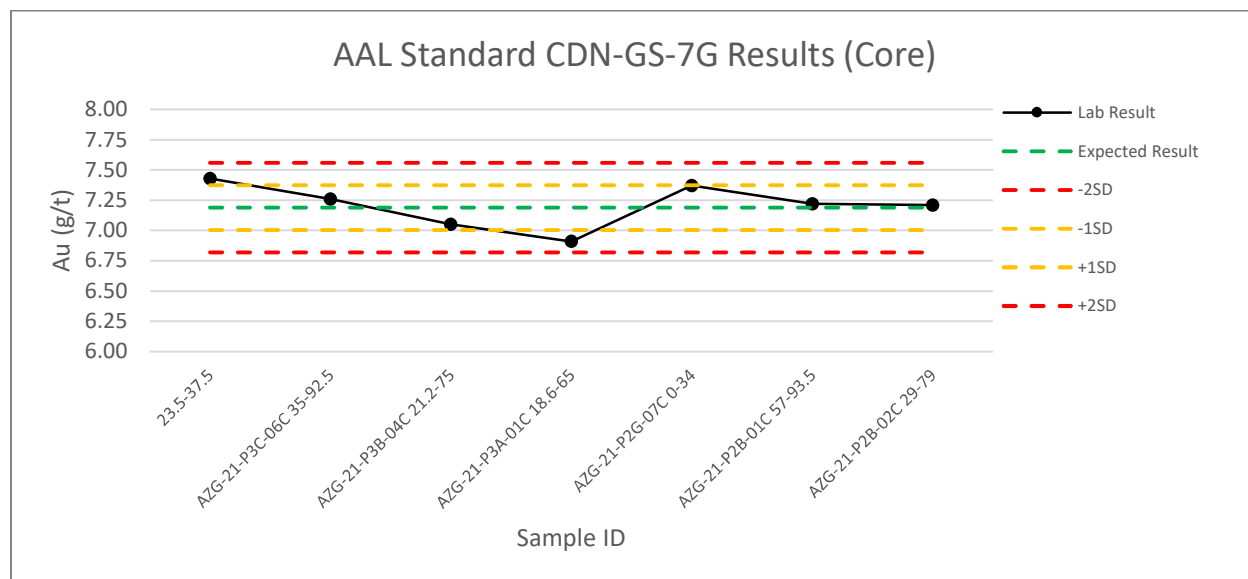
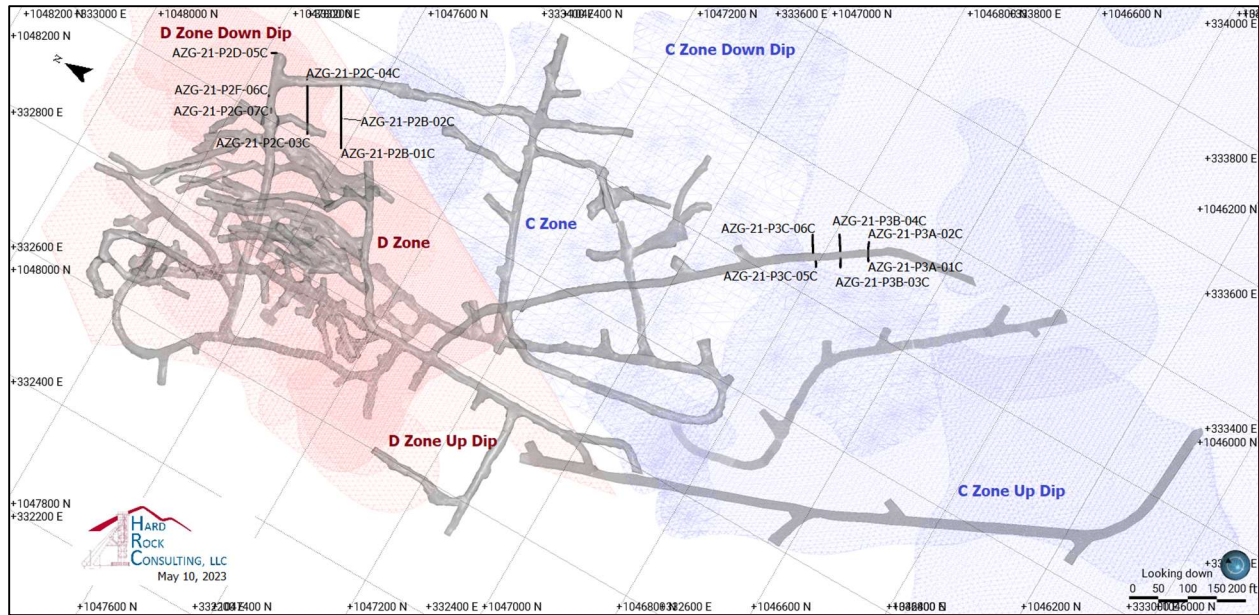


Figure 6-29 Gold Assay Results from CDN-GS-7G

All 13 drillholes intersected mineralized material proximal to the modeled domains. Only two drillholes, AZG-21-P3C-05C and AZG-21-P2B-01C, did not intersect gold mineralization greater than 2.92 g/t (0.10 oz/ton). Figure 6-30 shows the location of the infill core drilling and Table 6-22 summarizes some significant intercepts from the infill core drilling program. The drillholes from this program are included in the current geologic model and mineral resource estimate.



Source: Hard Rock Consulting, 2023

Figure 6-30 Location of 2021 Core Infill Drillholes

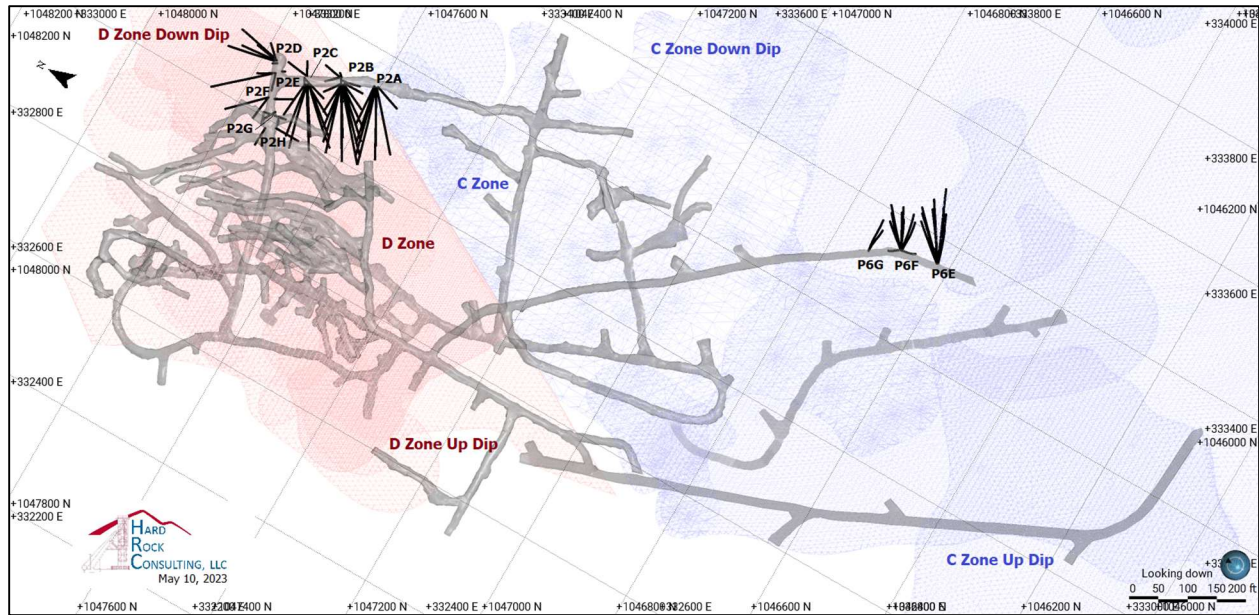
Table 6-22 Significant Results from the 2021 Underground Infill Core Drilling

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	gram/tonne	ft	m
AZG-21-P3A-01C	28.6	35.0	6.4	2.0	0.04	1.4	6.0	1.8
and	40.0	42.5	2.5	0.8	0.127	4.3	2.4	0.7
AZG-21-P3A-02C	70.0	77.5	7.5	2.3	0.310	10.6	5.7	1.7
includes	72.5	75.0	2.5	0.8	0.555	19.0	1.9	0.6
AZG-21-P3B-03C	14.0	63.0	49.0	14.9	0.133	4.6	45.0	13.7
includes	14.0	22.5	8.5	2.6	0.429	14.7	7.8	2.4
and	19.5	22.5	3.0	0.9	0.894	30.6	2.8	0.8
also	30.0	35.0	5.0	1.5	0.208	7.1	4.6	1.4
also	47.5	52.5	5.0	1.5	0.199	6.8	4.6	1.4
also	55.0	63.0	8.0	2.4	0.090	3.1	7.3	2.2
AZG-21-P3B-04C	41.3	80.0	38.7	11.8	0.258	8.8	25.0	7.6
includes	43.9	50.0	6.1	1.9	1.179	40.4	3.9	1.2
and	46.4	50.0	3.6	1.1	1.563	53.5	2.3	0.7
AZG-21-P3C-05C	35.0	50.0	15.0	4.6	0.020	0.7	13.6	4.1
includes	40.0	42.5	2.5	0.8	0.064	2.2	2.3	0.7
AZG-21-P3C-06C	52.5	90.0	37.5	11.4	0.225	7.7	23.0	7.0
includes	55.0	57.3	2.3	0.7	1.435	49.1	1.4	0.4
and	87.2	90.0	2.8	0.9	0.814	27.9	1.7	0.5
AZG-21-P2B-01C	67.0	72.5	5.5	1.7	0.041	1.4	3.5	1.1
includes	67.0	69.5	2.5	0.8	0.085	2.9	1.6	0.5
AZG-21-P2B-02C	39.0	50.0	11.0	3.4	0.334	11.4	10.2	3.2
includes	39.0	42.5	3.5	1.1	0.522	17.9	3.3	1.0
and	45.0	47.5	2.5	0.8	0.504	17.3	2.3	0.7
AZG-21-P2C-03C	56.5	74.0	17.5	5.3	0.259	8.9	14.0	4.4
includes	59.0	66.5	7.5	2.3	0.504	17.3	6.0	1.9
and	59.0	61.5	2.5	0.8	1.154	39.5	2.0	0.6
AZG-21-P2C-04C	40.4	67.0	26.6	8.1	0.254	8.7	24.1	7.5
includes	40.4	42.6	2.2	0.7	1.425	48.8	2.0	0.6
AZG-21-P2D-05C	62.5	79.3	16.8	5.1	0.045	1.5	15.1	4.7
includes	67.5	70.0	2.5	0.8	0.161	5.5	2.3	0.7
AZG-21-P2F-06C	30.0	41.0	11.0	3.4	0.159	5.4	10.1	3.1
includes	32.3	37.5	5.2	1.6	0.520	17.8	4.8	1.5
AZG-21-P2G-07C	15.0	25.5	10.5	3.2	0.331	11.3	10.2	3.2
includes	17.5	23.0	5.5	1.7	0.593	20.3	5.3	1.7

* Calculated from average dip and dip direction of modeled domains.

6.3.12 RC Infill Drilling by Sabre Gold Mines Corp

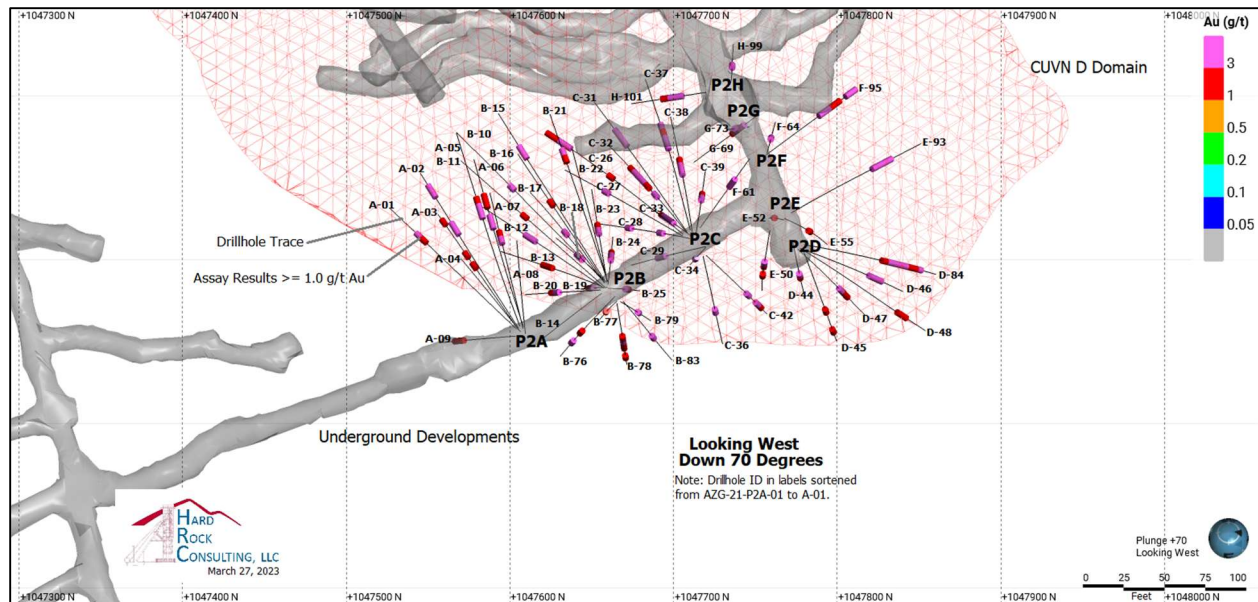
Between October 22, 2021, and December 17, 2021, SGLD completed 85 RC drillholes totaling 9,855 feet (3,004 meters) as a follow up to the infill core drilling earlier that year by AZG. Major Drilling was contracted to drill the holes using a Cubex drill rig. The holes were drilled from underground into targets of expected gold mineralization in the C zone and D zone with an average intercept spacing was 20 ft. Drillhole collar locations were professionally surveyed by Darling Engineering. Eighty of the 85 drillholes were surveyed down-the-hole using a RockTech SlimGyro™ continuous survey tool with readings taken every 20 ft. Two drillholes, AZG-21-P6A-01 and AZG-21-P6D-32, did not reach the target depth due to water issues and were not surveyed down the hole, sampled, or logged. Figure 6-31 shows the location of the RC infill drilling with station IDs labeled.



Source: Hard Rock Consulting, 2023

Figure 6-31 Location of 2021 RC Infill Drillholes

Sixty drillholes totaling 5,910 ft targeted mineralization in the D zone. The RC infill drilling targeted the same mineralized material as the infill core drillholes. Only six drillholes did not intersect mineralization grading greater than 1.7 g/t (0.05 oz/ton) gold. Forty-two of the drillholes intersected mineralization grading greater than 2.9 g/t (0.10 oz/ton) gold. Overall, the infill drilling successfully intersected the mineralized material identified in the geologic model and mineral resource estimate. Since the drilling targeted mineralization already classified as Measured and Indicated, and since the drilling confirmed the estimated grade in the area, the most appropriate use for the drilling in the D zone is for production modeling and detailed mine planning. Significant intercepts from the RC drilling in the D zone are shown in Table 6-23. A more detailed view of the D zone drilling with assays filtered to greater than 1 g/t is shown in Figure 6-32.



Source: Hard Rock Consulting, 2023

Figure 6-32 D Zone RC Infill Drilling

Table 6-23 Significant Intercepts from D Zone RC Infill Drilling

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
AZG-21-P2A-01	85.0	95.0	10.0	3.0	0.092	3.2	6.2	1.9
AZG-21-P2A-02	75.0	85.0	10.0	3.0	0.133	4.6	5.7	1.7
also	105.0	115.0	10.0	3.0	0.145	5.0	5.7	1.7
AZG-21-P2A-05	75.0	90.0	15.0	4.6	0.128	4.4	7.1	2.2
AZG-21-P2A-06	70.0	80.0	10.0	3.0	0.543	18.6	5.4	1.6
AZG-21-P2A-07	65.0	75.0	10.0	3.0	0.177	6.1	6.4	1.9
AZG-21-P2A-09	50.0	60.0	10.0	3.0	0.056	1.9	8.9	2.7
AZG-21-P2B-10	85.0	90.0	5.0	1.5	0.120	4.1	2.8	0.8
AZG-21-P2B-11	70.0	75.0	5.0	1.5	0.081	2.8	3.0	0.9
AZG-21-P2B-12	60.0	70.0	10.0	3.0	0.265	9.1	7.2	2.2
AZG-21-P2B-15	95.0	105.0	10.0	3.0	0.140	4.8	4.8	1.5
AZG-21-P2B-16	65.0	70.0	5.0	1.5	0.057	2.0	3.2	1.0
AZG-21-P2B-17	50.0	55.0	5.0	1.5	0.480	16.5	3.9	1.2
AZG-21-P2B-18	40.0	50.0	10.0	3.0	0.162	5.5	9.4	2.9
AZG-21-P2B-19	40.0	55.0	15.0	4.6	0.090	3.1	15.0	4.6
AZG-21-P2B-20	45.0	55.0	10.0	3.0	0.393	13.5	9.3	2.8
includes	45.0	50.0	5.0	1.5	0.724	24.8	4.7	1.4
AZG-21-P2B-21	85.0	95.0	10.0	3.0	0.118	4.0	5.1	1.6
AZG-21-P2B-23	45.0	55.0	10.0	3.0	0.156	5.3	8.0	2.4
includes	50.0	55.0	5.0	1.5	0.273	9.4	4.0	1.2
AZG-21-P2B-24	40.0	55.0	15.0	4.6	0.203	6.9	13.8	4.2
includes	45.0	50.0	5.0	1.5	0.381	13.1	4.6	1.4
AZG-21-P2B-25	45.0	55.0	10.0	3.0	0.082	2.8	9.0	2.8

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
AZG-21-P2B-76	70.0	75.0	5.0	1.5	0.252	8.6	4.4	1.4
AZG-21-P2B-77	50.0	55.0	5.0	1.5	0.065	2.2	4.9	1.5
AZG-21-P2B-78	50.0	60.0	10.0	3.0	0.117	4.0	8.7	2.6
AZG-21-P2B-79	50.0	55.0	5.0	1.5	0.267	9.1	4.5	1.4
AZG-21-P2B-83	55.0	60.0	5.0	1.5	0.654	22.4	3.9	1.2
AZG-21-P2C-26	95.0	115.0	20.0	6.1	0.315	10.8	11.6	3.6
includes	95.0	105.0	10.0	3.0	0.574	19.7	5.8	1.8
AZG-21-P2C-27	65.0	70.0	5.0	1.5	0.399	13.7	3.8	1.2
AZG-21-P2C-28	25.0	30.0	5.0	1.5	0.105	3.6	4.4	1.3
also	55.0	60.0	5.0	1.5	0.206	7.1	4.4	1.3
AZG-21-P2C-29	45.0	55.0	10.0	3.0	0.439	15.0	9.2	2.8
AZG-21-P2C-31	75.0	90.0	15.0	4.6	0.289	9.9	9.6	2.9
AZG-21-P2C-32	40.0	75.0	35.0	10.7	0.144	4.9	28.1	8.6
includes	55.0	70.0	15.0	4.6	0.245	8.4	12.1	3.7
AZG-21-P2C-33	35.0	55.0	20.0	6.1	0.392	13.4	19.5	6.0
includes	40.0	50.0	10.0	3.0	0.629	21.6	9.8	3.0
AZG-21-P2C-34	45.0	55.0	10.0	3.0	0.576	19.7	9.8	3.0
AZG-21-P2C-36	60.0	65.0	5.0	1.5	0.850	29.1	3.8	1.2
AZG-21-P2C-37	60.0	80.0	20.0	6.1	0.296	10.1	12.4	3.8
includes	60.0	70.0	10.0	3.0	0.526	18.0	6.2	1.9
AZG-21-P2C-38	50.0	65.0	15.0	4.6	0.425	14.5	11.4	3.5
includes	50.0	60.0	10.0	3.0	0.603	20.7	7.6	2.3
AZG-21-P2C-39	45.0	55.0	10.0	3.0	0.212	7.3	8.8	2.7
AZG-21-P2C-42	65.0	90.0	25.0	7.6	0.187	6.4	18.6	5.7
includes	80.0	85.0	5.0	1.5	0.698	23.9	3.7	1.1
AZG-21-P2D-44	60.0	70.0	10.0	3.0	0.077	2.7	9.0	2.8
AZG-21-P2D-45	70.0	75.0	5.0	1.5	0.076	2.6	3.9	1.2
also	95.0	100.0	5.0	1.5	0.065	2.2	3.9	1.2
AZG-21-P2D-46	75.0	90.0	15.0	4.6	0.165	5.7	10.1	3.1
AZG-21-P2D-47	65.0	80.0	15.0	4.6	0.494	16.9	11.1	3.4
includes	65.0	70.0	5.0	1.5	1.390	47.6	3.7	1.1
AZG-21-P2D-48	95.0	105.0	10.0	3.0	0.065	2.2	5.6	1.7
AZG-21-P2D-84	75.0	115.0	40.0	12.2	0.233	8.0	25.0	7.6
includes	95.0	100.0	5.0	1.5	0.964	33.0	3.1	1.0
AZG-21-P2E-50	50.0	55.0	5.0	1.5	0.498	17.1	4.5	1.4
AZG-21-P2E-52	45.0	50.0	5.0	1.5	0.083	2.8	5.0	1.5
AZG-21-P2E-55	50.0	55.0	5.0	1.5	0.052	1.8	4.3	1.3
AZG-21-P2E-93	80.0	100.0	20.0	6.1	0.452	15.5	11.9	3.6
includes	80.0	90.0	10.0	3.0	0.775	26.5	6.0	1.8
AZG-21-P2F-61	25.0	35.0	10.0	3.0	0.350	12.0	8.3	2.5
includes	30.0	35.0	5.0	1.5	0.571	19.6	4.2	1.3
AZG-21-P2F-64	30.0	35.0	5.0	1.5	0.277	9.5	4.7	1.4
AZG-21-P2F-95	50.0	85.0	35.0	10.7	0.493	16.9	16.0	4.9
includes	75.0	80.0	5.0	1.5	2.792	95.6	2.3	0.7
AZG-21-P2G-69	15.0	25.0	10.0	3.0	0.230	7.9	8.4	2.6
AZG-21-P2G-73	15.0	25.0	10.0	3.0	0.276	9.5	9.1	2.8
includes	15.0	20.0	5.0	1.5	0.477	16.4	4.6	1.4
AZG-21-P2H-99	15.0	20.0	5.0	1.5	0.210	7.2	4.3	1.3
AZG-21-P2H-101	15.0	25.0	10.0	3.0	0.262	9.0	5.6	1.7

* Calculated from average dip and dip direction of modeled domains.

AZG-21-P2A-01	85.0	95.0	10.0	3.0	0.092	3.2	6.2	1.9
AZG-21-P2A-02	75.0	85.0	10.0	3.0	0.133	4.6	5.7	1.7

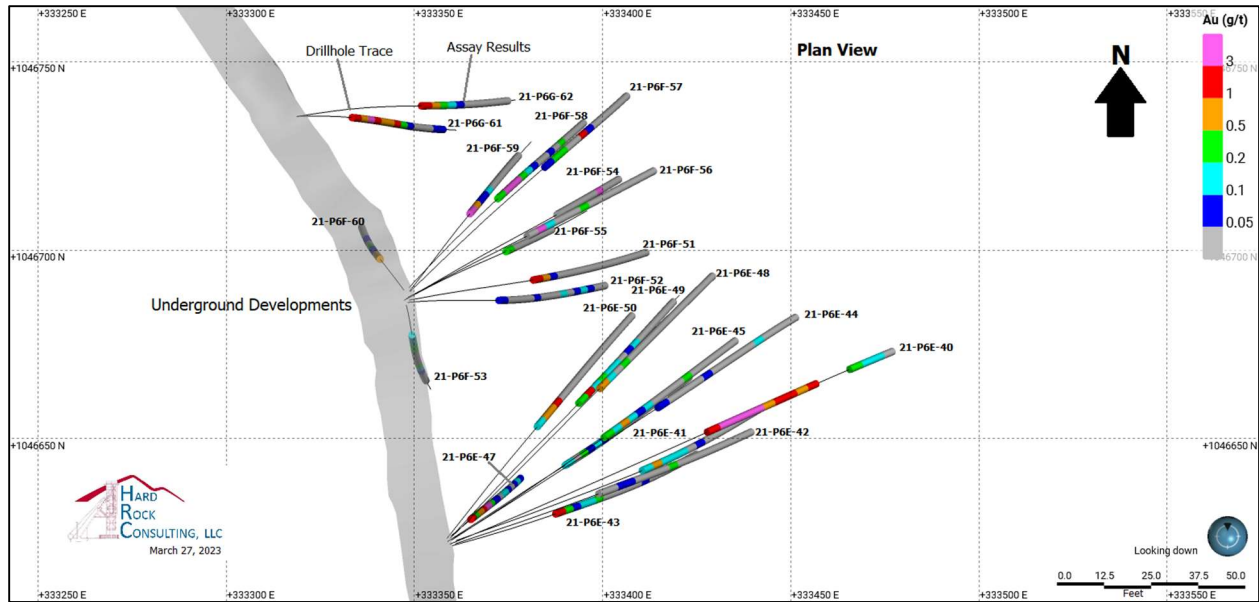
Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
also	105.0	115.0	10.0	3.0	0.145	5.0	5.7	1.7
AZG-21-P2A-05	75.0	90.0	15.0	4.6	0.128	4.4	7.1	2.2
AZG-21-P2A-06	70.0	80.0	10.0	3.0	0.543	18.6	5.4	1.6
AZG-21-P2A-07	65.0	75.0	10.0	3.0	0.177	6.1	6.4	1.9
AZG-21-P2A-09	50.0	60.0	10.0	3.0	0.056	1.9	8.9	2.7
AZG-21-P2B-10	85.0	90.0	5.0	1.5	0.120	4.1	2.8	0.8
AZG-21-P2B-11	70.0	75.0	5.0	1.5	0.081	2.8	3.0	0.9
AZG-21-P2B-12	60.0	70.0	10.0	3.0	0.265	9.1	7.2	2.2
AZG-21-P2B-15	95.0	105.0	10.0	3.0	0.140	4.8	4.8	1.5
AZG-21-P2B-16	65.0	70.0	5.0	1.5	0.057	2.0	3.2	1.0
AZG-21-P2B-17	50.0	55.0	5.0	1.5	0.480	16.5	3.9	1.2
AZG-21-P2B-18	40.0	50.0	10.0	3.0	0.162	5.5	9.4	2.9
AZG-21-P2B-19	40.0	55.0	15.0	4.6	0.090	3.1	15.0	4.6
AZG-21-P2B-20	45.0	55.0	10.0	3.0	0.393	13.5	9.3	2.8
includes	45.0	50.0	5.0	1.5	0.724	24.8	4.7	1.4
AZG-21-P2B-21	85.0	95.0	10.0	3.0	0.118	4.0	5.1	1.6
AZG-21-P2B-23	45.0	55.0	10.0	3.0	0.156	5.3	8.0	2.4
includes	50.0	55.0	5.0	1.5	0.273	9.4	4.0	1.2
AZG-21-P2B-24	40.0	55.0	15.0	4.6	0.203	6.9	13.8	4.2
includes	45.0	50.0	5.0	1.5	0.381	13.1	4.6	1.4
AZG-21-P2B-25	45.0	55.0	10.0	3.0	0.082	2.8	9.0	2.8
AZG-21-P2B-76	70.0	75.0	5.0	1.5	0.252	8.6	4.4	1.4
AZG-21-P2B-77	50.0	55.0	5.0	1.5	0.065	2.2	4.9	1.5
AZG-21-P2B-78	50.0	60.0	10.0	3.0	0.117	4.0	8.7	2.6
AZG-21-P2B-79	50.0	55.0	5.0	1.5	0.267	9.1	4.5	1.4
AZG-21-P2B-83	55.0	60.0	5.0	1.5	0.654	22.4	3.9	1.2
AZG-21-P2C-26	95.0	115.0	20.0	6.1	0.315	10.8	11.6	3.6
includes	95.0	105.0	10.0	3.0	0.574	19.7	5.8	1.8
AZG-21-P2C-27	65.0	70.0	5.0	1.5	0.399	13.7	3.8	1.2
AZG-21-P2C-28	25.0	30.0	5.0	1.5	0.105	3.6	4.4	1.3
also	55.0	60.0	5.0	1.5	0.206	7.1	4.4	1.3
AZG-21-P2C-29	45.0	55.0	10.0	3.0	0.439	15.0	9.2	2.8
AZG-21-P2C-31	75.0	90.0	15.0	4.6	0.289	9.9	9.6	2.9
AZG-21-P2C-32	40.0	75.0	35.0	10.7	0.144	4.9	28.1	8.6
includes	55.0	70.0	15.0	4.6	0.245	8.4	12.1	3.7
AZG-21-P2C-33	35.0	55.0	20.0	6.1	0.392	13.4	19.5	6.0
includes	40.0	50.0	10.0	3.0	0.629	21.6	9.8	3.0
AZG-21-P2C-34	45.0	55.0	10.0	3.0	0.576	19.7	9.8	3.0
AZG-21-P2C-36	60.0	65.0	5.0	1.5	0.850	29.1	3.8	1.2
AZG-21-P2C-37	60.0	80.0	20.0	6.1	0.296	10.1	12.4	3.8
includes	60.0	70.0	10.0	3.0	0.526	18.0	6.2	1.9
AZG-21-P2C-38	50.0	65.0	15.0	4.6	0.425	14.5	11.4	3.5
includes	50.0	60.0	10.0	3.0	0.603	20.7	7.6	2.3
AZG-21-P2C-39	45.0	55.0	10.0	3.0	0.212	7.3	8.8	2.7
AZG-21-P2C-42	65.0	90.0	25.0	7.6	0.187	6.4	18.6	5.7
includes	80.0	85.0	5.0	1.5	0.698	23.9	3.7	1.1
AZG-21-P2D-44	60.0	70.0	10.0	3.0	0.077	2.7	9.0	2.8
AZG-21-P2D-45	70.0	75.0	5.0	1.5	0.076	2.6	3.9	1.2
also	95.0	100.0	5.0	1.5	0.065	2.2	3.9	1.2
AZG-21-P2D-46	75.0	90.0	15.0	4.6	0.165	5.7	10.1	3.1
AZG-21-P2D-47	65.0	80.0	15.0	4.6	0.494	16.9	11.1	3.4
includes	65.0	70.0	5.0	1.5	1.390	47.6	3.7	1.1
AZG-21-P2D-48	95.0	105.0	10.0	3.0	0.065	2.2	5.6	1.7

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
AZG-21-P2D-84	75.0	115.0	40.0	12.2	0.233	8.0	25.0	7.6
includes	95.0	100.0	5.0	1.5	0.964	33.0	3.1	1.0
AZG-21-P2E-50	50.0	55.0	5.0	1.5	0.498	17.1	4.5	1.4
AZG-21-P2E-52	45.0	50.0	5.0	1.5	0.083	2.8	5.0	1.5
AZG-21-P2E-55	50.0	55.0	5.0	1.5	0.052	1.8	4.3	1.3
AZG-21-P2E-93	80.0	100.0	20.0	6.1	0.452	15.5	11.9	3.6
includes	80.0	90.0	10.0	3.0	0.775	26.5	6.0	1.8
AZG-21-P2F-61	25.0	35.0	10.0	3.0	0.350	12.0	8.3	2.5
includes	30.0	35.0	5.0	1.5	0.571	19.6	4.2	1.3
AZG-21-P2F-64	30.0	35.0	5.0	1.5	0.277	9.5	4.7	1.4
AZG-21-P2F-95	50.0	85.0	35.0	10.7	0.493	16.9	16.0	4.9
includes	75.0	80.0	5.0	1.5	2.792	95.6	2.3	0.7
AZG-21-P2G-69	15.0	25.0	10.0	3.0	0.230	7.9	8.4	2.6
AZG-21-P2G-73	15.0	25.0	10.0	3.0	0.276	9.5	9.1	2.8
includes	15.0	20.0	5.0	1.5	0.477	16.4	4.6	1.4
AZG-21-P2H-99	15.0	20.0	5.0	1.5	0.210	7.2	4.3	1.3
AZG-21-P2H-101	15.0	25.0	10.0	3.0	0.262	9.0	5.6	1.7

* Calculated from average dip and dip direction of modeled domains.

Twenty-five drillholes totaling 3,945 ft targeted mineralization in the C zone. The RC holes targeted mineralized material further southeast along strike of the infill core drillholes. Twelve drillholes did not intersect mineralization grading greater than 1.7 g/t (0.05 oz/ton) gold. Seven drillholes intersected mineralization grading greater than 2.9 g/t (0.10 oz/ton) gold. Significant intercepts from the RC drilling in Panel 6 are shown in Table 6-24. A more detailed view of the C zone drilling is shown in Figure 6-33.

The most significant limitation in discussing the results from the RC infill drilling was the predetermination of sample intervals. Rock chips for sampling were not collected outside of the predetermined sample interval range. Several drillholes start sampling in well mineralized material, therefore, the actual thickness of mineralization is not known. The issue is most pronounced in the C zone infill drilling. Sample intervals were predetermined based on the estimation domain model and the mineral resource estimate. The mineral resources were classified as Measured and Indicated. The poor performance from the C zone infill drilling is likely due to the location of the estimation domain being higher in elevation than currently modeled. The area of mineralization targeted by the infill drilling is informed by 16 drillholes, with an average intercept grade of 3.36 g/t (0.098 oz/ton) gold. The location uncertainty is the result of five drillholes completed by Cyprus in 1985 not being surveyed down-the hole and assumed vertical. While deviation in vertical drilling is generally not significant, any deflection in the drilling would cause the mineralized intercept to be higher in elevation than the assumed vertical drillhole. Since the C zone RC infill drilling was only partially sampled and non-sampled areas cannot be assayed, the drilling is not suitable for validation of the mineral resource estimate or production modeling purposes.



Source: Hard Rock Consulting, 2023

Figure 6-33 C Zone RC Infill Drilling

Table 6-24 Significant Intercepts from C Zone RC Infill Drilling

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
21-P6E-40	125.0	145.0	20.0	6.1	0.474	16.2	5.3	1.6
21-P6E-47	60.0	75.0	15.0	4.6	0.108	3.7	10.6	3.2
21-P6E-50	100.0	105.0	5.0	1.5	0.083	2.9	2.4	0.7
21-P6F-51	85.0	90.0	5.0	1.5	0.053	1.8	2.8	0.8
21-P6F-53	70.0	75.0	5.0	1.5	0.745	25.5	4.1	1.3
21-P6F-54	145.0	150.0	5.0	1.5	0.103	3.5	2.7	0.8
21-P6F-56	75.0	80.0	5.0	1.5	0.107	3.7	2.1	0.6
21-P6F-57	130.0	135.0	5.0	1.5	0.079	2.7	2.4	0.7
21-P6F-58	90.0	105.0	15.0	4.6	0.786	26.9	8.2	2.5
includes	90.0	100.0	10.0	3.0	1.124	38.5	5.4	1.7
21-P6F-59	75.0	80.0	5.0	1.5	0.388	13.3	3.0	0.9
21-P6G-61	40.0	75.0	35.0	10.7	0.052	1.8	21.3	6.5
includes	50.0	60.0	10.0	3.0	0.089	3.1	6.1	1.9

* Calculated from average dip and dip direction of modeled domains.

The sample handling followed similar procedures as described in the internal report “2017 QA/QC Procedures and Results, Copperstone Mine”, dated March 13, 2018, and prepared under the direction of AZG’s and SGLD’s drilling program supervisor, Mr. R. Michael Smith, SME-RM. The RC infill drilling was initially overseen by a geologist employed by SGLD with experience on the Project. Three contract geologists with no prior experience on the Project were hired to assist with the day-to-day operations of the RC drilling including logging, sampling, and insertion of QA/QC material. Early into the program, the geologist employed by SGLD left the Project, leaving only the contract geologists overseeing the drilling on a day-to-day basis. Intervals were pre-selected for analysis prior to drilling based on the modeled estimation domains. Samples were collected in five-foot intervals by drill helpers at the drill site. Sample bags were placed in a five-gallon bucket under the secondary port of the cyclone & rotary splitter in order to direct proper sample delivery. Samples were not collected in zones not designated for sample analysis, although representative samples of the RC chips were taken for the entire drillhole on every five-foot interval for geologic characterization. The samples are then stacked on a pallet and moved to a loading area where they are loaded into bins, awaiting shipment for analysis. All of these activities take place in a fenced-in area at the mine under 24-hour security. Samples were shipped to assay labs directly from site. Chain-of-custody documentation is employed, with the geologist, site manager, delivery employee and lab recipient signing for custody and receipt. The samples are placed onto pallets and wrapped for shipment; the pallets are inspected for integrity throughout shipment to laboratories.

Paper and digital copies of drill logs, survey data, shift reports and sample transmittal forms are stored at the geology office and in digital form in the SharePoint files for Copperstone.

The RC samples were submitted to ALS in Tucson, Arizona. ALS is ISO/IEC 17025 accredited and is independent of SGLD. Samples were prepared by first weighing the samples, logging them into the ALS global tracking system and then dried in ovens. Samples are then crushed to 70% passing 2mm, split using a Boyd rotary splitter, and then the split is pulverized to 85% passing 75 microns. Samples were analyzed for gold by fire assay at ALS Reno, Nevada. If the resulting gold assay exceeded 10 ppm, the sample was re-assayed

with a gravimetric finish. All samples, except QA/QC duplicates, were re-analyzed for gold and copper by cyanide leach at ALS Vancouver, British Columbia.

SGLD submitted 54 samples from the 2021 RC infill drilling for metallic screen analysis. The samples came from both the D zone and C zone drillholes. Figure 6-34 shows the comparison between the two assay methods. Similar to the metallic screen analysis completed in 2017 (Figure 6-5), the samples show good agreement between the assays with no bias towards either method. Two outlier results were identified and show higher gold grades in the fire assay than the metallic screen analysis. The results from both metallic screening studies suggest the presence of coarse gold does not impact the reliability of gold assays on the Project.

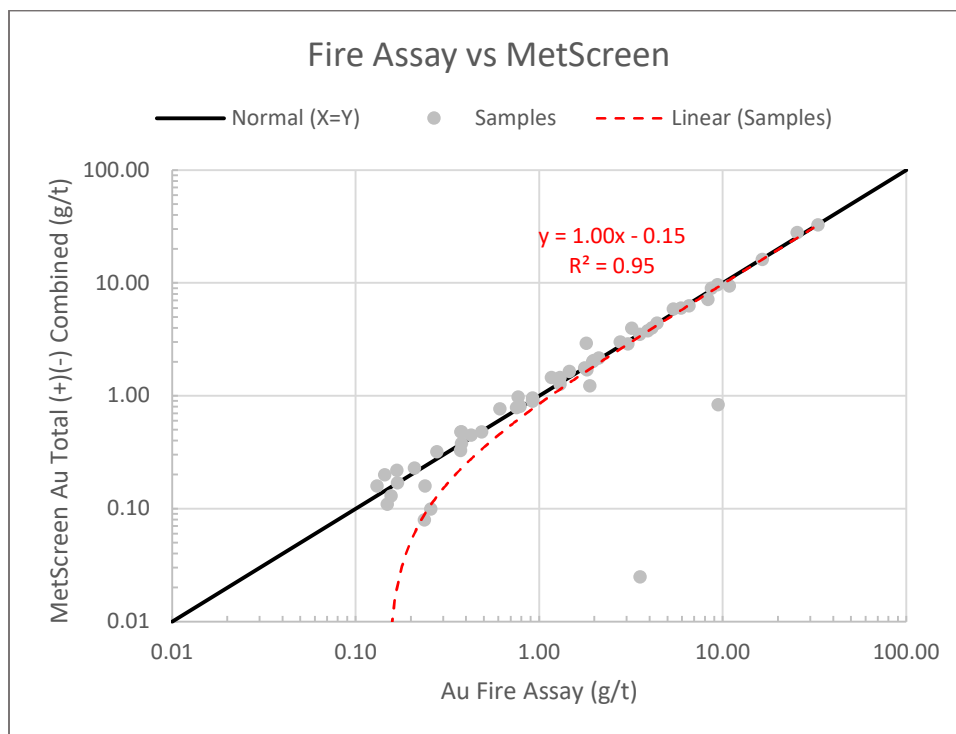


Figure 6-34 Metallic Screen Assay Comparison - 2021

SGLD geologists inserted 122 QA/QC samples into the sample stream as a check of ALS's accuracy and precision. The QA/QC samples consisted of 75 duplicates, 25 blanks from commercial bricks, as well as certified blanks from Rocklabs and Shea Clark Smith, and 22 SRM. The overall coverage of QA/QC samples was 11%, an insertion rate of approximately 1 in 9.

Figure 6-35 shows the gold fire assay results for 32 duplicates compared to their original. The remaining 43 duplicates only reported results for gold and copper by cyanide leach, and those plots shown in Figures 6-36 and 6-37 show similarly good comparisons. The duplicate values are shown to plot close to the normal line and the R^2 correlation coefficient of 0.9986 confirms good agreement between the original and the duplicate results. Three low grade samples had results outside of acceptable limits, a success rate of 90.6%.

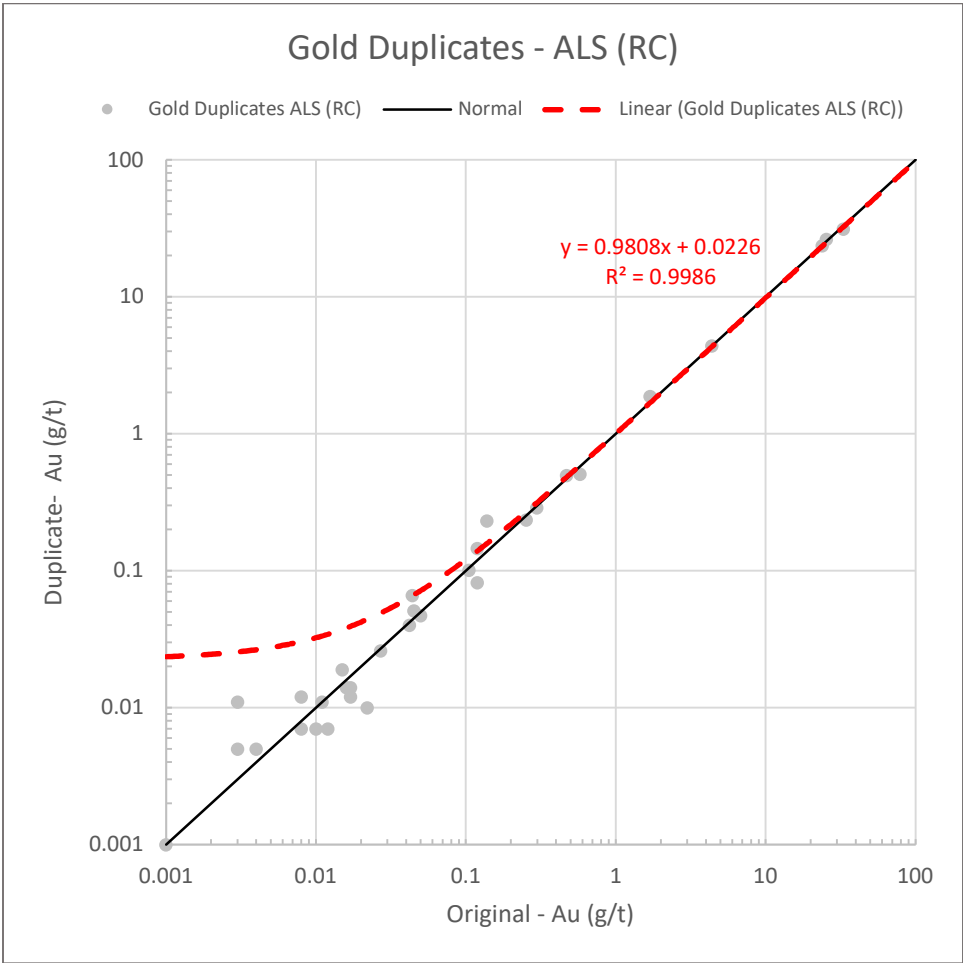


Figure 6-35 Gold by Fire Assay Results from RC Duplicates

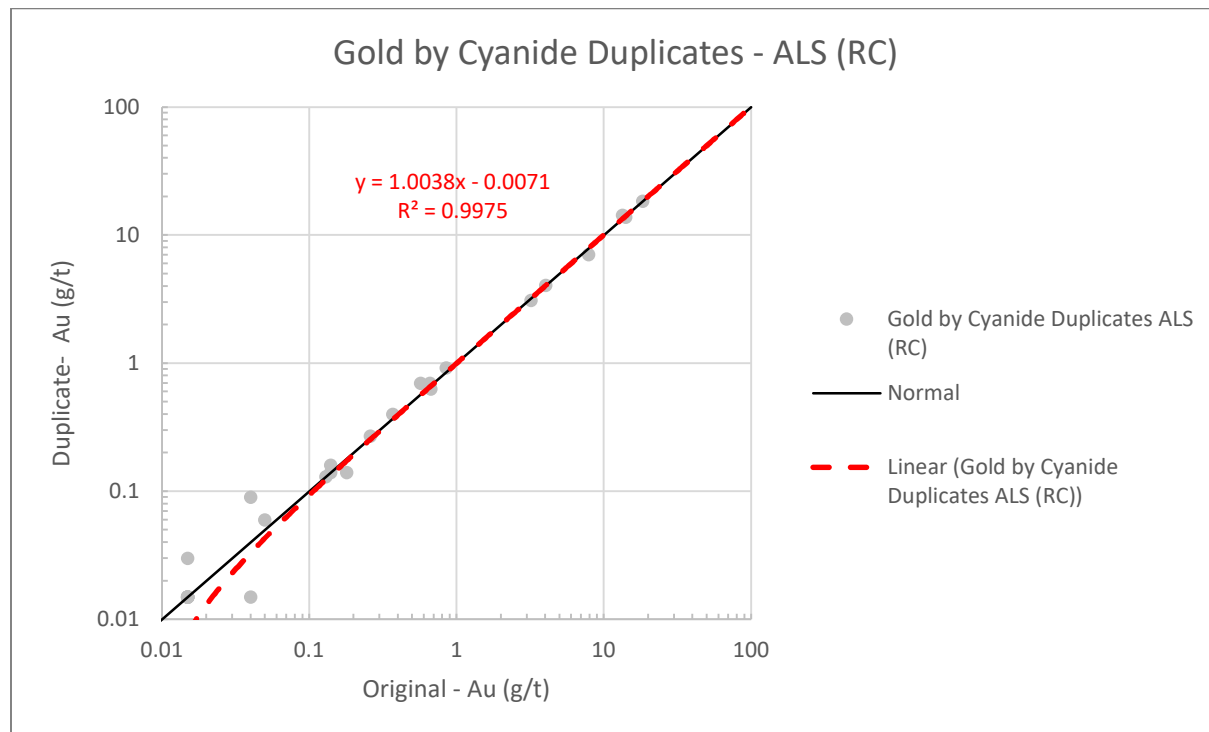


Figure 6-36 Gold by Cyanide Leach Assay Results from RC Duplicates

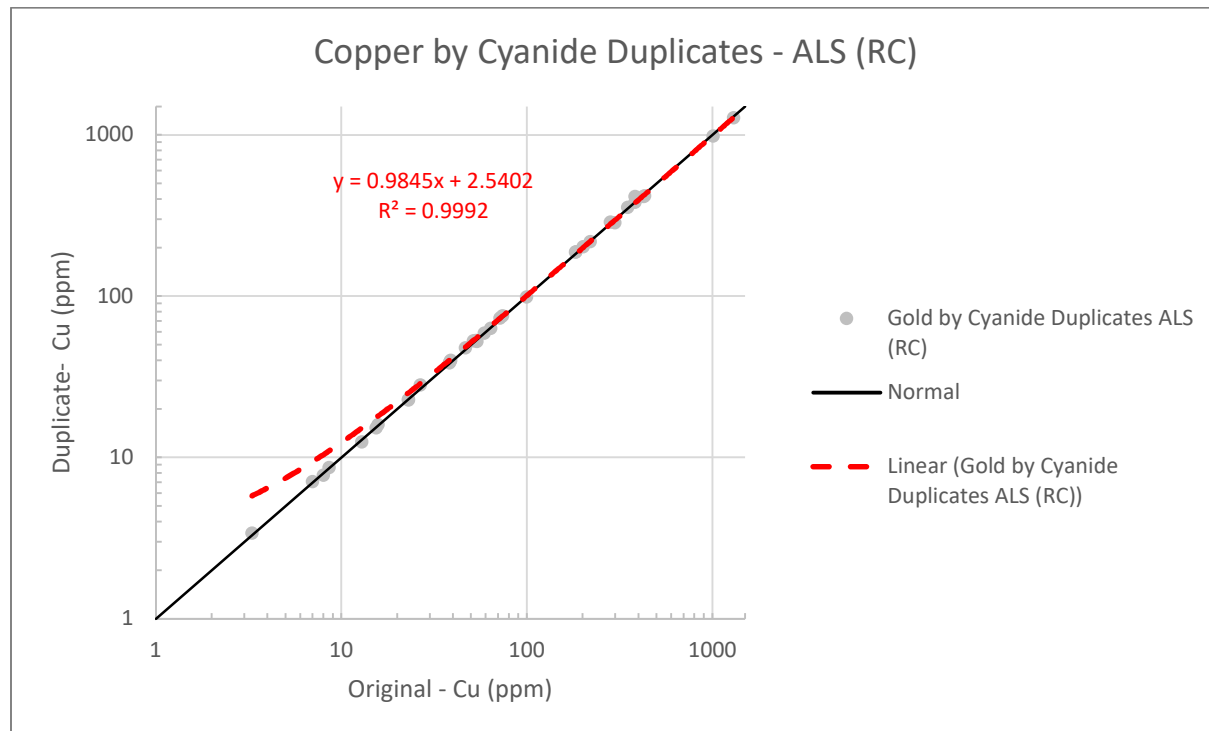


Figure 6-37 Copper by Cyanide Leach Assay Results from RC Duplicates

In addition to the 25 blanks inserted by SGLD geologists, ALS personnel inserted an additional 76 blanks, bringing the total blanks up to 101, an insertion rate of approximately 1 in 11. The results of the blanks are plotted in Figure 6-38. Of the 101 blanks, 92 reported values within the acceptable limit of 0.01 g/t Au. Of those, 70 reported values at or below the detection limit 0.001 g/t Au. Nine blanks reported gold values above 0.01 g/t, all of those blanks were inserted by SGLD. The success rate of the combined blank analysis was 91%.

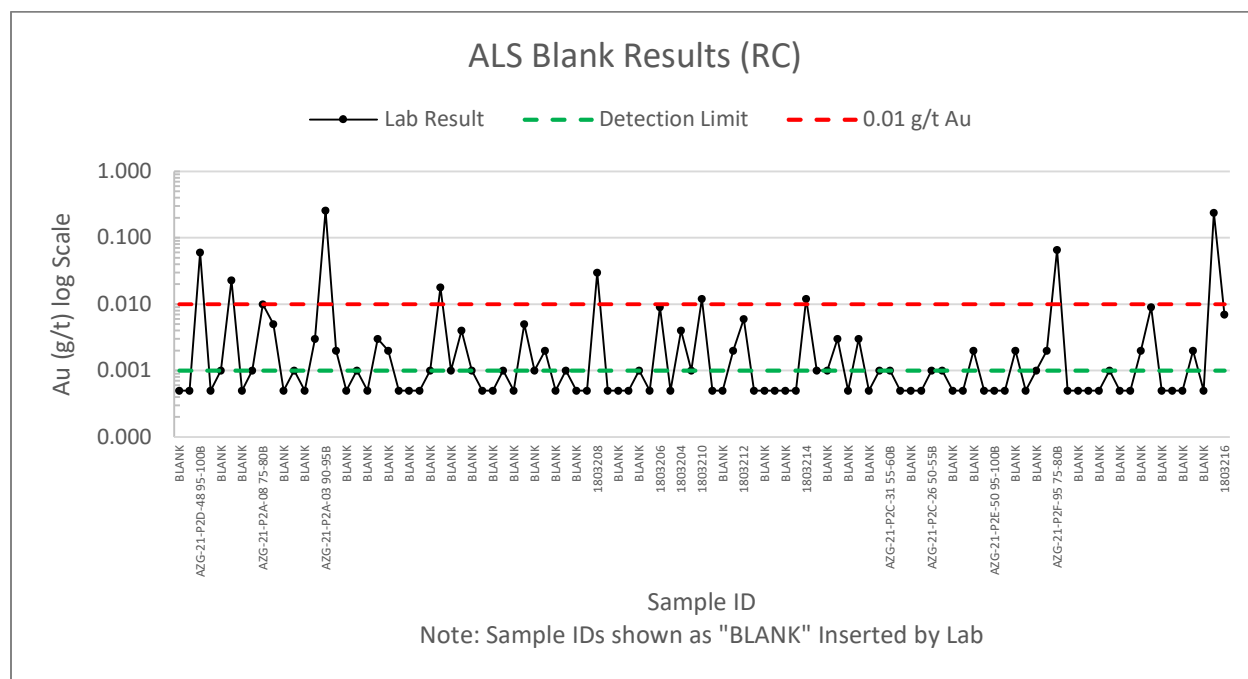


Figure 6-38 Gold Assay Results from RC Blanks

SGLD geologists inserted 22 SRM samples from CDN Resource Laboratories Ltd. of two different types into the sample stream. CDN-GS-P4G is a low-grade gold standard with a certified value of 0.468 g/t and a two standard deviations range of +/- 0.052 g/t by fire assay. Figure 6-39 shows the eleven results from that standard, and 5 results were outside the 2 standard deviation range. CDN-GS-7G is a high-grade gold standard with a certified value of 7.19 g/t and a two standard deviations range of +/- 0.37 g/t by fire assay. Figure 6-40 shows the eleven results from that standard, and all but one result reported within acceptable limits.

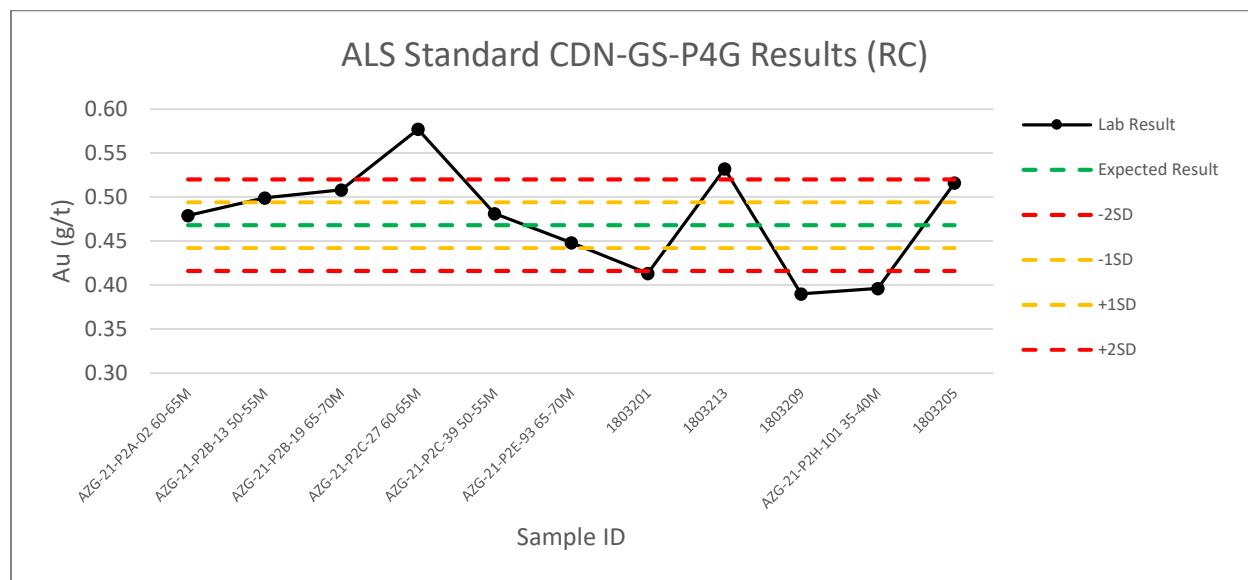


Figure 6-39 Gold Assay Results from CDN-GS-P4G

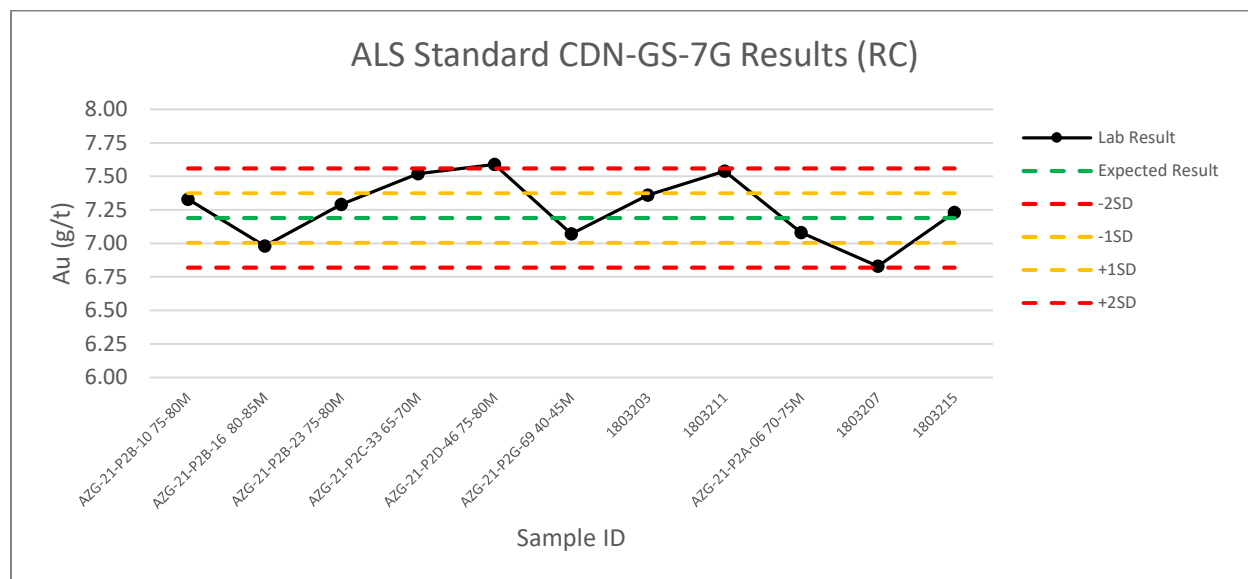


Figure 6-40 Gold Assay Results from CDN-GS-7G

In addition to the standards inserted by SGLD geologists, ALS inserted an additional 179 gold standards of eight different types, 268 copper by cyanide standards of eight different types, and 274 gold by cyanide standards of eight different types. A limitation to the validation effort for the laboratory standards was the absence of certificates for some of the SRM. When a certificate could be found, analysis of the performance of the SRM was completed in the same manner as the SGLD SRM. When a certificate for an SRM could not be found, the performance of the SRM was measured by calculating the average and standard deviation of the laboratory results. The upper limit and lower limit were established by adding or subtracting two

standard deviations from the calculated average. The gold SRM had a 95% success rate, the copper by cyanide SRM had a 97% success rate and the gold by cyanide SRM had a 96% success rate.

In conclusion, the combined SGLD and ALS QA/QC for gold analysis by fire assay covers 30% of the RC samples submitted with an insertion rate of 1 QA/QC sample for every 3 chip samples. The combined insertion rate for duplicates was 1 in 35, 1 in 11 for blanks, and 1 in 6 for SRM. The success rate for both duplicates and blanks were 91%, and the success rate for SRM was 93%. A total success rate of 92%. The combined evidence of the QA/QC analysis confirms the assay results reported by ALS for the RC samples are reliable and suitable for reporting purposes. Future drilling programs are recommended to have a higher insertion rate of at least 1 in 15 for blanks and standards prior to submission to the assay laboratory

6.4 Drillhole database

In 2017, AZG (as Kerr) commenced logging new drillhole lithology, alteration, mineralization, and structural information into a simplified and standardized format. SGLD continued the use of this format in its RC infill drilling program. Efforts have been made to bring historical drilling information into the same format. SGLD established a formal drillhole database for the Project using MxDeposit® software in 2024.

6.5 Historical Production

Ackerman (1998) reported production by Cyprus at Copperstone of 514,000 oz of gold from 6,200,000 tons (5,600,000 metric tonnes) grading 0.089 oz/ton of gold. This mill feed was produced via open pit surface mining methods, and the gold bearing mineralized material was processed using heap leach and tank leach methods. In 2011 American Bonanza constructed a 450 stpd flotation mill on site and in 2012 started underground mining from two previously developed declines located in the bottom of the Cyprus open pit. American Bonanza's mining focused on the D zone which is to the north of the open pit. From January 2012 to July 2013 American Bonanza produced approximately 16,900 oz of gold from 163,000 tons grading 0.104 oz/ton of gold. The authors are unaware of any other production information available for the Copperstone Project.

6.6 Historical Mineral Resource and Mineral Reserve Estimates

All previously reported mineral resource estimates for the Copperstone Project are superseded by the mineral resource estimate presented in Section 14 of this report. The historic estimates prior to February 2025 pre-date current NI 43-101 reporting requirements, do not conform to modern CIM definition standards, and are not considered relevant nor worthy of further discussion in this report.

A Mineral Resource Estimate for the Copperstone Project was announced January 2025 by Mining Americas as part of the NI 43-101 Preliminary Economic Assessment. Table 6-25 shows a summary of this estimate.

Table 6-25 Mineral Resource Statement for the Copperstone Project, La Paz County, Arizona, U.S.A., Hard Rock Consulting, LLC, February 15, 2023

Classification	Mass		Gold		
	Tons	Tonnes	Troy Ounces	Average Grade	
				t. oz/sh. ton	g/t
Measured	827,000	750,000	196,000	0.237	8.12
Indicated	503,000	457,000	104,000	0.207	7.09
Measured + Indicated	1,330,000	1,207,000	300,000	0.226	7.74
Inferred	1,069,000	970,000	197,000	0.184	6.30

The Mineral Reserve outlined in Section 15 is the maiden Mineral Reserve, previous Reserves do not conform to modern CIM definition standards and are not considered relevant nor worthy of further discussion in this report.

7 GEOLOGICAL SETTING AND MINERALIZATION

Much of the following text is modified and/or directly excerpted from PhD dissertations prepared by Knapp (1989) and Salem (1993). **Mr. Segerstrom** has reviewed this information and available supporting data and documentation in detail and finds the **information and** discussions presented **in these dissertations** to be reasonable and suitable for use in this report.

7.1 Regional Geology

As well described by Knapp (1989), the region of west-central Arizona records a complex history of deformation, metamorphism, and magmatism along the southwestern edge of the North American craton. This area was the site of extensive basement-involved folding and thrusting in the Maria fold and thrust belt during Late Cretaceous time. Episodes of southeast and south-directed folding and thrusting were followed by phases of north-vergent deformation, and syntectonic metamorphism was characterized by greenschist to lower amphibolite facies. Lower crustal levels experienced partial melting due to this crustal thickening, and large volumes of granitic plutons were intruded at higher crustal levels. Subsequently, large portions of middle crustal rocks were exposed in middle Tertiary time as the result of major crustal extension in the regional Whipple-Buckskin-Rawhide detachment terrain. This deformation was typically accompanied by greenschist facies, retrograde metamorphism, basin formation, and both basaltic and granitic magmatism.

The Copperstone Project is situated at the northern tip of the Moon Mountains in west-central Arizona (Figure 7-1), regionally within the Basin and Range geo-physiographic province, and within the westernmost extent of the Whipple-Buckskin-Rawhide detachment system. The Whipple-Buckskin-Rawhide detachment system is centrally located within the Maria fold and thrust belt (Reynolds et al., 1986), which extends from southeastern California to central Arizona. Mid-Tertiary low-angle normal faults (detachment faults) are recognized as significant regional structures in this portion of the Basin and Range, where major detachment faults are associated with mylonitization of lower-plate rocks and brittle faulting and rotation of upper-plate rocks. In general, mylonitic foliations are low-dipping and contain well-developed northeast-plunging mineral lineations. Upper plate rocks as young as mid-Tertiary dip moderately to the southwest and are cut by northeast-dipping normal faults.

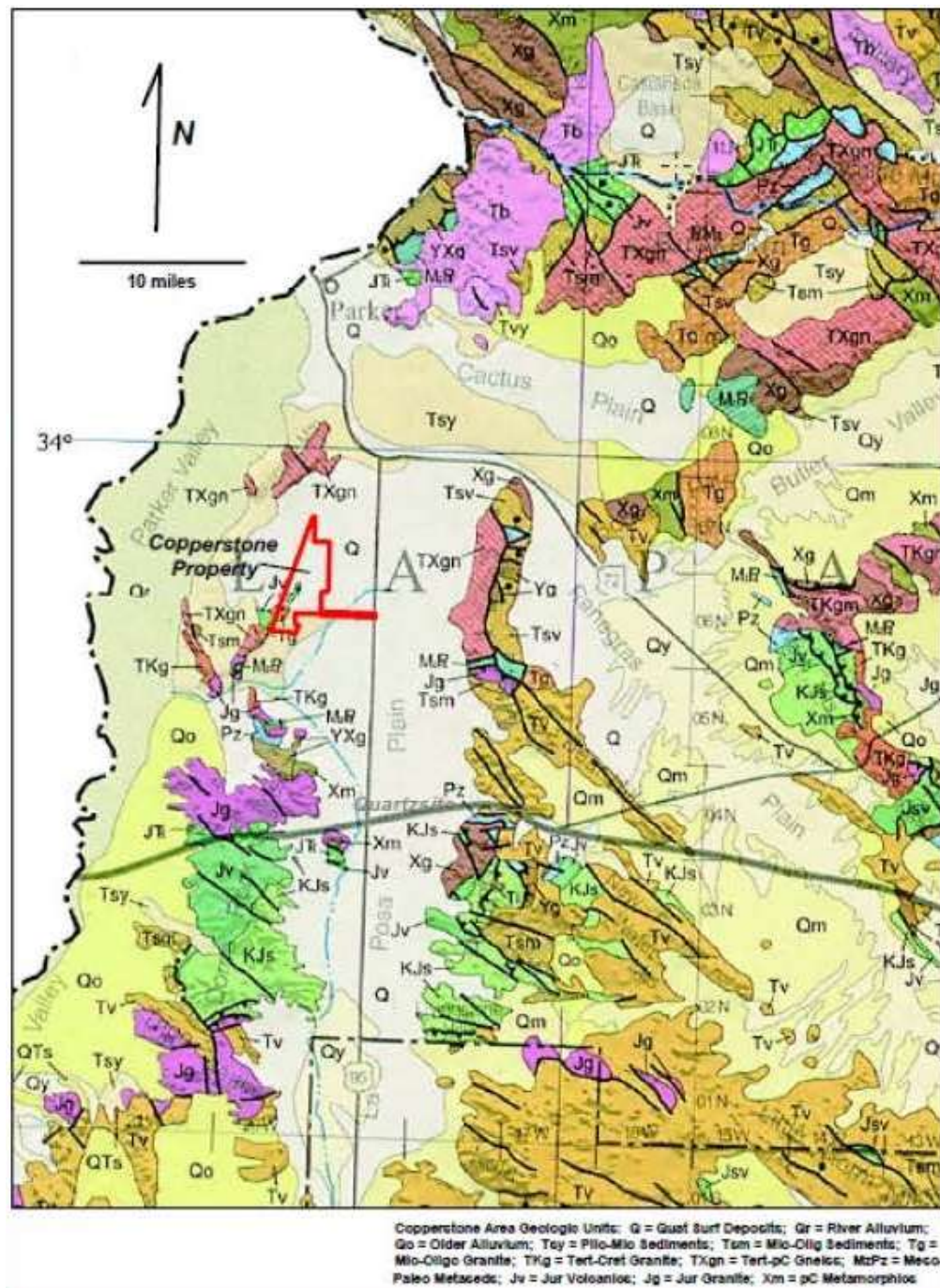


Figure 7-1 Regional Geologic Setting of the Copperstone Project (Pawlowski, 2005)

Both Mesozoic thrust faulting and Tertiary detachment faulting are documented in the Moon Mountains and demonstrate that thrust faulting and associated amphibolite facies metamorphism occurred in Late Cretaceous time, based on U-Pb zircon geochronology. The Moon Mountain detachment fault is exposed in the northern Moon Mountains roughly 1.5 miles south of the Copperstone Project area. The Moon Mountain and correlative Copper Peak detachment faults in the northern Moon Mountains constitute the western limit

of the Whipple-Buckskin-Rawhide detachment system at this latitude. Ductile Tertiary fabrics below the Moon Mountain and Copper Peak detachment structures indicate that footwall rocks were exhumed from considerable depths. Granitic rocks from the footwall of the Copper Peak detachment constrain the termination of ductile fabric development to -21 Ma (Knapp, 1989).

The major structure in the southern Moon Mountains is the Mesozoic Valenzuela thrust system, which dips moderately southeast. Movement on the thrust was multi-staged, with apparent evidence of south and north directed phases of movement (Knapp, 1989). Late Cretaceous thrusting of the Valenzuela resulted in Jurassic quartz syenites and Precambrian gneisses/schists overlying deformed Paleozoic sediments metamorphosed to the lower amphibolite facies. Basement-involved thrusting, as evidenced by the Valenzuela system, and crustal thickening in the Maria fold and thrust belt was broadly coeval with arc magmatism associated with subduction of the Farallon Plate in Late Cretaceous time. Thrusting may have been active as early as 80-90 Ma (Reynolds et al., 1986), but appears to have ended by 70 Ma. Arc magmatism made an eastern sweep through the southwestern Cordillera of North America during Cretaceous time, and was probably at the longitude of present-day western Arizona during the Late Cretaceous (Coney and Reynolds, 1977). On a local scale, thrusting does not appear to have been controlled by thermal anomalies due to arc magmatism. Much of the magmatic activity in the Maria fold and thrust belt appears to be syn- to post-kinematic and may represent a response to crustal thickening.

Igneous rocks of the Moon Mountains are representative of Jurassic, Late Cretaceous, and mid-Tertiary magmatism, and help to constrain the timing and significance of events in the crustal evolution of the region. Significantly, the locus of deformation does not appear to be strongly controlled by the presence of magmatism in the Maria fold and thrust belt. Per Knapp (1989), magmatism is better interpreted as a post-tectonic response to crustal thickening. The presence of magmatic activity during the early stages of crustal extension implies that heat was being introduced to the crust during this time and may have been integrally related to the cause of extension.

7.2 Local and Property Geology

In the vicinity of the Copperstone Project, the Moon Mountain detachment fault carries sedimentary and volcanic rocks of Paleozoic, Mesozoic, and Tertiary age over a ductilely deformed footwall consisting primarily of granitic intrusive rocks. The top of the granitic lower plate rocks is marked by the brecciated Copper Peak granite, which is exposed over an area of roughly 2 km² surrounding and to the south of Copper Peak, in the northeastern part of the Moon Mountains. The northern margin of this unit is truncated by the Moon Mountain detachment fault. A weakly to strongly developed tectonic fabric is present over much of the exposed extent of the granite and is characterized by flattened and stretched quartz grains and deformed potassium feldspar (Knapp, 1989).

The age of the Copper Peak granite is inferred by Knapp to be younger than Jurassic, based on intrusive relations with the Jurassic quartz porphyry, and Tertiary, based on compositional and textural similarity to other biotite-bearing granites in the Moon Mountain area. A small stock of porphyritic biotite granite intrudes the Copper Peak granite and is differentiated from it by higher biotite content and consequent dark color. This granite carries a locally developed foliation consisting of flattened quartz grains, but the rock is generally

not foliated, and is interpreted by Knapp to be late- to post- tectonic with respect to the development of mylonites in the footwall of the Moon Mountain detachment fault.

The hanging wall rocks are made up of metamorphosed Jurassic quartz latite porphyry (138 to 205 m.y., Spencer et al., 1988) as fault bounded slivers. The quartz latite porphyry is intruded by the Tertiary granitic lower plate rocks (Knapp, 1989). Limited outcrops of Jurassic quartz latite porphyry along the hanging wall of the Moon Mountain detachment fault were recorded by Knapp (1989), who infers that this unit overlies the metasediments. The sedimentary rocks and sediments that form the Moon Mountains range in age from Precambrian to recent and are represented by carbonate, quartzite, conglomerate, sandstone, and unconsolidated gravel that occurs in the Copper Peak area as small outcrops in the hanging wall of the Moon Mountain detachment fault, where they are variably faulted, tilted, and brecciated (Knapp, 1989).

7.2.1 Lithology

The primary lithologic units within the Copperstone Project area are Precambrian to Tertiary amphibolite metasediments, volcanics, and granitic intrusive rocks, with lesser amounts of sedimentary and volcanic supracrustal lithologies. Sheared and brecciated granite along the plane of the low-angle **structural zone** separates the lower plate mid-Tertiary granitic rocks from upper plate rocks, which **consist of** (from bottom to top) of Triassic phyllites and metasediments, Jurassic quartz latite porphyry, Miocene sediments and olivine basalt. The basal unit encountered is described as a chlorite phyllite to calcareous chlorite phyllite, with a maximum known thickness of up to 75 to 90 m (Salem, 1993). At the Copperstone mine, only upper plate rocks are exposed (Figure 7-2).

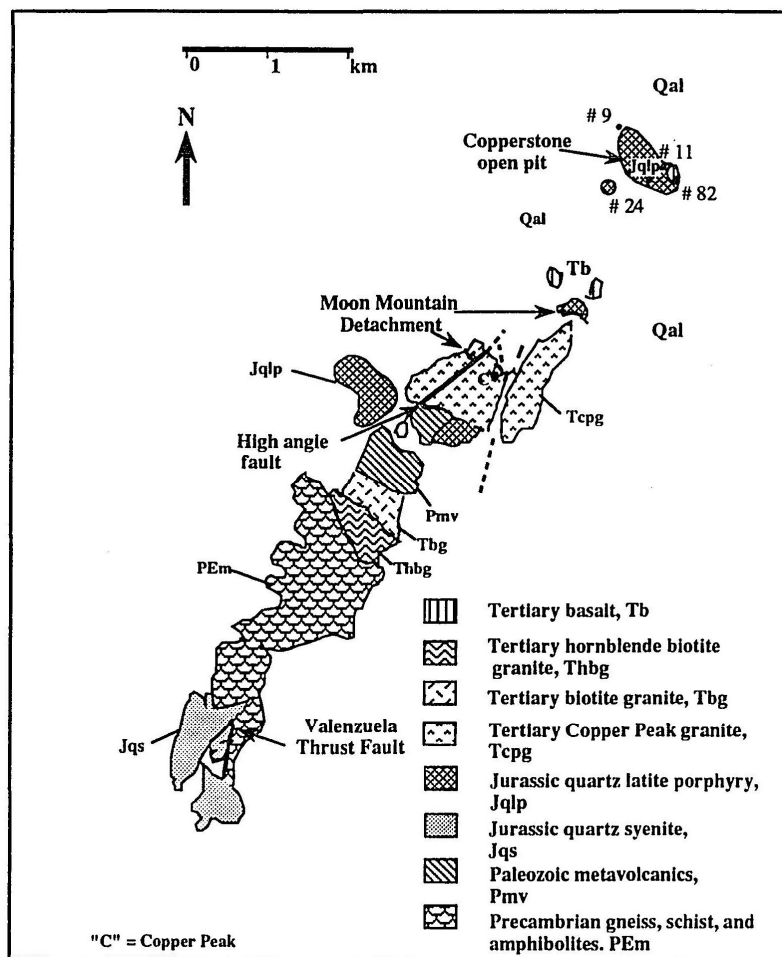


Figure 7-2 Geologic Map of the Copperstone Project (Salem, 1983)

The following detailed lithologic descriptions are modified from Salem (1993):

7.2.1.1 Triassic Phyllite

This rock is correlated with the lower unit of the Triassic Buckskin Formation of the Rawhide-Buckskin Mountains (Reynolds and Spencer, 1989). This unit is exposed only at the bottoms of drillholes on the northwest side of the pit underneath the Triassic chlorite schist, quartzite, and marble. Its thickness from drillhole data is 250 - 295 ft, but its lower contacts are unknown. The phyllite is fine-grained quartz and chlorite with elongate oriented crystals or narrow aggregated lenses. Feldspar also occurs as porphyroblasts, but mainly of plagioclase and less orthoclase, with variations in size from 0.008 to 0.047 in. Chlorite is well developed in this rock as parallel flakes and sometimes as pods that segregate to green (chlorite) and white (quartz and feldspar) bands. Two types of chlorite occur in this unit, clinocllore and pennine. The characteristic textural features in this unit are the ellipsoidal or lensoid shape of quartz and feldspar in a fine-grained matrix that is brecciated and sheared to an augen texture. Pseudomorphs of hematite after pyrite are well identified in this rock as disseminated fine cubic grains, ranging in size from 0.002 - 0.008 in. Phyllite rock is replaced in part by carbonates.

7.2.1.2 Triassic Metasediments

Triassic metasedimentary rocks are interlayered units of marble, quartzite, and chlorite schist that attain a maximum known thickness of up to 108 ft.

Marble consists predominantly of calcite with minor amounts of siderite-ankerite. Calcite ranges from 0.002 - 0.016 in and is arranged in an equigranular granoblastic mosaic. Grains have irregular margins and tend to form interlocking to complexly sutured aggregates. Calcite shows twin lamellae and rhombohedral cleavage in thin sections. The grain size varies from very fine-grained to coarse-grained with anhedral to subhedral form and high birefringence. Quartz occurs in the matrix as very fine to fine grains and as quartz veinlets. Along the contact between quartz veinlets and calcite is recrystallized coarse-grained calcite. Quartz-specularite veins are also well represented in this rock as replacements; specularite is crystalline to lath-like, mostly altered to earthy hematite. Brecciated carbonate rocks display open-space fillings by quartz-specularite, along with hematite and limonite.

Quartzite is predominantly composed of quartz with minor minerals such as biotite and chlorite. Quartz grains range in size from 0.23 to 0.024 in, are equigranular, granoblastic, and only slightly interlocking with equant grains. Some grains are coarser, anhedral, or as clusters that are scattered through a fine-grained matrix. Quartzose schists interbedded in the chlorite schist have a distinct foliation and parallelism marked by thin parallel films of sericite, crystalline muscovite, biotite, or chlorite. Chlorite forms individual flakes, and also appears in platy, radial aggregates. Undulatory extinction that indicates cataclastic effects is also found. This rock is locally replaced by quartz-specularite mineralization. The specularite is crystalline, with lath-like form, and ranges in size from 0.001 to 0.005 in. Reynolds and Spencer (1989) break out a Quartzite Member of the Buckskin Formation that appears to correlate well with this unit at Copperstone.

The chlorite schist is composed of chlorite as the principal micaceous mineral with quartz. Muscovite and biotite are also present up to a maximum of 2%. The rock is made up of segregated bands formed by chlorite, magnetite, and quartz. Chlorite is green, slightly pleochroic from yellowish green to green, as flakes, and with a greenish brown birefringence. The species clinochlore, a magnesium-rich chlorite, is identified by X-ray diffraction along with anomalous birefringence and positive sign of elongation. Quartz is anhedral, granular, ranges in size from 0.002 to 0.008 in, and has wavy extinction. Zircon and magnetite occur as accessory minerals. Zircon is characterized by its high relief, sub-rounded form, and high birefringence: it ranges in size from .007 to 0.05 in and is altered along grain borders to hematite. Biotite is partially to completely altered to chlorite. Small-scale folding of the original schistosity in which the micaceous minerals were aligned has produced a crenulation cleavage formed by chlorite-muscovite-biotite minerals. Carbonate-veinlets and quartz specularite occur as replacement and open-space fillings, and earthy hematite has replaced specularite. This chlorite schist is correlated with the lower member chlorite schist of the Triassic Buckskin Formation in the Planet-Swansea area (Reynolds and Spencer, 1989).

7.2.1.3 Jurassic Quartz Latite Porphyry

Quartz latite occurs as a series of weakly metamorphosed flows that vary structurally from massive to laminated. This unit represents the main host rock to the gold of the Copperstone deposit especially where it is brecciated along fault planes and shear zones. Microscopically, the quartz latites are holocrystalline rocks

with porphyritic or seriate textures. Quartz, K-feldspar, plagioclase, biotite, and magnetite are the phenocryst phases, commonly in glomero-porphyritic texture. Quartz varies in size from 0.04 – 0.40 in, is granular, and shows wavy extinction. Embayed quartz is also found, which indicates a magmatic origin. Very fine-grained quartz, sericite, and hematite partly to completely replace the quartz. Orthoclase and microcline phenocrysts range in size from 0.02 – 0.31 in.

7.2.1.4 Miocene Sediments

Spencer et al. (1988) indicates the presence of monolithologic sedimentary breccias derived from the Jurassic quartz latite porphyry adjacent to Copper Peak: such breccias also occur at Copperstone. These terrigenous rocks were associated with subaerial basin deposition during the development of the regional detachment system of west-central Arizona and southeastern California. Tertiary age is inferred from lithology and clast composition and comparison with sediments of known Tertiary age (Spencer et al., 1988; Knapp, 1989). The breccias are composed of angular to subangular rock fragments varying in size from millimeter up to cobble sizes, set in a normally subordinate matrix made up of smaller rock pieces, mineral fragments, and powder. Strong hematitization that occurs as earthy hematite along fractures and as open-space fillings as well as hematite-carbonate mineralization is well displayed. The breccia represents one of the main host rocks to gold deposition at Copperstone as open space fillings along the hanging wall of the Copperstone listric fault.

7.2.1.5 Olivine Basalt

Tertiary basalt is dark reddish-brown to black in color. Its thickness reported from the drillholes is up to 490 ft, and it is mainly restricted to the southeastern part of the mine, extending to the northeastern part of the Copper Peak area in the Moon Mountains. Knapp (1989) could not find a relationship between the basalt and the surrounding bedrock, but according to Cyprus' data from drilling, basalt is in contact with Jurassic quartz latite porphyry and its derived breccias. The proposed age by Knapp for the basalt is late Oligocene to mid Miocene. These lavas have a range in composition from 43.2 to 53.4% SiO₂, and they consequently exhibit variable mineralogical characteristics although most specimens are texturally remarkably uniform. The highest percent of silica is in basalts with quartz cavity fillings. The rocks are hypo- to holocrystalline fine-grained rocks with intersertal or intergranular textures. The dominant phenocryst phases are plagioclase, olivine, clinopyroxene, hornblende, and magnetite. Euhedral to subhedral zoned plagioclase laths form an interlocking framework packed with granular olivine, augite, and magnetite. The plagioclase phenocrysts range in size from 0.012 – 0.024 in and show lamellar twinning. Augite commonly occurs as discrete subhedral short prisms or granules that range from 0.001 – 0.012 in. with slight pleochroism from colorless to greenish. The basalts in the mine occur with varying degrees of oxidation and alteration which give a reddish color to the rocks in hand specimen.

7.2.2 Mineralization

Gold mineralization in the Copperstone Project has historically been produced from the Copperstone fault. The gold mineralization described in this report has been found to be restricted to the immediate vicinity of the Copperstone fault (also referred to as the Copperstone shear or the Copperstone structure), a moderate to shallow, northeast-dipping, semi-planar zone of shear and brecciation. interpreted by previous authors as a listric splay in the hanging wall of the Moon Mountain detachment fault, which has not been penetrated in drilling to date.

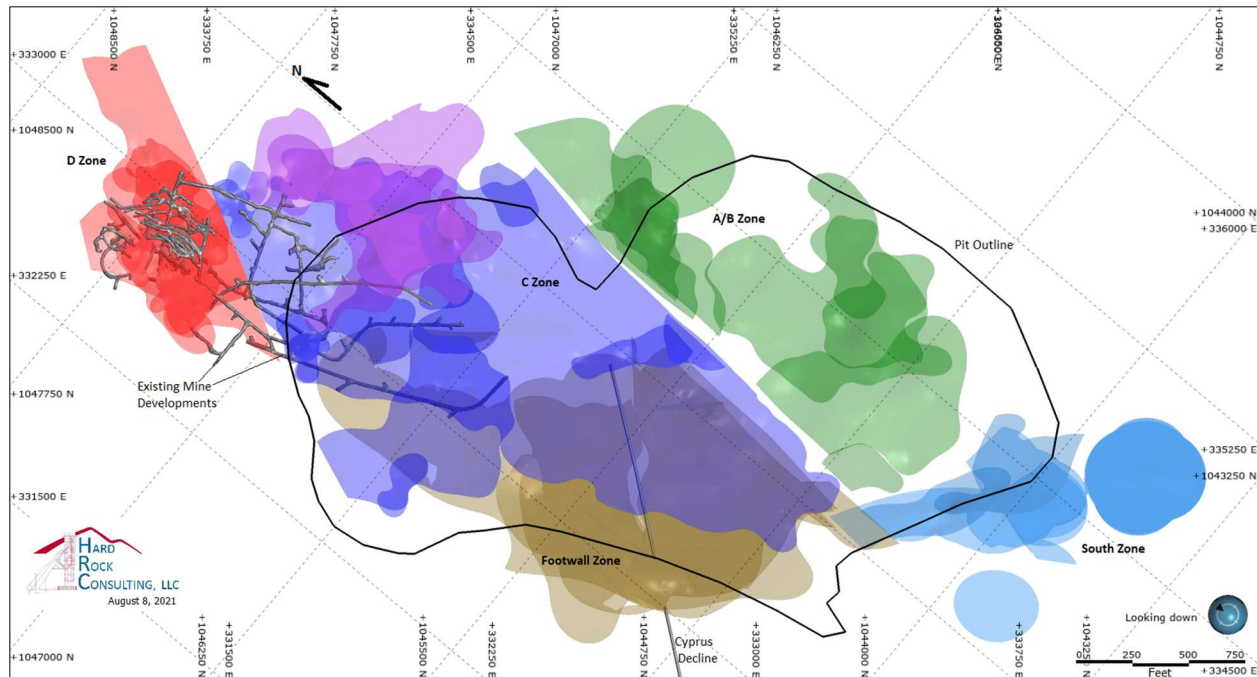
Segerstrom interprets the Copperstone fault as a multistage structure that accommodated both late Cretaceous compressional (thrust) as well as mid-Tertiary extensional (detachment) movement. Note that Sillitoe (2025) evaluated the Copperstone Project and interpreted the Copperstone Fault and associated structures to be thrust faults. In addition, Sillitoe states that the gold mineralization is primarily related to the Cretaceous thrusting, and not the later detachment faulting.

The Copperstone fault strikes about N30° to 60°W and dips from 20° to 50° to the northeast. The associated brecciated fault zone ranges from 45 ft to 185 ft in width with characteristic fault gouge, multi-phase breccia textures, shear fabric, and intense fracture sets across this width (MDA, 2000).

Gold mineralization at Copperstone has been hypothesized to be in the hanging wall of the Moon Mountain detachment fault, which has not been penetrated in drilling to date. The current authors have not seen evidence that supports the interpretation that the Copperstone fault is related to the Moon Mountain detachment fault.

The Copperstone fault presents strong evidence of shearing, with schistose textures and fault gouge. Brecciation is observable in the open pit, as are steeply northeast dipping, northwest-striking fractures and narrow shear zones. Mineralization is known to occur in association with both the primary Copperstone fault zone as well as high-angle, secondary fault structures. All mineralization appears to be cut off at the southeastern edge of the pit by a northeast-striking fault that dips to the southeast. Most of the fractures in the volcanic sequence are highly irregular and discontinuous, but the Copperstone fault zone has remained a dependable target for exploration and mining.

Segerstrom's conceptual geologic model interprets the Copperstone structure as part of a multistage, low-angle fault system related to late Cretaceous compression and subsequent overprinting by mid-Miocene extension (seen in Figure 7-4). This Copperstone fault system is presently considered the primary control/conduit for mineralization at the Copperstone Project. The mineralized Copperstone fault is continuously present across the pit area from the A, B and C zones and may extend even farther south across the sparsely drilled South zone. It appears to break down or splay upon entering the D zone and there are indications of some up-dip flattening in the northern C zone. The distribution of mineralization within the Project area, as represented by the A, B, C, and D zones, is shown in plan view on Figure 7-3.



Source: Hard Rock Consulting, 2021

Figure 7-3 Mineralized Zones within the Copperstone Project Area

7.2.2.1 A, B, and C Zones

Mineralization in the A, B, and C zones occurs along the primary Copperstone fault as well as secondary sub-parallel structures within the zone of shear. In addition, underground mapping has shown a number of steeper northwest-trending faults and fractures that localize alteration and mineralization in and around quartz-Fe oxide+/-Cu oxide veins. Observations show that where such high-angle structures intersect the low-angle (Copperstone fault) structures, a favorable site is prepared. Where the Copperstone structure is disrupted, a dilatant zone may occur, resulting in higher grade and thickness of the gold mineralization.

Similarly, there are northeast-trending faults which appear to offset and localize mineralization, resulting in mineralized shoots at intersections with the Copperstone fault. These northeast-trending faults occur as thin to wide crushed zones that may capture a drill string resulting in exaggerated mineralization widths. This can especially be the case where the drill pattern is based on a southwest drillhole orientation. Long intervals of elevated gold values in quartz latite, without continuity to nearby holes, suggest such high-angle controls.

7.2.2.2 D Zone

The D zone contains large imbricate slices of interbedded limestone and sandstone, of which the limestones have been largely replaced by specularite, earthy hematite and silica. In many drillholes, silica-magnetite-specularite-chlorite replacement bodies occur in two limestone layers of variable thickness, but generally no more than 5-10 ft. In some locations iron oxides form a matrix in silicified limestone but nearby there may be evidence for direct replacement of limestone by iron oxides. It is possible that some of the silicified

limestone is actually a pure white quartzite that has been brecciated. This would mean that silicification does not precede iron-oxide introduction.

Elevated gold grades are associated with limestone replacement bodies over areas of significant size, likely due to the extreme distortion and reactivity of the limestone. The slices of this sedimentary package have dimensions of up to tens to hundreds of feet in strike length and tens of feet of thickness. The imbricate slices are conformable to the Copperstone shear, having been caught up in the shearing with local rotation, tension gashes and associated deformation. A component of compressional deformation within this shear is evidenced by drag folding seen in Figure 7-4. This sedimentary package is located between hanging wall volcanic rocks and footwall phyllite. These sedimentary rocks are absent to the southeast in the Copperstone C, B and A zones.

The upper and lower contacts of the limestone replacement bodies are almost always tectonic. The upper contact with the volcanic rocks is also mostly tectonic but, in some locations, there is no evidence of shearing. The age of the limestone is also in doubt. It has been assumed as Triassic previously, but some are now suggesting that it is Permian in age and related to the Kaibab Limestone on the Colorado Plateau to the north. The interbedded nature of limestone with quartzite supports this idea, but the relationship to the adjacent phyllites is problematic.

Immediately north of the D zone is a northeast-trending fault zone which apparently terminates the Copperstone fault and associated secondary structures. Whether the Copperstone fault is offset or actually terminated by this fault is presently unknown, but it appears that the sedimentary sequence thickens considerably and may not be penetrated by the Copperstone fault.



Figure 7-4 Underground exposure of the D Zone showing drag fold within the Copperstone Structure

7.2.3 Alteration

7.2.3.1 Chlorite Alteration

A 2004 petrographic report by Larson (2004) shows a strong association of gold with chlorite. In the D zone, there are significant streaks and blebs of dark green chlorite that blend in with the dark-colored iron oxide masses. It is reported that these chlorites are iron-rich. The current explanation is that the chlorite forms later in a cross-cutting phase that includes gold, and its composition is influenced by the amount of iron in the pre-existing rock. Larson (2004) further reports that chlorite occurs in other settings at Copperstone, such as in the chlorite schist and phyllite, where it is not associated with gold. Other forms of chlorite have also been noted as low-grade alteration on fractures in the volcanics and as thin selvages along some quartz veins in the volcanics that are not associated with significant gold mineralization.

7.2.3.2 Magnetite Mineralization

Magnetite shows some correlation with gold grades in the D zone. As a visual guide to mineralization, the presence of magnetite has worked in logging, but there are significant deviations. Hole KER-17U-77 intersected a magnetite replacement body that had depleted gold and copper values, well below the average waste rocks at Copperstone. A more detailed review showed that two samples, while still silicified contained a significant degree of remnant calcium carbonate. It also contained more manganese oxides than normal and even had (relatively) anomalous values of some epithermal elements such as thallium. Preliminary conclusions are that, while magnetite may be favorable for the deposition of gold, it is not a sufficient condition. It is assumed to be early in the sequence of events but is not necessarily directly related to gold.

7.2.3.3 Copper Oxide Mineralization

Copper mineralization has been intersected within both the hanging wall and footwall zones of Copperstone. The majority of copper mineralization intersected by drilling is of oxide state and is understood to be primary. Oxide copper occurrence in drilling intercepts ranges from moderately coarse vein fill within structural zones to residual staining along thin margins. Predominant copper mineral occurrence is identified as malachite and chrysocolla. Intercepts containing native copper do occur at Copperstone as infill within veins and within dilational shears. Sulfide copper mineralization such as bornite and chalcopyrite are found at Copperstone but in lower occurrence relative to both oxide and native speciation. Recent metallurgical testing of historic samples from B zone materials did identify a small area containing anomalous concentration of sulfidic copper. Copper assay data was not typically collected during historic drilling and sampling programs. Kerr incorporated copper assays into the 2017 exploration program and planned to continue collecting copper assay data during future drilling and sampling.

A direct association of gold with copper is not established even though both occur in economic concentrations. Virtually every dataset shows no correlation between the two metals. While they may occur within the same vicinity of a drillhole, they seldom occur in significant grades in a single body.

The geochemistry of the Copperstone deposit may be amenable to a factor analysis treatment to discover elemental associations that are not visible to the casual eye. As more multi-element data is obtained, this should be carried out on a domainial basis by the recognized zones.

8 DEPOSIT TYPES

The Copperstone deposit is presently best described as a Cretaceous and mid-Tertiary, multistage, low-angle fault-hosted, epithermal gold deposit.

Salem (1993) suggests that the Copperstone deposit might be further classified as a new sub-set of volcanic-hosted epithermal precious-metal deposits, postulating that Copperstone was created during a late stage of detachment faulting, and that localization of gold deposition was controlled by boiling. Gold deposition was, according to Mahmoud's well-presented interpretation, related to the circulation of brine fluids driven by hot mid-Tertiary granitic lower plate rocks that may have also contributed water and or metals, causing ascending brines to move along the N- to NE- dipping Moon Mountain detachment fault at the Copper Peak area to the south. The fluids continued to ascend along the Copperstone listric fault and a series of high angle NE and NW faults that crosscut the Copperstone listric fault. This structure acted as conduit that was kept open by ongoing faulting without deposition until the fluids reached the boiling stage, perhaps as a result of decompression. The boiling fluid mixed with another less saline fluid within the mineralized horizon, a mixing that led to the precipitation of gold mainly along the brecciated hanging wall and footwall of the Copperstone listric fault, as open space-fillings and replacement mineralization (Salem, 1993).

Segerstrom, an author of this report, and previously Sillitoe (2025), did not see the evidence needed to support Salem's (1993) suggestion that the Copperstone mineralization be classified as a new sub-set of volcanic-hosted epithermal precious-metal deposits.

At Copperstone, alteration associated with gold mineralization is mainly chloritization, silicification, and potassic alteration (Salem, 1993). Hypogene mineralization can be divided into 3 paragenetic stages: early amethyst-quartz-chlorite-specularite-hematite-AuO; late fine-grained euhedral quartz adularia-chrysocolla ± malachite ± magnetite ± chalcopyrite-pink fluorite-barite-ankerite-calcite-AuO; and barren quartz-pale green fluorite-barite-hematite. Gold occurs as free particles or as encapsulations in amethyst and late fine-grained quartz. Electron microprobe microscopy has revealed the composition of chlorites associated with the gold deposits in quartz-chlorite-specularite-Au veins as iron rich chlorite with Fe:Mg ratios of 5:1, along with the presence of Cu incorporated in its structure. Fluid inclusion studies of amethyst, late fine-grained quartz, and fluorite indicate that gold was deposited at an average temperature of 290°C from boiling fluids of apparent salinities ranging from 11.7 to 19.9 wt% NaCl equivalent, ending with 190°C and 25.5 wt% NaCl equivalent (Salem, 1993).

9 EXPLORATION

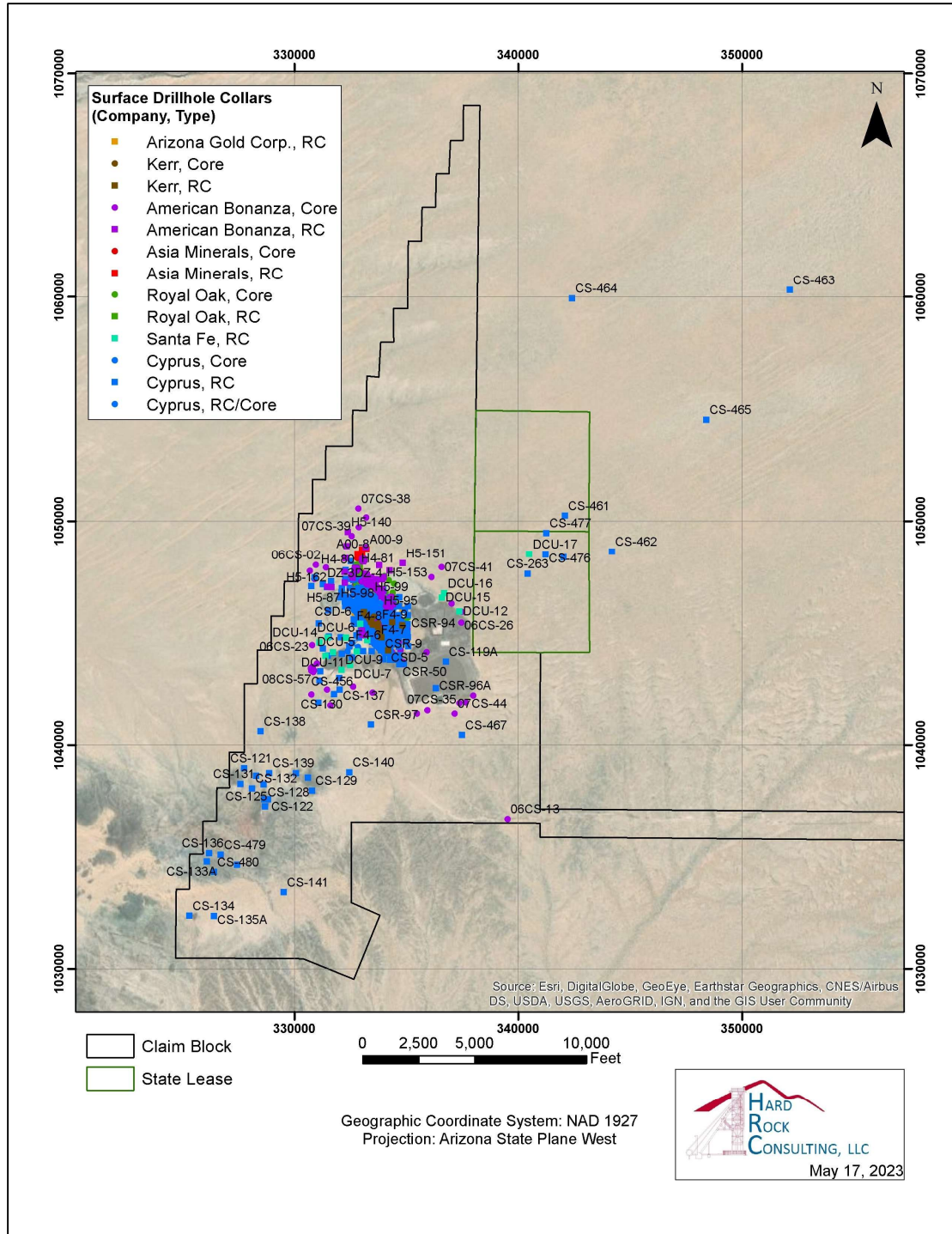
9.1 Exploration Conducted on behalf of MAI

To date, no surface exploration has been conducted at the Copperstone Project by, or on behalf of, Mining Americas Inc. Historical exploration completed by previous owners and operators is described in detail in Section 6 of this report.

10 DRILLING

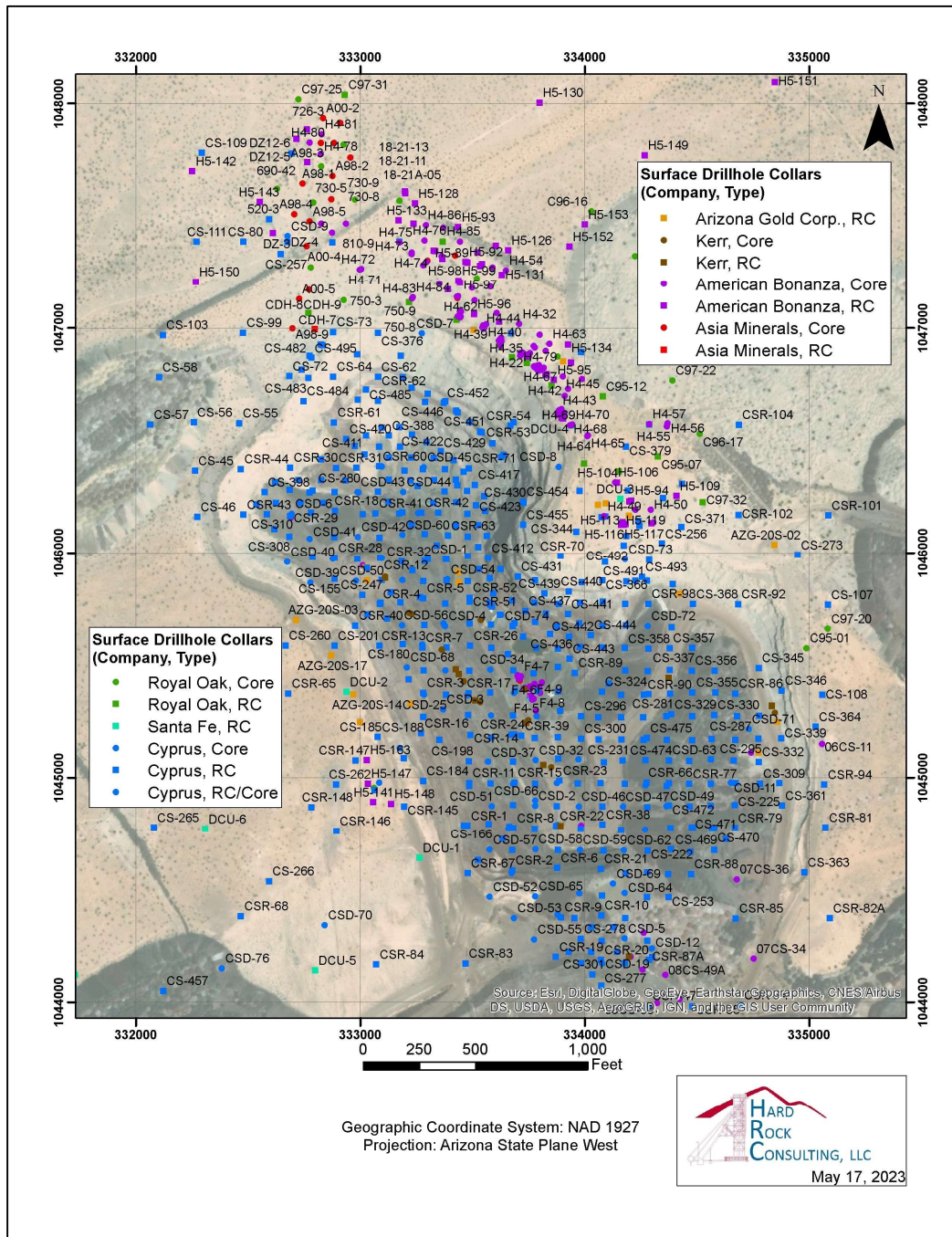
To date, no drilling for the purpose of exploration has yet been carried out at the Copperstone Project by, or on behalf of Mining Americas Inc. Historical drilling exploration, which was carried out under previous owners and operators, is discussed in detail in Section 6 of this report.

Figures 10-1 through 10-3 show drillhole collar locations for all drilling programs except Newmont and Southwest Silver. Figure 10-1 shows surface drillhole collar locations for the entire property. Figure 10-2 shows surface drillhole collar locations for drilling in the area of the open pit. Figure 10-3 shows underground drillhole collar locations. A complete list of drillhole collar locations, orientations, type, year, and operator are available in Appendix B - Section 9 Drill Hole Collar Locations.



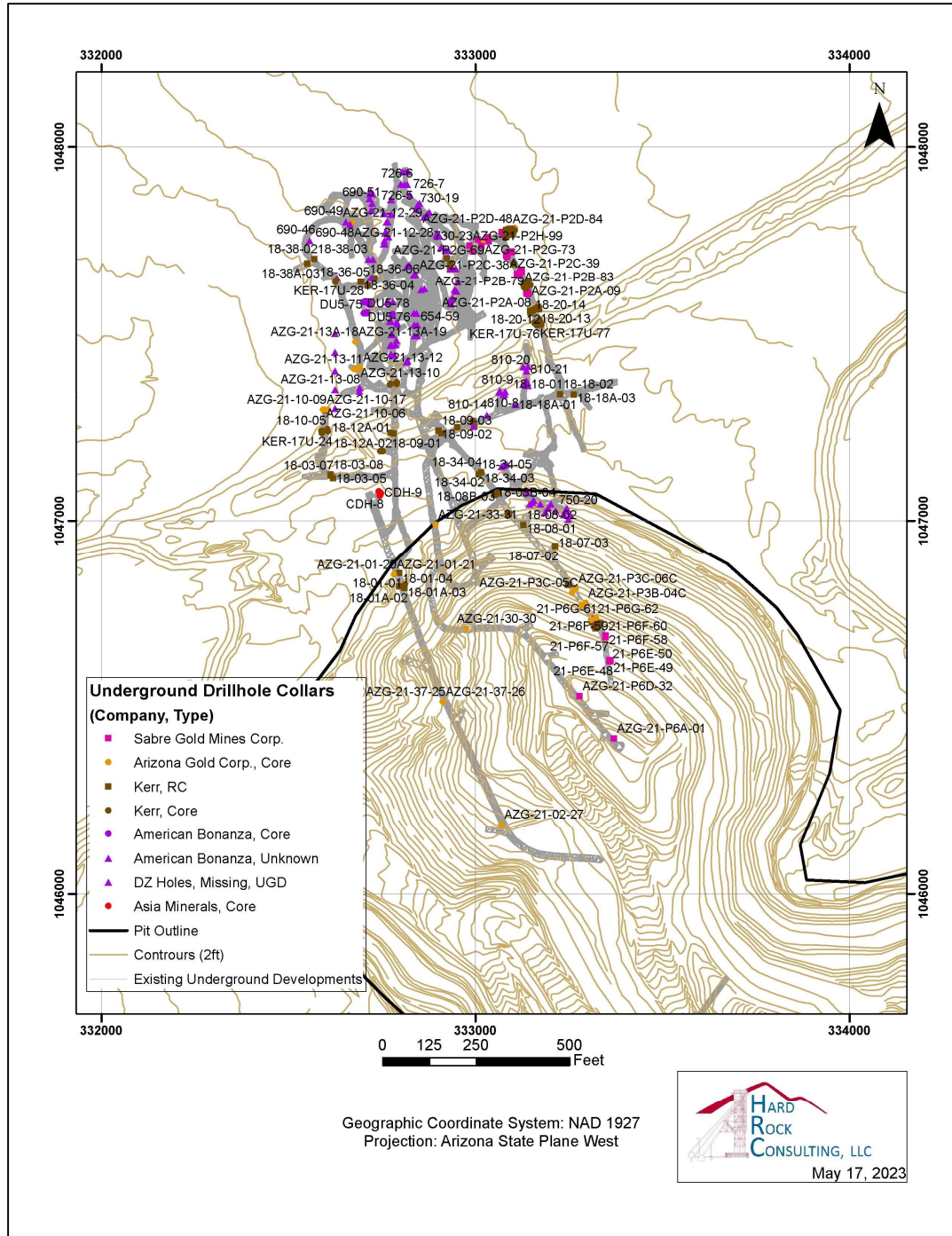
Source: Hard Rock Consulting, 2023

Figure 10-1 Plan View of Surface Drilling for the Property



Source: Hard Rock Consulting, 2023

Figure 10-2 Surface Drillhole Locations in the Area of the Open Pit



Source: Hard Rock Consulting, 2023

Figure 10-3 Underground Drillhole Locations

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

To date, no sampling has been carried out at the Copperstone Project by, or on behalf of Mining Americas Inc. Historical sampling, which was carried out under previous owners and operators, is discussed in detail in Section 6 of this report.

11.1 Opinion on Adequacy of Historical Sample Preparation, Analysis and Security

No drillhole data exists for the Newmont and Southwest Silver drilling. Detailed sampling procedures, QA/QC, and sample security are not available for the drilling completed by Cyprus, Santa Fe, and Royal Oak. However, the re-assay of Cyprus pulps in 2006, and duplicate quarter core sampling in 2011 verify the assay results. The QP finds the samples from Cyprus, Santa Fe, and Royal Oak suitable for use in mineral resource estimation. Documentation detailing sampling methods, QA/QC procedure, and sample security is summarized in Section 6 of this report for Asia Minerals and American Bonanza. Other than the production drilling completed by American Bonanza in 2012 and 2013, the samples are suitable for inclusion into the mineral resource estimate. Drillhole samples collected post-2015 (Kerr Mining, Arizona Gold Corp., Sabre Gold Mines) have sufficient associated supporting documentation for a meaningful evaluation of accuracy and reliability and are suitable for inclusion into the mineral resource estimate.

12 DATA VERIFICATION

12.1 Site Investigation

Lawrence Segerstrom, QP and co-author of this report, accompanied by Miguel Cardona, VP Exploration for Mining Americas, carried out an on-site inspection of the Copperstone Project for 3 days, January 14th through 16th. 2025.

While on site, Mr. Segerstrom focused on an inspection of surface and underground bedrock exposures, checked drillhole collar locations, and examined the on-site core and RC chips processing and storage facilities. Segerstrom also reviewed with on-site staff geologists details of the geology, as well as the geologic model.

Field observations during the site visit confirm previous reports on the geology of the Project area in terms of bedrock lithologies and alteration types (as described in Section 7 of this report). In the opinion of Mr. Segerstrom, the differences between the drillhole collar locations measured during the visit and the corresponding locations in the database are acceptable for the work described in this technical report See Table 12-1: Field Audit of Selected Collar Locations and Appendix C - Section 12 Drill Collar Audit.

Table 12-1 Field Audit Of Selected Collar Locations

HOLE_ID	COORDINATES IN DATABASE		ACTUAL FIELD COORDINATES		DIFFERENCE	
	Y	X	Y	X	Y	X
H5-135	333,906.00	1,046,597.31	333,915.67	1,046,603.84	9.67	6.53
KER-20S-03	332,715.31	1,045,699.55	332,716.49	1,045,712.16	1.18	12.61
KER-20S-14	332,970.33	1,045,368.18	332,971.46	1,045,369.82	1.14	1.64
KER-20S-18	333,507.49	1,046,989.96	333,512.50	1,046,987.72	5.01	-2.24

NOTE: COORDINATES SPCS ARIZONA WEST NAD 27

However, as stated above, Mr. Segerstrom, an author of this Technical Report, differs with previous project reports on the interpretation of the mineralized structural zone, preferring to call it a low-angle fault or shear zone rather than a detachment fault (further discussed in Section 7 of the Report. Observation of an underground exposure of the main mineralized structure, suggests that there was an earlier thrust component to this low-angle mineralized zone, which in places was superimposed by a low-angle normal component.

12.2 Database Audit

To verify the integrity of the project's assay database for the purposes of Mineral Resource Estimation, an independent, ground-up data verification audit was conducted by the QP. The objective was to cross-reference the digital assay database against primary, unmodified laboratory certificates and historical logs to identify potential transcription, unit conversion, or systemic data-entry errors.

Verification was conducted in strict accordance with comprehensive Standard Operating Procedures (SOPs) for geological data handling. A randomized, multi-generational subset of the database was selected to ensure continuous data quality across different operators and laboratory facilities spanning from 1988 to 2022. Primary source documents utilized in this audit included:

- **Modern Digital PDF Certificates (2018–2022):** ALS Global (formerly ALS Minerals) and American Assay Laboratories (AAL).
- **Historical Laboratory Excel Logs (2003–2005):** CUDH, DU, and H-series drilling campaigns.
- **Legacy Paper Certificates (1988–1996):** Inspectorate America Corporation and Geo-Monitor Inc.

Internal Database Review and Preliminary Corrections

Prior to the external audit, an internal preliminary mineral resource evaluation identified several structural inconsistencies within the master assay file.

The internal review noted that the median ratio between the database's gpt and opt fields was 34.28. Dividing this median value by 31.10 (grams per troy ounce) yields 1.10, which corresponds to the ratio between a metric long ton and an imperial short ton. This confirmed that the database's opt assays were recorded in imperial units, whereas the gpt assays were mathematically stored as grams per metric long ton. To ensure compatibility with imperial-system block-model densities and tonnages, the internal team generated normalized grade fields (AUOPT and AUGPT_S) specifically for mine planning.

Furthermore, the internal review flagged anomalous, repeated assay values of 0.034 g/t and 0.103 g/t across multiple drillholes. For the preliminary estimate, these corrupted 0.103 g/t values were temporarily replaced with "missing" (null) data to prevent the artificial generation of low-grade volumes in the block model. The internal team recommended a full review of the database before undertaking definitive resource estimation work.

Independent Verification Results & Statistical Summary

Following the internal review, the QP conducted a definitive verification audit. The master assay database contains a total of 91,092 sampled intervals. The audit successfully extracted and verified **4,871 intervals**, representing **5.35%** of the total database. This surpasses the generally accepted minimum industry standard of a 5% spot check.

Upon initial review, the raw database exhibited a 75.8% exact match rate. However, a detailed review of the discrepancies revealed that the majority of variances were not errors, but rather the result of acceptable, systematic QA/QC protocols applied during historical database compilation. When accounting for these acceptable procedural variances, the historical master database demonstrated an **adjusted accuracy rate of 95.78%**.

Procedural Variances (Acceptable)

The audit identified two primary categories of acceptable variance between the primary certificates and the master database:

1. **High-Grade Averaging & Overlimits:** In highly mineralized intervals, primary ICP results frequently triggered laboratory overlimits. The database correctly prioritized and superceded these values with secondary Gravimetric or Metallic Screen Fire Assays. Furthermore, where multiple gravimetric re-runs were conducted on a single high-grade pulp due to nugget effect, the database accurately recorded the mathematically weighted average of the runs rather than the single primary column value.
2. **Below Detection Limit (BDL) Standardization:** Historical laboratories utilized varied formatting for non-detects (e.g., <0.001, <5 ppb, or -0.005 oz/t). The database managers systematically standardized these entries to positive hardcoded limits (e.g., 0.001 g/t or 0.005 g/t), ensuring clean numeric processing for block modeling.

Identification and Remediation of Database Errors

By isolating the true errors from the procedural variances, the QP identified two specific, remediable issues within the dataset:

- **Transcription and Interval Shift Errors:** In isolated historical instances (e.g., hole AZG-21-P2A-06), data-entry inadvertently copied internal laboratory QA/QC rows (duplicate pulp runs and blanks) from the certificates and pasted them directly into the depth-interval columns. This caused a downward spatial shift, misaligning true gold grades with their corresponding geological intervals.
- **Resolution of the Export Corruption:** The QP's audit identified the root cause of the 0.103 g/t anomalies flagged by the internal team. A systemic mapping error during a recent (April 2024) database export process had globally overwritten hundreds of legitimate high-grade intercepts with a static value of 0.103 ppm.

Remediation: These values were already accounted for by the previous database alteration prior to estimation. A full review of the database is recommended to re-introduce missing high-grade values for future resource efforts.

Qualified Person's Opinion on Data Adequacy

The independent data verification program confirmed that the data collection, sampling, and analytical protocols employed across the various drilling campaigns were executed to industry standards. The errors identified during the internal review and the subsequent database audit were highly isolated, quantifiable, and successfully remediated by the QP. Some high-grade values have been missed in this estimation but will be re-inserted upon a full database review and compilation.

It is the opinion of the QP that the remediated assay database is robust, historically continuous, and fully adequate for the purposes of Mineral Resource Estimation.

12.3 Adequacy of Data

Based on the results of the site investigations by the QP's and data validation efforts, the QP considers MAI's drilling and sampling data, as contained in the current Project database, to be reasonably accurate and suitable for use in estimating mineral resources. Results of the manual audit indicate a minor and acceptable

error rate. The QP found that the records for the Project were organized and available upon request. It is recommended that the company perform a full review of historical data so as to recapture some missing high-grade values that have been lost through transcription.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing programs have been performed for the Copperstone project since 1986, completed by multiple metallurgical laboratories that are listed in this report. The project has changed ownership several times and undergone metallurgical testwork programs at each stage. The mine has also operated 2 times in the past, during the 1980's and 2011 for a period producing gold dore. Three NI 43-101 reports have been issued on the property (2011 FS, 2018 PFS and 2025 PEA) which include details of testwork completed up to 2022 by RDi. This section will summarize testwork completed on the project. The information contained in Sections 13-1 to 13-12 is largely excerpted from (Hardrock, 2025). The reader is encouraged to review the previous technical reports for more information.

13.1 Historical Metallurgy Testwork

Since 1986, several metallurgical studies have been completed to guide selection of an appropriate processing method and to optimize process operating parameters. Samples of various types of mineralized material, at variable grades, from the four main mineralized zones were prepared and sent to various laboratories and testing facilities for metallurgical testing. The laboratories and tests completed are summarized below:

- Hazen Research – 1986 – Cyanide Leaching
- Hazen Research – 1995 – Mineralogy
- Resource Development Inc. – 1999 – Cyanide Leaching
- McClelland Laboratories – 2000 – Cyanidation, Flotation, Gravity
- Echo Bay Minerals – 2001 – Flotation
- McClelland Laboratories – 2005 – Flotation, Gravity, Grinding, Rheology
- CAMP – 2009 – Gravity, Flotation
- Resource Development Inc. – 2018 – Flotation, Gravity, Cyanidation, Acid Leaching, Grinding, Thickening, Mineralogy
- Resource Development Inc – 2019 – Locked Cycle Cyanidation, Thickening, Filtration, Carbon Stripping
- Resource Development Inc. – 2021 – Grinding, Cyanidation, Thickening, Rheology
- Resource Development Inc – 2022 – Leaching, CCD Testing, and Cyanide Destruction Testing

The tests have consisted of cyanidation studies including bottle roll and column leach tests, gravity concentration tests, flotation studies including flotation only and flotation tests followed by cyanidation, particle size optimization, and reagent consumption tests.

13.2 Hazen Research (1986) – Original Open Pit Heap and Agitated Leach Test Work

In 1986, Hazen carried out test work on six-inch diameter core samples of breccia and silicified rock types that formed the basis of the original project pre-feasibility for open pit mining and low-grade oxide heap leaching and agitation leaching. Therefore, the data and mineralogy are not relevant to the new high-grade underground mining and processing project.

The studies were conducted on three composite core samples representing the different mineralized material type lithologies being considered in the deposit: brecciated, silicified, and mixed. The composites assayed 0.09 oz/ton to 0.127 oz/ton, and typically contained negligible silver and total sulfur (0.01%) and about 0.2% total copper. Cyanide consumption was about 0.16 kg/t to 0.39 kg/t. Relatively good cyanidation (column and bottle roll) and flotation responses were obtained in this work.

13.3 Hazen Research (1995)

In 1995, Hazen Research conducted mineralogical investigations on three samples (zones unidentified). Based on the presence of oxide copper minerals concerns were raised for cyanidation of the mineralized material, including higher-than-normal cyanide consumption and difficult cyanide destruction. It was recommended that future process development efforts must consider these factors. Gravity/flotation may prove to be a viable option; however copper extraction may prove difficult, due to the resistance of the copper minerals to flotation. As well, the calcite content raised concerns about high acid consumption, should acid leaching be considered. Overall, the sample mineralogy and conclusions appear to be consistent with subsequent analysis and test work conducted in future studies.

13.4 Resource Development Inc (1999)

Preliminary characterization and cyanidation/leaching studies were conducted in 1999 by Resource Development Inc. (RDI). The two composite samples were selected from core intervals within the “Hanging Wall” and D zone mineralization. The characterization-level testing on the two samples was not considered conclusive. However, it provided insight regarding the level of potential cyanide consumption associated with copper minerals that commonly accompany the gold mineralization at Copperstone. The Hanging Wall sample was from intervals of C zone hole extension east of pit, high grade, volcanic hosted quartz latite porphyry (manganese oxide and specular hematite). The D zone hole was hosted by limestone replaced by silica, iron oxide and with copper oxide mineralization (visible gold, chrysocolla, specularite, hematite, chlorite, silicification).

The 24-hour leach test results indicated that the Hanging Wall sample consumed a minor amount of cyanide (0.004 kg/mt of feed), while cyanide consumption in the D zone sample was significantly higher (1.085 kg/mt). Although both samples contained copper, the D zone sample contained approximately five times more copper than the Hanging Wall sample. The amount of cyanide-soluble copper in the D zone sample was similarly high, suggesting that the presence of soluble copper minerals that locally accompany gold mineralization will consume cyanide and should be studied further. Poor accountability was noted between assayed and calculated gold head which may be due to nugget effects as visible gold was noted in several samples. Possibly metallic assay would have given better assay reconciliations. Table 13-1 shows the summary test results of the cyanide leach testing.

Table 13-1 RDI - 1999 Cyanide Leaching Results

Zone	Lab	Date	Test	Wt Kg	Grade		Ag oz/T	Cu % Total	Parameter		Recovery %			CN Consumption	
					Au oz/T	Au (Calc)			Grind	Time (hr)	Au	Ag	Cu	g/l	kg/t
C-zone HW	RDI	1999	Direct Cyanidation	2.6	0.22	0.33	0.04	0.05	150M	24	83	27	5	1.0	0.004
D-zone	RDI	1999	Direct Cyanidation	9.4	0.77	0.56	0.06	0.24	150M	24	91	49	16	1.0	1.085

13.5 McClelland Laboratories (2000)

In 2000, McClelland Laboratories were commissioned by American Bonanza to conduct a gravity, cyanidation and flotation process evaluation on a 30-kilogram (64 pound) composite of representative D zone mineralized material. The composite was prepared with the objective of obtaining a composite gold grade of >1.0 oz/ton, copper content of >0.5%, and representative host rock/chemical reactivity characteristics of silicified limestone. The highlights of the test work were as follows:

- A total of 10 drillhole intervals from the D zone (from four holes Ag8-2, 3 5, and 13) were composited into a single test sample.
- No mineralogical study was conducted.
- Gold head grade was determined by metallics assay and the reconciliation with test work calculated head assay was good. Small free gold particles were observed in both gravity and flotation products.
- Both whole ore cyanidation and a single bulk flotation test gave good recoveries.
- Optimum cyanide concentrations were about 1.5 g/l, which is high. This was required to improve the recovery rate (at 48 hr), which otherwise at 0.5 g/L would take 72 hr. Cyanide consumptions were high at about 3 kg/t because of cyanide soluble copper dissolution. No copper leach balances, or speciation data were provided to support this, but it was consistent with current and past test work observations. There was about 0.6% Cu in the sample.
- The gravity work indicated high (about 50%) potential gold recovery, but this is inconclusive because the weight recoveries were also very high.
- A bulk float test recovered 94% gold into a 16.7 % feed weight concentrate at 200 mesh. Copper flotation metallurgy or concentrate cyanidation was not investigated.
- The sample solids specific gravity was quite high at 3.37 because of the relatively high hematite content.
- Conventional thickening tests on leach residue indicated good flocculation and rapid settling characteristics.

A single Ball Mill Work Index of 13.7 kWh/ton indicated that the material was of moderate hardness with respect to ball milling. Table 13-2 shows the summary results of the McClelland labs metallurgical tests during the year 2000.

Table 13-2 McClelland - 2000 Cyanidation and Flotation Test Results

Zone	Lab	Date	Test	Wt Kg	Grade		Ag oz/T	Cu % Total	Parameter		Recovery %			CN Consumption	
					Au oz/T	Au (Calc)			Grind	Time (hr)	Au	Ag	Cu	g/l	kg/t
D-zone Composite	McClelland	2000	Gravity/Cyanidation	30	1.03	0.82	NA	0.60	270M	72	99			0.5	2.877
			Gravity/Cyanidation		1.03	0.85			270M	72	99			1.5	3.286
			Gravity/Cyanidation		1.03	1.18			100M	72	93			1.5	3.173
			Direct Cyanidation		1.03	1.00			200M	72	92			1.5	3.173
			Float		1.03	1.18			200M		94				

13.6 Echo Bay Minerals (2001)

In 2001, Echo Bay Minerals evaluated flotation processing Copperstone mineralized material at their McCoy/Cove mill on behalf of American Bonanza. The highlights of the test work were as follows:

- A flotation test was conducted on a single 2 kg high-grade sample, but its origin was not identified.
- Flotation recovery was reasonable at 87% into a 4% feed weight concentrate. Possibly recovery could have been higher with an alternate frother, higher weight recovery and finer grind. However, the objective was to study the response of the mineralization using McCoy/Cove operating parameters.

Copper (0.56% Cu in head) recovery to the concentrate was only about 9%. This was consistent with the previous flotation/concentrate cyanidation work. The poor copper flotation recovery suggests copper is present predominantly as chrysocolla. This is important because chrysocolla is poorly recoverable (up to about 15%) in conventional flotation. Table 13-3 shows the summary results of the McCoy/Cove lab metallurgical tests during 2001.

Table 13-3 McCoy/Cove - 2001 Flotation Test Results

Zone	Lab	Date	Test	Wt Kg	Grade		Ag oz/T	Cu % Total	Parameter		Recovery %		
					Au oz/T	Au (Calc)			Grind		Au	Ag	Cu
D-zone Composite	McCoy/Cove	2001	Float	2.0	1.51	0.75	NA	0.56	140M		88	NA	9

13.7 McClelland Laboratories (2005)

In early 2005, McClelland Laboratories Inc. were commissioned to prepare two new D zone and Hanging Wall metallurgical composites, using samples provided by American Bonanza. The HW zone sample consisted of 43 intervals from 7 drillholes. The D zone sample consisted of 51 intervals from 3 holes. The intervals consisted of half cores. McClelland crushed the samples to 10 mesh and blended each zone for a master composite. Separate bulk samples were taken for the Bond's ball mill work and abrasion indices tests.

The program was originally designed to be conducted in two phases. Phase 1 was scoping in nature to test the effects of grind size on gravity recovery, flotation and direct cyanidation. Phase 2 tests also included Bond's ball mill work and abrasion indices tests, thickening and pulp rheology tests.

13.7.1 Phase 1

Gravity concentration, whole ore gravity/cyanidation and gravity/flotation and flotation concentrate cyanidation investigations were conducted and reported by McClelland. An initial Phase 1 of testing was conducted in March 2005, but the results were inconsistent and inconclusive, and this was followed up by a similar Phase 1.5 program in June 2005.

Gravity recoveries were relatively low in Phase 1 but improved in Phase 1.5. Phase 1.5 was undertaken to test methods of reducing the cyanide consumption in direct cyanidation and to improve flotation recoveries into a lower weight concentrate. The Phase 1 direct cyanidation tests produced good gold recoveries but at very high cyanide consumptions and extended leach times, due to the presence of soluble copper in the mineralized material. Flotation work was conducted to investigate the production of a smelter concentrate. A secondary objective of the flotation test work was to investigate the potential to separate gold and copper using flotation and produce a concentrate that could be cyanided more economically. Gold flotation recoveries were reasonable and concentrate cyanidation recoveries and cyanide consumptions were very good. The highlights of the study are summarized as follows:

- *Samples and Characterization.* The D zone composite comprised of seven drillhole core intervals from two drillholes. The Hanging Wall composite comprised of twelve drillhole core intervals from two drillholes and twenty-five intervals of coarse assay rejects from five holes, which were a reasonably wide range compared to the D zone composite. Copper was also an important issue relative to process selection. Additional work was recommended to better define the relationship between cyanide consumption, flotation and copper mineralization (speciation). This was also recommended in earlier programs. Copper speciation should be done on some sample sub-composites to help identify copper mineralization trends.
- *Gravity.* The gravity recoverable gold potential in the 2005 composite samples appeared relatively low at about 8-15% at typical weight recoveries (0.02% to 0.22%) used to produce a direct smelt material. However, gravity was not considered to be the primary recovery process, more as an insurance against occasional nugget gold and other unrecoverable gold by direct cyanidation or flotation. Gravity gold recovery showed some potential to improve overall recovery in the test work. However, to achieve acceptable gravity recoveries, weight recoveries were high and the concentration ratios low because of the concomitant recovery of iron present as magnetite/hematite in the samples. Improved panning in Phase 1.5 in some instances successfully eliminated the magnetite, and further work on gravity was recommended because it appeared to have the potential to add 1% to 2% to the overall recovery from gold that may not float well. Possibly, the gravity concentrate will not be a viable direct smelt material due to its high magnetite content and a separate intensive cyanidation unit will be required. However, an industrial-scale centrifugal gravity unit could be more selective in eliminating magnetite, versus the batch laboratory Knelson unit used in the test work, and further work on this concept was recommended. Previous work on other D zone and Hanging Wall zone samples suggested visible gold was present. Gravity potential may be understated by initially grinding in a single stage to 200 mesh followed by testing.

- *Direct Leach.* Extended direct leach times were required (72 hr to 96 hr) to achieve good gold recoveries of 95-98% and cyanide and lime consumptions were very high. Cyanide consumptions were about 3 kg/t for D zone material and 7 kg/t for HW zone material. This was consistent with earlier work and indicative of the presence of elevated levels of soluble copper. Copper levels and cyanide consumption were highest in the new Hanging Wall sample. Both 150 mesh and finer 200 mesh grinds gave good recoveries, and no trends were noted.
- *Flotation.* Only bulk rougher flotation concentration test work was conducted. Bulk gold flotation recoveries were reasonable, and at a grind of 200 mesh, direct flotation recoveries of about 89-90% were indicated and when combined with gravity 92% to 96%. Gravity appeared to have the potential to add 1% to 2% to the overall recovery from gold that may not float well. However, some flotation optimization potential was indicated that may offset this. Additional kinetic tests were recommended to assess float time and grind requirements and optimize mass pulls. Finer grinding to 200 mesh appeared to improve recovery but the results are inconclusive. Similar reagent suites were used in both phases, but Phase 1.5 utilized different reagent addition rates and pull techniques. Some cleaner work was also recommended to investigate the potential to produce a direct-smelt product or optimize cyanidation economics. Copper and sulfur balances should also be done. Assuming copper mineralization is predominantly chrysocolla, it was expected copper recovery will be low using conventional flotation processing and this appears to be the case based on the relatively low cyanide consumption (relative to whole ore) in the concentrate cyanidation test work.
- *Concentrate Leaching.* The flotation concentrates from all the Phase 1 tests were combined in a single sample to provide sufficient material for assaying and meaningful metallurgical tests. The flotation concentrates from Phase 1.5 were likewise combined. The flotation concentrates leached very well and in 72 hr produced recoveries of 98.3% to 99.4% and with much lower cyanide consumptions than direct cyanidation. This suggested the copper mineralization did not float well, which was consistent with earlier flotation work done by Echo Bay Minerals, and that it may be present predominantly as chrysocolla. However, no copper speciation or copper assays were done to be able to assess this. Copper and other base metals will accumulate in the leach solution if recycled and provision should be made in the design to bleed and destroy cyanide to remove this. Additional work will be required to assess the bleed and cyanide destruction requirements.

13.7.2 Phase 2

The following Phase 2 tests were completed to investigate other opportunities and provide some of the data necessary for mill design. Rheology and Settling work was done to support an underground paste backfill option.

- *Magnetite Recovery.* Dawson investigated the production of a by-product magnetite product from tailings. Only about 5% of the iron in the feed reported to the final concentrate. The sample was highly oxidized, and it was assumed that most of the iron is present as hematite. No further work was recommended.
- *Rheology.* The rheology of a rougher tailing underflow sample was assessed by Pocock Industrial showed Bingham Plastic characteristics at all solids concentration tested. Experience shows that thickener underflow solids concentrations exhibiting a yield value in excess of 30 Pascals ("Pa") are not considered practical for standard thickener design, as pumping and torque problems will likely result. This threshold occurred at slightly above 60% solids for the Flotation Rougher

Tailings. In a full-scale plant, underflow solids concentrations exceeding 60% solids should be avoided unless specialized equipment is employed to handle the higher viscosities.

- *Settling.* Static thickening tests by Pocock Industrial explored the effect of variations in flocculent dose, and feed solids concentrations for conventional thickener design. Thickener unit area sizing on properly flocculated thickening tests performed on the Flotation Rougher Tailings material generally fell below the minimum range of 0.125 m²/tonnes per day to 0.150 m²/tonnes per day recommended by Pocock Industrial for full scale conventional thickeners. This recommended design range corresponded to underflow solids concentrations from 50% to 60% solids by weight. Dynamic (High Rate) thickening tests were also performed on the Flotation Rougher Tailings sample to determine the recommended hydraulic design basis and expected overflow suspended solids concentrations.
- *Bond's Work Index:* The recommended four buckets (two of each) of D zone and HW zone samples were submitted by McClelland to Metso Minerals for Bond ball mill Work Index (BWI) and abrasion index (Ai) testing. The D zone sample BWI indicated it was softer than the HW zone (12.98 kWh/ton and 15.04 kWh/ton) but much more abrasive with an Ai of 0.4464 and 0.1508 respectively. Overall, this material can be ranked as of medium hardness and abrasivity with respect to ball milling and wear rates should not be expected to be excessive.

Table 13-4 shows the summary results of the McClelland labs metallurgical tests during 2005.

Table 13-4 McClelland - 2005 Metallurgical Test Results

Zone	Lab	Date	Test	Wt Kg	Grade		Ag oz/T	Cu % Total	Parameter		Recovery %			CN Consumption				
					Au oz/T	Au (Calc)			Grind	Time (hr)	Au	Ag	Cu	kg/t				
D zone	McClelland	2005	Grav/Leach	122.5	0.51	0.53	0.06	0.57	150M	48/72	96/98	>47	NA	3.195				
			Grav/Leach						200M	48/72	94/96	>47		3.354				
			Grav/Leach+NH4OH						200M	48/72	90/95	>47		3.559				
			Grav/Float						150M		89							
									200M		89							
Hangingwall	McClelland	2005	Grav/Leach	103.0	0.33	0.33	0.03	0.90	150M	72/96	91/97	>32		7.422				
			Grav/Leach						200M	72/96	90/98	>29		7.100				
			Grav/Leach+NH4OH						200M	72/96	96/98	>48		7.918				
			Grav/Float						150M		88							
									200M		91							
D zone+HW	McClelland	2005	Conc Cyanidation			11.23			200M	48/72	89/99			0.477				

13.8 CAMP (Center for Advanced Mineral and Metallurgical Processing) (2009)

This testing was undertaken to review and verify the previous metallurgical work completed on Copperstone deposit, identify gaps, and expand incomplete testing. The testing attempted to confirm grinding requirements, gravity testing expectations, and optimize flotation requirements. The results were used to optimize the final flowsheet and metallurgical balance for when Bonanza operated the underground mine at Copperstone. The following identifies the test work:

- *Samples and Characterization.* Composites were developed from across the C and D zones to generate as representative of sample as possible for verification testing. The samples were pulled

from core from both the C and D zones. For the C zone testing, 22 samples from 5 holes were used. From the D zone, 34 samples from 14 holes were used. Samples were picked to generate a representative sample of mineralized material across both zones.

- *Gravity.* The gravity recoverable gold potential in the 2009 composite samples appeared very good at about 25-40% at typical weight recoveries (1.0% to 1.25%) used to produce a salable material with a gold content from 8 to 15 oz/ton. However, to achieve acceptable gravity recoveries, as weight recoveries were slightly higher than expected, options such as a Gemini Table to remove gangue prior to further processing may be required.
- *Flotation.* Only a single bulk rougher flotation concentration test was conducted on each sample to verify previous results. Bulk gold flotation recoveries of 70 to 75% were reasonable at a grind of 200 mesh, and when combined with gravity, 86 to 94% recoveries were achievable. The flotation tails were consistent with that of previous testing with much higher head grade. Additional kinetic tests were recommended to assess float time, grind requirements and optimize mass pulls. Other reagent schemes should also be evaluated to maximize final recovery for flotation.
- *Grinding.* Grinding of the D zone and C zone samples identified a Bond's ball mill Work Index of 12.8 and 14.3 kWh/ton respectively. These work indexes are in the same range as previous testing. No further Bond grind testing was recommended on the D and C zones.

Table 13-5 shows the summary results of the CAMP labs metallurgical tests during 2009.

Table 13-5 CAMP - 2009 Metallurgical Test Results

Zone	Lab	Date	Test	Grade		Ag oz/t	Cu % Total	Parameter	Recovery %		
				Au oz/T	Au (Calc)			Grind	Au	Ag	Cu
D zone	CAMP	2009	Grav/Float	0.33	0.64	0.93	0.85	200M	28/88	47	NA
C zone	CAMP	2009	Grav/Float	0.34	0.41	0.88	0.92	200M	37/86	27	NA

13.9 Resource Development Inc (2018)

The primary objective of the test work was to confirm that the flotation processing option was the optimal method and technically viable in the various gold mineralization zones. As discussed previously in 2011, American Bonanza constructed a 450 stpd flotation mill on site and in 2012 started underground mining from two previously developed declines located in the bottom of the Cyprus open pit. American Bonanza's mining focused on the D zone which is to the north of the open pit. From January 2012 to July 2013, American Bonanza produced approximately 16,900 oz of gold from 163,000 tons of mineralized material grading 0.104 oz/ton of gold. The RDI 2017 test program was expanded to also evaluate the technical/economic aspects of the alternative processing options, namely leaching of flotation concentrate and whole ore leaching to produce doré at site. These options also had the potential of producing copper as a by-product.

The metallurgical test work undertaken on the samples from the four mineralized zones (A, B, C, and D) and a master composite, prepared with samples from zones C and D which constitutes the majority of the resources at the Project, included head analysis, mineralogy, gravity, flotation, grindability, and cyanide

leaching of mineralized material and flotation concentrate. The summary of the test results indicated the following:

- **Head Assays** – The head analysis of Composites A, B, C, D, and the master composite are shown in Table 13-6. The composites for the four zone samples assayed 2 g/t Au to 8 g/t Au, with the master composite (higher grade sample) averaging 20.6 g/t Au.

Table 13-6 Head Assays of Zone Samples Tested

Element	Composite				
	A	B	C	D	Master
Au, g/t					
Assay #1	2.051	6.056	8.022	4.303	20.782
Assay #2	2.030	5.947	8.128	-	20.610
Average	2.040	6.00	8.075	4.303	20.696
Ag, g/t					
Assay #1	0.3	3.4	25.4	1.6	1.6
Assay #2	0.3	3.02	26.0	-	1.6
Average	0.3	3.3	25.7	1.6	1.6
Cu, %					
Cu _{AcidSol} , %	0.0193	0.271	1.196	0.55	0.262
Cu _{CNSol} , %	0.0013	0.238	0.0175	0.0071	0.023
Cu _T , %	0.058	0.602	1.384	0.7045	0.351
S _{Total} , %	0.05	0.23	0.05	0.02	0.05
S _{Sulfide} , %	<0.01	0.10	0.02	0.02	0.02
S _{Sulfate} , %	0.05	0.13	0.03	<0.01	0.04
C _{Total} , %	0.12	<0.01	0.06	0.44	0.18
C _{Organic} , %	0.02	<0.01	0.04	0.05	0.04
C _{Inorganic} , %	0.10	<0.01	0.02	0.39	0.14

The copper values in these samples varied from 0.058% to 0.7%. The copper in Composite B was primarily secondary or primary copper whereas the copper in other composites was basically oxide copper.

- **Mineralogy** – The purpose was to determine the bulk mineralogy of two samples, namely, Composite B and the Master Composite, with an emphasis on gold and copper mineralogy. Each sample was prepared as a standard polished thin section for analysis by reflected/transmitted light microscopy.

Although there were some differences in copper mineralogy between the samples, the general bulk mineralogy and gold occurrence was essentially the same. The master composite contained the lowest concentration of copper mineralogy. Chalcopyrite occurred as small grains locked in quartz with a grain size up to 15 microns, but was rare. One grain identified was attached to

pyrite. Chrysocolla was present as blueish green liberated fragments up to 1 mm in size and as small pockets in secondary quartz. A fragment of earthy iron oxide with inclusions of a highly anisotropic phase with a yellow brown tint may represent delafossite/tenorite. The most notable copper mineral in the master composite sample was two thin seams of native copper that measure approximately 1 mm in quartz with iron oxide.

The composite B sample contained the highest concentrate of copper which occurred as sulfides. The most prominent type was chalcocite. Chalcocite occurred as large, liberated masses measuring over 1 mm and as small interstitial patches in quartz. The chalcocite was generally associated with red oxide. Much of the chalcocite carried small rod-shaped inclusions of covellite and rarely, minor relict chalcopyrite. Small drop shaped chalcopyrite grains up to 10 microns were also seen locked in quartz. Grains of liberated bornite with a grain size of 8 microns to approximately 300 microns occurred as a trace. The bornite shows moderate to strong alteration/replacement by chalcopyrite and covellite/digenite.

In the Master composite sample, three grains of gold were identified. Two grains were locked in quartz with a grain size of 5 microns to 9 microns. One large 25-micron grain sat between blades of specular hematite. In the Composite B sample, two grains of gold were identified. One 5-micron grain was associated with chalcocite. One large 24-micron grain sat in a granular matrix of quartz and iron oxide. Although difficult to determine, these gold grains may actually be liberated. Table 13-7 outlines the major and minor mineralogy for each sample. Concentrations of individual phases are based on petrography.

Table 13-7 Percent Mineralogy of Composite B and Master Composite

Client Sample no.:	Copperstone MC	Copperstone Comp B
Quartz	46	42
Hematite/Goethite	30	25
Plagioclase	9	10
K-Feldspar	5	8
Chlorite	4	10
Muscovite	3	5
Calcite	3	*
Chalcopyrite	*	*
Chrysocolla	*	-
Bornite	-	*
Covellite	-	*
Delafossite/Tenorite	*(?)	-
Chalcocite	-	*
Digenite	-	*
Rutile	*	*
Magnetite	*	*
Leucosene	*	*
Native Cu	*	-
Au	*	*

- **Rougher Flotation** – Gold from the composites can be floated using simple reagent suite consisting of potassium amyl xanthate (PAX), Aeropromotor 404 and a frother. The flotation test results indicate that the finer the grind, the higher the gold recovery but lower the

concentrate grade. The majority of the gold floats in the first three minutes of flotation. Sulfidization of the feed material made slight improvements in overall gold recovery. Assay by size data of the flotation tails indicates that the majority of the gold losses are in the coarser particle sizes while the majority of copper is in the fine fraction. Flotation test results for composites A through D varied from 87% to 91% Au. Flotation results for the master composite are shown in Table 13-8

Table 13-8 Master Composite Flotation Test Results

Product	Cumulative Flotation Time, min	Cumulative Recovery %				Cumulative Grade		
		Wt.	Au	Ag	Cu	Au g/t	Ag g/t	% Cu
Grind, P ₈₀ = 200 Mesh (Test 25)								
Conc. 1	3	1.2	86.1	46.1	14.7	664.1	32.2	4.61
Conc. 2	6	2.2	88.7	50.3	18.2	370.0	19.0	3.09
Conc. 3	9	2.9	89.5	51.9	20.2	284.2	14.9	2.60
Conc. 4	12	3.8	90.0	53.7	22.7	215.1	11.6	2.21
Cal. Feed	-	100.0	100.0	100.0	100.0	9.19	0.8	0.37

- **Cleaner Flotation** – Tests were completed with material from the master composite and Composite D. A combination of approaches was utilized to achieve maximum gold recovery while providing a concentrate grade of over 350 g/t. The individual cleaner results achieved a range of recoveries from 86.3% to 87.6% Au. The maximum gold recovery with a grade of over 350 g/mt Au would result from the combined rougher concentrate and cleaned scavenger concentrate with regrind. An overall recovery of 87.6% was observed with a concentrate grade of 584 g/t Au.
- The flotation tailings from tests 3 and 11 representing Composites A and D were subjected to sulfuric acid leach for extraction of oxide copper. The acid consumption was extremely high (20 to 36kg/t) and the pH was still higher than 2 which is required for copper leach. An XRF analysis indicated the presence of MgO and CaO, thereby indicating the presence of carbonates, which make the acid leach process for copper extraction uneconomical.
- **Whole Ore Leach Tests** – Cyanidation of the mineralized material recovered 88% to 97% gold for all composite samples except B. The extraction was only 9.2% for Composite B. The sample and composite for B zone was considered an anomaly, and not representative of the B zone area of the deposit.

Cyanide leach tests of the master composite at various grind sizes indicated the gold extraction improved as the particle size becomes finer. A particle size of 200 mesh exhibited the highest extraction at 97.1%. Based on these results, an economic evaluation of the processing options indicated that the whole ore leach process was the best option for this deposit.

Leach tests were completed at lower cyanide levels in an effort to decrease consumption. Initial tests were started at 0.5 g/L, 0.75 g/L, and 1.0 g/L NaCN and allowed to decay. The cyanide was consumed after the initial 6 hours of leach time and the tests were then maintained at 0.25 g/L NaCN. Gold extractions for all tests were significantly lower than previous results at a

maintained concentration of 1.0 g/L NaCN. Additional tests were conducted at maintained levels of 0.5 g/L, 0.75 g/L, and 1.0 g/L NaCN. Tables 13-9 and 13-10 present the cyanidation leach test results.

Table 13-9 WOL Results - Composite Samples

Process Parameter	Composite											
	A			B			C			D		
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
Extraction %												
6 hr	90.7	12.7	4.5	1.5	0	12.4	11.7	9.6	4.4	30.2	5.2	8.8
24 hr	95.4	12.9	4.7	4.9	0.3	21.9	61.4	35.4	8.3	73.3	10.6	16.3
48 hr	97.0	14.4	5.1	9.2	0.3	33.0	97.4	58.5	14.2	88.0	12.6	24.5
Residue, g/t	0.07	1.0	592	3.68	4.6	4060	0.11	0.6	13165	0.22	2.8	5270
Cal. Feed, g/t	2.27	1.2	624	4.05	4.6	6056	4.08	1.4	11300	1.82	3.2	6977
Reagent Consumption, kg/t (48 hr)												
NaCN	0.242			4.128			3.033			3.627		
Lime	3.739			4.819			3.747			4.125		

Table 13-10 WOL Results – Master Composite

Process Parameter	Particle Size								
	P ₈₀ 100 mesh			P ₈₀ 150 mesh			P ₈₀ 200 mesh		
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
Extraction %									
6 hr	34.5	19.7	5.2	31.6	19.1	5.6	10.9	2.5	17.1
24 hr	64.9	41.1	8.5	69.4	38.8	9.4	38.2	18.5	24.8
48 hr	88.1	62.8	12.3	84.5	52.3	11.8	97.1	50.8	29.3
Residue, g/t	0.41	0.4	9860	0.58	0.6	9720	0.12	0.6	2780
Cal. Feed, g/t	3.47	1.1	11247	3.75	1.3	11019	3.99	1.2	3933
Reagent Consumption, kg/t (48 hr)									
NaCN	3.374			3.425			2.700		
Lime	3.997			3.655			1.560		

- **Gravity** – Test work on gravity recovered a small portion of the gold present in the mineralized material at a concentrate grade that is not direct smeltable. Hence, gravity process was not incorporated in the recommended process scheme. The fine gold is directly cyanide leachable and hence there was no need to have a gravity circuit.
- **Grindability** – Bond's ball mill work indices were determined for the master composite sample. The average value of six historical values and the two present work indices was 14.0. The current ball mill will be able to process +/- 840 stpd of mineralized material to produce a product of P₈₀ of 200 mesh at this work index.

Based on the 2018 test work, a conceptual process flowsheet was developed and is given in Figure 13-1. The proposed process flowsheet consists of crushing and grinding the mineralized material to P₈₀ of 200 mesh and sending the slurry to a pre-leach thickener. The thickener underflow will be sent to a series of leach tanks where gold and cyanide soluble copper could be extracted. The scoping level test work indicates that SART process can recover copper and thereby reduce cyanide consumption. The flowsheet is adaptable to incorporate the SART process between the leach tanks and the CIP tank. Following the CIP process the leach residue will be thickened to recover process water containing cyanide which will be recycled back to the leach tanks thereby reducing the overall cyanide consumption. In addition, the potential incorporation of the SART process will recover copper thereby improving overall project economics. The thickener underflow (leach residue) will be subjected to cyanide destruction before pumping the slurry to the tailings pond.

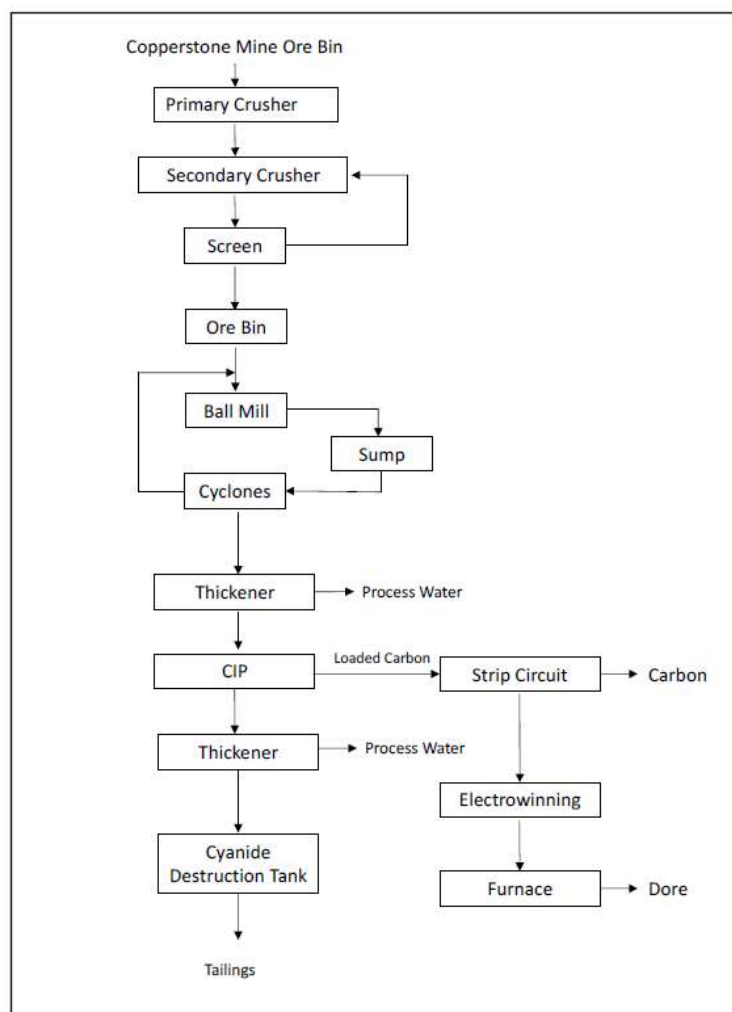


Figure 13-1 Whole Ore Leach Process Flow Diagram

13.10 Resource Development Inc (2019)

The primary objective of the test program undertaken in 2019 was to determine the effect of copper on the leach process and to determine reagent consumptions. The testing included locked-cycle cyanide leaching with carbon adsorption, carbon stripping, thickening and filtration tests, and cyanidation destruction tests.

The highlights of the test results indicated the following:

- Head assay results for the composite sample, a blend of zones C and D, determined the gold grade to be 7.12 g/t Au. Copper content was 0.283% Cu with approximately 38% of that being cyanide soluble.
- Locked cycle leach tests indicated that an average of 98.2% of the gold was extracted based on the calculated head assay from the CIC data. Copper extractions averaged 13.8% for the cyanide decay tests and 15.2% for the maintained tests.
- The increased copper content of the locked cycle leach solutions did not appear to impede the extraction of gold. The copper content continued to increase in the circuit along with the WAD cyanide levels, reaching a maximum of 3200 mg/L in the final cycle. Slightly higher copper and WAD CN grades were observed in the maintained cyanide tests as compared to the decay tests.
- Cyanide consumptions were higher for the tests with the maintained concentrate versus the tests that were allowed to decay. The decay tests averaged a consumption of 1.75 kg/t, while the maintained tests averaged 1.97 kg/t. Cyanide consumptions could potentially be reduced with leach times of less than 48 hours.
- Lime consumptions averaged 1.68 kg/t for the decay tests and 1.42 kg/t for the maintained tests when the barren solution was recycled. Actual plant consumptions would be slightly lower due to better process control.
- Carbon loading tests indicate that approximately 99% of the gold in the pregnant solution was loaded to the activated carbon within 1 hour. Less than 1.5% of the copper in solution was loaded on the carbon. Free cyanide levels up to 2000 ppm did not significantly effect loading kinetics and only slight reduced the amount of copper loading. The 2.0 g/L NaCN solution loaded 0.7% of the copper while the 1.0 g/L solution loaded 1.3%.
- Thickening test work indicates a final settled density of 65% solids utilizing a high molecular weight anionic polymer. A thickener with a unit area of 0.045 m²/tonne/day would be needed to achieve a underflow density of 60% solids
- A maximum percent solids of 78.7% was achieved during filtration testing. A filtration rate of approximately 50 dry lb/ft²/hr was accomplished with vacuum filtration. Stable filter cake was produced at higher than 78% solids, otherwise it would lose integrity when stress was applied.
- A total of 0.45 kilograms of water for each kilogram of leach residue lowered the wash solution to 0.82 mg/L Au, 180 mg/L free cyanide, and 235 mg/L WAD CN. Approximately 0.6 kg of fresh water/kg residue would be needed to efficiently remove the residual metals from the leach residue.
- The cyanide destruction process successfully reduced the residual cyanide to acceptable levels. Sodium metabisulfite consumption was 1.5 kg/t, and copper sulfate consumption was 0.12 kg/t. Consumptions could be reduced with more efficient washing of the leach residue.

- The carbon stripping process removed approximately 81.3% of the gold, 94.5% of the silver, and all of the copper loaded on the carbon. A copper grade of 405 mg/L in the strip solution indicates that pretreatment of the carbon will be necessary.

13.11 Resource Development Inc (2021)

A trade-off study for the incorporation of the SART process in the flowsheet vs. direct cyanidation leaching indicated a CAPEX of +/- 5 million dollars for the SART process. In order to keep the project CAPEX low, it was decided that a direct cyanidation process would be employed, followed by either a CIC or MC process.

RDI completed additional metallurgical testing in 2021 to investigate whole ore leaching and develop process data for flow sheet optimization and equipment sizing. Leaching test work included particle size and cyanide concentration leach optimization. In addition, comminution testing, and characterization of the slurry and pregnant solutions were completed. Test work was also completed for sizing of downstream processes with thickening testing.

13.11.1 Sample Preparation and Characterization

Individual drillhole interval samples from zones A, B, C and D were received for testing. This represents all the zones which include Measured and Indicated mineral resources. The combined sample weights from each zone ranged from 43 kilograms to 79 kilograms. Individual samples from each zone were combined to create approximately 40 kilograms of composite sample to achieve an approximate gold grade of 6 g/t Au. Trace amounts of silver were present in the composites with values below 2.5 g/t Ag. Copper values ranged from a low of 0.13% in zone A to a high of 0.56% in zone C. The cyanide soluble copper values mirrored the total copper assays with approximately 6% of the copper soluble in zone A to as high as 31.5% in zone C. Zones B, C, and D contained small amounts of organic carbon, while zone A did not. None of the composite samples contained more than a trace of sulfide sulfur. Each composite sample was stage crushed to minus 6 mesh, thoroughly blended, and split into charges for test work. A representative split of each composite sample was submitted for head analyses. The head assay data is summarized in Table 13-11.

Table 13-11 Head Analysis of 2021 Composite Samples

Element	Zone A	Zone B	Zone C	Zone D
Au, g/t	7.521	8.015	6.996	6.855
Ag, g/t	1.1	0.3	2.4	1.9
Cu, ppm	1291	3346	5603	3026
CuCN Sol, ppm	72.47	491.6	1763	776.8
CTotal, %	0.02	0.61	0.35	1.06
COrganic, %	<0.01	0.15	0.06	0.17
CInorganic, %	0.02	0.47	0.29	0.89
STotal, %	0.07	0.06	0.08	0.28
SSulfide, %	<0.01	<0.01	<0.01	0.09
SSulfate, %	0.07	0.06	0.08	0.19
ICP Data				
Al %	4.85	6.14	4.02	1.86
Ca %	0.25	1.81	1.21	3.10
Fe %	11.41	10.30	14.68	25.36
K %	1.99	2.12	1.89	0.59
Mg %	1.55	0.77	0.62	0.80
Na %	0.68	1.11	0.62	0.25
Ti %	0.07	0.12	0.08	0.05
As ppm	<2	16	<2	<2
Ba ppm	4476	2936	4410	1071
Bi ppm	<2	<10	6	6
Cd ppm	11	10	15	27
Co ppm	11	14	10	8
Cr ppm	56	51	55	67
Hg ppm	0.76	0.49	1.58	3.28
Mn ppm	2446	11500	7965	2699
Mo ppm	<1	1	1	16
Ni ppm	13	12	8	6
Pb ppm	10	25	<10	25
Sb ppm	<2	<2	<2	<2
Se ppm	<5	6	<5	5
Sr ppm	134	191	168	191
Te ppm	14	14	19	22
V ppm	49	50	45	48
W ppm	77	53	74	61
Zn ppm	42	52	28	48

13.11.1.1 Bond Mill Work Index and Grind Study

A Bond's Ball Mill Work Index test was completed with each composite sample at a closed size of 100 mesh (150 microns). The work index results, summarized in Table 13-12, indicated that the samples would be considered medium hardness and were within similar ranges of previous test results.

Table 13-12 2021 Bond Ball Work Index Results

Sample	BWI (kWh/t)	Classification
Zone A Comp	12.96	Medium
Zone B Comp	12.92	Medium
Zone C Comp	14.29	Medium-Hard
Zone D Comp	11.34	Medium

13.11.1.2 Particle Size Optimization Leach Testing

A series of leach tests were completed with each composite sample at various particle sizes to determine optimal grind size. Bottle roll leach tests were completed at P80 100 mesh, 150 mesh, and 200 mesh. The leach conditions were 45% solids, pH 11, and 2.5 g/L NaCN (maintained) for 48 hours. Solution samples and process checks were completed at 6, 12, 24, 36, and 48 hours. After 48 hours, a sample of the final solution and residue from each test were submitted for gold, silver, and copper assay. The bottle roll leach results indicated the following:

- High gold extractions were observed in all composite samples. Gold extraction at 48 hours ranged from 93.5% to 99.3%. The gold leached quickly, with the maximum extraction obtained in approximately 24 hours. Gold extractions improved slightly from 100 mesh to 150 mesh, with little improvement observed between 150 mesh and 200 mesh.
- Silver extractions varied widely, ranging from 21.7% to 80.6% due to the low silver content in the samples.
- Copper extractions were low in zone A and B composites, ranging from 3.1% to 8.7%. Zone C and D composites extracted more copper, ranging from 24.6% to 27.4%.
- CIL testing provided similar metal extraction results to the standard leaches. A very small percentage of the copper was adsorbed onto the carbon.
- Cyanide consumption generally increased as the particle size decreased. The amount of cyanide soluble copper in each sample had a significant effect on the cyanide consumption. Low cyanide soluble copper in zones A and B consumed 0.8 kg/t and 0.98 kg/t respectively at a particle size of P80 150 mesh, while the high cyanide soluble copper in zones C and D exhibited consumption

of 3.89 kg/t and 2.18 kg/t respectively. Lime consumptions were consistent among all composite samples, averaging just under 4 kg/t.

- Gold grades in the leach solutions were observed to decrease after 24 hours of leaching. Larger decreases were consistent with higher copper values, which would suggest an interference when assaying the solutions. Residue assays between the CIL and non-CIL tests were consistent.

13.11.2 Cyanide Optimization Leach Testing

A series of leach tests were completed with each composite sample at various sodium cyanide levels to determine optimal reagent dosage. Bottle roll leach tests were completed at a particle size of P₈₀ 150 mesh and sodium cyanide levels of 1.5 g/L, 1.75 g/L, and 2.5 g/L NaCN (maintained and decay). The leach conditions were 45% solids and pH 11 for 24 hours. Solution samples and process checks were completed at 4, 8, 16, 20, and 24 hours. After 24 hours, a sample of the final solution and residue from each test were submitted for gold, silver, and copper assay. The cyanide dosage leach results indicated the following:

- High gold extractions (>97%) were observed in all composite samples at all cyanide additions with the exception of zone C. The high copper content in zone C caused low gold extraction in the 1.5 g/L and 1.75 g/L decay cyanide tests.
- In general, gold leached quickly with the highest extractions occurring at approximately 16 hours. Gold grades in the leach solutions were observed to decrease after approximately 16 hours of leaching.
- Silver extractions varied widely due to the low silver content in the samples and did not appear to be dependent upon cyanide concentration.
- Copper extractions were similar between the maintained and decay cyanide tests. Slightly less copper was extracted from the high copper zone C in the decay tests.
- Cyanide consumption increased as the amount of cyanide soluble copper in each sample increased. Cyanide consumption decreased by an average of 15% in the zone A, B, and D decay tests as compared to the maintained cyanide tests. The difference in the zone C tests averaged 40% which indicates there was not enough cyanide, specifically in the decay tests. Zones A, B and D consumed 0.45 kg/t to 1.99 kg/t, while the high cyanide soluble copper in zone C exhibited minimum consumption of 3.69 kg/t to achieve a gold extraction of 97%.

13.11.3 High Intensity Cyanide Leach Testing of High Soluble Copper Samples

Zone B samples from the original metallurgical program completed at RDI in 2018 indicated high levels of cyanide soluble copper which consumed cyanide and inhibited gold extraction at standard cyanide levels. CIL bottle roll tests were completed with high intensity cyanide to determine if gold extraction could be improved with higher cyanide levels. Bottle roll leach tests were completed at a particle size of P₈₀ 200 mesh and sodium cyanide levels of 7 g/L and 10 g/L NaCN (maintained). The leach conditions were 45% solids and pH 11 for 48 hours. Gold extractions of approximately 98% were obtained with the high cyanide soluble copper samples from zone B but cyanide consumptions were very high at 13.76 kg NaCN/t and 15.11 kg NaCN/t for the 7 g/L and 10 g/L tests. The leach data for each test is summarized in Table 13-13.

Table 13-13 Cyanidation Leach Test Results - Zone B High CN Soluble Copper

Process Parameter	BR 41 - CIL (P80 200 mesh) 7.0 g/L NaCN Maintained			BR 42 - CIL (P80 200 mesh) 10.0 g/L NaCN Maintained		
	Au	Ag	Cu	Au	Ag	Cu
Extraction %						
48 hr	97.9	83.6	94.8	98.1	80.7	94.0
Residue, ppm	0.12	0.3	292	0.10	0.4	356
Cal. Feed, ppm	5.63	1.8	5619	5.29	2.0	5893
Reagent Consumption, kg/t (48 hr)						
NaCN	13.756			15.111		
Lime	4.019			4.043		

13.11.4 Thickening and Rheology Testing

Thickening test work was conducted with each composite sample at a particle size of P₈₀ 150 mesh. The pH was adjusted to approximately 11 with hydrated lime prior to the addition of high molecular weight polymers (Polymer Ventures, DAF10). Initial thickening results with 35 g/t dosage of an anionic polymer produced final settled densities for zones A, B, and C of approximately 49% solids. The zone D results were similar to previous work conducted at RDI at 57% solids, while zones A, B, and C had not been previously tested. Additional thickening tests were conducted with non-ionic polymer, higher molecular weight anionic polymer, and lower dosages of the original polymer in an effort to increase the final density. Lowering the polymer dosage to 20 g/t increased the final solids density at 24 hours to approximately 55% for zones A, B, and C while zone D was similar at 59% solids. Thickener unit area ranged from 0.010 to 0.054 m²/tonne/day to achieve an underflow density of 50% solids for all zones. Zone D was able to achieve an underflow density of 55% solids with a unit area of 0.023 m²/tonne/day, while the other zones needed +/- 2.4 m²/tonne/day to achieve 55% solids. Viscosity measurements at varying pulp densities (40%, 45%, and 50%) were determined for all composites at a grind size of P₈₀ of 150 mesh with a Brookfield viscometer. The zone A sample provided the highest viscosity values, while zones B, C, and D all had similarly low viscosity. Reasonable viscosity values were observed in all samples at a pulp density of 45% solids.

13.12 Resource Development Inc (2022)

The primary objective of this study was to generate leaching process data for flowsheet optimization and equipment size utilizing the direct cyanidation, CCD, and MC process.

Composite samples of zones A, C, and D that were tested during the 2021 study were utilized for 2022 testing. Three new composite samples were created with the existing zone composites to approximate mineralized material blends that would be encountered during various years of mine plan, namely, years one, two, and three.

The production composites are described in Table 13-14 and the head analyses are given in Table 13-15. Bottle roll leach results on each composite at a particle size of P₈₀ 150 mesh, 45% solids, pH 11, and cyanide concentrations of 1.75 g/L and 2.5 g/L NaCN (maintained) for 48 hours were performed, and the data is given in Table 13-16. Additional bulk leach tests were completed for each composite to verify extraction kinetics and produce leach residue for thickening and detoxification testing. The test data are given in Table 13-17.

Table 13-14 Production Composites

Production Composite Description			
Element	Zone A	Zone C	Zone D
Comp 1	10%	70%	20%
Comp 2	40%	60%	0%
Comp 3	60%	20%	20%

Note: Composite numbers correspond to the year in the mine plan.

Table 13-15 Head Analyses

Head Analyses of Composite Samples			
Element	Comp 1	Comp 2	Comp 3
Au, g/t	5.811	5.931	5.622
Ag, g/t	1.2	1.0	1.2
CuCN Sol, ppm	1369	1018	560
Cu, ppm	4020	3420	2288
CTotal, %	0.40	0.18	0.30
COrganic, %	0.08	0.05	0.05
CInorganic, %	0.32	0.13	0.25
STotal, %	0.09	0.06	0.12
^s Sulfide, %	<0.01	<0.01	<0.01
^s Sulfate, %	0.09	0.06	0.12
ICP Data			
%			
Al	3.91	4.87	4.49
Ca	1.28	0.65	1.03
Fe	20.51	16.36	18.17
K	2.07	2.44	2.05
Mg	0.70	0.99	1.24
Mn	0.62	0.59	0.34
Na	0.39	0.49	0.46
Ti	0.12	0.13	0.13
ppm			
As	2	2	<2
Bi	<2	<2	<2
Cd	<1	<1	<1
Co	2	5	4
Cr	63	46	61
Hg	0.78	0.81	0.88
Mo	5	1	3
Ni	5	5	<5
Pb	<10	21	11
Sb	2	3	2
Se	5	<5	<5
Sr	140	135	150
Te	16	15	15
V	39	41	42
W	85	93	90
Zn	41	44	51

Table 13-16 Kinetic Cyanidation Leach Test Results

Kinetic Cyanidation Leach Test Results																		
Process Parameter	Comp 1 (P ₈₀ 150 mesh)						Comp 2 (P ₈₀ 150 mesh)						Comp 3 (P ₈₀ 150 mesh)					
	BR43			BR44			BR45			BR46			BR47			BR48		
	(1.75 g/L NaCN)			(2.5 g/L NaCN)			(1.75 g/L NaCN)			(2.5 g/L NaCN)			(1.75 g/L NaCN)			(2.5 g/L NaCN)		
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
Extraction %																		
6 hr	74.3	20.7	17.3	90.8	36	20.2	88.6	80.7	16.9	85.6	9.1	20.3	92.6	24.4	13.7	91.6	28.7	15.4
12 hr	95.5	22.4	21.2	100.4	33.6	22.8	99.3	82.6	19.4	97	9	22.2	98.3	25	14.7	97.6	29.3	16.3
24 hr	100.7	23.4	23.1	103.6	33.3	24.3	101.6	81.2	20.8	99.2	9.2	23.5	100.9	25.5	15.8	99.1	28.8	17.5
36 hr	98.4	23.3	23.4	99.9	32.9	24.4	99	82.9	20.7	95.7	9	23.2	100.3	26	15.9	97.9	29.4	18.1
48 hr	97.7	23.1	22.6	98	32.7	25.3	97.2	81.7	21.4	97.3	9.2	22.6	98.6	25.5	16.4	98.2	30	17.5
Residue, ppm	0.13	1.5	3200	0.1	0.8	3200	0.14	0.1	2800	0.14	3.4	2700	0.08	1	2100	0.1	0.8	1900
Cal. Feed, ppm	5.48	2	4136	5.23	1.2	4285	5.06	0.4	3563	5.3	3.7	3490	5.31	1.3	2512	5.61	1.1	2303
Reagent Consumption, kg/t (48 hr)																		
NaCN	2.686			3.039			2.134			2.575			1.219			1.491		
Lime	3.709			3.657			3.426			3.446			3.46			3.449		

Table 13-17 Bulk Cyanidation Leach Test Results

Bulk Cyanidation Leach Test Results																		
Process Parameter	Comp 1 (P ₈₀ 150 mesh)						Comp 2 (P ₈₀ 150 mesh)						Comp 3 (P ₈₀ 150 mesh)					
	BR49 (1.75 g/L NaCN)			BR50 (2.5 g/L NaCN)			BR51 (1.75 g/L NaCN)			BR52 (2.5 g/L NaCN)			BR53 (1.75 g/L NaCN)			BR654 (2.5 g/L NaCN)		
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
Extraction %																		
4 hr	69.2	49.7	15.6	84.2	65.3	17.9	81.2	46.1	17.8	79.1	65.3	17.4	85.7	61	12.8	87.5	64	14.4
8 hr	94.1	51.4	20.3	94.6	65.6	21.7	94	47.8	19.1	91.1	65.6	20.2	93.6	63.4	14.8	95.2	64.2	16.2
16 hr	98.6	53.1	22.5	99.1	67.9	23.3	97.6	49.4	23.6	98.4	65.9	21.5	97.4	63.7	15.8	98.3	64.5	17.3
20 hr	99.6	53.4	22.7	98.7	68.2	23.6	98.1	48.1	25.7	100.2	66.3	22	98.1	64.1	16.2	98.8	67	17.6
24 hr	98	52.2	22.9	98.3	68.6	23.7	98.2	49.9	21.8	98.1	66.6	21.9	97.8	64.4	16.4	98.5	65.2	17.5
Residue, ppm	0.14	0.4	3200	0.12	0.2	3100	0.12	0.4	2600	0.12	0.2	2600	0.14	0.2	1900	0.1	0.2	1800
Cal. Feed, ppm	6.73	0.8	4151	6.75	0.6	4064	6.63	0.8	3324	6.62	0.6	3330	6.3	0.6	2272	6.49	0.6	2182
Reagent Consumption, kg/t (24 hr)																		
NaCN	2.579			2.713			2.143			2.272			1.064			1.294		
Lime	3.408			3.405			3.403			3.408			3.405			3.408		

The leach residues from bulk leach tests were subjected to counter-current-decantation (CCD) testing to determine the settling rate and the stages of washing needed to reduce the precious metals and cyanide values to acceptable levels. The results are summarized in Tables 13-18 through 13-20.

Table 13-18 CCD Results Composite 1

CCD Results - Composite 1 Sample								
Sample	CCD Stage	Au Grade (mg/L)	Ag Grade (mg/L)	Cu Grade (ppm)	Free Cyanide (ppm)	WAD Cyanide (ppm)	Final Solids (%)	Unit Area (m ² /tonne/day)
Comp 1 (1.75 g/L)	Stage 0	5.29	0.35	764	1,760	2496		
	Stage 1	1.93	0.34	377.6	600	789	60%	0.015 @50% 0.033 @55%
	Stage 2	0.51	0.08	102.9	120	158	60%	0.017 @50% 0.041 @55%
	Stage 3	0.12	0.02	34.4	20	46.0	60%	0.017 @50% 0.036 @55%
	Stage 4	0.03	<0.01	9.0	10	11.0	58%	0.019 @50% 0.047 @55%
Comp 1 (2.5 g/L)	Stage 0	5.32	0.35	774	2,520	3943		
	Stage 1	2.10	0.15	392.6	960	1000	60%	0.016 @50% 0.037 @55%
	Stage 2	0.51	0.04	108.7	160	218	61%	0.015 @50% 0.033 @55%
	Stage 3	0.13	0.01	34.9	30	59.0	61%	0.016 @50% 0.037 @55%
	Stage 4	0.04	<0.01	9.9	10	13.0	59%	0.018 @50% 0.045 @55%

Table 13-19 CCD Results Composite 2

CCD Results - Composite 2 Sample								
Sample	CCD Stage	Au Grade (mg/L)	Ag Grade (mg/L)	Cu Grade (ppm)	Free Cyanide (ppm)	WAD Cyanide (ppm)	Final Solids (%)	Unit Area (m ² /tonne/day)
Comp 2 (1.75 g/L)	Stage 0	5.23	0.32	580	1,720	2083		
	Stage 1	1.61	0.17	312.0	600	726	59%	0.026 @50% 0.062 @55%
	Stage 2	0.51	0.04	84.6	80	144	60%	0.019 @50% 0.046 @55%
	Stage 3	0.14	0.01	27.8	20	41.0	59%	0.024 @50% 0.063 @55%
	Stage 4	0.03	<0.01	7.1	10	10.0	57%	0.021 @50% 0.066 @55%
Comp 2 (2.5 g/L)	Stage 0	5.21	0.32	586	2,440	3334		
	Stage 1	2.10	0.15	294.1	920	984	59%	0.021 @50% 0.048 @55%
	Stage 2	0.49	0.04	86.8	160	189	59%	0.022 @50% 0.049 @55%
	Stage 3	0.13	0.01	29.8	40	55.0	59%	0.021 @50% 0.049 @55%
	Stage 4	0.01	<0.01	8.2	20	13.0	57%	0.026 @50% 0.075 @55%

Table 13-20 CCD Results Composite 3

Composite 3 Sample								
Sample	CCD Stage	Au Grade (mg/L)	Ag Grade (mg/L)	Cu Grade (ppm)	Free Cyanide (ppm)	WAD Cyanide (ppm)	Final Solids (%)	Unit Area (m ² /tonne/day)
Comp 3 (1.75 g/L)	Stage 0	4.94	0.29	299	1,760	1607		
	Stage 1	1.58	0.13	160.0	640	552	57%	0.021 @50% 0.057 @55%
	Stage 2	0.46	0.03	45.0	120	109	57%	0.020 @50% 0.053 @55%
	Stage 3	0.12	<0.01	17.1	20	33.0	57%	0.025 @50% 0.082 @55%
	Stage 4	0.03	<0.01	5.2	10	8.00	57%	0.037 @50% 0.096 @55%
Comp 3 (2.5 g/L)	Stage 0	5.13	0.30	307	2,480	2356		
	Stage 1	1.91	0.13	161.9	960	730	58%	0.020 @50% 0.047 @55%
	Stage 2	0.47	0.02	44.7	160	130	58%	0.018 @50% 0.041 @55%
	Stage 3	0.10	<0.01	17.5	40	36.0	57%	0.023 @50% 0.077 @55%
	Stage 4	0.04	<0.01	5.4	20	9.00	56%	0.028 @50% 0.084 @55%

The fourth stage thickener underflow was subjected to Inco process (i.e. sulfur dioxide/air) for cyanide destruction. Sodium metabisulfide was added to the 50% solids slurry at 3 times the calculated stoichiometric level based on free cyanide titration results. Copper was not needed as a catalyst since the solution had 30ppm copper. The cyanide destruction results are summarized in Table 13-21.

Table 13-21 Cyanide Destruction Results

Cyanide Destruction Results				
Sample		Free Cyanide (ppm)	WAD Cyanide (ppm)	Total Cyanide (ppm)
Comp 1 (1.75 g/L)	0 hr	30	105	----
	0.5 hr	<5	23.4	----
	1 hr	<5	11.3	----
	2 hr	<5	10.5	----
	4 hr	1.14	1.17	1.31
Comp 1 (2.5 g/L)	0 hr	60	104	----
	0.5 hr	10	2.41	----
	1 hr	<5	<0.01	----
	2 hr	<5	<0.01	----
	4 hr	0.205	0.241	0.592
Comp 2 (1.75 g/L)	0 hr	20	79.3	----
	0.5 hr	<5	<0.01	----
	1 hr	<5	<0.01	----
	2 hr	<5	<0.01	----
	4 hr	2.30	2.38	2.67
Comp 2 (2.5 g/L)	0 hr	40	104	----
	0.5 hr	<5	5.13	----
	1 hr	<5	8.65	----
	2 hr	<5	8.51	----
	4 hr	4.30	4.44	4.73
Comp 3 (1.75 g/L)	0 hr	20	42.6	----
	0.5 hr	<5	<0.01	----
	1 hr	<5	<0.01	----
	2 hr	<5	<0.01	----
	4 hr	0.914	1.09	1.61
Comp 3 (2.5 g/L)	0 hr	30	55.9	----
	0.5 hr	<5	0.04	----
	1 hr	<5	<0.01	----
	2 hr	<5	0.01	----
	4 hr	0.215	1.06	3.21

The highlights of the test results indicated the following:

- Composite samples had gold grades ranging from 5.6 g/t to 5.9 g/t and trace amounts of silver. Total copper values ranged from 0.23% to 0.40% with approximately 30% cyanide soluble copper.
- High gold extractions were obtained in all composite samples. The gold leached quickly, with the maximum extraction generally achieved at 16 - 20 hours. The cyanide concentrations of 1.75 g/L and 2.5 g/L NaCN produced similar gold extractions ranging from approximately $\geq 98\%$. Slight increases in silver and copper extraction were achieved at the higher cyanide concentration. Cyanide consumptions increased by approximately 15% at the higher cyanide concentration after 24 hours of leaching.
- Four stages of CCD decreased the grade of the final solution to approximately 0.03 ppm Au, <0.01 ppm Ag, <10 ppm Cu, and <20 ppm free cyanide for all samples. The 2.5 g/L NaCN material exhibited similar results to the 1.75 g/L NaCN material, with slightly higher cyanide values. A wash ratio of approximately 6 kg water to 1 kg of solids was used for 3 stages of CCD, while 4 stages used a ratio of approximately 8 kg water to 1 kg of solids.
- Final settled solids of all samples ranged from 56% to 61%. Settling rates were slower with the Composite 2 and 3 samples due to the higher percentage of zone A as compared to the Composite 1 sample. The unit area required to achieve 50% solids ranged between 0.015-0.019 m²/tonne/day for Composite 1, 0.019-0.026 m²/tonne/day for Composite 2, and 0.018-0.037 m²/tonne/day for Composite 3. The required unit area to achieve 55% solids was approximately twice as much as the area required to achieve 50% solids.
- Cyanide detoxification with sodium metabisulfite at stoichiometric additions of 4.5X to 6X achieved total cyanide content of <5 ppm for all samples within 1 hour of retention time. Sodium metabisulfite consumption averaged 0.45 kg/mt for the 1.75 g/L NaCN samples and 0.73 kg/mt for the 2.50 g/L NaCN samples

14 MINERAL RESOURCE ESTIMATE

The following preliminary mineral resource evaluation was prepared for the Copperstone Project. The estimation has been verified by Scott Zelligan, P.Geo., with an effective date of December 1, 2025.

14.1 Drillhole Database

Drill hole and sample database were provided in .csv format. Files provided include:

- Collar and Survey tables
- Assay Tables
- Lithological Tables

Wireframes were provided in .dxf (AutoCAD) and .dm (Datamine) format. Block models were provided in .dm format. The estimation was produced in Datamine software and validated in Leapfrog Geo software. All data has been confirmed by the authors of this report. Assay data includes diamond drill core and rotary drill chip samples. The database contains results from 1,619 drill holes, and the estimate includes results from 1,219 drill holes completed on the Property up to the date of December 1, 2025.

14.2 Interpretation

The preliminary mineral resource evaluation defined mineralized volumes using indicator variables rather than manually constructed wireframes. This methodology was employed to account for the variability within the geological domains hosting the gold and best represent the drilled distribution of gold within the system. An analysis of the gold grade distribution in the 5-foot composites demonstrated that a 0.1 gpt threshold clearly separates mineralized rock from waste. Consequently, indicator variables IND01, IND02, IND1, and IND2 were established, corresponding to 0.1, 0.2, 1.0, and 2.0 gpt grade thresholds, respectively. These thresholds support the delineation of mineralized envelopes suitable for evaluating both open-pit and underground mining scenarios.

14.3 Wireframing and Contact Profiles

As delineated above, wireframes were generated using indicator estimation methods. A contact analysis was conducted between the different mineralized domains defined by the indicator variables. A hard boundary was observed between domains, indicating that each domain must be estimated utilizing only the composites strictly contained within its respective volume.

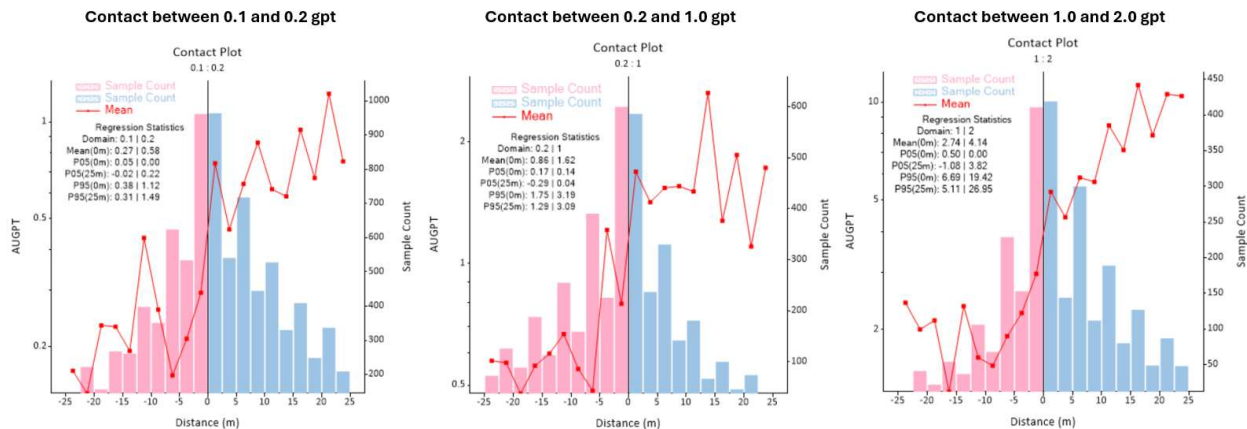


Figure 14-1 Contact Profiles

14.4 Exploratory Data Analysis

14.4.1 Compositing

Drill holes were composited to 5-foot interval lengths. 5 feet was the most common sampling interval. Additionally, an analysis comparing gpt grades and original sample lengths revealed that grades exceeding 30 gpt occur exclusively in samples shorter than 5 feet.

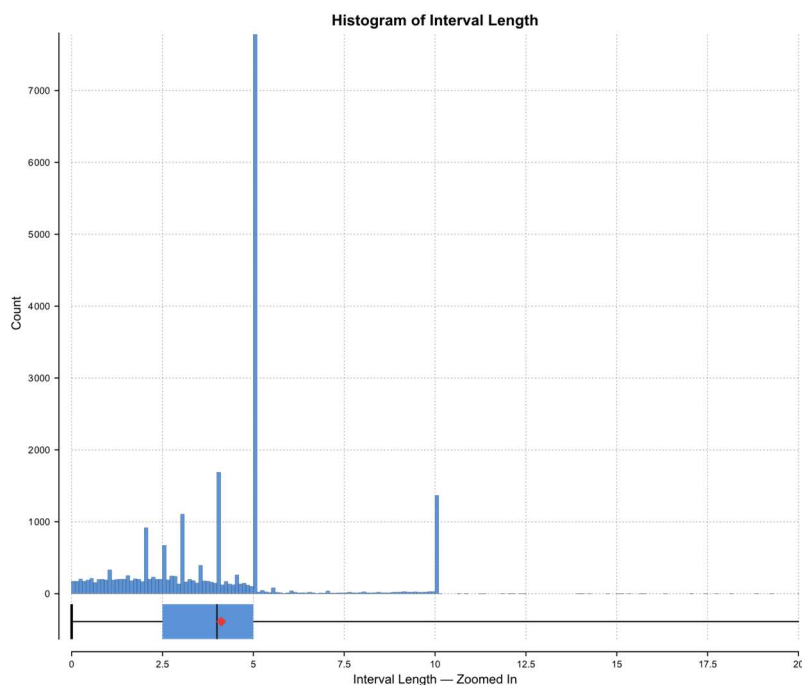


Figure 14-2 Interval Lengths

14.4.2 Outlier Management and Capping Strategy

Grade capping was employed to manage outliers. A capping analysis was performed on a per-domain basis utilizing probability-plot analysis and applying a 99% cumulative probability criterion.

- Domain 0.1 gpt: Capped at 2.0 gpt (Reduced Coefficient of Variation from 7.11 to 1.41).
- Domain 0.2 gpt: Capped at 6.7 gpt (Reduced CV from 4.27 to 1.61).
- Domain 1.0 gpt: Capped at 15.6 gpt (Reduced CV from 2.73 to 1.28).
- Domain 2.0 gpt: Capped at 66.2 gpt (Reduced CV from 3.95 to 1.49).

A new grade variable, AUGPTC, was created to incorporate these capped grades, successfully reducing the coefficient of variation (CV) to below 2 across the domains, making it suitable for the block model estimation.

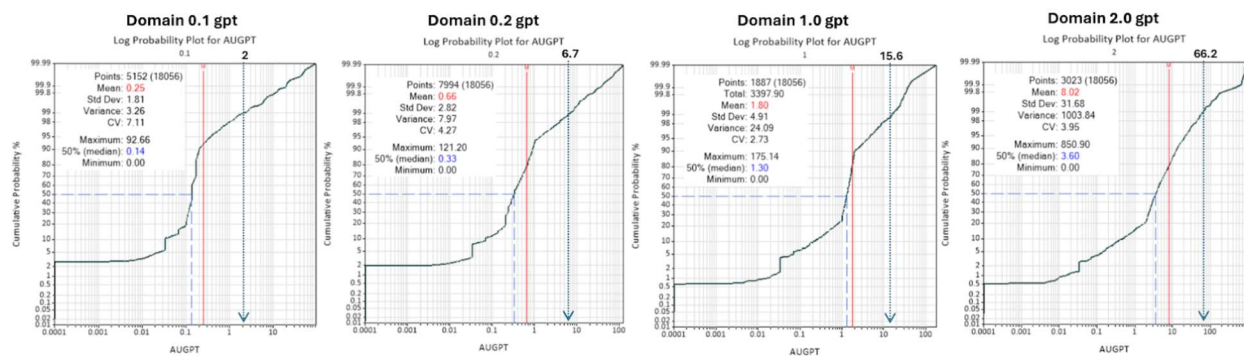


Figure 14-3 Log Probability Plots for Indicator Domains

Table 14-1 Raw Composite Statistics and Capped Composite Statistics

Field	Ore	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	COV
AUGPT	0.1	5152	0	92.66	0.25	1.81	7.11
AUGPT	0.2	7994	0	121.2	0.66	2.82	4.27
AUGPT	1	1887	0	175.14	1.8	4.91	2.73
AUGPT	2	3023	0	850.9	8.02	31.68	3.95
AUGPTC	0.1	5152	0	2	0.17	0.25	1.41
AUGPTC	0.2	7994	0	6.7	0.54	0.88	1.61
AUGPTC	1	1887	0	15.6	1.63	2.08	1.28
AUGPTC	2	3023	0	66.2	6.57	9.77	1.49

14.5 Density

Densities were applied to the block model by lithological type, utilizing provided lithology wireframes and provided historical technical reports. The specific gravity (SG) applied in short tons per cubic foot were:

- QAL: 0.054
- BSLT: 0.084
- FESTN: 0.091
- MTSD: 0.083
- PHYL: 0.083
- QLP: 0.084

14.6 Previously Extracted Material

The author was provided detailed wireframes of both the open-pit-minded surface extents, as well as the previously mined underground workings. Densities and tonnages were adjusted to zero in previously mined volumes to properly account for historical extraction. The models tracked historical open-pit blocks using the parameter ZONE=1, and historical underground workings using the parameter VOID=1.

14.5 Interpolation Plan

14.5.1 Block Model Parameters

A complete block model, inclusive of waste blocks, was generated to enable the assessment of waste-to-ore ratios for open-pit scenarios, as well as operational dilution for underground scenarios. The block model utilized a parent cell size of 20 ft by 20 ft by 5 ft. The model dimensions are defined in imperial units and rotated to facilitate direct use by mine engineers working with established parameters.

Table 14-2 Block Model Parameters

Axis	Origin (ULC)	Block Size (ft)	Sub-block Count	Minimum Block Size (ft)	Num Blocks	Boundary Size (ft)
X	331970.0	20	8	2.5	257	5140
Y	1047564.0	20	8	2.5	126	2520
Z	1710.0	5	Variable	0.005	306	1530
Dip Az	60					
Dip	35					

14.5.2 Search Parameters

The different indicators were estimated using inverse distance squared (ID2). Mineralized volumes were subsequently defined where blocks demonstrated a probability of being ore at or above 40%. Gold grades (utilizing the AUGPTC capped variable) were then estimated within each indicator-defined domain using ID2. Three search passes were applied during the estimation, guided by dynamic anisotropy defined by the previous mineralization wireframes:

- Pass 1: 82 ft by 82 ft by 19 ft, utilizing 4 to 9 composites from a minimum of two drill holes.
- Pass 2: 164 ft by 164 ft by 38 ft, utilizing 4 to 9 composites from a minimum of two drill holes.
- Pass 3: 820 ft by 820 ft by 190 ft, utilizing 3 to 12 composites from a minimum of one drill hole.

14.6 Resource Categorization

Mineral resource classification is the application of Measured, Indicated and Inferred categories, in order of decreasing geological confidence, to the resource block model. These are CIM definition standards (adopted by the CIM Council on May 10, 2014) for reporting on mineral resources and reserves, which are incorporated, by reference, in NI 43-101.

As per CIM (2014):

Measured 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.

Indicated 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.

Inferred 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.

These categories are applied in consideration of, but not limited to, drill and sample spacing, QA/QC, deposit-type and mineralization continuity. Mineral resources were preliminarily classified into Measured, Indicated, and Inferred categories using the following drill hole spacing parameters:

- Measured: Maximum drill hole spacing of an 82 ft by 82 ft grid.
- Indicated: Maximum drill hole spacing of a 164 ft by 164 ft grid.
- Inferred: Maximum drill hole spacing of a 200 ft by 200 ft grid.

14.7 Model Validation

A visual and statistical validation was completed to ensure that the estimated block model honors directional trends observed in the composites and that the block model is not over-smoothed or over- or under-estimated.

14.7.1 Statistics

As is typical in most estimates the grade and coefficient of variation between the estimate and the originating samples has lowered. This is common in part because sampling is inevitably clustered around high-grade areas, creating a bias in the input which is rectified geometrically in the estimation process. In general this model compares very favourably to the informing dataset.

Table 14-3 Block Statistics for Copperstone Mineral Resource

FIELD	ORE	NSAMPLES	MINIMUM	MAXIMUM	MEAN	STANDDEV	COV
AUGPT	0.1	546806	0.00	1.55	0.15	0.07	0.49
AUGPT	0.2	817936	0.01	5.57	0.46	0.27	0.59
AUGPT	1	146951	0.02	12.92	1.42	0.65	0.46
AUGPT	2	179169	0.33	65.00	6.14	4.54	0.74

Table 14-4 Statistical Comparison between informing Composites and Block Model

		Raw Composites		Capped Composites		Block Model	
FIELD	ORE	MEAN	COV	MEAN	COV	MEAN	COV
AUGPT	0.1	0.25	7.11	0.17	1.41	0.15	0.49
AUGPT	0.2	0.66	4.27	0.54	1.61	0.46	0.59
AUGPT	1	1.80	2.73	1.63	1.28	1.42	0.46
AUGPT	2	8.02	3.95	6.57	1.49	6.14	0.74

14.7.2 Population Distribution

Histograms are used to determine whether the population distribution has been accurately maintained in the estimation process. This ensures that the data has not been unnecessarily smoothed.

Histograms were reviewed for the overall captured sample population versus the overall estimated blocks. The population distributions compare favourably, with the normal reduction of the mean and shifting of the population to slightly lower grades.

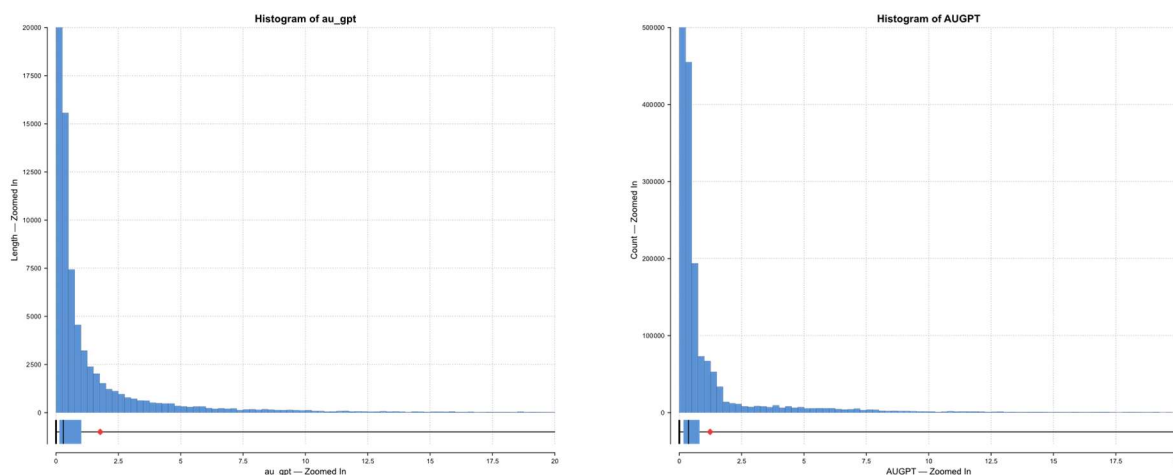


Figure 14-4 Au g/t Grade Distribution Comparison (Left: Composite Histogram Right: Block Histogram)

14.7.3 Sections

Sections confirm the correlation between drill results and estimated grades. Continuity seems logical and there are no glaring mismatches between drill hole grades and block model grades.

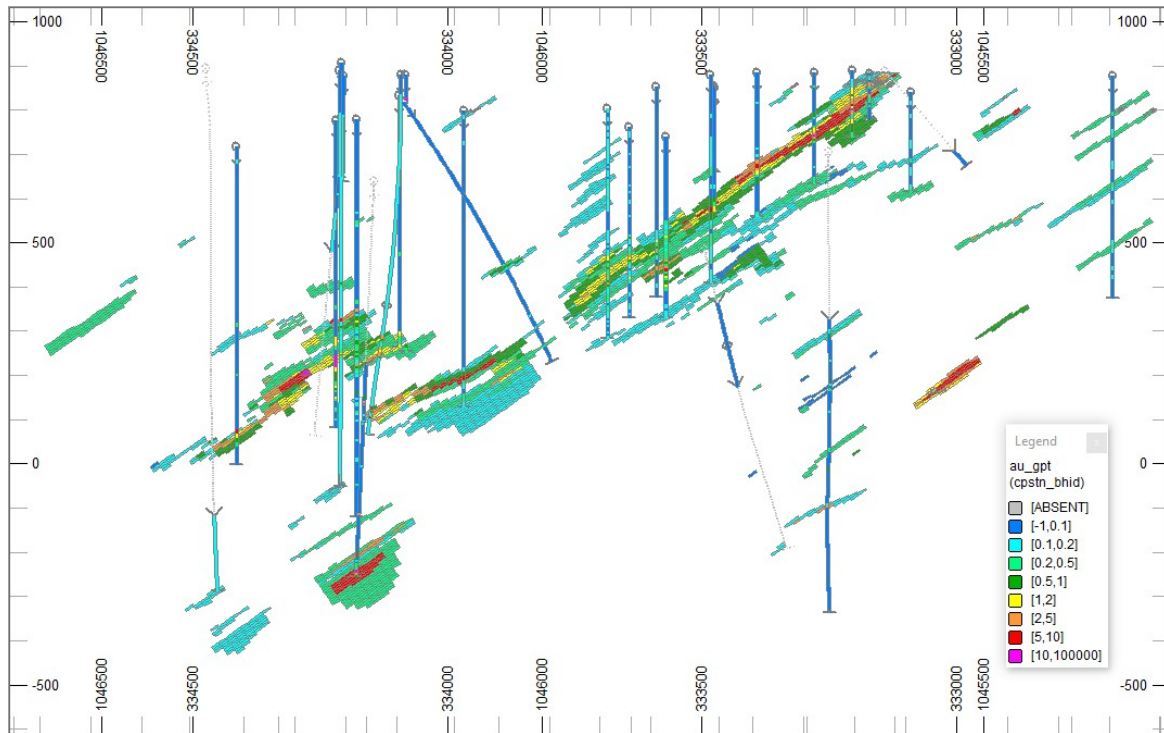


Figure 14-5 Rotated Section 1 (Azimuth 150 Dip -90)

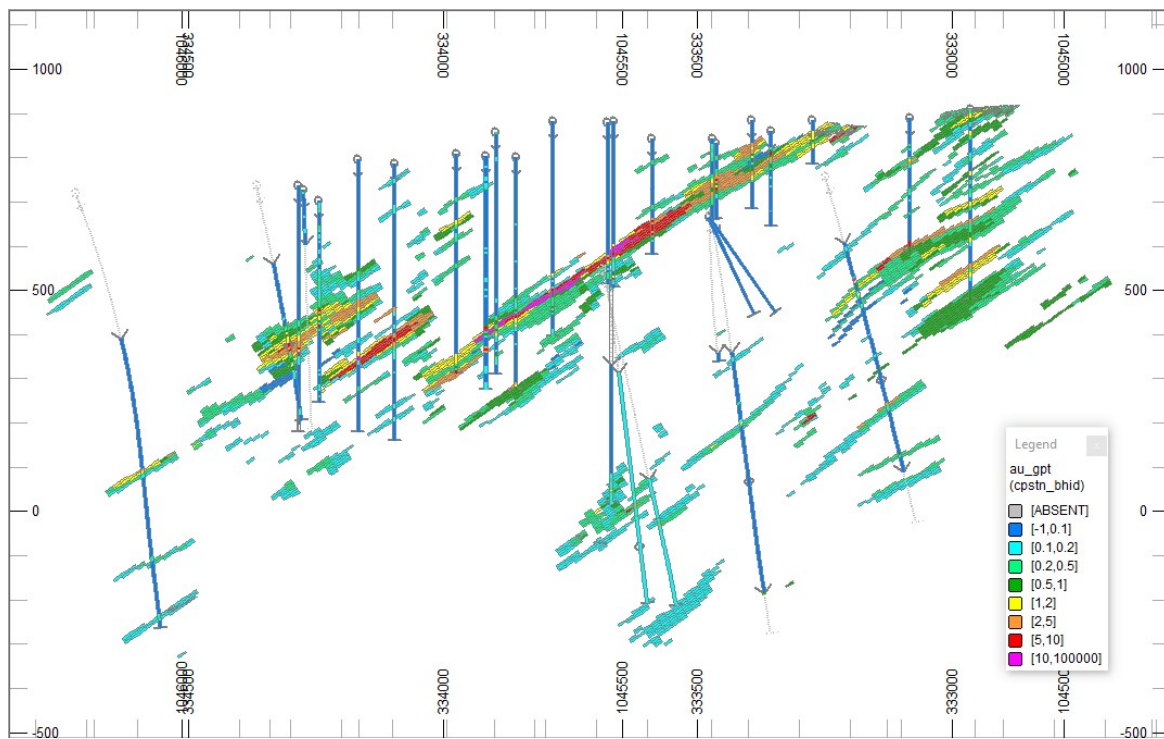


Figure 14-6 Rotated Section 2 (Azimuth 150 Dip -90)

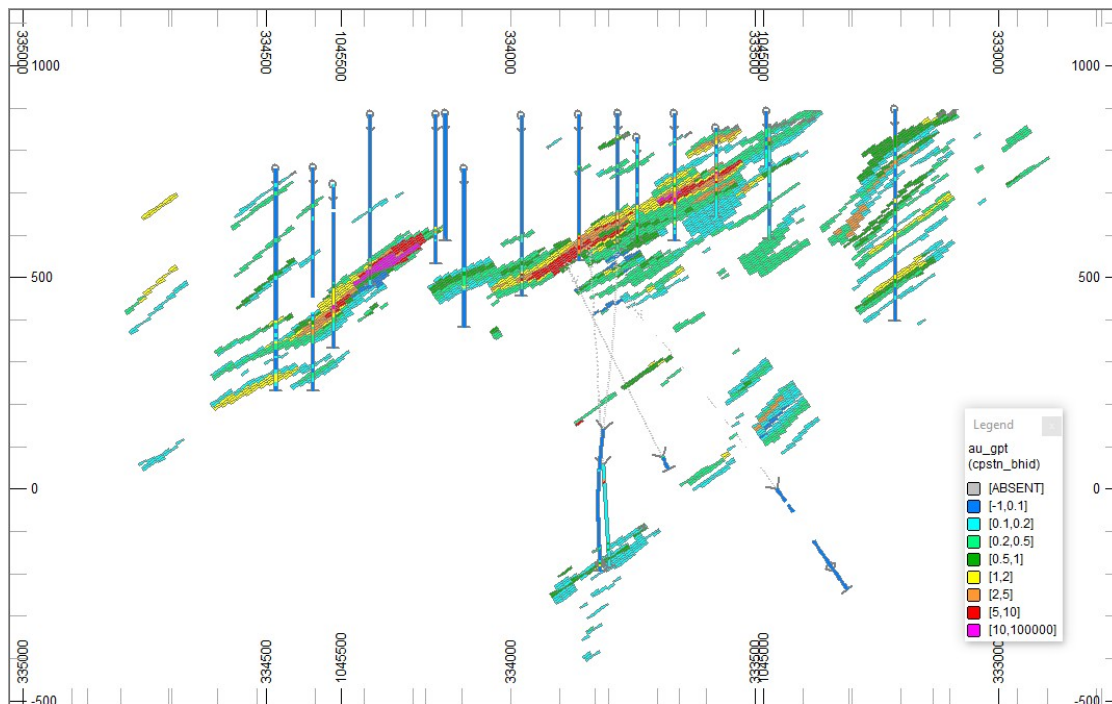


Figure 14-7 Rotated Section 3 (Azimuth 150 Dip -90)

14.8 Mineral Resource Reporting

The Copperstone mineral resource estimate (“MRE”) is presented in Table 14-4 below.

The MRE is reported at an underground mining cut-off of 0.064 oz/ton gold (2.2 g/t Au). Deswik Stope Optimizer was used as a guide to generate mineralized zones based on \$3,000/oz gold price and minimum mining widths and economic parameters consistent with the type of mineral system present at Copperstone.

Table 14-5 Mineral Resource Cut-off Parameters for Deswik Stope Optimization

Mineral Resource Cut-off		
Mining	\$/ore ton	\$ 102.87
Processing	\$/ore ton	\$ 42.64
G&A	\$/ore ton	\$ 16.39
Recoveries	ton	95.0%
Royalties	%	3.0%
Refining & Shipping cost	per/oz	\$ 12.00
Total cost	\$/ore ton	\$ 173.90
Gold Selling Price	oz	\$ 3,000.00
Minimum Mining Width	ft	9
Cut-off Grade	oz/ton	0.064

This Resource is considered by the author to have reasonable prospects for eventual economic extraction.

Table 14-6 Copperstone Mineral Resource

Classification	Tons (tons)*	Tonnes (tonnes)*	Au Grade (oz/ton)	Au Grade (g/t)	Contained Au (troy ounces)*
Measured*	2,423,000	2,198,000	0.15	5.08	359,000
Indicated*	2,046,000	1,856,000	0.13	4.54	271,000
M&I*	4,469,000	4,054,000	0.14	4.83	630,000
Inferred*	442,000	401,000	0.12	4.04	52,000

*Notes:

11. The effective date of the Mineral Resource Estimate is December 1, 2025
12. The Mineral Resource Estimate has been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) incorporated by reference into NI 43-101, and CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing or other relevant issues.
13. The mineral resource is reported at an underground mining cut-off of 0.064 oz/ton (2.2 g/t Au) within estimated blocks which meet the criteria of a minable shape (minimum mining width of 9ft). The cut-off is based on the following assumptions: a gold price of \$3,000/oz; assumed mining cost of \$102.87/ton (\$113.16/tonne), process costs of \$42.64/ton (\$46.90/tonne), general and administrative and property/severance tax costs of \$16.39/ton (\$18.03/tonne), refining and shipping costs of \$12.00/oz, a metallurgical recovery for gold of 95%, and a 4.5% gross royalty.
14. Resources are reported using at a minimum overall gold cut-off grade of 0.064 oz/ton Au (2.2 g/t Au). Internal dilution is incorporated into the MSO shape volumes as part of the estimated quantities and grades.
15. Previously mined areas were modelled based on all available records and were excluded from reporting of this mineral resource.
16. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Reserves are estimated from Measured and Indicated Mineral Resources. Inferred Mineral Resources are not included in the Mineral Reserves.
17. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and may be materially affected by modifying factors including but not restricted to mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.
18. There has been insufficient exploration to define the Inferred Resources tabulated above as an Indicated or Measured Mineral Resource, however, it is reasonably expected that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. There is no certainty that any part of the Mineral Resources estimated will be converted into Mineral Reserves.
19. Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.
20. Scott Zelligan, P. Geo. is responsible for reviewing and approving the Copperstone mine underground Mineral Resource Estimate. Mr. Zelligan is a Qualified Person ("QP") as set out in NI 43-101

15 MINERAL RESERVE ESTIMATE

The mineral reserves for the Copperstone underground gold project are estimated in accordance with the CIM Definition Standards. Measured mineral resources are converted to Proven reserves, while Indicated mineral resources are converted to Probable reserves. Inferred resources are excluded from the reserve calculation and are treated as waste.

15.1 Conversion Assumptions, Parameters and Methods

Deswik Stope Optimizer (Deswik SO) was used as a guide to define the economically extractable stope envelopes within the measured and indicated mineral resource. The optimizer output was refined to remove outliers and to merge areas together to create final cut-and-fill stope shapes. The following table summarizes the nominal stope dimensions.

Table 15-1: Deswik.SO Stope Shape Parameters

Parameter	Value
Stope height (ft)	10
Nominal width (ft)	14
Minimum width (ft)	9
Wall orientation	vertical (no shanty-backs)
Mining method	mechanized cut-and-fill (CRF/RF)

Dilution and Mining Recovery

The reserve tonnages incorporate a mining recovery of 95 % and external dilution of 10 % at zero grade, both consistent with industry experience for mechanized cut-and-fill operations in comparable geotechnical settings (average RMR76 of 45, Fair Rock classification; Langston and Associates, 2022). External dilution is applied first to the Deswik SO output to produce the reserve tonnage and grade reported in the reserve statement. The 95 % mining recovery is then applied to the contained metal to obtain the recoverable gold delivered to the processing plant. Internal dilution from sub-cut-off-grade material is inherent in the optimizer output and does not require a separate modifying factor. The modifying factors are summarized below.

Table 15-2: Recovery and Dilution Factors

Factor	Value	Application
Mining recovery	95%	Applied to contained metal in optimizer output
External dilution	10 % at zero grade	Applied to optimizer output
Internal dilution	Inherent	Embedded in optimizer stope shapes

Cut-off Grade Derivation

The following table summarizes the cost inputs used to derive the breakeven cut-off grade.

Table 15-3: Cut-Off Grade Input Parameters.

Cost Item	Value
Reference mining cost	US \$102.87 / ton mined
Processing cost (milling)	US \$42.64 / ton milled
G&A cost	US \$16.39 / ton milled
Total on-site operating cost	US \$161.81 / ton milled
Sales and refining cost	US \$12.00 / oz
Metallurgical recovery	95 per cent
Dilution	10 per cent at zero grade
Royalty and other gross proceed tax/payments	3 per cent of gross revenue
Gold price	US \$3,000 / oz

The breakeven calculation yields a cut-off grade of 0.0642 oz/ton, equivalent to 2.2 g/t Au. This cut-off grade is applied as an input parameter to Deswik SO to generate the economic shapes.

15.2 Mineral Reserve Statement

The following table presents the mineral reserve estimate for the Copperstone project as of [Effective Date].

Table 15-4: Mineral Reserve Statement

Classification	Tons (tons)	Tonnes (tonnes)	Au Grade (oz/ton)	Au Grade (g/t)	Contained Au (troy ounce)
Proven	1,159,000	1,052,000	0.15	5.08	172,000
Probable	972,000	882,000	0.14	4.61	131,000
Total (Proven & Probable)	2,132,000	1,934,000	0.14	4.87	303,000

Notes to Mineral Reserve estimate:

1. The effective date for the Mineral Reserve is December 1, 2025
2. The Mineral Reserve Estimate has been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) incorporated by reference into NI 43-101.
3. Costs were based on the operating costs ranges as outlined in Table 15-3.
4. Mineral Reserves were estimated using a gold price of \$3,000/oz, metallurgical recovery of 95%, and a cut-off grade of 0.0642 oz/ton Au (2.2 g/t Au). Mining recovery of 95% and external dilution of 10% at zero grade applied.
5. Inferred mineral resources are not mineral reserves and do not have demonstrated economic viability. Inferred resources are excluded from this reserve estimate and treated as waste.
6. Numbers are rounded to the nearest ton or ounce; rounding may cause minor discrepancies in totals.
7. Reserves are contained within the measured and indicated mineral resources. No additional material has been added.
8. Peter Szkilnyk, P.Eng., is responsible for reviewing and approving the Copperstone mine underground Mineral Reserve Estimate. Mr. Szkilnyk is a Qualified Person ("QP") as set out in the NI 43-101.

15.3 Factors That May Affect the Mineral Reserves

The mineral reserve estimate is subject to forward-looking assumptions regarding metal price, exchange rate, operating costs, metallurgical recovery, geotechnical behaviour, and regulatory conditions. Material changes to any of these assumptions could affect the tonnage, grade, or economic viability of the reserves. The QP is not aware of any legal, political, environmental, or other risks that could materially and adversely affect the potential development of the mineral reserves other than those described below.

- Metal price. Reserves are reported at a gold price of US \$3,000/oz. A sustained decline in the gold price below the breakeven threshold implied by the cut-off grade of 0.0642 oz/ton (2.2 g/t Au) would reduce the economically extractable portion of the resource.
- Operating costs. Reserves assume the mining, processing, G&A, and sales and refining costs disclosed in the Cut-off Grade Derivation subsection. Variation in diesel, labour, consumables, or contractor rates could change the breakeven cut-off grade and, in turn, the reserve tonnage.

- Metallurgical recovery. A metallurgical recovery of 95 % is assumed based on the test work disclosed in Section 13. Recovery shortfalls in production would reduce payable ounces without changing reserve tonnage.
- Mining recovery and dilution. Reserves apply 95 % mining recovery and 10 % external dilution at zero grade. Actual ground conditions, stope performance, or equipment selection may vary these factors.
- Geotechnical conditions. The Ground Control Management Plan (Langston and Associates, 2022) characterizes the rock mass as Fair (average RMR76 of 45). Localized variation in ground conditions could affect stope dimensions, dilution, or require additional ground support.
- Permitting and regulatory. The project operates under existing permits held by Mining Americas. Changes to permit conditions, reclamation requirements, or water-use allocations could affect the mine schedule.

16 MINING METHODS

16.1 Summary

The feasibility study underground mine plan for the Copperstone Project includes approximately 2,131,000 tons of mineralized mill feed to be extracted by underground mining in 7 years. The underground mine designs and schedule utilize measured and indicated mineral resources as part of the analysis. Mineral resources that are mineral reserves have demonstrated economic viability. The production schedule calls to produce 600 stpd for first 12 months and then ramping up to 1,000 stpd an annual production of 365,000 tons through the milling circuit. Mining recoveries of 95% were applied and dilution factors are calculated based on internal stope dilution calculations and external dilution factors of 10%. The mill feed will be placed on a stockpile at the crusher pad and a loader will be employed to feed the crusher at one twelve - hour shifts, seven days per week.

The mine plan for the mineralized material is based on the following criteria.

- Cut and Fill mining method using Rock Fill (“RF”) and Cemented Rock Fill (“CRF”);
- Cut-off grade of 2.2 g/t (0.0642 oz/ton) gold for underground mining;
- Production rate ramped up to 1,000 stpd;
- Development drifting and raising of approximately 124,031 ft for the Life of Mine (“LOM”);

The proposed mining method for the Copperstone Project is an overhand mechanized cut-and-fill approach utilizing cemented rock fill. This method was selected for its flexibility in efficiently extracting ore from zones with low vein dip angles. It also helps minimize dilution through precise geological control and disciplined mining practices.

Underground mining methods were evaluated with the objectives of minimizing dilution, capital and operating costs, and maximizing resource recovery while sustaining the planned mill throughput. The Copperstone mineralization is relatively shallow, with an average dip of approximately 38 degrees. Although certain zones exceed a 45-degree dip and can be mined using gravity-assisted methods, most of the deposit is too flat to be efficiently mined using longhole stoping.

The Copperstone Mine previously operated as an open pit from 1987 to 1993 under Cyprus, producing approximately 514,000 ounces of gold from 6,173,000 tons of mill feed at an average grade of 0.089 oz/ton Au.

In 2012, American Bonanza Gold Corp initiated underground mining via two declines developed from the base of the open pit, focusing primarily on the D Zone located north of the pit. Between January 2012 and July 2013, the operation produced approximately 16,900 ounces of gold from 163,000 tons of mill feed grading 0.104 oz/ton Au.

As a result of this historic underground mining and an exploration drift completed in 2017, approximately 12,800 feet of underground development access is currently in place. This infrastructure includes two declines extending from the pit, significantly reducing the amount of new development required to bring the mine into full production.

16.2 Geotechnical

Numerous geotechnical studies have been completed for the property, with the most recent conducted by Golder (2006), Call & Nicholas (2010), Langston and Associates (2012, 2021, and 2022), and Tierra Group International (2018). In January 2022, Langston and Associates prepared a Ground Control Management Plan (GCMP).

In 2018, Dr. Dermot Ross-Brown of Tierra Group International, Ltd. reviewed previous geotechnical work and conducted site visits to the existing underground workings to estimate ground support requirements and determine maximum stope opening dimensions for the mine plan. Dr. Ross-Brown completed three site visits to gain familiarity with the geology, assess underground conditions, and collect rock mass quality data in areas of potential stoping.

The primary tools used to assess rock quality were the Rock Mass Rating (RMR) system (Bieniawski, 1974; 1989; Gonzalez de Vallejo and Ferrer, 2011) and the Q-system (Barton, 2002; Norwegian Geotechnical Institute, 2015). Historically, limited emphasis has been placed on these classification systems in evaluating the stability of underground excavations. However, over the past two decades, both RMR and Q have become standard approaches in stope design. These methods have been referenced briefly in previous reports prepared for Copperstone.

Dermot Ross-Brown spent 5 days at the mine mapping RMR and Q in selected boreholes, with an emphasis on the potential mineralized zones and the 15 to 30 ft into the hanging wall and footwall of the intercept. Additional data was obtained from mapping exposures in the underground and in the pit. The observations of the four zones of mineralization for potential mining are described below:

- Zone A – Mostly found in the pit, this zone is very broken up, and mostly in a shear zone.
- Zone B – Competent, blocky rock mass with a well-defined jointing system that can be seen when coming up out of the pit near the maintenance office.
- Zone C – This is recognizable in the pit. Three exposures were visited underground, and one intersection seen in a borehole core.
- Zone D – This zone is only exposed in the historic underground workings and in the borehole cores.

Core and exposure mapping was carried out to obtain preliminary RMR and Q data to support stope design across all four zones. For each drillhole, Copperstone's geologist identified the mineralized interval, after which Tierra Group's engineer subdivided this interval, along with the hanging wall (HW) and footwall (FW), into geotechnical units—typically three to four per section—based on consistent geological and geotechnical characteristics.

RMR and Q logging were conducted for each unit, and the RMR values were averaged over the relevant intervals to estimate representative RMR values for the mineralized zone, as well as the HW and FW, for each borehole. A summary of the RMR data is presented in Table 16-1.

Table 16-1 Results of RMR By Zone

Code	Borehole No	Zone	Notes – Geology near mineralization	RMR76 HW	RMR76 Mineralized	RMR76 FW
20	KER-17U-30	B	phyllite/LS/QLP/fault	37	69	66
21	KER-17U-50	D	phyllite/LS/siltstone	40	40	NV
22	KER-17U-12	D	siltstone and sandstone/LS/phyllite	38	40	66
23	KER-17U-53	D	phyllite/faults/LS/phyllite	45	47	NV
24	KER-17U-69	D	QLP	73	NV	NV
25	KER-17U-06	D	D transitioning to Zone C; all QLP	33	57	38
26	CRD-04-07	C	All QLP	NV	50	NV
27	KER-17S-11	B	QLP/shear zone/phyllite	23	26	14
Average (for the eight boreholes)				41	47	46
Average Zone A				NV	NV	NV
Average Zone B				30	47	40
Average Zone C				NV	56	NV
Average Zone D				46	46	52
Notes: NV = No Value available; LS = Limestone; QLP = Quartz Latite Porphyry COPPERSTONE MINE – Estimates of RMR76 from the cores of eight boreholes, averaged over the width of the mineralized zone and up to 10 m either side into the HW and the FW. The recorded values assume DRY working conditions (this, in turn, assumes that the mine has been dewatered in the vicinity of the stopes prior to stoping). The averages are given for the different zones (A, B, C, and D). COPPERSTONE MINE – Estimates of RMR76 from the cores of eight boreholes, averaged over the width of the mineralized zone and up to 10 m either side into the HW and the FW. The recorded values assume DRY working conditions (this, in turn, assumes that the mine has been dewatered in the vicinity of the stopes prior to stoping). The averages are given for the different zones (A, B, C, and D).						

The average RMR values are in the range of 40 to 50, this is classified as ‘Fair Rock’ (Gonzalez de Vallejo and Ferrer, 2011). The averages reflect very little difference between the quality of the rock in the different zones. Both good and bad rock layers, intrusive weak and altered zones, and faults are seen in all four zones. Initial assumptions are that zones B and C may have better rock conditions than zones A and D, but the limited data available for zones B and C does not prove this at this time.

The Critical Span Design Curve (Paknalis, 2002 and Paknalis, 2014) is a useful chart for estimating the safe spans of stopes and other excavations for man-entry. Figure 16-1 illustrates RMR76 plotted along the x-axis and the design span plotted along the y-axis. The figure is based on numerous observations from excavations all over the world. The blue dots represent stable excavations while the red ones are unstable excavations. The blue-shaded zone is the transition zone between stable and unstable excavations.

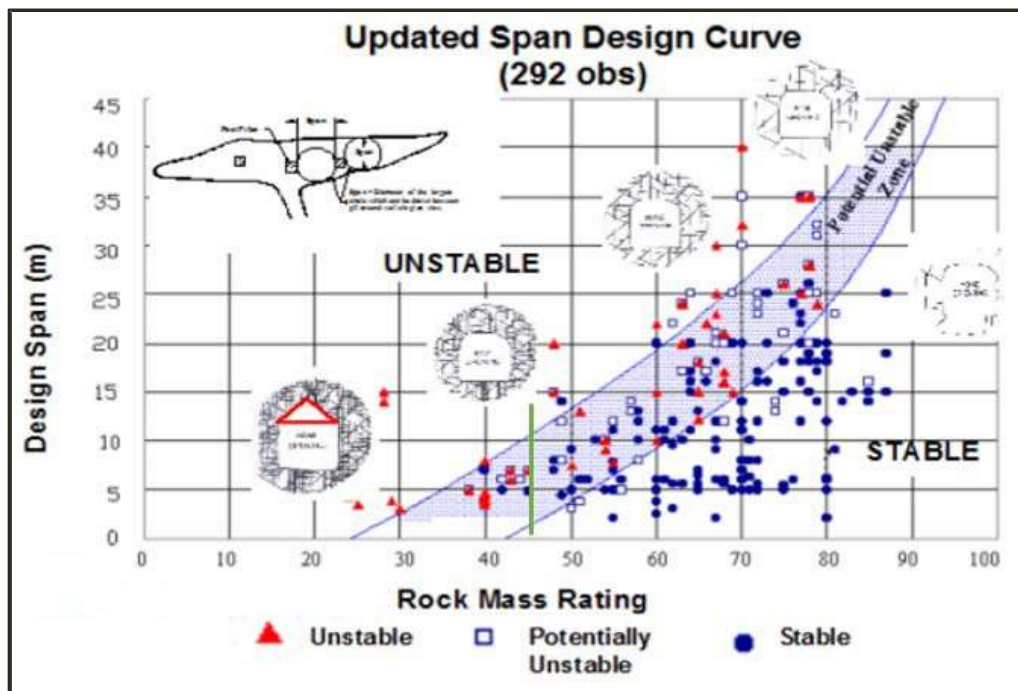


Figure 16-1 Critical Span Design Curve (Paknalis, 2002, updated 2014)

Note that the critical span is defined as the diameter of the largest circle that can be inscribed within the boundaries of an excavation. The referenced graph is based on short-term observations, typically ranging from three months to one year, and assumes either unsupported conditions or standard pattern bolting for local support (e.g., 1.8 m long bolts spaced at 1.2 m × 1.2 m).

In some cases, a correction is required for shallow jointing. Where joints are oriented nearly parallel to the back (less than 30°), a value of 10 should be subtracted from the RMR76 rating when using 1.8 m bolts installed in a regular pattern.

Subject to the limitations noted above, the estimated average RMR value of 45 for Copperstone is represented by a green vertical line on the graph, with a corresponding span of approximately 5 m (16 ft) falling within the transition zone. This indicates that spans of about 5 m are expected to remain stable in the very short term (i.e., one to two weeks), consistent with observations from existing drifts at Copperstone.

However, the risk of instability increases with larger excavation spans and delays in installing effective ground support. At Copperstone, excavations up to 5 m in width and height are expected to remain stable provided that local support is installed promptly. Stope length is less critical than width and height in this context.

From a design perspective, all excavations will require at least “normal support” in the back and, in some cases, partially down the walls. Normal support is defined as 1.8 m long bolts installed on a 1.2 m × 1.2 m pattern; split sets may be suitable where the excavation life is less than six months. Additional support, such

as mesh and occasional straps, will likely be required, and in poorer ground conditions, the use of steel fiber-reinforced (Sfr) shotcrete or shotcrete with steel mesh screen may be necessary.

Support requirements can be refined following each blast using the Q-system, as detailed by NGI (2015). For the PFS, it was assumed that 10% of development and stoping areas would require normal bolting (1.8 m × 1.2 m × 1.2 m), 70% would require bolting with mesh and occasional straps, and 20% would require bolting following the application of Sfr shotcrete or Sfr with steel mesh screen.

In 2021, Langston and Associates conducted a site visit and completed a stope span stability study. The analysis evaluated span increments of 15 ft and 25 ft for the back, and 15 ft and 30 ft along the hanging wall, to assess rock mass stability under various production heading geometries. The methods applied included two-dimensional finite element modeling, kinematic wedge analysis, and rock mass quality assessment.

Results indicated that back spans could be increased, limiting the need for multiple-pass mining to areas of poorer rock mass quality or unusually wide mineralized zones. The study also showed that cemented rock fill becomes load bearing after placement, contributing to deformation resistance and providing support pressure to the excavated openings.

At a 15 ft span in either the back or hanging wall, minimal rock mass yielding was observed in the back and none in the hanging wall. When spans increased to 25 ft in the back and 30 ft along the hanging wall, rock mass yielding remained minor, with displacements on the order of 2 to 3 mm. Evaluation of potential wedge failure and overall rock mass quality suggested that stability can be effectively managed by adapting standard development ground support practices to production headings.

16.3 Hydrological

In February 2010, Schlumberger Water Services (“SWS”) completed a Dewatering Evaluation report on the Copperstone deposit. The report provides an evaluation of the potential dewatering requirements for underground development at Copperstone. The description of the site hydrogeology and anticipated dewatering requirements below are partially excerpted from the SWS report.

16.3.1 Surface Water and Recharge

The Project is situated within the Tyson Wash watershed, with Tyson Wash located approximately five miles southwest of the mine site. No permanent or ephemeral surface drainage channels occur directly within the project area, although several small unnamed washes are present within the mine complex.

The existing open pit encompasses a catchment area of approximately 654 acres. In addition, two adjacent surface runoff catchments contribute drainage toward the mine area:

Catchment I – Located immediately north of the mine and covering more than 175 acres.

Catchment II – Located immediately south of the mine and covering more than 2,854 acres.

Average annual precipitation at the site is approximately 4.7 inches, with August accounting for an average of 0.84 inches. The nearest perennial surface water body is the Colorado River, located approximately nine miles west of the project area.

Based on regional studies, net groundwater recharge from direct precipitation infiltration is expected to be minimal, likely less than 0.2 inches annually. Assuming infiltration rates ranging from 0.5% to 3% of mean annual precipitation, estimated annual infiltration volumes are as follows:

- Pit Catchment (654 acres): Approximately 417,000 to 2.5 million gallons per year, equivalent to a continuous recharge rate of about 0.8 to 4.8 gpm.
- Catchment I (175+ acres): Approximately 2.85 to 17.12 million gallons per year, equivalent to a continuous recharge rate of about 5.4 to 32.6 gpm.

Stormwater ponding within the pit is not anticipated to be a significant concern due to the rapid infiltration observed from water previously pumped from the underground decline and discharged into the pit. However, this behavior indicates hydraulic connectivity with the underlying bedrock system and supports the need for an effective mine dewatering program.

16.4 Mine Dewatering

Planned mine development will extend from approximately 520 feet above mean sea level (amsl) to 120 feet below mean sea level, representing an additional depth of approximately 190 feet below the existing underground workings. At its deepest extent, mining will occur roughly 450 feet below the current groundwater table, corresponding to an estimated hydraulic pressure of approximately 210 psi.

Rock Quality Designation (RQD) data indicate an average classification of Fair Rock under dry conditions. If groundwater is not controlled, Rock Mass Rating (RMR76) values may decrease by approximately 8 points under saturated conditions and by as much as 15 points where flowing groundwater is present. Because rock masses with RMR76 values below 40 are generally considered difficult to mine and support, effective dewatering will be essential for maintaining ground stability and efficient mining operations.

The SWS hydrogeologic assessment concluded that the principal shear zone is relatively fractured and moderately permeable. The occurrence of warm groundwater suggests deep, undulating groundwater flow paths and localized zones of enhanced permeability. Structural boundaries within the system appear to limit regional recharge, indicating that dewatering inflows should stabilize or decrease over time.

SWS estimated mine dewatering requirements to range between approximately 150 and 300 gpm, including recharge contributions. To accommodate potential development-related inflow events and operational recirculation, additional contingency capacity has been incorporated into the design. Accordingly, the total installed pumping capacity is expected to range from approximately 300 to 450 gpm.

The existing dewatering system currently operates at an average rate of approximately 278 gpm. Water is pumped from the 810 Drift (elevation 160 ft) using twin 60-hp Tsurumi pumps through a 6-inch pipeline to the 730 Access Finger Cut sump (elevation 215 ft). Twin 150-hp Tsurumi pumps then lift the water through

a second 6-inch pipeline to a surface storage tank (elevation 590 ft). From there, a 200-hp Cornell pump transfers the water through a 10-inch pipeline to the tailings pond or designated water disposal area (elevation 894 ft). Localized dewatering of production stopes will be accomplished using portable 15- to 30-hp Tsurumi pumps.

Given current pumping rates and the fact that planned mining extends only approximately 190 feet deeper than existing workings, the SWS estimate of a 300 gpm dewatering system appears reasonable. While pump stations will require relocation as mining advances, significant increases in overall pumping capacity are not anticipated.

The mine plan also includes approximately 500 feet of additional development to construct a large water storage sump at the base of the C Zone. This facility will provide operational water storage and supplemental stormwater management capacity.

16.5 Mine Design

16.5.1 Stope Design

A range of underground mining methods was evaluated to identify an approach that minimizes dilution, capital requirements, and operating costs while maximizing resource recovery and maintaining the planned mill production rate.

The Copperstone mineralized zones are relatively flat lying, with an average dip of approximately 38 degrees. Although localized areas exceed 45 degrees and may permit limited gravity flow of broken material, the majority of the deposit is insufficiently steep to support an efficient long-hole stoping method. Consequently, a mechanized overhand cut-and-fill mining method utilizing both rock fill and cemented rock fill (CRF) has been selected for the Project.

Overhand cut-and-fill mining was chosen because of its flexibility and effectiveness in extracting mineralization at relatively low dip angles. The method also provides excellent control of mining dilution through selective extraction and close geological supervision. Mineralized stopes were designed using the Deswik Mineable Stope Optimizer (MSO), applying a gold price of US\$3,000/oz and a cut-off grade of 2.2 g/t Au (0.0642 oz/ton).

Mining will be conducted from a main ramp access. Initial stope development advances down-gradient to the ore-waste contact and then continues through the mineralized zone to establish the hanging wall contact. Once the hanging wall has been defined, longitudinal mining panels are developed perpendicular to the access drift and extracted along strike to the stope limits.

The stope design is based on a mining height of 10 ft, which was selected to minimize dilution, improve ground control, and accommodate the proposed mining equipment fleet. In narrower portions of the deposit, additional waste rock must be excavated along the hanging wall and footwall contacts to achieve complete ore recovery. While dilution generally increases with stope height, it decreases significantly as the width of the mineralized zone increases.

The width of the mineralized zones varies considerably throughout the deposit. Total stope widths range from approximately 9 ft to 102 ft, with an average width of approximately 30 ft. For stopes exceeding 16 ft in horizontal width, the mining span will be controlled by dividing each cut into multiple adjacent drift-and-fill passes. Following subdivision into individual mining passes, the average mining width is reduced to approximately 12 ft, providing improved ground control and enhanced selectivity while minimizing dilution.

16.5.2 Cut-off

The mine plan and stope designs were developed using Deswik's Mineable Stope Optimizer (MSO) based on the mining breakeven cut-off grade. The cut-off grade was calculated using a gold price of US\$3,000/oz, mining costs of US\$102.87/ton, processing costs of US\$42.64/ton, general and administrative costs plus property and severance taxes of US\$16.39/ton, refining charges of US\$12.00/oz, a metallurgical gold recovery of 95%, and a 3% royalty.

Based on these assumptions, the resulting breakeven cut-off grade is 0.0642 oz/ton Au, which was used to define mill feed material within the stope optimization and mine planning process.

16.5.3 Dilution

Mining dilution was incorporated into the stope designs through a combination of planned external dilution and calculated internal dilution. External dilution was applied at a rate of 10% and assigned a zero-grade value. In addition, the Mineable Stope Optimizer (MSO) was configured with a minimum mining width of 6 ft and a 1.5-ft dilution skin applied to both the hanging wall and footwall, resulting in a minimum stope width of 9 ft.

Internal dilution was calculated from material contained within the stope shapes that falls below the economic cut-off grade. A mining recovery factor of 95% was also applied to the stope tonnage and contained metal reported in the mine plan.

16.5.4 Development Design

The primary access ramp is developed within the footwall of the mineralized zones and serves as the principal access to the cut-and-fill mining areas. Main haulage drifts and ramps are designed with dimensions of 15 ft by 15 ft (4.6 m by 4.6 m) to accommodate 33-ton articulated haul trucks. Ramp gradients are limited to a maximum of 15% to ensure safe and efficient haulage operations.

The ramp layout minimizes sharp curves and turns to improve equipment productivity, traffic flow, and ventilation efficiency. Muck bays, approximately 30 ft deep, are strategically located adjacent to stope access points or at intervals of approximately 500 ft along the ramp to support development mucking activities. As mining advances, these excavations are repurposed for operational infrastructure, including sumps, transformer stations, storage areas, and exploration drill bays.

A new portal is planned at the base of the open pit during the second half of 2027 to provide primary access to Zone A. Existing drifts associated with the historic West and North portals, developed during previous mining and exploration programs, will be rehabilitated and incorporated into the main haulage network

during 2026. Utilization of these existing workings will reduce initial capital development requirements and accelerate access to the mineralized zones.

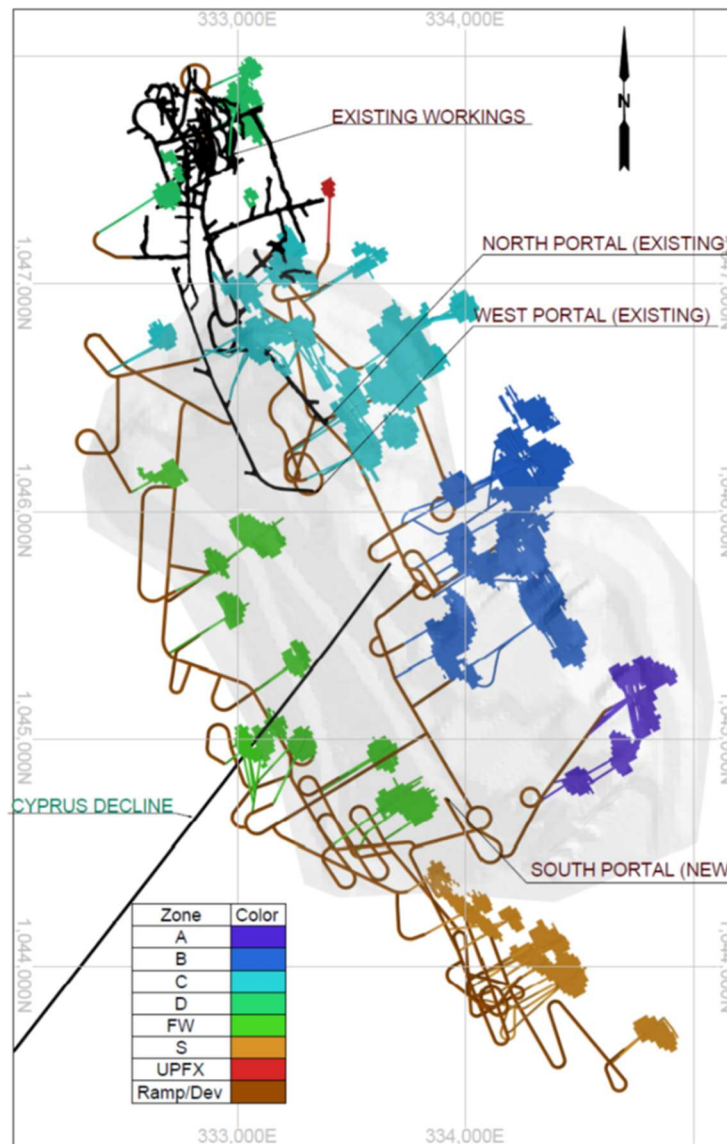


Figure 16-2 Underground Planned Development by Zone

The main haulage ramps are developed in the footwall and extend approximately 160 ft beyond the limits of the mineralization. Stope access ramps are designed at 10 ft by 10 ft (3.0 m by 3.0 m) to accommodate the selected jumbo drill and 3.9 yd³ load-haul-dump (LHD) equipment.

A nominal vertical level spacing of 60 ft has been adopted, allowing access to six 10-ft-high drift-and-fill mining lifts from a single access location. Access to the initial lift is established by advancing a ramp downward at a gradient of -15%. Subsequent lifts are accessed through a series of back-slashed ramps developed at a +15% gradient, providing efficient access to the remaining five mining horizons while minimizing development requirements.

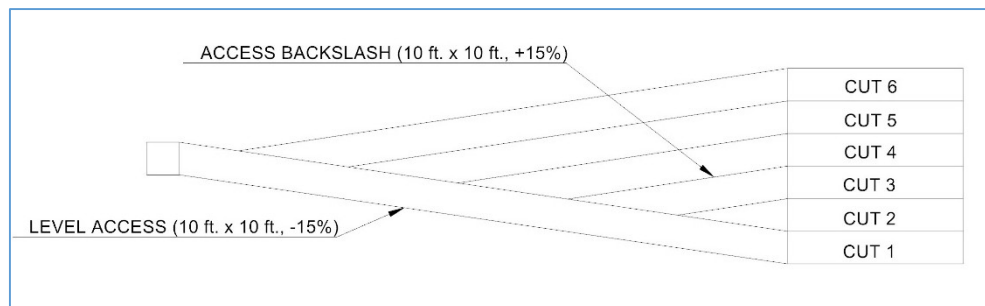


Figure 16-3 Stope Access Development

Stopes will be mined using a mechanized cut-and-fill method with cemented rock fill (CRF) and rock fill (RF), employing a sequence of 10-ft-high drift-and-fill lifts. The nominal stope dimensions are 10 ft in height and 14 ft in width, with a minimum mining width of 9 ft and near-vertical stope walls.

In areas where the mineralized zone exceeds 16 ft in width, extraction will be completed using multiple adjacent mining passes to maintain ground stability and control dilution. Following completion of each lift, cemented rock fill with an assumed bulk density of 1.9 t/m³ and a minimum 28-day unconfined compressive strength of 1 MPa will be placed to provide ground support, maintain stope stability, and allow safe extraction of subsequent lifts. Overall development schedule is summarized in table 16-2. Development quantities for the main haulage ramp include a 12 per cent growth allowance for muck bays and infrastructure cut-outs

Table 16-2 Development Footage

Activity	2026	2027	2028	2029	2030	2031	2032	2033	LOM Total
Main haulage ramp (15 x 15)	4,678	2,659	12,950	10,714	1,638	2,286	404	517	35,848
Stope access ramp (10 x 10)	3,381	2,885	5,225	8,685	3,754	4,715	359	679	29,683
Stope access back-slash (10 x 10)	130	4,936	10,884	13,563	11,958	10,284	4,115	2,630	58,501
Total	8,190	10,480	29,060	32,962	17,350	17,285	4,879	3,826	124,031

16.6 Production Schedule

The underground production schedule is built around an average mill feed rate of 600 ton/day in the ramp-up year and 1,000 ton/day from 2028 onward, following a mill upgrade. The schedule assumes 365 operating days per year with two 10-hour shifts per day on a four-on/four-off crew rotation. The production schedule was developed in Deswik.

16.6.1 Development Productivity

The development schedule is based on the advance rates summarized below. Development productivity assumptions are based on two 10-hour shifts per day, with the main haulage ramp achieving an average advance rate of 0.6 drill-and-blast rounds per shift.

Back-slash development achieves substantially higher advance rates by excavating from the upper face of the previously completed access ramp. This configuration enables continuous drilling, blasting, and mucking operations while improving equipment utilization. As a result, back-slash development can achieve approximately 31 ft of advance per round compared to approximately 9.3 ft per round for a conventional access heading.

16.6.2 Stoping Productivity

Each active stope face is expected to produce approximately 176 tons per day based on a mining cycle operating over two 10-hour shifts and completing one round per day. To achieve the planned production rate of 1,000 tons per day, six to seven active stopes will be mined concurrently. Maintaining multiple active production faces provides operational flexibility and accommodates backfill curing requirements, development delays, and other unplanned interruptions.

The nominal stope geometry consists of a 10-ft-high by 14-ft-wide heading. The mining cycle includes jumbo drilling, ANFO blasting, mucking with a 3.9 yd³ load-haul-dump (LHD) unit, haulage using 33-ton articulated trucks, and installation of required ground support. Overall cycle productivity is primarily constrained by bolting activities.

16.6.3 Annual Production Schedule

The production ramp-up period during 2026 and 2027 is focused on advancing the main haulage ramp, establishing critical mine infrastructure, and completing initial stope access development. Ore mined during 2026 will be stockpiled at the ROM stockpile close to the crusher and processed following a planned three-month mill commissioning and ramp-up period.

From 2028 through 2031, the operation is scheduled to achieve and maintain full production rates, with six to seven active stopes operating concurrently and peak mine production reaching approximately 1,148 tons per day. During this period, underground development will continue at a reduced rate to provide access to future mining areas and sustain production as completed stopes are backfilled and returned to the mining cycle.

Table 16-3 LOM Production Schedule

	2026	2027	2028	2029	2030	2031	2032	2033	LOM Total
Ore (tons)	47,619	276,505	411,543	419,032	414,340	407,245	137,757	17,586	2,131,627
Grade (oz/ton)	0.1344	0.1938	0.1523	0.1417	0.1232	0.1214	0.1276	0.135	0.1419
Contained Au (oz)	6,401	53,597	62,658	59,382	51,057	49,452	17,584	2,373	302,505
Waste (tons)	112,799	98,645	350,771	362,203	144,569	155,192	40,446	34,467	1,299,093
Development (ft)	8,190	10,480	29,060	32,962	17,350	17,285	4,879	3,826	124,031

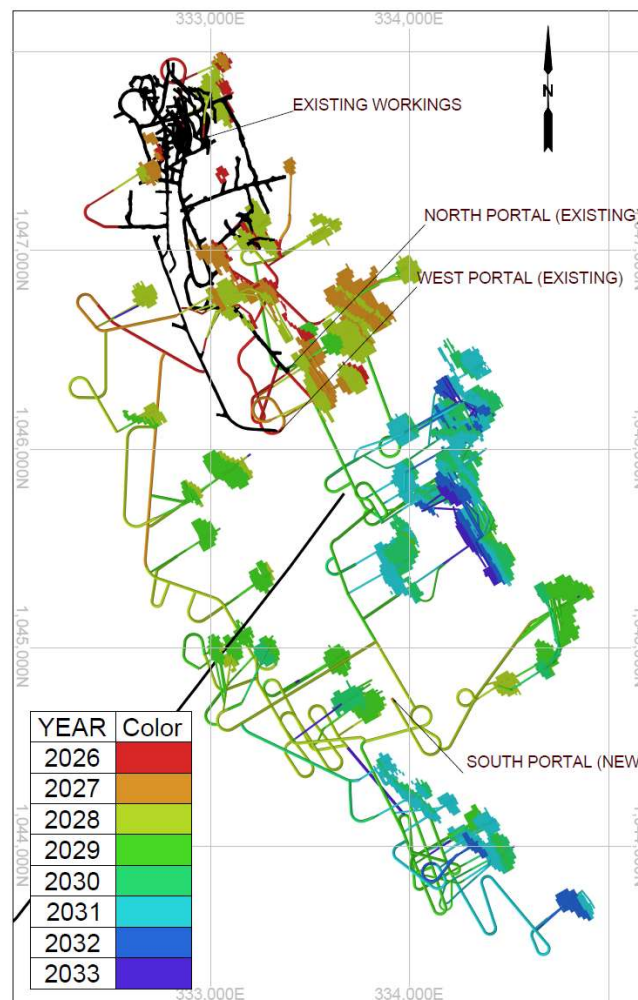


Figure 16-4 Schedule by Area

16.6.4 Backfill

The primary objective of the backfilling strategy is to achieve a consistently tight fill throughout wider cut-and-fill areas, particularly where multiple adjacent drifts are mined on the same lift. Inadequate fill in these areas could result in increased effective spans and a higher risk to stope back stability.

To ensure a uniform and tight fill, the mine design incorporates both cemented rockfill (CRF) and non-cemented rockfill (RF) methods. CRF is placed using a 3 m³ LHD equipped with a rammer, allowing for thorough compaction and tight filling along the full length of the stope. Backfill placement is scheduled at an average rate of 393 yd³ per day per stope. Considering that multiple stopes may be in the backfilling phase simultaneously, the overall average backfill rate over the life of the mine is estimated at 393 yd³ per day.

In 2021, MineFill Services conducted a study that included laboratory testing of aggregate materials to develop an appropriate backfill mix design, along with an evaluation of suitable backfilling methods given the challenging ground conditions and proposed stope geometries. Based on this work, MineFill established the following general specifications for CRF.:

- Aggregate intact rock density – 2.6 tonnes/m³
- CRF density – 1.9 tonnes/m³
- Aggregate top size of 6-inches
- Ideal coarse to fines – 60/40
- Water to cement ratio of 0.8 for jam fills
- Nominal binder content of 3% to achieve 1 MPa at 28 days.

A backfill cement slurry and screening plant is planned within the pit to process waste rock from both run-of-mine development and material stockpiled from previous open pit operations. This material will be screened and blended with slurry to produce cemented rockfill (CRF) for stoping areas requiring support.

All backfill will be placed by end-dumping and compacted tightly against the back or hanging wall, minimizing the risk of aggregate segregation. Nevertheless, MineFill recommends limiting fragment sizes to a maximum of 6 inches, as larger material can negatively affect backfill density, stiffness, and strength.

Although a strict aggregate grading specification is not required and a relatively coarse top size is acceptable, it remains important to ensure that the CRF contains an adequate proportion of fines (particles smaller than 10 mm). Without sufficient fines, the backfill may contain excessive void space, resulting in reduced strength.

The selection of suitable rock types for CRF aggregate is also critical. In general, phyllite should be avoided, while granite has been shown to be appropriate, as confirmed through laboratory testing of CRF mixtures.

16.6.5 Surface Mill Haulage

Mill feed haulage from the bottom of the open pit to the mill feed stockpiles is planned using a 4 m³ front-end loader and two 40-tonne articulated haul trucks. The loader will also be utilized to load underground haul trucks when backfill material is sourced from the open pit waste dump stockpile.

Surface haulage operations are based on two crews working a four-on, four-off rotation, with 10-hour day shifts and no night shift. Under these assumptions, two haul trucks are sufficient to meet the mill feed requirements.

16.7 Mine Ventilation

16.7.1 Existing System

The existing mine ventilation system utilizes a **150 hp Jet-Air** primary fan and a **100 hp Spendrup** fan to deliver fresh air through the West Portal. Intake air travels down the West Decline to the junction of the D Zone declines, West Decline, and North Decline, where the primary airflow is directed up the North Decline and exhausts through the North Portal.

At this intersection, a **50 hp** auxiliary fan directs fresh air through ventilation ducting into the D Zone, supplying the 730 Access Drift intersection. A **25 hp** auxiliary fan provides ventilation to the dewatering station located in the 730 Hangingwall Decline.

Ventilation to the C Zone is provided by a **100 hp Spendrup** fan, which forces air through ducting to the exploration drift developed during the summer of 2017.

The existing ventilation circuit has been measured to provide airflow of up to **110,000 cfm**, which is adequate for current development activities but will require expansion to support planned production levels.

16.7.2 Ventilation System Upgrade

The total mine ventilation requirement at the planned production rate of 1,000 stpd is estimated at approximately 200,000 cfm, as summarized in Table 16-4.

The proposed ventilation upgrade includes the construction of a dedicated fan drift and airlock system to establish an efficient flow-through ventilation circuit. The design consists of approximately 160 ft of new drift development from the West Decline, where a bulkhead and a 150 hp primary ventilation fan will be installed. This system is designed to deliver approximately 200,000 cfm at 4 to 8 inches water gauge (wg). As mining production expands, additional ventilation capacity and fan upgrades will be required to meet increasing airflow demands.

An airlock system, consisting of two Hoffman bi-fold doors equipped with PLC controls, electronic safety sensors, and hydraulic actuators, will be installed in the Main Decline (Figure 16-9). The airlock will maintain a positive ventilation seal while allowing uninterrupted movement of personnel and equipment through the intake airway.

Following commissioning of the new ventilation system, the temporary fans, ducting, and roll-up door at the West Portal can be removed, allowing the portal to serve as the primary haulage access. The North Portal will continue to function as the primary exhaust, while the A Zone, Cyprus Decline, and future return air raise will provide additional exhaust capacity for mining operations in the A, B, South, and Footwall zones.

As discussed previously, a ventilation raise is planned to deliver fresh intake air from the West Decline to the lower levels of the C Zone. Air will then travel through the existing exploration drift before exhausting through the North Decline, providing an efficient ventilation circuit for the expanding mine.

Additional underground ventilation infrastructure will be required to distribute airflow throughout the active mining areas. During the initial capital phase, 100 hp, 42-inch booster fans will be installed to maintain adequate airflow as mine development progresses. Smaller auxiliary ventilation systems, consisting of 75 hp stope fans and flexible ventilation ducting, will deliver fresh air directly to active working faces. Up to six auxiliary stope fans are expected to operate simultaneously during peak production.

The ventilation upgrades described above have been incorporated into the underground mine development and capital execution schedule and are expected to provide sufficient airflow to support the planned production rate while maintaining flexibility for future mine expansion.

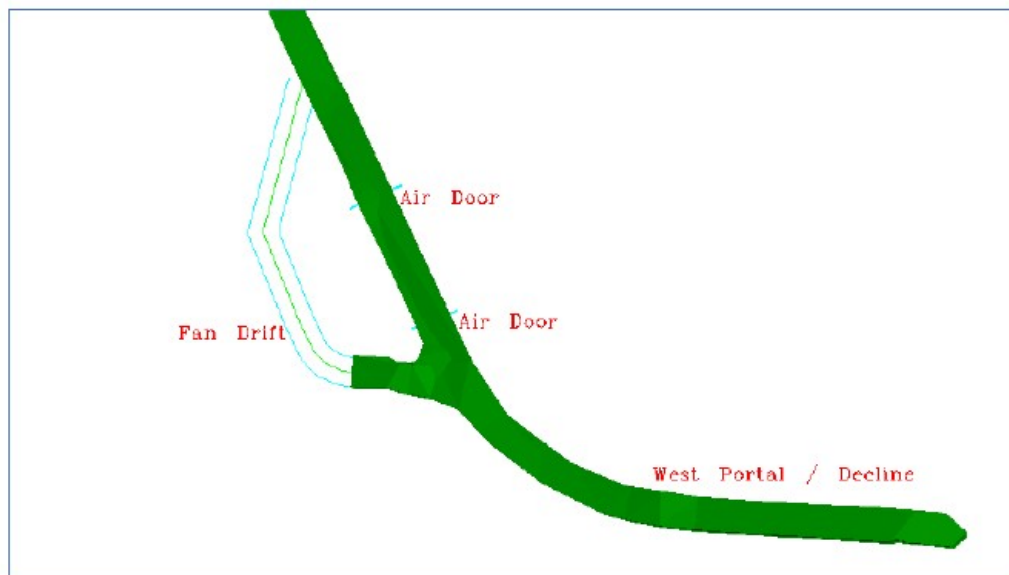


Figure 16-5 West Decline Fan Drift

Table 16-4 Ventilation Airflow Requirements

Manufacturer / Model #	Equipment Type	Engine Manufacturer	Power HP	Utilization Factor	CFM	Utilized CFM
Normet	ANFO Truck with Boom	Cummins	161	20%	13600	2720
Sandvik DD212i	Single Boom Jumbo	Deutz	74	10%	8800	880
Sandvik DD322i	Two Boom Jumbo	Deutz	134	10%	5100	510
Sandvik DD322i	Two Boom Jumbo	Deutz	134	10%	5100	510
Caterpillar R1300G	Underground LHD	Caterpillar	175	50%	13600	6800
Caterpillar R1300G	Underground LHD	Caterpillar	175	20%	13600	2720
Caterpillar R1700	Underground LHD	Caterpillar	345	50%	16200	8100
Caterpillar R1700	Underground LHD	Caterpillar	345	50%	16200	8100
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	80%	22000	17600
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	80%	22000	17600
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	80%	22000	17600
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	80%	22000	17600
Getman A64	U/G Fuel/Lube Truck	Mercedes	173	20%	7900	1580
Caterpillar D4K2 XL	Dozer with Jammer	Perkins	110	50%	11800	5900
Cat 120 Grader	Underground Grader	Caterpillar	110	20%	11800	2360
Cat 908 IT	UG IT	Caterpillar	75	25%	8000	2000
Cat 908 IT	UG IT	Caterpillar	76	25%	8000	2000
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Kubota RTV XG850		Kubota	54	20%	6300	1260
Total						124,660
Contingency						27,425
Grand Total						152,085

16.8 Electrical Services and Compressed Air

Bruno Engineering completed a review of the existing electrical system and identified the upgrades required to support a production rate of 1,000 stpd. Based on projected power demands, the main substation will be upgraded to a 15/20 MVA oil-filled transformer, along with the installation of a new capacitor bank and circuit breaker. Additional upgrades include a new motor control center, switchgear, overhead power line, and infrastructure to support new underground feeders.

For the ventilation system, a new PLC-based automation control system is planned, along with a new fan motor starter and feeder. The design also includes two new power centers, complete with switch houses and motor starter assemblies. New underground 15 kV Type MP-GC cables will supply power to these centers, along with additional feeder cables for jumbos, inline and stope fans, dewatering pumps, and face lighting.

Compressed air will be required in both development and production headings to operate jumbos, handheld jackleg and stoper drills for ground support installation, as well as for general utility needs throughout the mine. The existing surface compressor system will be utilized to meet these demands. The current unit, an Atlas Copco GA315 VSD (381 hp), is rated to deliver 1,394 ft³/min at 150 psi.

16.9 Mining Equipment

The mine plan is based on contract mining for the initial pre-production development and then for the first twelve months of production during Year 2. In month eight of Year one a fleet of underground mining equipment is planned to be purchased so the mine can transition to owner mining. The planned mine mobile equipment when the mine transitions to owner mining will consist of the diesel-powered equipment shown in Table 16-5.

A two-boom jumbo drill is planned for main ramp development, while two single-boom jumbos will be used for stope access and stope development headings. The electric/hydraulic drills will be operated by a single operator, with assistance from the mining crew during setup and teardown in each heading.

A mixed fleet of four 3.9 yd³ LHDs and two 8 yd³ LHDs is planned for development and mill feed production headings. In addition, two smaller 1.7 yd³ LHDs will be utilized in narrow mill feed headings to minimize dilution. The LHDs will muck development and production headings to the designated muck bay for each area. Once mucking is complete, the bolting process can begin. Material stored in the muck bays is then loaded onto haul trucks for transport either to the surface mill feed stockpile or to designated underground backfill areas. Development waste will be stockpiled in muck bays near production zones and later used for cemented rock fill (CRF), optimizing turnaround time in the headings. A fleet consisting of two 45-ton trucks and two to three 35-ton ejector box trucks will be required to haul mill feed to surface and transport waste material to the backfill areas as needed. The larger trucks will primarily be used for ore haulage and waste development, while the 35-ton ejector box trucks will be dedicated to backfill placement and waste movement.

Support equipment will consist of two bolters, two scissor lifts for hanging utilities, boom truck, lube/fuel truck, shotcrete sprayer, underground road grader, explosives truck, four buggy vehicles and a pallet handler.

As discussed previously a 5 yd³ loader will be located in the pit bottom to load mill feed into two 40 tonne articulated dump trucks for transportation to the mill. There is also a small track dozer planned to help maintain the stockpiles on surface.

Fixed capital equipment required for the mine will consist of the ventilation system upgrade, electrical system upgrade, expanded mine dry, CRF (cemented rock fill) binder plant, compressor and the mine dewatering system.

Table 16-5 Mining Equipment

Manufacturer / Model #	Equipment Type	Engine Manufacturer	HP	KW	Engine Type
Normet	ANFO Truck with Boom	Cummins	161	120	Diesel
Sandvik DD212i	Single Boom Jumbo	Deutz	74	55	Diesel
Sandvik DD322i	Two Boom Jumbo	Deutz	134	100	Diesel
Sandvik DD322i	Two Boom Jumbo	Deutz	134	100	Diesel
Caterpillar R1300G	Underground LHD	Caterpillar	175	130	Diesel
Caterpillar R1300G	Underground LHD	Caterpillar	175	130	Diesel
Caterpillar R1700	Underground LHD	Caterpillar	345	257	Diesel
Caterpillar R1700	Underground LHD	Caterpillar	345	257	Diesel
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	446	Diesel
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	446	Diesel
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	446	Diesel
Caterpillar AD45	U/G Articulated Haul Truck	Caterpillar	598	446	Diesel
Getman A64	U/G Fuel/Lube Truck	Mercedes	173	129	Diesel
Caterpillar D4K2 XL	Dozer with Jammer	Perkins	110	82	Diesel
Cat 120 Grader	Underground Grader	Caterpillar	110	130	Diesel
Cat 908 IT	UG IT	Caterpillar	75	56	Diesel
Cat 908 IT	UG IT	Caterpillar	76	57	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel
Kubota	RTV XG850	Kubota	54	40.3	Diesel

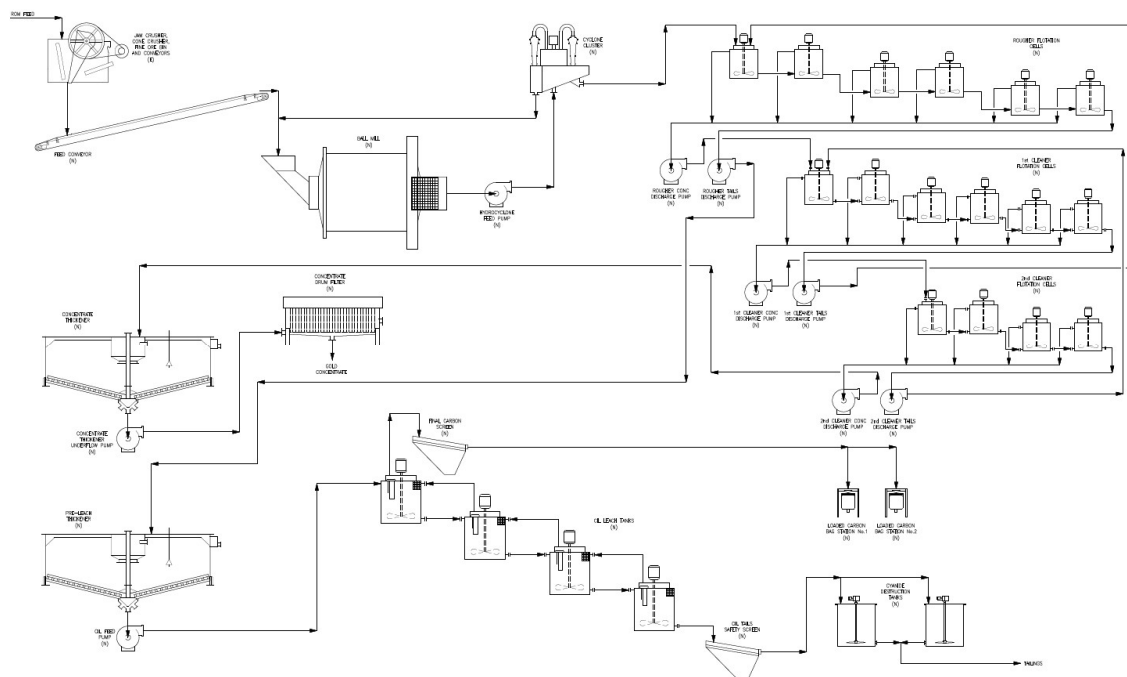
17 RECOVERY METHODS

17.1 Process Development

Process selection has been based on historical metallurgical testing for the Copperstone deposit as outlined in Section 13. The selected process flowsheet (see Figure 17-1) includes a crushing and grinding circuit with flotation that will produce a high-grade gold concentrate for off-site processing and product sales. The second circuit incorporated into the processing facility after the flotation is a bulk cyanide leaching (CIL) of the flotation tailings. The CIL plant will produce loaded carbon that will be screened and bagged to be processed offsite at the Company's Nevada operations site (Pan Mine ADR).

A significant amount of prior metallurgical testwork was completed for the Copperstone project and reviewed by the current authors. The testwork results for the current flowsheet indicate an overall combined gold recovery of approximately 96%, with the concentrate from the gold flotation circuit accounting for 75-80%, and the remaining gold in the concentrate tailings transferring to the CIL for final residual gold recovery of 85-90% (from flotation tailings). Based upon economic and technical analysis, the combined gold extraction method with flotation and tailings leaching was chosen as the best processing option for the Copperstone deposit. The proposed "hybrid" process offers relatively low operating costs compared with other processing options previously considered for the mine as well as maximum flexibility to accommodate expected variations in the deposit mineralogy. In addition, the elimination of on-site ADR, gold smelting, and refining facilities significantly reduces initial capital expenditures. By transporting gold-bearing products to off-site facilities for further processing, the project can leverage existing Company infrastructure and processing capacity, resulting in a cost-effective reduction in both capital and operating requirements while maintaining operational efficiency.

Figure 17-1 Flotation and Tailings Leach Process Flow Diagram



The proposed flowsheet takes advantage of the existing primary crusher, secondary crusher, screen, ore bin, and conveyors. This equipment is currently being refurbished by the company. The remaining processing plant will be reconstructed using equipment sourced from suppliers to provide an initial throughput capacity averaging 600 tons per day with the capacity to expand up to 900 tons per day once operations are underway. The existing ball mill will be removed and replaced with a larger mill required for the target throughput and target grind size to the flotation circuit. The design implements a pre-leach thickener followed by 4 leaching tanks providing 24 hours of total residence time with space to allow for additional expansion in the future. The loaded carbon product will be batch screened, separated and bulk-bagged to be processed off-site. The new “hybrid” plant and the carbon recovery process will be installed as per the equipment listed in Tables 17-1.

17.2 Process Plant Equipment

The list of new and existing equipment required for the flotation and tailings leach process is given in Table 17-1.

Table 17-1 List of Major Process Equipment

Major Equipment List		
Primary Jaw Crusher	150 HP 32 inch X 40inch	existing
Secondary Cone Crusher	300 HP 4 ft Symons	existing
Triple Deck Screen	40 HP 6 ft X 16 ft	existing
Fine Ore Bin	500 tons	existing
Ball Mill	11ft x 15ft 1000 HP	refurbish
Hydro-cyclones	2x 15 inch	new
Rougher Flotation	1200 cuft	new
Cleaner 1	400 cuft	new
Cleaner 2	200 cuft	new
Concentrate Thickener	15 ft	new
Concentrate Filter	150 ft2	new
Tailings Pre-Leach Thickener	30 ft	new
CIL Tanks (4)	22X 28ft	new
	75,000 gal (each)	new
Cyanide Destruction (2)	12 x 15 ft	new
Loaded Carbon Safety Screen	6 x12 ft	new
Lime Tank (Mixing)	10x 12 ft	new

17.3 Process Description

The Copperstone Mine will restart the process plant using existing infrastructure from previous operations where available including the existing crushing plant which is currently being refurbished (see Table 17-2). Run-of-mine (ROM) ore is delivered to the jaw crusher feed hopper either directly from the mine or by loader from stockpiles located close to the crushing plant. Jaw crusher discharge reports to a crusher screen operating in closed circuit with a cone crusher; screened product is conveyed to a crushed ore bin, from which a conveyor delivers ore to an 11' x 15' primary ball mill fitted with a 1000 horsepower motor. The ball mill operates in closed circuit with the hydro-cyclones, reducing the ore to a maximum nominal P_{80} of 100 - 125 μm (120 -150 mesh). The process plant will operate with an overall design availability of 92% availability.

Cyclone overflow feeds a conditioning tank and then a flotation circuit consisting of roughers and two stages of cleaning, producing a high-grade gold concentrate with a mass pull of up to 4%. The gold concentrate is dewatered in a thickener, with the thickener underflow at 50% solids pumped to plate-and-frame filter for discharging to a concrete pad for final bulk container loading and shipment off-site. Flotation concentrates are shipped offsite to a third-party processor for toll processing and gold recovery.

Flotation tailings report to a tailings pre-leach thickener to remove excess water ahead of cyanide leaching, with thickener underflow pumped at 45% solids to the carbon-in-leach (CIL) tanks. Slurry is combined with cyanide solution pumped from the pre-mixed reagent holding area. Dissolved gold is recovered through counter-current solution flow and intermittent carbon advancement in the CIL tanks. Mixing interstage screens separate slurry from the coarser carbon while allowing solution contact for loading. Loaded carbon is screened, drained, and bagged for stripping and refining off-site. Leached tailings are pumped to be detoxified (cyanide destruction via SMBS and aeration) prior to being discharged to the existing tailings storage facility (TSF). Recovered process solution (water) is returned to the plant.

Table 17-2 Process Design Criteria

Process Design Criteria		
Process Plant Operations	24/7	Hours/ Days per Week
Run-of-Mine (ROM - initial)	219,000	Tons per Year
Plant Feed Rate (initial)	600	Average Tons per Day
Design Capacity	1000	Shorts Tons per Day
ROM Specific Gravity	3.17	lb/ft ³
ROM F80	3-5	inches
Plant Availability	92	%
Crushing Plant Capacity	100	Tons per hour
Crushing Plant Operation	12	Hours per day
Fine Ore Bin (FOB)	500	tons
Processing Mill (F80)	1/2	inch
Bond Work index	14.2	kWh/st
Cyclones (P80)	120 (125)	Mesh (microns)
Flotation Conc. (Recovery)	~4	% mass pull
Pre-Leach Thickening	295	Settling Rate, ft ² /tons per day
	45	Underflow Density, Wt%
Leach Tanks	24	Design Residence Time, hr
	1.45	Feed Specific Gravity
Tailings Deposition	40	Feed Percent Solids, Wt%

17.4 Plant Operations

The current site plan for the mill and operations consists of standard 2-stage crushing followed by closed circuit ball mill grinding with a gravity circuit placed in the cyclone underflow. Although, the plant was designed to process ore at an average rate of up to 1000 tons per day (approximately 330,00 tons per annum); initially the plant will operate at a maximum of 600 tons per day producing approximately 20 tons of concentrate (20 tons flotation concentrate at 4 to 8 oz/t. A gravity circuit may be introduced in the future if required and produce a concentrate up to 15 oz/t. The planned layout is shown in Figure 17-2 and an isometric view is shown in Figure 17-3. The design implements a pre-leach thickener followed by 4 leaching tanks providing 24 hours of total residence time. The tails from the flotation circuit will be pumped to a pre-leach thickener prior to the carbon in leach (CIL) circuit for cyanide addition and fine gold dissolution and recovery. The new CIL leaching plant will be constructed to the north of the existing plant footprint, next to the flotation plant location.

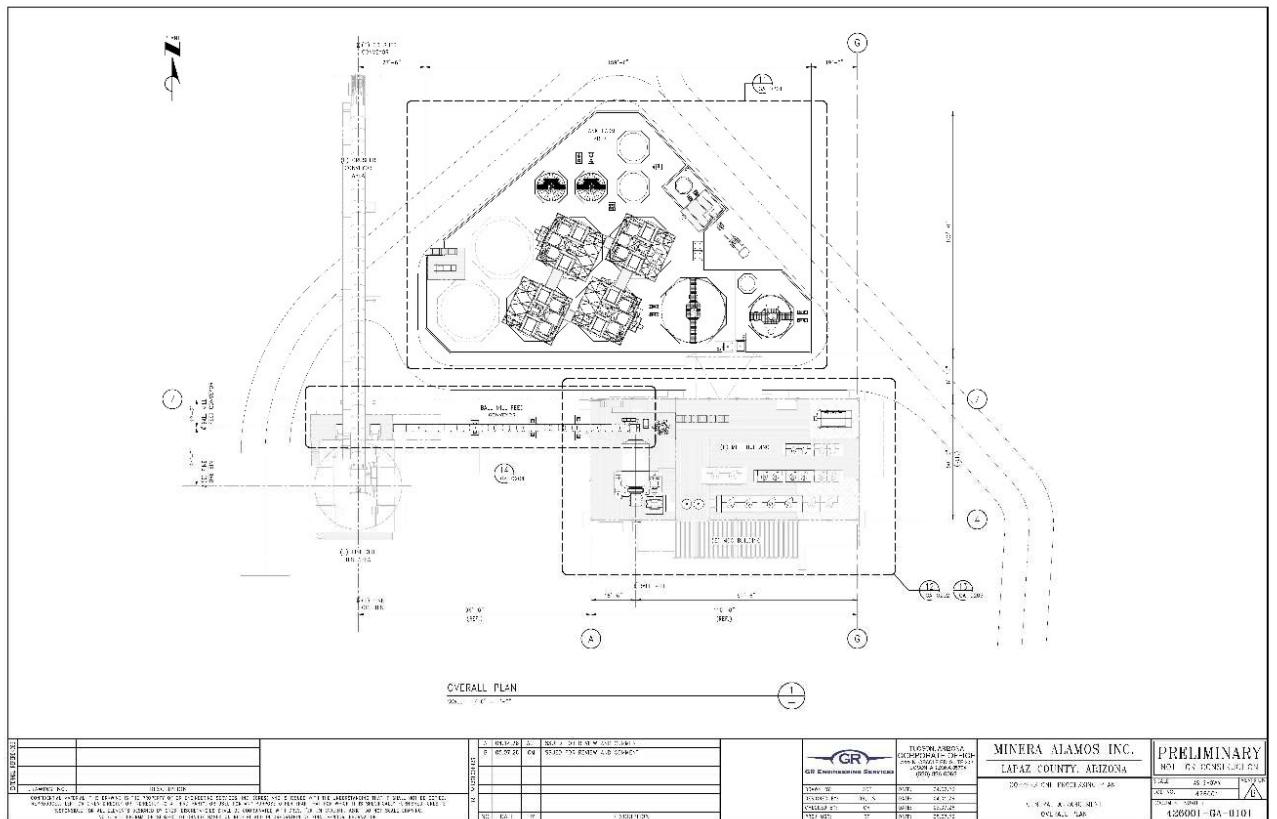


Figure 17-2 Plant Layout

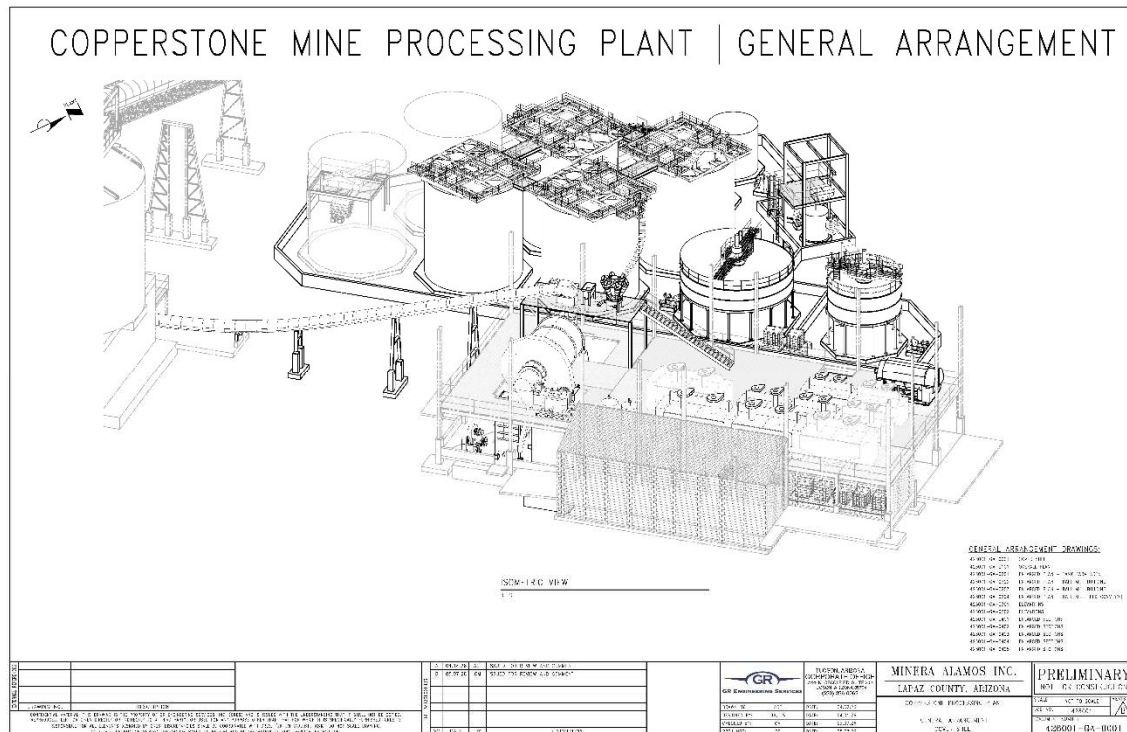


Figure 17-3 Process Plant Isometric View

17.5 Plant Services

17.5.1 Blower Air and Compressed Air

Blowers will supply low pressure process air to the flotation cells. Compressors will supply medium pressure process air the CIL tanks if required.

17.5.2 Plant and Instrumentation Air

Air compressors will provide plant and instrument air for the process plant. Plant air receivers will act as a buffer storing air to account for variations in demand prior to being distributed throughout the process plant. Instrument air will be dried before being stored in the instrument air receivers and distributed throughout the plant.

17.5.3 Fresh and Fire Water

Refer to Section 18.

17.5.4 Domestic Water

Refer to Section 18.

17.5.5 Gland Seal Water

Water for the gland seal water system will be supplied from the raw water tank. The gland seal water tank will store and distribute gland water to the plant with gland seal water pumps in a duty-standby configuration. To prevent particulates from causing damaged gland seals throughout the plant, the water feeding the gland water tank will pass through 25 µm particulate filters.

17.5.6 Process Water

Process water will consist of concentrate thickener overflow, flotation tailings thickener overflow, tailings reclaim water, and fresh makeup water as required. Process water will be stored in the process water storage tank and distributed by the process water pumps, in a duty: standby configuration, to consumption points in the grinding and flotation circuits.

17.6 Metallurgical Accounting

The plant feed and tailings products will be sampled manually which will allow the plant to determine an accurate metal balance. A weigh scale is currently installed on the ball mill feed conveyor will measure the primary crushed material tonnage. A manual belt cut sampling point on the ball mill feed conveyor will allow for the collection of a mill feed head grade sample for cross-validation with the calculated head grade. This sample will also be utilized to determine and review the moisture content of the mill feed. Water supplied and used in the various areas will continuously be monitored.

Metallurgical samplers will be used to generate shift composite samples that will be assayed for plant metallurgical accounting. The following process streams will be equipped (as a minimum) with metallurgical samplers:

- Cyclone overflow
- Flotation concentrate
- Flotation tailings
- CIL tailings

On an hourly basis, reagent usage rates to unit operations will be measured and accumulated using flowmeters. Reconciliation of the reagents used over relatively long periods will be achieved by delivery receipts and stock takes.

17.7 Plant Consumption

17.7.1 Water

Approximately 100 to 200 gpm of raw water is required for site and process plant makeup water. Water is sourced from either as reclaim water from the TSF polishing pond, underground development discharge water, or from 3 locally drilled water wells.

17.7.2 Energy

The power demand for the process plant, along with the rest of the project, will be provided by grid power. The average power demand for the process plant is approximately 3 MW. The average power demand does not reflect the instantaneous power demand for equipment start-up and power plant capacity sizing.

17.7.3 Reagents and Steel Consumables

Grinding and crushing liners and grinding media consumption rates are listed in Table 17-3. Reagent storage, mixing and pumping facilities will be in designated areas near the flotation and leaching circuits. Flotation and gold leaching reagents are summarized in Table 17-4.

Table 17-3 Crushing and Grinding Consumables

Steel Consumption		
Description	Consumption	
Jaw Crusher Liners	Set/Year	1
Cone Crusher Liners	Set/Year	1
Ball Mill Liners	Set/Year	1.5
Grinding Media	Lbs. /ton feed	1.25

Table 17-4 Hybrid Plant Reagent Consumptions

Reagents (Flotation and Leaching)		
Description	Consumption	
PAX Collector	Lbs./ton	0.165
MIBC Frother	Lbs./ton	0.45
R208	Lbs./ton	0.085
Cyanide, NaCN 20%	Lbs./ton	2
Hydrated Lime, 90% Ca(OH) ₂ w/w	Lbs./ton	2.5
SMBS, Na ₂ SO ₅	Lbs./ton	0.5
Flocculant	Lbs./ton	0.3

18 PROJECT INFRASTRUCTURE

The Copperstone deposit is favorably located across flat, sandy desert. The main line of the Santa Fe railroad passes to the north of the property.

Adjacent to the existing site access road, water pipelines from existing site surface wells, and an existing overhead commercial power line is routed to the project site. All three have current permitted and renewable federal Right of Ways.

Existing site infrastructure includes an office building, warehouse/shop for mining surface equipment, laboratory building, change house, septic system, and various shipping containers, which act as secure storage for MAI's reverse circulation and diamond drill core logging boxes. Incoming commercial 69 kV overhead electrical power is delivered to a power substation located on site. Water is currently delivered from three water wells to an existing 375,000 gal storage tank located in the same area. The three water wells are covered under State of Arizona - Department of Water Resources registration numbers 55-514525, 55-514526, and 55-908563.

Within a 35-mile radius of Copperstone, several communities including Parker and Quartzsite, Arizona and Blythe, California, are equipped to provide housing, shopping and schools for mine personnel and their families.

18.1 Office, Mine Shop, Warehouse/Shop and Laboratory

The office complex is located to the west and south of the process plant. See Figure 18-1 for the current infrastructure layout.

The facility serves as the administrative center for the operation and houses personnel from accounting, purchasing, human resources, health and safety, environmental compliance, engineering, and general management.

Dedicated office space for senior mine operations and maintenance personnel is provided within the mine maintenance shop, while the laboratory includes an office for the Laboratory Manager.

The administrative offices are equipped with both internal and external communication systems to support site operations, management functions, and coordination with off-site personnel and stakeholders.

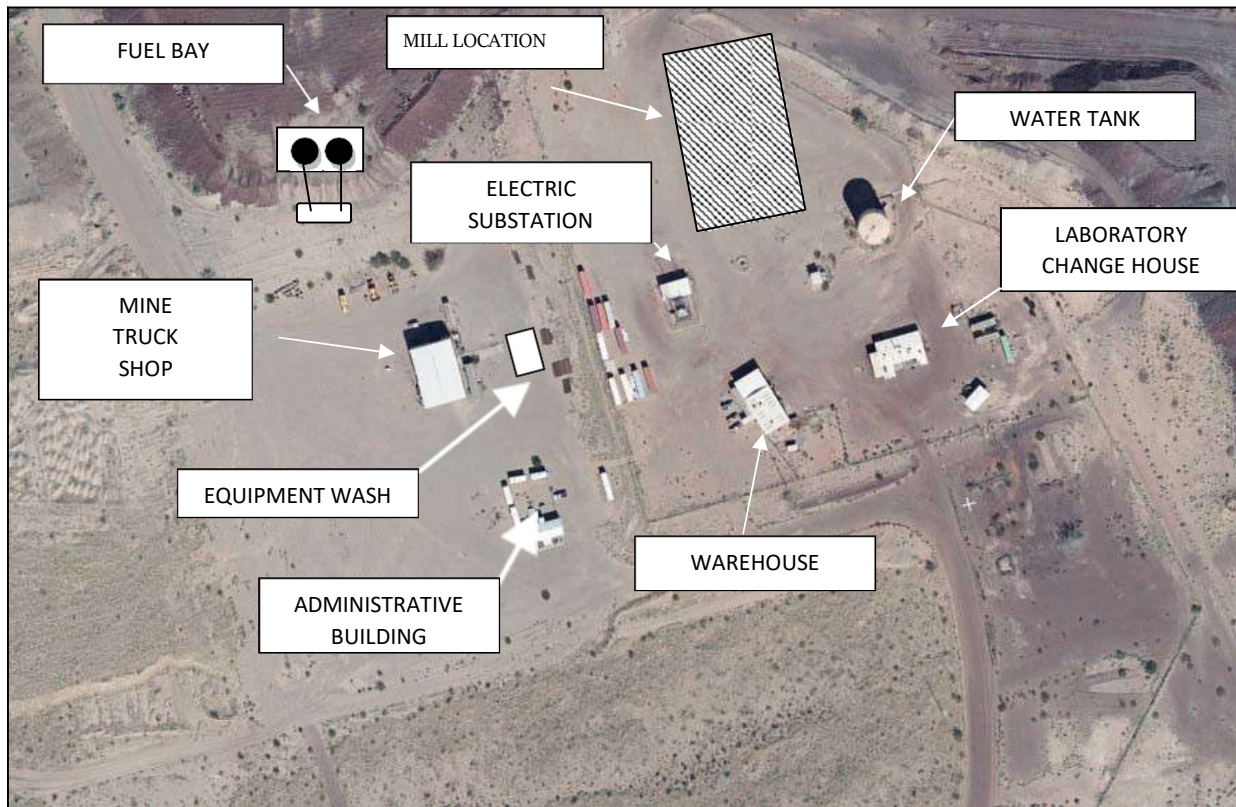


Figure 18-1 Ancillary Facilities

18.1.1 Office

The currently constructed administration building is approximately 1,950 ft² and will house all the administrative and management personnel. The building is a single-story pre-engineered, prefabricated, multi-trailer building with corrugated metal roofing and siding located just north of the entrance to the plant, outside the fence line. Visitor parking is provided outside the fence line at the office facilities. Employee parking is provided as necessary within the facilities area. This configuration will allow many of the site's vendors and other visitors to access management and operating personnel without entering the process plant or mine area.

18.1.2 Mine Maintenance Shop

The maintenance shop consists of a pre-engineered steel Butler-type building with metal siding and a total floor area of approximately 5,400 square feet. The structure is insulated with two-inch fiberglass insulation installed throughout the roof and wall systems, with plywood protection provided along the lower wall sections.

The facility is located on the northwest side of the site infrastructure area and is equipped with two vehicle service bays, compressed air distribution piping, welding stations, office space, dedicated spare parts storage areas and a dedicated tool storage area. The shop is designed as a drive-through facility to facilitate

equipment movement, with steel rails embedded in the concrete floor to protect the slab from damage caused by tracked equipment.

The mine shop serves as the primary maintenance facility for the operation and will be used to perform routine servicing, repairs, and preventative maintenance on all heavy mobile mining and support equipment.

18.1.3 Warehouse

The warehouse is located at the southern end of the site facilities area and comprises approximately 2,600 square feet of floor space. The building is a pre-engineered steel structure with metal siding and serves as the central receiving, storage, and distribution facility for site supplies, excluding process plant reagents. The warehouse includes indoor storage areas for parts and consumables, office space, and a dedicated tool crib and small-parts inventory area.

A fenced storage yard located immediately south of the warehouse is used for the storage of large equipment components, bulk materials, and other items suitable for outdoor exposure. The exterior storage area consists of a compacted gravel surface designed to support material handling and equipment access.

Most reagents and chemicals delivered to the Project are intended for use in the process plant or laboratory. These materials are initially received through the warehouse and subsequently transferred to their respective facilities for storage and use. Bulk process reagents are stored in a dedicated storage area attached to the process plant.

18.1.4 Laboratory/Change House Area

The analytical laboratory is a single-story, pre-engineered steel building with corrugated metal roofing and siding located adjacent to the employee change house immediately east of the process plant entrance. The facility encompasses approximately 1,750 square feet and includes a sample preparation area, wet chemistry laboratory, fire assay and metallurgical laboratory, environmental laboratory, offices, lunchroom, and restroom facilities. A 10-foot-wide double door at the north end of the building provides access for receiving and handling samples within the preparation area.

The sample preparation area is physically separated from the analytical laboratory by an interior partition wall and is equipped with sample dryers, crushers, pulverizers, sample splitters, and a dust collection system designed to capture and contain airborne particulates generated during sample processing.

The analytical laboratory contains wet chemistry facilities, reagent storage, balance rooms, and analytical instrumentation required for metallurgical and environmental testing. The laboratory also includes designated systems for the collection, storage, and management of chemical wastes. Disposal of laboratory waste materials is conducted in accordance with applicable regulatory requirements based on the nature of the waste generated. Office space for laboratory personnel is incorporated within the facility.

The employee change house is a single-story pre-engineered steel structure with corrugated metal roofing and siding located adjacent to the laboratory within the secured plant area. Employees Park in a designated parking area inside the site fence and access the change house via a short walkway of approximately 50 feet. The facility contains approximately 1,750 square feet of floor space and provides separate men's and women's locker rooms, showers, and restroom facilities.

A fully permitted septic system with a design capacity of approximately 1,500 gallons per day services the laboratory and change house facilities.

Laboratory waste streams are generally compatible with the milling process and, where appropriate, are returned to the process circuit for treatment and disposal within the operating system

18.2 Other Necessary Infrastructure

18.2.1 Lube Oil Tank Farm

The lubricants and oil storage is located outside on the northeast side of the mine shop. This area contains tanks or totes for engine oil, waste oil, hydraulic oil, gear oil, DEF and anti-freeze. These are stored in their individual containers, and all are within a single steel containment unit.

18.2.2 Diesel Fuel/Gasoline Dispensing Area

Diesel fuel for the equipment fleet is to be located north of the mine shop area outside the building on the old waste facility. Pumps and dispensers will be provided for fueling. A drive-through dispensing system will be provided for vehicles and for filling the fuel/lube truck. Two diesel storage tanks (10,000 gallons each) will be used to store diesel fuel. The gasoline dispensing area is with the diesel dispensing area. All fuel storage is constructed with containment to prevent hydrocarbon contamination of the surrounding area. These storage areas drain to a central collection sump where all spills are collected.

18.2.3 Wash Bay

A dedicated vehicle wash facility is located west of the mine maintenance shop and consists of a contained concrete wash pad designed to accommodate both heavy and light vehicles. The facility is configured as a drive-through station and is equipped with pressure washers, associated hoses, and water storage tanks to support equipment cleaning operations.

The wash pad is graded toward a central collection area to ensure that wash water and sediments are fully contained and prevented from being discharged to the surrounding environment. Collected solids, including mud and debris, are captured in a concrete sump designed for periodic cleanout using a front-end loader or similar equipment. This configuration provides effective management of wash water and sediments while supporting environmental compliance and site housekeeping requirements.

18.3 Explosives Storage

Separate magazines are provided for blasting powder and detonator caps. The powder and cap magazine is provided by the blasting contractor and will meet all appropriate ATF requirements. The magazines exceed code requirements and are separated by at least 200 ft with intervening separation berms.

The location of the magazines is directly north of the current pit entrance next to the mine waste storage facility. This area is remote and is shielded from the mill and admin areas to the south by a waste facility and from the pit traffic due to location near the waste dump.

18.4 Power

The electrical power supply for the Project facilities is provided by Arizona Public Service Company (“APS”). APS is the main electric utility service provider for the entire facility.

The existing 69 kV power line was designed for a mill and heap leach facility of larger size and equipment and has sufficient line capacity for the new operation. The power line is owned by APS with a right-of-way permitted by the BLM for the power line corridor. An on-site substation is currently operational and located to the west of the existing mill site. Limited changes are expected to the current power supply system.

The average continuous draw for the Copperstone mine and process facilities is estimated to be 3,433 kW and will require a minimum transmission voltage of 13.8 kV. Power requirements are expected at the mine, mill, and facilities and will require 4160, 480, and 220/110 voltage. Power to the new Leach plant will be through underground concrete-incased ductwork.

Table 18-1 Power Requirements

Area	HP	Connected kW	Diversity	Ave. Draw kW
Water Wells	500	373	0.75	280
Mine	2,290	1,708	0.75	1,281
Crusher	570	425	0.75	319
Grinding	697	520	0.82	426
Leaching	1,348	1,005	0.82	824
Buildings and other	455	339	0.75	254
Total				3,433

APS has maintained and repaired the existing substation. No repairs are expected to the electrical line or substation infrastructure.

18.5 Water

Fresh water for the facility will be supplied from dewatering of the underground mine and/or from three existing Patch Living Trust permitted wells located east of the Copperstone mine near the intersection of the Copperstone Mine Road and Arizona Highway 95.

The Patch Living Trust water wells are permitted under Arizona Department of Water Resources (ADWR) and are currently valid and can be located under the ADWR - GWSI Well Information site ID 514525, 514526, and 517883.

The water balance for the site indicates a net surplus of water, primarily as a result of water produced from the mine. It is currently anticipated that a permitted shallow evaporation pond/infiltration gallery will be constructed close to the site for use in evaporating excess water as required.

Table 18-2 Water Balance

Copperstone Water Balance		
Startup water needed for plant	350	gpm
Mine	107	gpm
Plant	52	gpm
Road etc.	57	gpm
Total	217	gpm
Water produced from mine	278	gpm
Excess water	61	gpm

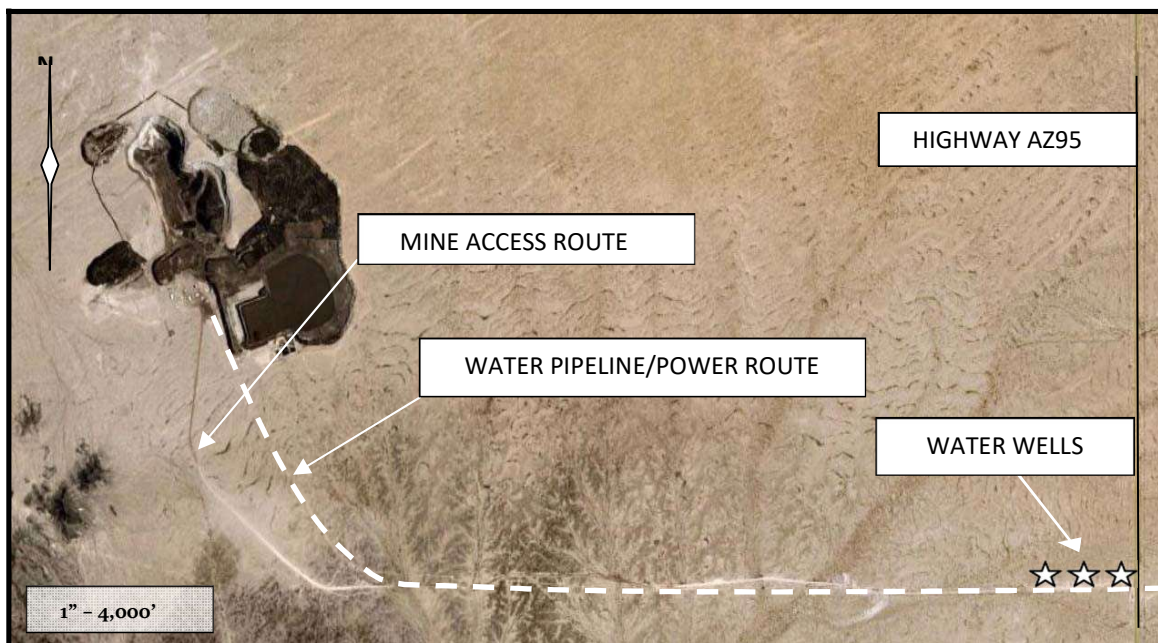


Figure 18-2 Water Well Location

Excess water from the mine that is not used for mineral processing or site dust control will be transported to an evaporation pond/infiltration gallery. Water from the existing water wells is transported to the mill site via a currently installed pipeline and booster system.

Water is supplied from the fresh/fire water tank to the facility by pump and level indicator. Fresh water will be distributed to:

- Gland seal water tank and horizontal centrifugal pumps for seal water for mechanical equipment,
- Buildings for non-potable water, and
- Fire water distribution system in the mill site and admin areas.

Decant collected from the thickeners and filtering circuit is collected in the process water holding tank and recycled to the process circuit. Water in the TSF discharges to the reclaim water pond. This process water will be pumped from the reclaim pond to the process water holding tank and will be distributed by pump and pipeline to mill usage points.

Drinking water is trucked to the site using a local drinking water vendor.

18.6 Communications

Cell phone telephone systems are provided for communications between the on-site offices in the main office, laboratory and mine shop area. Supervisors' and mechanics' vehicles and the process plant and crusher will be equipped with mobile radio units. Supervisors will be equipped with hand-held radios.

- A satellite internet system provides the project with external VOIP and data communications.
- Cell phone communications are currently available on site using a local provider.
- Hand-held radio communication through FM radio communications is currently permitted.
- A leaky feeder system is planned for underground communications.

18.7 Roads

The existing main access road to the site is Arizona Highway 95, a two-lane paved highway that connects the nearby towns of Parker and Quartzsite, Arizona. The primary mine access road (approximately 5 miles) is an existing, well-traveled dirt road that terminates at the Copperstone mine. The access road is regularly maintained and considered to be accessible year around. Several facility and in-pit roads will continue to be used to access all areas of the mine, plant and administrative facilities.

18.8 Site Drainage

Diversion ditches are currently in place around the entire site and protect the surface facilities from damage due to rainfall runoff and to help control sediment runoff from previously disturbed areas of the project. No changes to these facilities are expected.

18.9 Fencing

The entire site including the mine area and waste dumps is fenced with barbed wire. A total of 45,000 ft of this fence is in place and requires minor maintenance. The main facilities are fenced using approximately 500 ft of 6-ft high chain link fence which ties into the barbed wire fence.

18.10 Tailings Storage Facility

The existing Tailings Storage Facility (TSF) was originally designed by Schlumberger and permitted by the Arizona Department of Environmental Quality (ADEQ) with a total storage capacity of approximately 1.24 million tons. The facility was initially constructed as Phase 1A, consisting of a 17-ft-high embankment and a portion of the ultimate approved footprint, providing storage capacity for approximately 366,000 tons of tailings.

During operations in 2012 and 2013, approximately 101,500 tons of tailings were deposited within the facility. Based on the planned processing rate and projected mine production schedule, the remaining capacity of the existing Phase 1A facility is sufficient to support slightly more than one year of operation. Consequently, construction of the Phase 1B expansion will be required during the first year of renewed operations.

To accommodate continued tailings production, an additional downstream raise of approximately 4 ft is anticipated by the third year of operation. These staged expansions will provide the storage capacity required for the planned life while maintaining compliance with the approved TSF design criteria.

Figures 18-3 and 18-4 illustrate the planned progression of the TSF from the existing facility configuration through its ultimate permitted development.

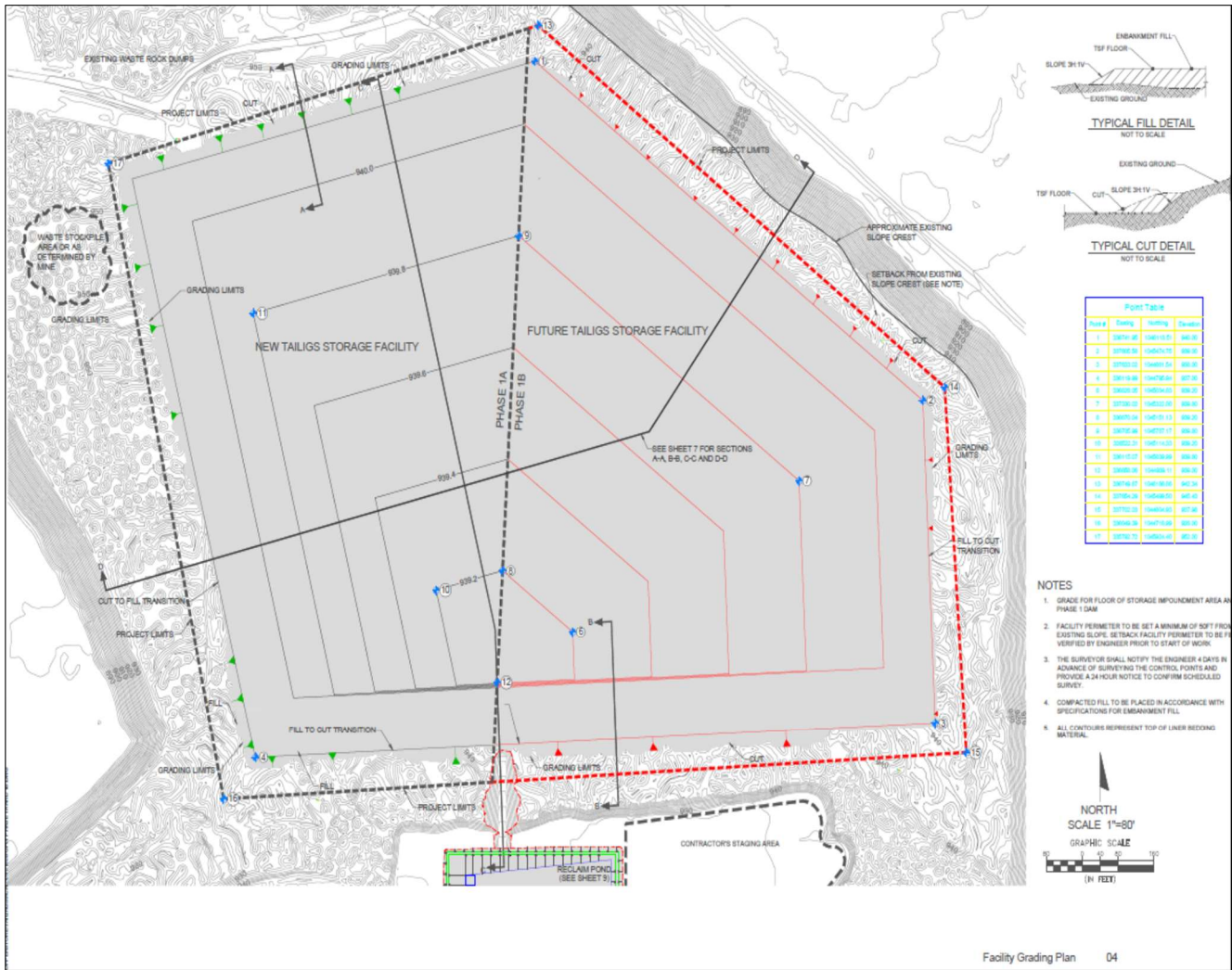


Figure 18-3 Existing TSF Phase 1A (left) and 1B (right)



19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Gold markets are mature: global markets with reputable smelters and refiners are located throughout the world. Concentrate from the flotation circuit will be shipped to a port on the west coast for shipping. Loaded carbon will be shipped to the Company's Pan Mine for processing and sold as doré.

The gold doré produced at the Company's Pan Mine site will be refined under the existing agreements with Ashai Refineries USA. It is reasonable to assume that the product from the Project will be salable. A selling price of \$3,500/oz. for gold has been used to develop this PFS, which is substantially less than the 36-month (US\$ 4,295) and 60-month (US\$ 4,463) trailing averages at the end of June 2026, as tabulated from public data from the website www.kitco.com. The month-end closing spot price at that time was \$4,075/oz. for gold. The historical gold prices are presented in Figure 19-1.



Figure 19-1 Historical Gold Prices (3 Month)



Figure 19-2 Historical Gold Prices (6 Month)

Table 19-1 Anticipated Refining and Transportation Assumptions

Description	Units	Value
Au Payable in Dore	%	99.9
Flotation Au Payable	%	99.7
Flotation Refining Charge	\$/oz	10
Flotation Shipping Charge	\$/mt	225

19.2 Commodity Price Projections

Precious metals are freely traded globally in many markets and are highly liquid commodities. Metals prices and foreign exchange rates used in the economic analysis are in line with recently published technical reports and are within both the 24 month and 36-month trailing averages.

A gold price of \$3,500/oz has been used in the economic assumptions.

19.3 Contracts

No contracts have been negotiated at this stage of the Project except for a gold streaming agreement with Star Royalties which is discussed in more detail in Section 4.2. As part of the agreement Star receives 4% of annual gold production at a gold price of 25% of spot.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

To support approval of the 2008 Mine Plan of Operations (MPO), the Updated Interim Management Plan, and associated state environmental permitting requirements anticipated in Q3 2026, a series of environmental baseline studies and assessments were completed, including:

- Water Resources Assessment (surface water and groundwater)
- Threatened and Endangered Species and Wildlife Assessment
- Air Quality Assessment
- Biological Survey
- Cultural Resources Assessment

These studies were conducted to characterize existing environmental conditions, evaluate potential project impacts, and provide supporting documentation for the federal and state permitting process.

20.1.1 Water Quality

Water quality samples were collected as part of the hydrogeologic investigation and from the point-of-compliance (POC) monitoring wells established under Aquifer Protection Permit (APP) No. P106172. These data provide the baseline characterization of groundwater quality at the project site and serve as a benchmark for ongoing environmental monitoring. Representative analytical results from the POC wells are summarized in Tables 20-1 and 20-2.

Table 20-1 POC-1 Test Data

POC-1					
Date Sampled	11/29/2017	UNITS	CALCULATED AL & AQL LIMITS		
Parameter			POC-1	POC-1	AWQS
Groundwater Level			AL	AQL	mg/L
Water elevation (amsl)		ft	520		
O & G	2	mg/L			
pH (lab)	8.2				
Specific Conductivity (lab))	1580	uS/cm			
TOC	1	mg/L			
TDS	898	mg/L			
Alkalinity	146	mg/L			
Antimony	0.0014	mg/L	0.005	0.006	0.006
Arsenic	0.0027	mg/L	0.04	0.05	0.05
Barium	0.022	mg/L	1.6	2	2
Beryllium	0.00005	mg/L	0.0032	0.004	0.004
Boron	1.1	mg/L			
Cadmium	0.0001	mg/L	0.004	0.005	0.005
Calcium	33.9	mg/L	30.5	NONE	
Chloride	199	mg/L	205	NONE	
Chromium	0.01	mg/L	0.08	0.1	0.1
Copper	0.01	mg/L	0.02	NONE	
Fluoride	4.68	mg/L	NONE	6.3	4
Hardness	104	mg/L	97	NONE	
Iron	0.02	mg/L			
Lead	0.0001	mg/L	0.04	0.05	0.05
Magnesium	4.8	mg/L	5.2	NONE	
Manganese	0.005	mg/L			
Mercury	0.0002	mg/L	0.0016	0.002	0.002
Nickel	0.008	mg/L	0.08	0.1	0.1
Nitrate	0.46	mg/L	1.70	10	10
Nitrite	0.01	mg/L	0.8	1	1
Nitrate + Nitrite as N	0.46	mg/L	1.70	10	10
Potassium	4	mg/L			
Selenium	0.0024	mg/L	0.040	0.050	0.05
Silver	0.01	mg/L	0.02	NONE	
Sodium	275	mg/L	291	NONE	
Sulfate	295	mg/L			
Thallium	0.0001	mg/L	0.0016	0.002	0.002
Zinc	0.01	mg/L			

Table 20-2 POC 2 Test Data

POC-2					
Date Sampled	11/29/2017	UNITS	CALCULATED AL & AQL LIMITS		
Parameter			POC-2	POC- 2	AWQS
Groundwater Level			AL	AQL	mg/L
Water elevation (amsl)		ft	590		
O & G	2	mg/L			
pH (lab)	8.1				
Specific Conductivity (lab)	1460	uS/cm			
TOC	1	mg/L			
TDS	838	mg/L			
Alkalinity	155	mg/L			
Antimony	0.0005	mg/L	0.005	0.006	0.006
Arsenic	0.0058	mg/L	0.04	0.05	0.05
Barium	0.031	mg/L	1.6	2	2
Beryllium	0.00005	mg/L	0.0032	0.004	0.004
Boron	1.34	mg/L			
Cadmium	0.0001	mg/L	0.004	0.005	0.005
Calcium	31.3	mg/L	36.1	NONE	
Chloride	178	mg/L			
Chromium	0.01	mg/L	0.08	0.1	0.1
Copper	0.01	mg/L			
Fluoride	5.52	mg/L	NONE	5.5	4
Hardness	98	mg/L	113	NONE	
Iron	0.03	mg/L			
Lead	0.0001	mg/L	0.04	0.05	0.05
Magnesium	4.7	mg/L	5.5	NONE	
Manganese	0.005	mg/L	0.011	NONE	
Mercury	0.0002	mg/L	0.0016	0.002	0.002
Nickel	0.008	mg/L	0.08	0.1	0.1
Nitrate	0.02	mg/L	2.00	10	10
Nitrite	0.01	mg/L	0.8	1	1
Nitrate + Nitrite as N	0.02	mg/L	2.00	10	10
Potassium	4.2	mg/L	4.7	NONE	
Selenium	0.0024	mg/L	0.040	0.050	0.05
Silver	0.01	mg/L	0.02	NONE	
Sodium	252	mg/L	307	NONE	
Sulfate	248	mg/L	351	NONE	
Thallium	0.0001	mg/L	0.0016	0.002	0.002
Zinc	0.01	mg/L			

Water quality analyses indicate that groundwater at the site is slightly alkaline, with an average pH of approximately 8.0. With the exception of fluoride, which is naturally elevated in the local hydrogeologic environment, all analyzed parameters were below the Arizona Aquifer Water Quality Standards (AWQS). Overall, the groundwater quality is considered good and is well suited for use as process water.

20.1.2 Air Quality

The primary potential source of air quality impacts from the Project will be fugitive dust generated by the crushing and screening circuit. Dust emissions from these activities will be controlled through the use of water spray bars at the jaw crusher to precondition and pre-wet mill feed material prior to processing.

Dust generated by mining operations is expected to be relatively minor and will be limited primarily to underground workings, mill feed stockpiles, the tailings storage facility, and site haul roads. Dust generation from the temporary coarse ore stockpile is anticipated to be minimal because the mineralized material delivered from underground will contain sufficient moisture. Should the material dry during storage, dust will be controlled through periodic water application and stockpile spraying.

Dust emissions from the tailings storage facility will be managed through controlled tailings deposition practices and maintenance of moisture within the ponded areas. During operations, exposed tailings surfaces will be kept wetted as necessary, and upon closure the tailings surface will be covered with approximately 6 inches of crushed waste rock to provide long-term dust suppression.

Fugitive dust generated from haul roads will be mitigated through routine watering and road maintenance. Milling and gold recovery operations are conducted as wet processes and therefore have minimal potential for dust generation. Gold extraction is accomplished using a dilute sodium cyanide solution within the leaching circuit, resulting in negligible atmospheric emissions. Overall, projected air emissions from the Project are well below applicable regulatory thresholds.

20.1.3 Noise

The primary source of operational noise at the Project will be the crushing circuit. Potential noise impacts from crushing activities are minimized by the remote location of the processing facility, which is situated well away from sensitive receptors such as residences, recreational areas, and other occupied facilities.

Noise generated by underground mining operations will be largely confined below surface and is not expected to be perceptible beyond the limits of the existing open pit. Similarly, vibration associated with blasting activities is expected to be minimal due to the relatively small explosive charges required for underground development and production blasting. As a result, blast vibrations are not anticipated to be noticeable beyond the Project boundary.

20.1.4 Surface Water Management

The Copperstone mine is situated in a desert area of low precipitation with very flat sandy soils with low runoff capacity. Drainages downgradient from the mine site are ephemeral in nature. The nearest Waters of the United States are Tyson Wash and the Colorado River which are 7 and 12 miles away from the mine, respectively. Due to these factors the mine maintains a no discharge status from a stormwater perspective.

The mine will maintain zero discharge of process waters at the facility. All process fluids are contained with engineered systems. Environmental Monitoring

Environmental monitoring will be carried out during the life of the project to ensure compliance with all permit conditions and current best practices. The environmental program for the Copperstone mine will include:

- Quarterly monitoring of two POC wells installed down gradient of the designated area of impact in the APP to monitor for mine related contaminants in the groundwater.
- Routine inspections of the process fluid systems as specified in the APP.
- Annual inspection of storm water control systems,
- Routine air quality control equipment performance testing as required by the Air Quality Permit.
- Routine monitoring of opacity as specified in the air quality permit. The frequency and extent of the monitoring program may be modified during the permit modification process, in particular the Aquifer Protection Permit and the Air Quality Permit.

20.1.5 Hazard Operations Plans

The Copperstone Mine will develop and implement operational plans and procedures to address fluid management, worker health and safety, employee training, emergency response, environmental monitoring, reporting requirements, and other key operational practices. These management systems may be documented in various formats, including formal operating manuals, standard operating procedures (SOPs), inspection checklists, signage, work orders, training materials, and other site-specific guidance documents. Together, these measures will support safe, efficient, and environmentally responsible operation of the Project

20.2 Disposal, Monitoring, and Management Plans

20.2.1 Disposal Facilities and Operations

The August 2025 Mine Plan of Operations submittal describes the mine's planned mining and processing facilities. Most operations, including mining, processing, and exploration—were previously approved under earlier MPOs. The 2025 update seeks approval for a modification to the mill' process circuit.

20.2.2 Closure and Post-Closure

Through production, closure, and post-closure, process water, surface water, and groundwater within the area of impact will be closely managed. A closure notification and plan will be submitted to ADEQ as specified in the APP. The closure plan will address process fluid management, facility decommissioning and waste disposal, reclamation of the tailings storage facility and waste rock disposal facilities, underground mine closure, and the cessation of dewatering and infiltration. Mill and process plant fluids will be managed as part of the decommissioning process.

To control stormwater and limit erosion and sediment transport from disturbed areas during the pre-production, production, and post-production periods, stormwater management, erosion, and sediment control BMPs will be employed as appropriate, including:

- Diversion of stormwater run-on away from mine facilities to prevent stormwater contact with disturbed areas;
- Construction of erosion control berms;
- Site grading to route stormwater to constructed diversion channels
- Construction of runoff collection basins as required.

20.3 Permitting

20.3.1 Permit History/Background

A summary of the environmental permits currently in place and which are necessary to operate the mine according to the parameters stipulated in the 2008 MPO is provided in Section 4.3. The mine was operated as a fully permitted, underground gold mine for approximately 7 months in 2013, with all required permit approvals for the mining, crushing, and surface flotation milling of approximately 70,000 tons of mineralized material. The operation was reportedly plagued with operational issues that resulted in lower than planned production tonnage and grades. The mine elected to revert to “idle” status, per the requirements of the Interim Management Plan in the MPO. An Updated Interim Management Plan was approved by the BLM on August 12, 2019, and the Copperstone Project continues to operate according to the requirements of this plan. MAI submitted an updated MPO (Aug 2025) to the BLM to incorporate an additional leach circuit. BLM is currently reviewing the document.

20.3.2 Compliance History

The Aquifer Protection Permit (APP) remains active and requires ongoing groundwater monitoring, including routine sampling of downgradient point-of-compliance (POC) wells and submission of quarterly and annual monitoring reports to the Arizona Department of Environmental Quality (ADEQ).

Elevated fluoride concentrations were first observed in the POC wells in June 2018. The Arizona Aquifer Water Quality Standard (AWQS) for fluoride is 4.0 mg/L. Subsequent investigations, including the Parker Basin Fluoride Study, determined that elevated fluoride concentrations are a naturally occurring regional groundwater characteristic. The study found elevated fluoride levels in eight of nine wells sampled throughout the basin. Sampling of the Project’s POC wells and seven nearby wells identified fluoride concentrations ranging from 3.7 mg/L to 7.3 mg/L. The most recent monitoring results reported fluoride concentrations of 6.6 mg/L in POC-1 and 4.1 mg/L in POC-2. The Parker Basin Fluoride Study and a request to revise the fluoride exceedance thresholds were submitted to and approved by ADEQ.

The Project’s air quality permit requires monthly Method 9 opacity observations and reporting to ADEQ whenever the processing plant is operating. The Company has maintained a strong compliance record and has consistently met its monitoring, data collection, and reporting obligations. Upon renewal, the air permit

is expected to remain valid through 2028. A minor permit modification will be required following completion of the final processing plant design to incorporate any design-specific changes.

20.3.3 Description of Applicable Permits, Amendments, and Approvals

The current MPO includes a maximum throughput of 600 tons per day, which meets the allowed tons by the current air permit of 210,000 ton per annum. The current MPO includes a stand-alone underground mine and associated process facilities all located on BLM managed land. As long as the throughput does not exceed the permitted quantity and the basic configuration of the mining method, process method, process fluid management and tailings management does not change, all of the required environmental permits are in place and do not require revision. However, the proposed modifications to the mill circuit, spurred an August 2025 MPO update to the BLM. The MPO review is ongoing. BLM has requested that this MPO update summarize all previous approvals.

In January of 2020, the U.S. Bureau of Land Management (“BLM”) issued a Decision of Record (“DOR”) based on a Finding of No Significant Impact (“FONSI”) formally approving Kerr’s Mining Plan of Operation (“MPO”) at the Copperstone Mine. Following Kerr’s application in June of 2018, the BLM conducted an Environmental Assessment of the Copperstone Mine as required by the National Environmental Policy Act. This process involved several independent studies to evaluate the effect of the project including cultural and biological resources, traffic, noise, water and air quality. Additionally, the BLM provided for public comment allowing the public to review study results, discuss the proposed plans with Kerr’s representatives and submit formal comments to BLM. The following changes are pending and under review of the new MPO for the Copperstone mine:

- Increase of gold mill feed production from the current allowable limit of 450 tons per day to 600 tons per day.
- Use of flotation for recovery of copper and gold from mill feed, using typical copper/gold flotation cell,
- Use of cyanide for recovery of gold from mill feed using captive steel tanks located in the gold mill feed processing facility.
- Storage of stabilized tailings produced from the processing facility.
- Construction and use of a water evaporation and infiltration basin to be used to manage surplus water generated from underground operations.

In 2009 American Bonanza posted a reclamation bond of \$1.7 million. The Project is not subject to any other known environmental liabilities, and the QP knows of no other existing or potential future significant factors or risks (permitting, environmental, or otherwise) that might affect access, title, or the right or ability to perform work on the Project.

Table 20-3 Copperstone Mine Permit Summary

Permit	Approval/Permit #	Granting Agency
Mine Plan of Operations	Latest Revision Approved January 2020 to include cyanide usage, increased plant throughput to 600 stpd, and use of evaporation/infiltration basins.	Bureau of Land Management (BLM) Yuma District
Hazardous Waste RCRA	EPA ID AZD982500910, Number Issued for Life of Mine	EPA
Fuel Storage	Authorization for Life of Facility	EPA
Rights-of-Way Permits	AZA 32505 and AZA 32506, Issued to Angie Patch Survivor Trust, Patch Living Trust, and Patch Daniel L Credit Trust July 2018, for 20-year terms.	BLM
AZPDES 2010 Multi-Sector General Permit	MSGP No Discharge Certification	Arizona Department of Environmental Quality (ADEQ)
Air Quality Control Permit	Registration No. 73215 as amended June 3, 2025 Good for 5 years.	ADEQ – Air Quality
Aquifer Protection Permit (APP)	Permit No. P106172 as amended December 2024. Valid for life of facility.	ADEQ – Groundwater Protection
Well permits	Issued for Life of Mine	Arizona Department of Water Resources (ADWR)

In order to proceed with the redevelopment of the Copperstone mine, it is anticipated that MAI will have to obtain the following permit modifications and approvals:

- Air Quality Permit Minor Modification
 - Minor Modification to process plant with new tank emission calculations
- MPO Addendum
 - Update process circuit. Summarize all previous MPO authorizations, updates disturbance table and reclamation cost estimate
 - APP amendment has been authorized for a new Fluoride limit for the Copperstone mine; in which MAI has obtained an increase to the Arizona state limits.

Copperstone certifies as a no discharge facility which negates the need for a storm water pollution plan (SWPPP)

MAI will obtain Arizona State Mine Inspector's approval of the MPO amendment addressing new infrastructure and disposal facilities and plans for post-mining reclamation of those facilities. The MPO and the APP contain provisions for the closure and reclamation of facilities upon the cessation of mining activity are discussed in Section 20.4 of this report.

20.4 Remediation and Reclamation

The Copperstone Mine Reclamation Plan must follow 43 CR 3809 as stipulated in the MPO. The reclamation regulatory requirements are also dictated by the Arizona Mined Land Reclamation Act, the BLM regulations, and the APP Program, although other regulatory requirements may contribute mitigation elements.

The current MPO is designed to use previously disturbed land from former mining activities. A reclamation plan and associated reclamation cost estimate are provided in the currently approved MPO and are also part of the August 2025 MPO submittal under review by the BLM.

As described in the MPO, all planned land use is currently within the disturbed land area of the original Copperstone mine project boundary. The areas include:

- Administrative area,
- Tailings facility, mine openings, and
- Waste facilities.

The total land use acreage is estimated at approximately 130 acres which is entirely within the original disturbed footprint. MAI's Reclamation Plan addresses reclamation for all planned activities, including the removal of buildings, underground mining facilities, process facilities, roads, pipelines, electrical lines, and reclamation of the land used. MAI is not responsible for disturbance by previous owners/operators, including the old tailings/heap leach areas, the old reclaim pond, unused sections of the old waste dumps and facilities and full backfill of the pit. The previous operator was Cyprus Mining which was acquired several years ago by Freeport McMoRan Inc. (FMI). FMI completed reclamation of the old TSF/leach pad (the former leach pad was constructed on top of the tailings impoundment) and construction of a new drain down collection pond in 2104. FMI continues to collect samples and monitor closure of legacy facilities and maintains an active APP permit with ADEQ and an agreement with the BLM.

Reclamation of mining and exploration operations may include recontouring, ripping, stabilization, seedbed preparation; growth media application; and revegetation. The reclamation procedures for the Copperstone Project incorporate five basic components:

- Establishment of all mining related activities in or on previous disturbed acreage;
- Stable topographic surface and drainage conditions that are compatible with the surrounding landscape and serve to control erosion;
- Establishment of soil conditions most conducive to establishment of a stable plant community through stripping, stockpiling, and reapplication of suitable growth medium;
- Revegetation of disturbed areas to compatible with proposed post-mining land uses; and
- area.

20.4.1 Contouring and Shaping

Final grading will create landforms that are stable, do not allow for pooling or ponding, and blend with the surrounding disturbed topography. Final grading will minimize erosion potential and additional surface disturbance and will facilitate the establishment of post-mining vegetation. Straight lines will be altered to provide contours which are visually and functionally compatible with the surrounding terrain.

20.4.2 Seedbed Preparation

Seedbed preparation will take place after grading, stabilization, and growth media placement. Procedures used in seedbed preparation will include:

- Loosening of compacted surfaces from ripping.
- Using tillage implements, as needed, for all areas to be reclaimed

The prepared surfaces will be seeded using a broadcast seeder and/or rangeland drill, depending on the working area and steepness of slope.

20.4.3 Seeding/Planting

Revegetation activities will be performed in the fall through late winter to take advantage of winter moisture. For broadcast applications, the seeder will be followed by dragging a light chain or other means to provide some soil cover of the seed. When possible, a range land drill will be used for more effective seeding. The rocky terrain and soil materials in the Project area may dictate use of broadcast seeding.

20.4.4 Seed Mixtures and Application Rates

The selection and development of seed mixtures of grasses, forbs, and shrubs will focus on native species suitable for the desert climate and low moisture of the area. The seed mixture will be developed in consultation with BLM and may be adjusted to develop different plant communities in successive seedings. Proper range management, after reclamation, is an integral part of long-term diversity development.

20.5 Reclamation Design

20.5.1 Underground Mine

The underground openings will be bulkheaded approximately 75 ft into the mine entrances and each entrance filled with material from the immediate area from top to bottom and out to the entrance. The material will be placed higher and wider than the entrance so there are no openings or gaps.

The vertical vent shafts will be bulkheaded approximately 50 ft from the top of the shaft with steel sets and poured concrete. The bulkhead will be approximately 10 ft thick with 40 ft of material from the surrounding area placed in the existing hole. The dirt will be filled to level with the surrounding terrain and signs placed to identify potential hazards. The bulkheads will be of sufficient engineering design to ensure they will not collapse or create a public safety hazard.

20.5.2 Tailings Storage Facility

When the TSF use is completed, the structure will be leveled to a 3H:1V slope, capped 2 to 3 ft of capping material consistent with the current material used by the previous operator (1-ft minus crushed gravel). The cap will ensure that any precipitation received in the tailings pond area effectively runs off into areas that do not contain previous tailings or heap leach material thus eliminating infiltration into the old tailings/leach pads. The area will be seeded in accordance with the appropriate seed mixtures and application rates.

20.5.3 Reclaim Pond

After the tailings pond decant has stopped flowing from the TSF into the reclaim pond, the pond solution would be allowed to evaporate in place. When evaporation is complete, the pond bottom will be sliced open

to allow drainage and the sides cut and brought into the center of the pond and buried. The reclaim pond will be filled in with material from the immediate area.

20.5.4 Buildings

All buildings will be dismantled, prepared for shipment and transported off-site. Following the removal of all structures and buildings, four concrete pads will remain in place:

- Maintenance shop floor,
- Warehouse floor,
- Laboratory/Change-house floor, and,
- Pad between storage units.

Each pad will be fractured thoroughly by ripping with a bulldozer and all protruding steel reinforcement will be cut-off and disposed of in-place. All concrete will be covered with at least two feet of waste rock material hauled from the adjacent waste rock dump and will be covered in accordance with the county solid waste disposal requirements and permits. All other storage units and buildings are modular in nature and will be removed intact.

20.5.5 Crusher/Plant

The crusher facility is composed of a jaw and cone crusher, screen, and a system of conveyor belts leading to a crushed mill feed storage bin. The facility is portable and will be removed from site at the end of the project. The retaining wall to the north of the crusher station and the crushed mill feed storage bin will be broken down with steel components removed from site for scrap and inert concrete buried in place.

The mill is composed of concrete containment pads and associated ball mill, tanks, flotation circuit and a CIL circuit. Closure of the plant area will consist of several activities. Milling and mineralized material processing equipment will be sold and removed. All structural steel will be removed from site for scrap. All concrete pedestals will be broken down, and protruding reinforcing steel will be cut off and buried in-place.

The mill facilities area covers approximately 9 acres. Material from the waste piles to the north and sand stockpiles to the immediate west of the maintenance building will be hauled and spread over the entire area to an approximate thickness of two feet. The area will be seeded with an approved BLM seed mixture but based on experience at the site, it is expected that natural propagation of vegetation will provide the best reclamation.

20.5.6 Fuel Storage

The above-ground fuel storage facility contains a total of four tanks including two 10,000-gallon tanks, a 6,000-gallon gasoline tank, and a 3,000-gallon waste oil tank. Unused liquid will be collected from the tanks and hauled off-site for recycling in compliance with all applicable state and federal regulations. The tanks and all associated liner material will be removed from the fuel storage area and shipped off-site. The entire facility area will be inspected for evidence of petroleum hydrocarbon contamination once the buildings and tanks have been removed. Contaminated soil or gravel will be identified by its discoloration from exposure

to fuel, oil, grease or coolant and laboratory testing. This material will be excavated, loaded into trucks and hauled off-site for disposal in accordance with applicable laws and regulations.

20.5.7 Roads

The main haulage road out of the pit, all other links in the road network around the mine, and all remaining exploration roads and compacted surfaces will be contoured as near as possible to the surrounding terrain and revegetated. Any water diversion structures will be removed, and the natural drainage patterns restored. Water bars or other structures may be left in place to reduce any undue erosion. Public access roads will be left, or returned to the pre-mining condition and location, if practicable given the post-mining topography.

20.5.8 Other

Water for dust suppression and domestic use is supplied from one above-ground storage tank. When no longer needed, the tank will be drained, concrete busted and buried, and the tank transported from the site.

The explosives-storage area consists of a bulk-ANFO storage silo and two magazines. All tanks, magazines, and materials will be removed at the end of the project. The ANFO bin and magazines will then be hauled from the site. Sewage will be pumped from the septic tank and the top cover of the tank will be removed. The concrete walls will be ripped with a dozer and the tank backfilled with earth to the level of the top of the ground.

A review of the electrical transmission line will be completed with Arizona Public Service (APS) utility near the end of mine life. APS will identify the best way to remove the power line and poles to an area appropriate for their and the BLM's end user needs.

All other areas of the Copperstone site not affected by MAI's operations are to be left as is with no further disturbance.

20.6 Reclamation Costs

The cost estimate reclamation was prepared by SGLD as required by the Bureau of Land Management's bonding policies and with Arizona statute Title 27, Chapter 5, Article 1, Section 27-932. The reclamation cost estimate was developed by SGLD using the State of Nevada Standardized Reclamation Cost Estimator (Version 1.1.2) and reviewed against cost for local Arizona contractors. Reclamation activities and associated costs used in this estimate include:

- Fluid management,
- Earthwork,
- Revegetation,
- Demolition, decontamination and disposal,
- Post-reclamation maintenance/monitoring,
- Mobilization/demobilization,
- Development of a long-term fluid management system, and
- Agency administrative costs.

The direct operational and maintenance costs are calculated to total \$1,632,049. As this is a brownsfield site previously operated by Cyprus Mining Company, MAI plans to only reclaim the areas outlined in the MPO. The cost to reclaim other areas of the site that were undisturbed by MAI are not included in this estimate.

The current MPO update will include a new reclamation cost estimate.

20.7 Social or Community Impact

The Copperstone mine is an established operation with a long history of positive economic impact on the surrounding local communities. The QP knows of no other social or community related requirements or plans for the Project.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The Copperstone Project is focused on developing the existing Copperstone Mine (placed in care and maintenance in 2014) and purchased by Mining Americas Inc. in January, 2025). The Copperstone Project is located in La Paz County, Arizona, roughly 19 miles north of the town of Quartzsite, AZ. The study contemplates refurbishing existing infrastructure, increasing mill capacity from 450s stpd to 600 stpd then to 1000 stpd (in Year 2), and restarting underground mining operations.

A consolidated capital cost estimate was prepared for the Project based on underground mining operations with a process plant capacity of 600 stpd for the first year of operations then increasing to 1000 stpd in year 2. Installing or upgrading supporting infrastructure and increasing tailings capacity over 2 phases (during the sustaining capital period). The existing plant supported a flotation and CIL process flowsheet however all equipment with a few exceptions have been removed. The basis of estimate for the plant contemplates new equipment however the Company expects to consider used equipment to reduce the risk of cost overruns.

Costs are expressed in Q2 2026 US Dollars, with no allowances for price escalation or currency fluctuations. All values are shown in US Dollars unless otherwise noted.

The capital cost estimate was categorized into initial and sustaining capital. Initial capital comprises all capital expenditures (CapEx) incurred prior to the commencement of mill production. Sustaining capital comprises all capital expenditures incurred from the commencement of mill production through to the end of the mine life, including mine closure costs.

Owners Cost have been included in the initial capital indirect costs and include such things as:

- Insurance
- Detailed Engineering
- Pre-production Labour

21.2 Capital Costs

A summary of the initial and sustaining capital costs over the LOM are shown below in Table 21-1.

Table 21-1 Capital Cost Summary

Capital Costs (\$US Millions)	Initial	Sustaining	Total LOM
Underground Mine - Infrastructure	8.4	35.5	43.9
Underground Mine - Development	21.1	10.8	32.0
Tailings Management Facility	0.0	4.9	4.9
Mineral Processing Plant	16.3	0.0	16.3
On-Site Infrastructure	1.4	0.0	1.4
Total Direct Costs	47.3	51.2	98.5
Owner Costs and Reclamation	0.0	5.0	5.0
Project Indirect Costs	3.8	1.9	5.7
Contingency	5.5	3.6	9.0
Total Indirect Costs	9.2	10.5	19.8
Exploration	0.7	8.0	8.8
Total	57.2	69.8	127.0

21.3 Basis of Estimate

The consultants who assisted in contributing to the estimated quantities and capital costs are as follows:

- GR Engineering Services Inc (“GRES”) – General site, Process Plant and Tailings
- Small Mine Development (“SMD”) – Mining

The overall capital cost estimate developed in this Pre-Feasibility Study represents an accuracy range between -25% and +25%.

Table 21-2 Estimate Contributions By Area

Area	Contributor
Mining	SMD/MAI
Site Development	GRES/MAI
Process Plant	GRES
On-site Infrastructure	GRES/MAI
Off-Site Infrastructure	MAI
Project Indirects	SMD/GRES/MAI
EPCM	GRES/MAI
Owners Costs	MAI
Contingency	SMD/GRES/MAI

21.3.1 Major Cost Types

Direct Costs—includes for the supply and construction of all permanent project facilities

- Labour Hours - on-site installation hours to perform the task
- Burdened Labour Costs – Labour hours X burdened labour costs¹
- Bulk Material Costs—material required for the task (e.g., engineered fill, concrete, structural steel, bulk items)
- Plant Equipment—all equipment costs related to the mining, process (e.g. crushers, ball mill, transformers etc...)
- Sub-Contract Costs—catch all category to pick up costs that cannot be split into labour and material (e.g., sub-contractor quotations and lump sum quotations).
- Construction Equipment Costs – Temporary construction equipment needed to construct (cranes, loaders, pickup trucks, forklifts, welding equipment etc...)

Indirect Costs—includes costs that are not readily allocated to individual direct cost items. These costs include both, time-based (driven by the Project schedule) and fixed costs. The following are typical items to be included with indirect costs:

- Common distributable costs (e.g., temporary construction facilities and services, construction equipment, commissioning spares, freight, insurance, and EPCM services)
- Some Owner's costs

Owner's Costs—Costs to be incurred by the Owner during the pre-production phase of the Project.

Contingency—A cost allowance, to cover necessary work within the defined scope of the Project that cannot be identified or itemized at this stage of the Project development.

21.3.2 Estimating Methodology

The estimate has been developed using industry-standard estimating practices appropriate for a PFS-level engineering study and reflects the current level of project definition, engineering design maturity, execution planning, and available market information.

The estimate includes all major components required for mine development and operation, including underground development, surface infrastructure, process facilities, utilities, mobile equipment, construction support, indirect costs, owner's costs, G&A costs and contingency allowances.

¹ Labor rate costs are based on information from the U.S. Department of Labor, Davis-Bacon Wage Determinations, Arizona (La Paz County), AZ20260007/1 <https://sam.gov/wage-determination/AZ20260007/1>

The estimate is based on the following project information and engineering deliverables:

- Mine production schedule and life-of-mine plan
- Underground mine layouts and development sequencing
- Geotechnical and hydrogeological assessments
- Ventilation and dewatering designs
- Process plant design criteria and flowsheets
- Equipment lists and mechanical design criteria
- Surface infrastructure layouts
- Electrical single-line diagrams and load lists
- Preliminary construction schedule
- Vendor budget quotations for major equipment packages
- Contractor pricing and recent comparable project benchmarks
- Current labour, fuel, material, and freight pricing

The estimate is presented in Q2 US Dollars unless otherwise noted.

The estimate was developed using a combination of

- Quantity take-offs
- Unit-rate based estimating
- Vendor budget quotations
- Factored estimating methodologies
- First-principles calculations
- Historical cost benchmarking from comparable mining projects

Where engineering definition permitted, costs were developed from measured quantities and installation unit rates. Where design information remained preliminary, factored or benchmarked-based methodologies were applied using appropriate engineering judgment.

Major equipment pricing was obtained from budgetary supplier quotations where available. In the absence of vendor quotations, pricing was derived from recent comparable and industry databases.

21.4 Initial Capital

Initial capital for this study includes items purchased in Year -1 before production and construction activities. Production is estimated to start in month 1 of year 1. Initial capital costs were developed for the plant upgrade, infrastructure needs, fixed and mobile equipment, IT, G&A and mining. The low initial capital costs for the project reflect the in-place mine development, infrastructure, buildings, and equipment which is a beneficial aspect of the Copperstone mine. Total initial capital is estimated at \$55.9 million.

21.4.1 Initial Mine Capital Cost

Initial mine equipment and mine infrastructure capital costs are presented in Table 21-2 by unit cost, units required and total cost. The mine plan is based on using a mining contractor for the initial pre-production development and continued for the first 10 months of production during Year 1. As a result, the underground mining fleet for the transition to owner mining is included in the sustaining capital costs and not part of the initial capital costs. The initial capital costs include the surface equipment required for the transfer of mineralized material from the portal to the mill, electrical upgrades, ventilation upgrades, dewatering upgrades, remaining underground rehabilitation that needs to take place and the initial mine development. Capital equipment and infrastructure items as outlined in Table 21-2 are also included in the initial capital costs.

Underground mine development costs were estimated on a development meter basis using excavation profiles, ground support classifications, and planned development sequencing from the mine design.

Development unit rates incorporated:

- Drilling and blasting
- Mucking and haulage
- Ground support installation
- Scaling and rehabilitation
- Ventilation installation
- Electrical reticulation
- Water and compressed air services
- Dewatering infrastructure
- Labour, supervision and consumables

Development productivity assumptions were based on:

- Anticipated ground conditions
- Planned mining methods
- Equipment fleet configuration

- Ventilation constraints
- Historical performance from comparable operations

Table 21-3 Initial Mine Department Capital Costs

Area	Description	# Initial Units	# Additional Units	# Total Units	\$/Unit (\$US_1,000's)	Initial Capital Cost (\$US_1,000's)
Mine	Front Loader 4m3 - New	1	0	1	\$837	\$209
Mine	Articulated Truck 40t - New	2	0	2	\$540	\$270
Mine	5,000 gallon water truck - Used	1	0	1	\$120	\$30
Mine	Normet-U/G Fuel/Lube Truck	1	0	1	\$762	\$191
Mine	Caterpillar D4K2 XL-Dozer with Jammer	1	0	1	\$320	\$80
Mine	Surface dozer - D6 Used	1	0	1	\$270	\$68
Mine	Dewatering pumps	3	0	3	\$65	\$195
Mine	Surface Substation	1	0	1	\$916	\$916
Mine	Electrical Cable from main area to pit to ug	1	0	1	\$591	\$591
Mine	Surface facility electrical upgrades	1	0	1	\$295	\$295
Mine	Expanded Dry	1	0	1	\$382	\$382
Mine	Bobcat	1	0	1	\$55	\$55
Mine	Survey Equipment	1	0	1	\$200	\$200
Mine	Software-Mine Planning	2	0	2	\$60	\$120
Mine	Software-AutoCAD	3	0	3	\$2	\$6
Mine	Software-Vent Sim	1	0	1	\$5	\$5
Mine	Computer Hardware	9	1	10	\$3	\$27
Mine	Plotter and printers	1	0	1	\$10	\$10
Mine	Leaky Feeder	1	0	1	\$363	\$363
Mine	UG Mine Contractor Mobilization	1	0	1	\$242	\$242
Mine	Rehab of existing workings to prep for production	1	0	1	\$288	\$288
Mine	Degob various access points	1	0	1	\$153	\$153
Mine	C Zone Failure Area Clean up	1	0	1	\$51	\$51
Mine	Main fan, W-5150-C, 150 hp, 78" x 30" x 1200 rpm	3	0	3	\$46	\$126
Mine	West Portal Vent Door Wall	2	0	2	\$85	\$170
Mine	West Portal Vent Doors	2	0	2	\$59	\$118
Mine	Fan Drift & Slabbing for airlock	1	0	1	\$102	\$102
Mine	Air door & install North/West decline intersect	1	0	1	\$85	\$85
Mine	Underground MPCs	2	0	2	\$345	\$690
Mine	North Portal Construction/North	1	0	1	\$760	\$760
Mine	West Portal Intake Fans 2026	2	0	2	\$79	\$159
Mine	CF Access Fans, years 2026	5	0	5	\$35	\$173
Total Mine						\$7,128
Total Mine Development						\$21,125
Total Initial Mine Capital						\$28,253

21.4.2 Initial Process Plant Capital Cost

The processing method selected is a flotation process followed by cyanide leaching of the flotation tails. New or used equipment for the process plant will be purchased and installed with the exception of the much of

the existing plant steel and electrical, crushing circuit up to and including the fine ore bin, water tanks, and the substation.

Table 21-4 Initial Plant Capital Costs

Area	Description	# Initial Units	# Additional Units	# Total Units	\$/Unit (\$US_1,000's)	Initial Capital Cost (\$US_1,000's)
Plant	Front Loader 4m3- Plant	1	0	0	\$837	\$209
Plant	Constructed costs (field direct & indirect)	1	0	1	\$15,607	\$15,607
Plant	Assay Lab Refurbishment	1	0	1	\$500	\$500
Total Plant						\$16,316

21.4.3 Initial G&A Capital Cost

The General and Administration costs are items that the owner will be managing or supplying with respect to general facility upgrades, software, pre-production labour etc... Table 21-5 lists the initial capital for the G&A infrastructure

Table 21-5 Initial G&A Capital Costs

Area	Description	# Initial Units	# Additional Units	# Total Units	\$/Unit (\$US_1,000's)	Initial Capital Cost (\$US_1,000's)
G&A	Pickups	6	0	6	\$60	\$360
G&A	Septic system Expansion	1	0	1	\$30	\$30
G&A	Technical Services Office Building	1	0	1	\$147	\$147
G&A	Site IT upgrades	1	0	1	\$100	\$100
G&A	Site Access Road - Resurface and Dust Abatement	1	0	1	\$0	\$0
G&A	Warehouse Tool Cage/Room	1	0	1	\$21	\$21
G&A	Refurbish Vehicle Wash Bay	1	0	1	\$16	\$16
G&A	Line Fuel station/storage	1	0	1	\$3	\$3
G&A	Repair fresh water tank	1	0	1	\$35	\$35
G&A	Repair Existing Water Supply System	1	0	1	\$62	\$62
G&A	Upgrade Potable water system	1	0	1	\$25	\$25
G&A	Re-Furbish Guard shack	1	0	1	\$5	\$5
G&A	Install Auto Gates and re-configure Fencing	1	0	1	\$45	\$45
G&A	First Aid Room	1	0	1	\$15	\$15
G&A	First Fill PPE & Safety Gear	1	0	1	\$25	\$25
G&A	Used Ambulance	1	0	1	\$15	\$15
G&A	First Fill Warehouse Tools Allowance	1	0	1	\$85	\$85
G&A	Water Stand at Highway Wells	1	0	1	\$23	\$23
G&A	Operations Staff During Construction	1	0	1	\$399	\$399
Total G&A						\$1,410

21.4.4 Initial Indirects and Contingency Capital Cost

Indirect costs were estimated separately from direct construction costs and include:

- Project management

- Design engineering
- Drafting
- Procurement
- Commissioning services required to construct and commission the processing facility
- As-built drawings
- Spare parts
- First fills and consumables
- Mobilization/demobilization
- Temporary field facilities
- Temporary utilities
- Testing services
- Material storage
- Project supervision
- Administration labour
- Communications
- Light vehicles
- Cleanup
- Safety costs

The overall field indirect cost estimates are outlined in Table 21-5 and 21-6.

Table 21-6 Indirect Cost Breakdown

Item	Basis
Feasibility & Detail Engineering	10% allowance
Procurement	1.5% of fixed plant equipment
Third-party materials testing	Fixed allowance
Startup & Commissioning	1.5% factor
Spare Parts	5% of fixed plant equipment
First Fills	Fixed allowance

Table 21-7 Initial Capital Indirect Costs

Indirect Items	Initial Capital Cost (\$US_1,000's)
Engineering	1,858
Procurement	109
Materials Testing	15
Startup & Commissioning	279
Spare Parts	363
First Fills	412
Applicable Sales Taxes	732
Total EPCM Indirects	3,768

Contingency distributions as per Table 21-8.

Table 21-8 Contingency Allocations

Contingency Item	Basis
Plant Construction	25%
Tailings Construction	25%
Mine Infrastructure, G&A	12%
Mine Equipment	15%
Mine Development	20%

21.5 Sustaining Capital

Sustaining capital has been estimated across all major operational areas, including the underground mine, process plant and tailings storage facility. The scope and timing of sustaining capital expenditures have been developed in accordance with the mine plan, equipment replacement schedules and applicable regulatory requirements. The sustaining capital requirements by year and area are presented in Table 21-9.

Table 21-9 Sustaining Capital Costs

Sustaining Capital \$US_1,000's	Year 1 2027	Year 2 2028	Year 3 2029	Year 4 2030	Year 5 2031	Year 6 2032	Year 7 2033	Year 8 2034	Total
Mine - Equipment	\$16,371	\$7,950	\$7,368	\$3,121	\$710	\$0	\$0	\$0	\$35,519
Mine - Development	\$2,481	\$3,924	\$3,247	\$496	\$693	\$0	\$0	\$0	\$10,841
Plant	\$0	\$0	\$0	\$0	\$0	\$0	\$0	\$0	\$0
G&A	\$0	\$0	\$0	\$0	\$0	\$0	\$0	\$0	\$0
Indirects	\$571	\$792	\$584	\$0	\$0	\$0	\$0	\$0	\$1,948
Tailings	\$2,940	\$33	\$1,900	\$0	\$0	\$0	\$0	\$0	\$4,873
Contingency	\$2,664	\$560	\$181	\$94	\$85	\$0	\$0	\$0	\$3,584
Exploration	\$5,323	\$2,713	\$0	\$0	\$0	\$0	\$0	\$0	\$8,035
Reclamation Closure	\$0	\$0	\$0	\$0	\$0	\$0	\$0	\$5,000	\$5,000
Total Sustaining Capital	\$30,350	\$15,971	\$13,280	\$3,712	\$1,488	\$0	\$0	\$5,000	\$69,801

The following categories of sustaining capital have been considered in the cost estimates presented in this report:

21.5.1 Underground Mine – Equipment and Transition to Owner Mining

Sustaining capital for the underground mine includes provisions for major mobile equipment component rebuilds and additional equipment requirements necessary to sustain planned mining rates throughout the life of mine.

A significant sustaining capital event is planned in Month 10 of Year 1, when a fleet of underground mining equipment will be purchased to facilitate the transition from contract mining to owner-operator mining. The planned mobile equipment fleet at the time of the transition is summarized in Table 16-5. This transition is a key operational milestone and is expected to reduce unit mining costs over the remainder of the mine life relative to the contract mining rates.

The underground mine equipment fleet has been incorporated into the economic model as a capital lease, structured on a three-year term at an interest rate of 10% per annum with a 25% down payment required at the time of acquisition. Lease payments have been modelled accordingly and are reflected in the project cash flow.

21.5.2 Underground Mine - Development and Infrastructure

Sustaining capital requirements also include continued primary mine development necessary to maintain planned advance rates and access to planned ore zones throughout the mine sequence. This includes ongoing development associated with the A Zone portal, which forms a key access and infrastructure component of the underground mine. Sustaining capital provisions for underground development cover drilling, blasting, mucking, ground support, ventilation extensions, and related services required to maintain the development

schedules assumed in the mine plan. The sustaining capital underground development requirements by year and area are presented in Table 21-10.

Table 21-10 Capital Mine Development Requirements and Costs

Development Capital	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Feet	4,678	2,659	12,950	10,714	1,638	2,286		
\$/ft	\$ 4,516	\$ 933	\$ 303	\$ 303	\$ 303	\$ 303	\$ -	\$ -
\$/m	\$ 14,812	\$ 3,061	\$ 994	\$ 994	\$ 994	\$ 994	\$ -	\$ -

21.5.3 Process Plant

There are no sustaining capital purchases for the plant as it is assumed to run without need of upgrade for the current life of mine. Costs associated with equipment maintenance, rebuilds etc... are included in the operating costs.

21.5.4 Tailings Storage Facility

The TSF sustaining capital accounts for the two planned expansion and raise required over the life of the mine to maintain adequate tailings storage capacity (Expansion 1B and raise 2A). Each raise includes costs associated with earthworks, design, construction management, mechanical, piping and instrumentation installation.

21.5.5 Site Infrastructure and General

No sustaining capital has been contemplated for the site infrastructure with respect to replacement of roads, water supply, treatment systems power supply infrastructure etc... beyond the initial capital purchases and construction due to the short mine life and average life expectancy of the systems.

21.5.6 Reclamation and Closure

Reclamation and closure costs have been estimated at \$5 million and are included within the sustaining capital provisions. A reclamation bond in the amount of \$1.6 million has been posted for the Project. Progressive reclamation obligations and any capital expenditure required during operations to fulfill environmental or permit conditions will be further defined and incorporated into a site specific Environmental Management Plan.

All sustaining capital estimates have been developed based on vendor quotations, external engineering assessments, and industry benchmarks where applicable, and are reported in US dollars. Contingency has been applied to sustaining capital line items. The sustaining capital requirements by year and area are presented in Table 21-8.

21.6 Operating Cost Estimates

Operating costs for the Project have been estimated for all aspects of the mining operation, including underground mining, process plant operations, and general and administrative (G&A) expenses. Costs are presented on a per-short ton-mined basis for mining and on a per-short ton-milled basis for processing and G&A and reflect the two distinct operational phases of the Project, an initial contract mining period followed by a transition to owner mining in Month 10 of Year 1. The operating cost estimates by year and area are presented in Table 21-10.

Table 21-11 Total Operating Cost Summary

Operating Costs	\$/oz Au	\$/ton ore
Total Mining	-\$649.05	-\$88.65
Total Processing	-\$157.98	-\$21.58
Total Site G & A	-\$52.89	-\$7.23
Transportation and Refining	-\$13.35	-\$1.82
Bi-product credit (silver)	\$0.00	\$0.00
Cash Operating Costs	-\$873.27	-\$119.28
Royalties	-\$157.50	-\$21.51
Production Taxes	-\$33.60	-\$4.59
Total Operating Costs	-\$1,064.37	-\$145.38
Corporate General/Admin	-\$10.30	-\$1.41
Reclamation cost - prorated	-\$17.17	-\$2.35
Exploration costs-sustaining	-\$2.45	-\$0.34
Capital costs - sustaining	-\$244.12	-\$33.34
All-In-Sustaining-Costs	-\$1,338.41	-\$182.82

21.6.1 Mine Operating Costs

During the initial period of operations, underground mining will be conducted by a contracted mining company. Contract mining costs have been estimated based on a negotiated contract rate and include all labour, equipment, consumables, supervision and associated services required to execute the mining plan. The use of contract mining during the rehabilitation and initial ramp-up period reduces the Project's upfront capital requirements and allows the operation to build operational knowledge prior to transitioning to an owner-operated fleet.

In month 10 of Year 1, the Project transitions to owner mining with the purchase of a dedicated underground mobile equipment fleet, as described in Section 16. Following the transition, mine operating costs reflect the direct costs of an owner-operated workforce and equipment fleet, including labour (operators, supervisors, and maintenance personnel), equipment operating costs (fuel, lubricants, tires, and wear parts), drilling and blasting consumables, ground support materials, and mine services. Owner mining unit costs are expected

to be lower than contract mining rates over the life of mine, reflecting the elimination of contractor margin and the optimization of the fleet to match planned production rates. Mining operating costs also include the costs associated with continued primary and secondary development necessary to maintain adequate ore exposure and access to planned stoping areas in accordance with the mine plan. Development costs include all drilling, blasting, mucking, ground support, ventilation and associated services.

Mine operating costs are calculated from base principals for equipment, consumables, supplies, services and manpower requirements based on the mine schedule. Equipment costs are calculated based on required hours of operation to meet the production schedule and hourly costs for equipment components, supplies, consumables and manpower. Diesel costs were estimated at \$3.50/gallon. Mine maintenance costs are principally based on manufacturer's recommendations, and component replacement and cost. The Company has also used data from operating mines of similar size when developing the operating costs for the mine. The costs detailed by department over the life of the mine are shown in Table 21-12 and Figure 21-1 shows the distribution of these costs by department.

Table 21-12 Mine Operating Costs

Department	Average Yearly Costs	\$/ton Mill Feed	\$/oz Au Sold
Mine G&A	\$3,027,908	\$9.23	\$67.60
Secondary Development	\$5,152,441	\$15.71	\$115.03
UG Stopping	\$9,614,518	\$29.32	\$214.64
Backfill	\$1,372,914	\$4.19	\$30.65
UG Mill Feed Transport	\$720,575	\$2.20	\$16.09
Surface Haulage	\$2,908,818	\$8.87	\$64.94
Mine Services & Maint.	\$3,671,329	\$11.20	\$81.96
Engineering	\$413,422	\$1.26	\$9.23
Geology	\$1,482,392	\$4.52	\$33.09
Contingency	\$709,108	\$2.16	\$15.83
Grand Total	\$29,073,426	\$88.65	\$649.05

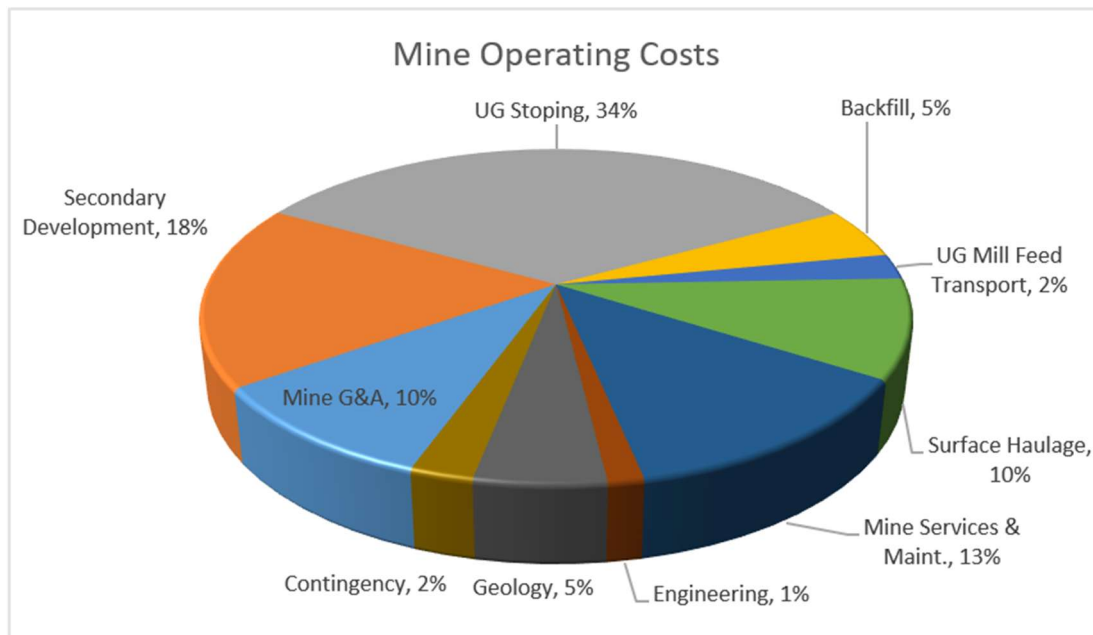


Figure 21-1 Mine Operating Cost Distribution

Table 21-13 shows the distribution of the estimated mine operating costs by cost category for a typical year. On average the mine operating costs are estimated \$90.64 per ton of mineralized material mined or \$663.6 per ounce of gold sold when averaged over the life of mine.

Table 21-13 Mine Operating Costs by Category

Category	Average Yearly Costs	\$/ton	\$/oz Au Sold
Labor & Benefits	\$14,446,754	\$44.05	\$322.51
Labor Alloc	\$0	\$0.00	\$0.00
Energy	\$896,889	\$2.73	\$20.02
Fuel & Lubes	\$3,154,246	\$9.62	\$70.42
Wear Parts	\$503,305	\$1.53	\$11.24
Maint. Parts	\$931,975	\$2.84	\$20.81
Materials/Supplies	\$8,300,196	\$25.31	\$185.30
Blasting Supplies	\$691,119	\$2.11	\$15.43
Service	\$779,889	\$2.38	\$17.41
Lease	\$21,000	\$0.06	\$0.47
Contingency	\$0	\$0.00	\$0.00
Grand Total	\$29,725,374	\$90.64	\$663.60

21.6.2 Plant Operating Costs

Processing costs were estimated by GRES using a detailed equipment-based cost model that incorporated hourly rates for equipment power, supplies, consumables, maintenance and manpower. Reagent usages and wear material estimates were estimated based on factors determined from laboratory testing. Power was estimated at \$0.100/Kwh based on current rates in the area. The costs details by department over the life of the mine are shown in Table 21-14 and Figure 21-2 shows the distribution of these costs by department.

Table 21-14 Process Plant Operating Costs

Department	Average Yearly Costs	\$/ton	\$/oz Au Sold
Laboratory	\$519,618	\$1.58	\$11.60
Crushing & Conveying	\$550,188	\$1.68	\$12.28
Milling	\$1,592,125	\$4.85	\$35.54
Flotation	\$1,480,423	\$4.51	\$33.05
Leaching	\$1,371,875	\$4.18	\$30.63
Reagents	\$539,489	\$1.65	\$12.04
Tailings	\$400,151	\$1.22	\$8.93
Power	\$0	\$0.00	\$0.00
Admin	\$329,634	\$1.01	\$7.36
General Plant	\$290,035	\$0.88	\$6.47
Grand Total	\$7,073,539	\$21.57	\$157.91

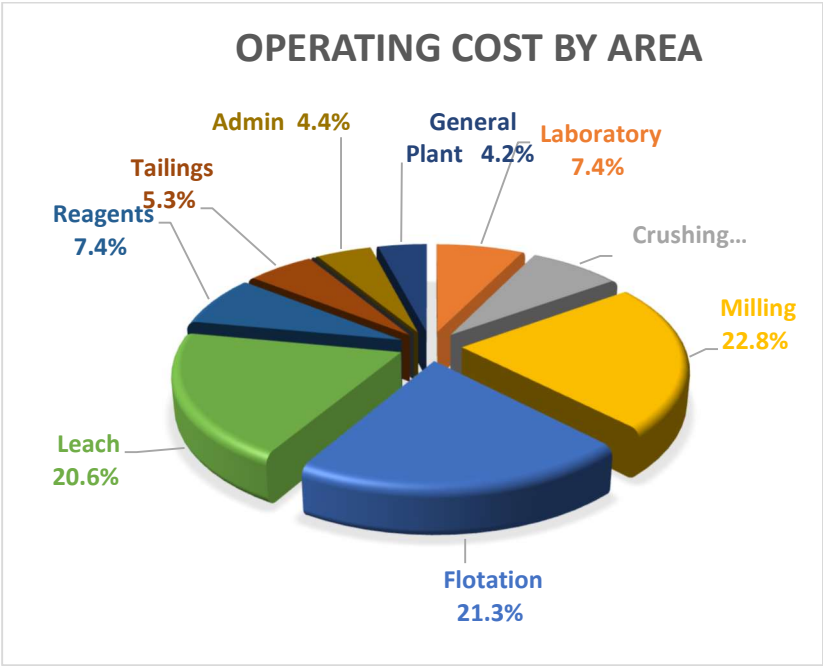


Figure 21-2 Process Operating Costs Distribution

Table 21-15 shows the distribution of the estimated plant operating costs by cost category for a typical year and Figure 21-3 shows the distribution of these costs by category. On average the plant operating costs are estimated \$21.58 per short ton of mineralized material milled or \$157.98 per ounce of gold sold.

Table 21-15 Process Plant Operating Costs by Category

Category	Average Yearly Costs	\$/ton	\$/oz Au Sold
Labor	\$2,771,710	\$8.45	\$61.88
Operating Consumables	\$1,698,833	\$5.18	\$37.93
Power	\$1,576,540	\$4.81	\$35.20
Plant Maintenance	\$634,374	\$1.93	\$14.16
Plant Mobile Equipment	\$31,655	\$0.10	\$0.71
Laboratory	\$360,426	\$1.10	\$8.05
Grand Total	\$7,073,539	\$21.57	\$157.91

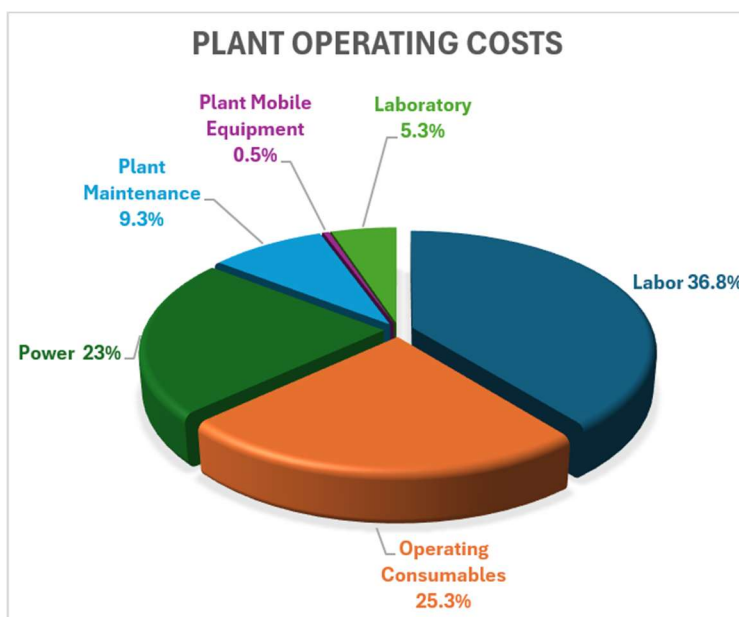


Figure 21-3 Plant Operating Costs Distribution

21.6.3 G&A Operating Cost

The G&A costs have been developed from the Company's knowledge and experience as well as data from similar size operations. The major G&A cost component is staff and labor, but G&A also covers such things as security, office equipment, heat and lighting, communications, overtime, property insurance, office supplies, computer system license fees, admin building maintenance, janitorial services, outside services and allowances for travel and meetings. These costs are fixed as they are not tied directly to the number of

ounces produced or tons milled. The costs details by department over the life of the mine are shown in Table 21-16 and Figure 21-4 shows the distribution of these costs by department. On average the G&A operating costs are estimated to be \$7.23 per ton of mineralized material milled or \$52.89 per ounce of gold sold. In addition to the direct G&A operating costs, variable costs for Property Tax (\$0.40/t, \$2.90/oz) and Mine Service Tax (\$4.19/t, \$30.70/oz) are also considered outside of the table presented.

Table 21-16 G&A Operating Costs

Department	Average Yearly Costs	\$/ton	\$/oz Au Sold
Admin	\$1,052,927	\$3.21	\$23.51
Human Relations	\$103,310	\$0.32	\$2.31
Security & Safety	\$497,799	\$1.52	\$11.11
Accounting	\$93,252	\$0.28	\$2.08
Purchasing	\$160,445	\$0.49	\$3.58
Environmental	\$276,932	\$0.84	\$6.18
Contingency	\$54,617	\$0.17	\$1.22
Grand Total	\$2,239,283	\$6.83	\$49.99

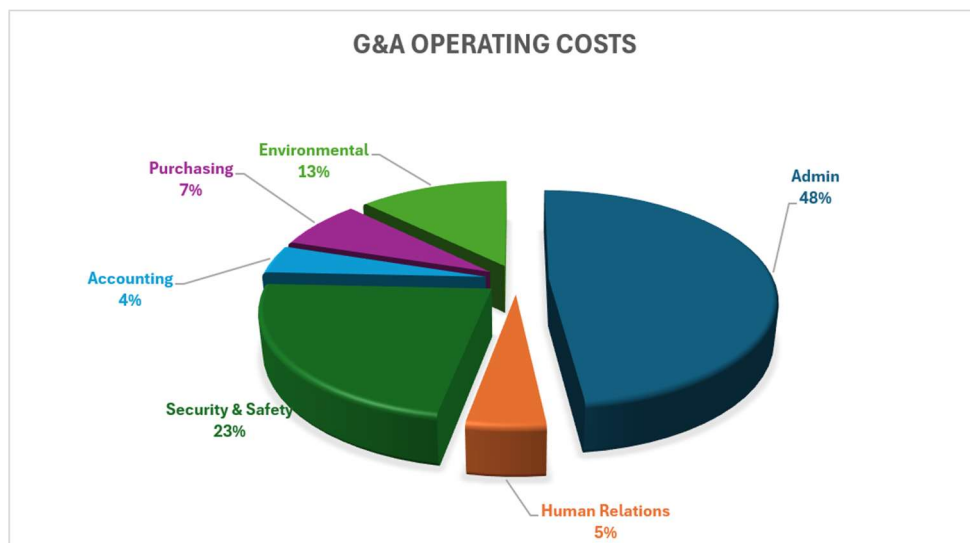


Figure 21-4 G&A Operating Costs Distribution

21.6.4 Labour

Operating labor rates and burdens percentages are based on mines of similar size and compared to the Company's other operation (Pan Mine) as well as industry data for labor rates in mines in Arizona. Staffing

levels and rates for the life of the operation are shown in Table 21-17 through 21-20. Overtime was estimated at 2.5% and payroll burdens were estimated at 38% with a 50% bonus for the underground miners.

Table 21-17 Manpower Requirements for Mine Operations

Manpower Summary				Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
				Rate	2026	2027	2028	2029	2030	2031	2032	2033
Mining G&A												
<u>1</u>	Mine Superintendent	Mine G&A	\$180,000	1	1	1	1	1	1	1	1	0
<u>5</u>	Mine Foreman	Mine G&A	\$145,000	0	2	5	5	5	5	5	5	0
<u>0</u>	Mine Assistant	Mine G&A	\$40,000	0	0	0	0	0	0	0	0	0
	Maint. Superintendent	Mine Mntnc	\$110,000	0	0	0	0	0	0	0	0	0
Mine Salaried					1	3	6	6	6	6	6	0
Mining G&A												
<u>1</u>	Mine Clerk	Mine G&A	\$23.00	1	1	1	1	1	1	1	1	0
Development												
<u>4</u>	Miner 1	Development	\$55.00	3	2	4	4	2	0	2	2	0
<u>7</u>	Miner 2	Development	\$55.00	5	4	6	7	4	0	4	4	0
<u>4</u>	Helper	Development	\$35.00	3	2	4	4	2	0	2	2	0
Development					11	8	14	15	8	8	8	0
Sesondary Development												
<u>2</u>	Jumbo	Ore Access	\$80.00	2	2	2	2	2	2	2	1	0
<u>4</u>	Miner 2	Ore Access	\$45.00	3	4	4	4	4	4	4	2	0
	Miner 3	Raises	\$22.50	0	0	0	0	0	0	0	0	0
Raises					5	6	6	6	6	6	3	0
U/G Production												
<u>11</u>	Miner 1	UG Stoping	\$45.00	8	11	11	11	11	11	11	11	0
<u>11</u>	Miner 2	UG Stoping	\$45.00	8	11	11	11	11	11	11	11	0
<u>3</u>	Helper	UG Stoping	\$35.00	2	3	3	3	3	3	3	3	0
<u>3</u>	Backfill	Backfill	\$35.00	2	3	3	3	3	3	3	2	0
<u>0</u>	Backfill helper	Backfill	\$25.00	0	0	0	0	0	0	0	0	0
<u>4</u>	Blaster	UG Stoping	\$40.00	1	2	3	3	3	3	3	1	0
Stoping					22	30	31	31	31	31	28	0
Haulage												
	Miner 1	Surface Ore Ha	\$30.00	4	6	8	8	8	8	6	4	0
	Miner 2	Ore/Waste Trai	\$25.00	0	0	0	0	0	0	0	0	0
<u>2</u>	Miner 3	Ore/Waste Trai	\$35.00	4	4	6	6	5	4	4	2	0
<u>2</u>	Miner 4	Ore/Waste Trai	\$35.00	4	4	6	6	5	4	4	2	0
<u>0</u>	Hoistman	Ore/Waste Trai	\$32.00	0	0	0	0	0	0	0	0	0
Haulage					12	14	20	20	17	16	14	8
Mine Services												
<u>3</u>	U/G Helper - Supplies&Cleanup	Mine Services	\$30.00	2	3	3	3	3	3	3	2	0
Mine Maintenance												
<u>3</u>	Lead Mechanic	UG Stoping	\$65.00	2	3	3	3	3	3	3	3	0
<u>3</u>	Heavy Equipment Mechanic	UG Stoping	\$52.00	2	3	3	3	3	3	3	3	0
<u>2</u>	Mechanic Helper	UG Stoping	\$22.50									
<u>6</u>	Mechanic/Welder	UG Stoping	\$45.00	5	6	6	6	6	6	6	6	0
<u>0</u>	Planner/Clerk	UG Stoping	\$19.50	0	0	0	0	0	0	0	0	0
<u>2</u>	Electrician	UG Stoping	\$55.00	2	2	2	2	2	2	2	2	0
Total Mine Maintenance					11	14	14	14	14	14	14	0
Total Mine Operations					64	78	95	97	86	77	83	69

Table 21-18 Manpower Requirements for Engineering and Geology

Manpower Summary				Rate	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
					2026	2027	2028	2029	2030	2031	2030	2031	2032
<u>Engineering</u>													
<u>1</u>	Sr Mining Engineer	Engineering	\$125,000	1	1	1	1	1	1	1	1	1	0
<u>1</u>	Jr Mining Engineer	Engineering	\$105,000	1	1	1	1	1	1	1	1	1	0
<u>0</u>	Chief Surveyor	Engineering	\$110,000	0	0	0	0	0	0	0	0	0	0
<u>1</u>	Surveyor Tech	Engineering	\$24.00	1	1	1	1	1	1	1	1	1	0
	Engineering			2	3	3	3	3	3	3	3	2	0
<u>Geology & Grade Control</u>													
<u>1</u>	Sr Geologist	Geology	\$125,000	1	1	1	1	1	1	1	1	1	0
<u>2</u>	Ore Control Geologist	Geology	\$125,000	2	2	2	2	2	2	2	2	1	0
<u>0</u>	Assayer	Assay Lab	\$25.00	0	0	0	0	0	0	0	0	0	0
<u>0</u>	Sample Prep	Assay Lab	\$23.00	0	0	0	0	0	0	0	0	0	0
<u>3</u>	Sampler	Geology	\$24.00	2	3	3	3	3	3	3	3	2	0
	Geology			5	6	6	6	6	6	6	6	4	0
Total Mine OP Eng Geo					7	9	9	9	9	9	9	5	0

Table 21-19 Manpower Requirements for Process Plant Operations

Manpower Summary				Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
				Rate	2026	2027	2028	2029	2030	2031	2032	2033
Processing Plant												
Salary Personnel												
<u>1</u>	Plant Superintendent	Plant G&A	\$165,000	0	1	1	1	1	1	1	1	0
<u>2</u>	Plant Foreman	Plant G&A	\$115,000	0	2	2	2	2	2	2	1	0
<u>0</u>		Primary Crush	\$75,000	0	0	0	0	0	0	0	0	0
<u>1</u>	Metallurgist	Floatation	\$110,000	0	1	1	1	1	1	1	1	0
<u>0</u>	Plant Foreman	Grinding	\$90,000	0	0	0	0	0	0	0	0	0
<u>0</u>	Plant Foreman	Thickening/Co	\$90,000	0	0	0	0	0	0	0	0	0
<u>1</u>	Assay Lab Manager	Assay Lab	\$110,000	0	1	1	1	1	1	1	1	0
<u>1</u>	Plant Mntnce Forman	Plant Mntnc	\$135,000	0	1	1	1	1	1	1	1	0
Process Salary				0	6	6	6	6	6	6	5	0
Hourly Personnel												
<u>1</u>	Instrument Technician	Plant G&A	\$45.00	0	1	1	1	1	1	1	1	0
<u>2</u>	Crushing Operators	Primary Crush	\$33.08	0	2	2	2	2	2	2	1	0
		Primary Crush	\$33.08	0	0	0	0	0	0	0	0	0
		Primary Crush	\$33.08	0	0	0	0	0	0	0	0	0
	Laborers	Primary Crushing	\$33.08	0	0	0	0	0	0	0	0	0
<u>4</u>	Operator	Floatation	\$33.08	0	4	4	4	4	4	4	2	0
		Floatation	\$17.00	0	0	0	0	0	0	0	0	0
<u>2</u>	Plant Operator	Grinding	\$33.08	0	2	2	2	2	2	2	1	0
		Grinding	\$33.08	0	0	0	0	0	0	0	0	0
		Grinding	\$33.08	0	0	0	0	0	0	0	0	0
		Grinding	\$33.08	0	0	0	0	0	0	0	0	0
<u>2</u>	Plant Operator	Thickening/Co	\$33.08	0	2	2	2	2	2	2	1	0
		Thickening/Co	\$33.08	0	0	0	0	0	0	0	0	0
<u>0</u>	Plant Operator	Paste Backfill	\$26.00	0	0	0	0	0	0	0	0	0
		Paste Backfill	\$20.90	0	0	0	0	0	0	0	0	0
<u>0</u>	Plant Operator	Refinery	\$33.08	0	0	0	0	0	0	0	0	0
		Refinery	\$33.08	0	0	0	0	0	0	0	0	0
		Refinery	\$33.08	0	0	0	0	0	0	0	0	0
<u>4</u>	Assayer	Assay Lab	\$32.32	0	4	4	4	4	4	4	4	0
<u>2</u>	Sample Prep	Assay Lab	\$19.00	0	2	2	2	2	2	2	2	0
<u>2</u>	Mechanic	Plant Mntnc	\$36.79	0	2	2	2	2	2	2	1	0
<u>2</u>	Mechanic Helper	Plant Mntnc	\$31.70	0	2	2	2	2	2	2	1	0
<u>1</u>	Electrician	Plant Mntnc	\$36.79	0	1	1	1	1	1	1	1	0
Process Hourly				0	22	22	22	22	22	22	15	0
Total Process Operations				0	28	28	28	28	28	28	20	0

Table 21-20 Manpower Requirements for G&A and Totals for Project

Manpower Summary			Rate	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
				2026	2027	2028	2029	2030	2031	2030	2031	2032
General and Administrative												
0	Mine Manager	Admin	\$210,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0.5	Environmental Manager	Environmenta	\$145,000	0.0	0.5	0.5	0.5	0.5	0.5	0.5	0.3	0.0
0.5	HR Manager	Human Relatic	\$125,000	0.0	0.5	0.5	0.5	0.5	0.5	0.5	0.3	0.0
1	Safety Superintendent	Security & Safe	\$150,000	0.8	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.0
0	Safety Clerk	Security & Safe	\$70,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0.5	Controller	Accounting	\$124,000	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.3	0.0
0.5	Purchasing Manager	Purchasing	\$110,000	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.3	0.0
1	Environmental Tech	Environmenta	\$100,000	0.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.0
0	IT Tech	Admin	\$75,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0	Payroll	Accounting	\$27,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0	Accounts Payable	Accounting	\$27,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0	Admin Assistant	Admin	\$17,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0	Janitor	Admin	\$18,000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
4	Safety/Security	Security & Safe	\$28,000	3.0	4.0	4.0	4.0	4.0	4.0	4.0	2.0	0.0
1	Warehousemen	Purchasing	\$30,000	0.8	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.0
Total G&A				5	9	9	9	9	9	9	6	0
Total Salary				6	18	21	21	21	21	21	17	0
Total Hourly				70	107	120	122	111	102	108	83	0
Total Operations				76	124	141	143	132	123	129	100	0

22 ECONOMIC ANALYSIS

22.1 Cautionary Statement

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates
- Assumed commodity prices and exchange rates
- Mine production plans
- Projected recovery rates
- Sustaining and operating cost estimates
- Assumptions as to closure costs and closure requirements
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed
- Unrecognized environmental risks
- Unanticipated reclamation expenses
- Unexpected variations in quantity of mineralized material, grade, or recovery rates
- Geotechnical and hydrogeological considerations during mining being different from what was assumed
- Failure of plant, equipment, or processes to operate as anticipated
- Accidents, labour disputes, and other risks of the mining industry.

22.2 Methodologies Used

The Project has been evaluated using a discounted cashflow (DCF) analysis based on a 5% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including pre-production costs, operating costs, taxes, royalties, and streaming costs. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the midpoint of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metals price, discount rate, head grade, recovery, total operating cost, and total capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 21 in Q2 2026 USD. The exchange rate used in the economic model is 1.38 CAD/USD. The economic analysis has been performed on a constant dollar basis with no inflation or escalation.

22.3 Assumptions

The economic analysis assumes the Project is a 100% equity basis. All dollar amounts in this analysis are expressed in US dollars (US\$), unless otherwise specified.

The economic analysis includes the entire Project life. The valuation date on which the net present value (NPV) and internal rate of return (IRR) are measured is the start of Year -1.

Details of the capital and operating cost estimates are described in Section 21. The production schedule used for the economic analysis is described in Section 16.

Total mine life is 7.5 years (1 year of construction, 6.5 year of operation)

All metal products will be sold in the same year they are produced

Project revenue will be derived from the sale of gold concentrate and gold doré

Currently, there are no contractual refining arrangements or concentrate sales agreements

22.4 Introduction

The economic evaluation of the Project was carried out using a financial model developed by Mining Americas based on production schedule, application of operating, capital, and sustaining costs, as discussed earlier in this document. The financial model generated an estimate of annual pre-tax and post-tax cash flows and Mining Americas has undertaken an internal review of both pre-tax and post-tax economic evaluation models.

Key economic indicators were developed for both pre-tax and post-tax, and include the Project's NPV, IRR, and payback period (time in years to recover the initial capital investment once operations commence). The base-case results apply the following key inputs:

- Gold: \$3,500/oz
- Discount Rate: 5%
- Exchange Rate: C\$ 1.00 to US\$ 0.72

Table 22-1 summarizes the base case results of the analysis

Table 22-1 Analysis Base Case Results

Operating Costs	Unit	Value
Average Milling Rate	t/d	600
Total Mining	\$/t milled	88.65
Total Processing	\$/t milled	21.58
Total Site G & A	\$/t milled	7.23
Transportation and Refining	\$/t milled	1.82
Royalties	\$/t milled	21.51
Production Taxes	\$/t milled	4.59
Corporate General/Admin	\$/t milled	1.41
Reclamation cost - prorated	\$/t milled	2.35
Exploration costs-sustaining	\$/t milled	0.34
Capital costs - sustaining	\$/t milled	20.93
Pre-Tax NPV 5.0%	US\$ Million	\$425.73
Pre-Tax Internal Rate of Return	%	124.1%
Pre-Tax Payback Period	Years	1.11
Pre-Tax Payback Multiple	number	10.30
After NPV 5.0%	US\$ Million	\$341.5
After Internal Rate of Return	%	110.5%
After Payback Period, Years	Years	1.13
After Payback Multiple	number	7.87

22.4.1 Taxes

State, local, and federal taxes, including income taxes and the Arizona Transaction Privilege Tax, Mining (Severance Tax), and tax loss carry-forwards have been considered in this study, and are included in the economic analysis.

22.4.2 Working Capital

An estimate of working capital has been incorporated into the economic analysis based on one month's forecasted operating costs. The allowance is intended to provide sufficient liquidity to fund operating expenditures during the period between commencement of operations and receipt of revenues. The estimate includes expenditures associated with mining, processing, general and administrative activities, consumables, and other routine operating costs. Working capital excludes depreciation, financing costs, income taxes, and other non-cash items. The working capital requirement is estimated at \$3.5 million, based on the first full year of forecast operating costs.

22.4.3 Royalties

A 1.5% production royalty and the existing Streaming Agreement, which receives 4% of annual gold production at a gold price of 25% of spot have been calculated on the gross proceeds, less transportation and refining costs as required by underlying agreements provided by MAI.

22.4.4 Transportation and Refining Charges

The following off-site and sales terms used for the economic analysis were determined using industry benchmarks and standard industry practices summarized in Table 22-2.

Table 22-2 Transportation and Refining Charges

Description	Units	Value
Au Payble in Dore	%	99.9
Flotation Au Payable	%	99.7
Flotation Refining Charge	\$/oz	10
Flotation Shipping Charge	\$/mt	225

22.4.5 Corporate Income Taxes

United States and State corporate taxes have been considered in the economic analysis.

22.5 Economic Analysis

The projected total lifespan of the Project is 7.5 years: 1 year of pre-production and construction, and 6.5 years of operations. Approximately 302,505 oz of gold is projected to be mined, with 291,161 oz recovered and produced for sale. An initial capital investment of \$57.2 million, including contingency/working capital is projected. Following the AISC guidelines, life-of-mine average base case Cash Operating Cost is projected to be \$873/oz of gold sold. The AISC life-of-mine average base case Total Operating Cost (including royalties and production taxes), is expected to be \$1,064/oz. The AISC is projected to be \$1,338/oz.

Table 22-3 Copperstone Project Total Operating Cost/ounce Gold & per ton Mill Feed

Operating Costs	\$/oz Au	\$/ton ore
Total Mining	-\$649.05	-\$88.65
Total Processing	-\$157.98	-\$21.58
Total Site G & A	-\$52.89	-\$7.23
Transportation and Refining	-\$13.35	-\$1.82
Bi-product credit (silver)	\$0.00	\$0.00
Cash Operating Costs	-\$873.27	-\$119.28
Royalties	-\$157.50	-\$21.51
Production Taxes	-\$33.60	-\$4.59
Total Operating Costs	-\$1,064.37	-\$145.38
Corporate General/Admin	-\$10.30	-\$1.41
Reclamation cost - prorated	-\$17.17	-\$2.35
Exploration costs-sustaining	-\$2.45	-\$0.34
Capital costs - sustaining	-\$244.12	-\$33.34
All-In-Sustaining-Costs	-\$1,338.41	-\$182.82

As previously mentioned, the gold price used in the economic analysis (US \$3,500/oz Au). The economic analysis was done on an annual cashflow basis; the cashflow output is shown in Table 22-4.

Table 22-4 Copperstone Project Schedule and Cash Flow

Note: All Dollars are in US	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Life-of-Mine
Production Schedule										
MINE PRODUCTION										
Tons Ore Mined	47,619	276,505	411,543	419,032	414,340	407,245	137,757	17,586	-	2,131,627
Au, oz/tn	0.13	0.19	0.15	0.14	0.12	0.12	0.13	0.13	-	0.142
Tons LG Ore Mined	-	-	-	-	-	-	-	-	-	-
Au, g/tn	-	-	-	-	-	-	-	-	-	-
Development Feet	8,190	10,480	29,060	32,962	17,350	17,285	4,879	3,826	-	124,031
Development Waste	112,799	93,645	350,285	362,203	1,569	192	40,446	33,467	-	1,298,407
Total Tons Mined	160,419	376,150	761,628	781,235	558,909	562,437	178,203	52,054	0	3,430,034
Stockpile Balance	-	-	-	-	-	-	-	-	-	-
Ag, g/tn	-	-	-	-	-	-	-	-	-	-
Au, g/tn	-	-	-	-	-	-	-	-	-	-
PROCESS PRODUCTION										
Tons Ore Processed	-	209,844	366,015	365,015	365,015	365,015	366,015	94,910	-	2,131,627
Au, oz/t	-	0.194	0.153	0.142	0.123	0.121	0.139	0.144	-	0.142
Income Statement										
REVENUE										
Contained Oz Au to Mill	-	40,737	55,956	51,749	44,953	44,339	51,058	13,713	-	302,505
Saleable Oz Au, post 99.9% Refinery credit	-	39,209	53,857	49,809	43,267	42,677	49,144	13,199	-	291,161
Gross Revenue	-	137,231,123	188,600,613	174,330,521	151,435,787	149,367,916	172,002,966	46,195,816	-	1,019,064,742
Transportation and Refinery Charges	-	634,466	804,255	757,329	681,508	674,660	253,716	79,798	-	3,865,732
Net Refined Revenue	-	136,596,657	187,696,358	173,573,192	150,754,279	148,693,256	171,749,250	46,116,018	-	1,015,179,010
Royalties	-	2,058,467	2,827,509	2,614,958	2,271,537	2,240,519	2,680,044	692,937	-	15,285,971
Star Royalties- Royalty	-	4,116,934	5,655,018	5,229,916	4,543,074	4,481,037	5,160,089	1,385,874	-	30,571,942
Net Revenue	-	130,421,256	179,213,830	165,728,319	143,939,669	141,971,708	164,009,117	44,037,206	-	969,321,097
OPERATING EXPENSES										
Mining										
Mine G&A	-	437,758	2,472,306	3,203,381	3,194,942	3,194,942	3,194,942	788,186	-	19,681,401
Secondary Development	-	1,456,230	7,582,347	5,161,493	6,671,964	5,181,455	5,018,636	327,998	-	33,490,868
UG Stopping	-	1,707,964	14,378,759	9,587,106	9,633,685	9,735,682	9,639,543	6,445,550	-	1,366,076
UG Ore Transport	-	328,455	2,514,216	1,283,981	1,301,122	1,279,155	1,216,899	865,377	-	8,923,943
Surface Haulage	-	117,931	711,750	952,308	967,250	967,250	967,250	967,250	-	4,883,738
Mine Services & Maint	-	-	1,844,550	2,985,155	2,954,299	3,148,310	3,162,785	3,872,425	-	18,907,316
Backfill	-	447,876	4,266,586	4,278,753	4,346,990	4,285,695	4,225,169	1,744,773	-	23,863,637
Engineering	-	110,776	440,001	441,036	440,001	440,001	313,272	62,154	-	2,687,243
Geology	-	303,339	1,403,443	1,712,940	1,728,050	1,717,364	1,701,205	905,137	-	9,635,551
Contingency, @ 2.5%	-	122,758	890,351	740,155	780,458	748,746	739,160	485,806	-	4,609,205
Total Mining	-	5,633,087	36,504,309	30,346,308	31,996,782	30,698,600	30,305,574	19,918,027	-	188,977,267
Processing & ROM handling										
Labour	-	3,077,929	3,054,389	3,077,929	3,077,929	3,077,929	2,520,508	612,409	-	18,499,024
Laboratory	-	224,319	391,636	390,666	390,666	390,666	391,636	101,554	-	2,280,841
Crushing & Conveying	-	96,436	168,367	167,907	167,907	167,907	168,367	43,659	-	980,549
Milling	-	761,008	1,328,633	1,325,003	1,325,003	1,325,003	1,328,633	344,524	-	7,737,807
Flotation	-	693,922	1,211,508	1,208,198	1,208,198	1,208,198	1,211,508	314,153	-	7,055,686
Leaching	-	626,836	1,094,384	1,091,394	1,091,394	1,091,394	1,094,384	283,781	-	6,373,565
Reagents	-	102,726	179,347	178,857	178,857	178,857	179,347	46,506	-	1,044,497
Tailings	-	12,579	21,961	21,901	21,901	21,901	21,961	5,695	-	127,898
Power	-	-	-	-	-	-	-	-	-	-
Admin	-	6,289	10,980	10,950	10,950	10,950	10,980	2,847	-	63,949
General	-	180,294	314,773	313,913	313,913	313,913	314,773	81,623	-	1,533,199
Contingency, @ 0.0%	-	-	-	-	-	-	-	-	-	-
Total Processing	-	5,782,338	7,775,978	7,786,618	7,786,618	7,786,618	7,242,097	1,636,750	-	45,997,016
Site General & Administration										
Admin	-	1,105,589	1,108,618	1,105,589	1,105,589	1,105,589	1,054,964	258,090	-	6,844,026
Human Relations	-	115,970	116,288	115,970	115,970	115,970	77,049	14,298	-	671,514
Security & Safety	-	542,873	544,360	542,873	542,873	542,873	428,135	91,710	-	3,235,695
Accounting	-	104,680	104,967	104,680	104,680	104,680	69,549	12,906	-	696,140
Purchasing	-	173,232	173,706	173,232	173,232	173,232	144,207	32,053	-	1,042,893
Environmental	-	294,294	295,100	294,294	294,294	294,294	261,689	66,095	-	1,800,060
Contingency, @ 2.5%	-	58,412	58,576	58,416	58,416	58,416	50,890	11,879	-	355,005
Total G&A	-	2,395,049	2,401,615	2,395,053	2,395,053	2,395,053	2,086,481	487,031	-	14,555,334
Property Tax										
Property Tax	-	82,793	120,955	125,655	107,008	100,520	94,090	87,455	-	844,054
Mine Severance Tax	-	-	1,059,879	1,682,438	1,516,963	1,271,341	1,256,965	1,702,703	-	8,938,947
Total Operating Costs	-	5,115,880	45,862,530	42,331,916	43,822,950	42,558,620	41,844,730	31,043,398	-	259,312,617
Operating Margin (EBITDA)	-	5,115,880	84,558,727	136,881,914	121,905,368	101,681,049	100,126,970	132,965,719	-	710,008,479
Development Deduction	-	14,787,543	1,736,881	2,746,832	2,272,584	347,439	484,857	-	-	22,376,135
Amortization	-	633,752	708,190	825,911	923,307	938,198	958,977	3,642,460	-	9,589,772
Depreciation	-	-	7,003,709	12,451,878	9,918,064	8,059,735	7,548,264	6,192,339	-	54,805,548
Reclamation Deduction	-	-	-	-	-	-	-	-	-	-
Salvage Income	-	-	-	-	-	-	-	-	-	-
Interest Expense	-	102,141	1,028,788	1,132,776	540,478	57,342	-	-	-	2,861,524
Income - before NOL & Perc Depletion	-	20,639,315	74,081,159	119,724,516	108,250,936	92,278,336	91,134,873	125,814,403	-	620,375,500
Net Operating Loss Adjustment	-	20,639,315	74,081,159	17,258,156	-	-	-	-	-	70,700,000
Depletion	-	-	-	26,882,075	24,859,248	21,590,950	21,295,755	24,601,368	-	125,834,977
State Income Tax	-	-	-	5,290,800	5,837,418	4,948,117	4,888,736	7,084,912	-	29,668,636
Federal Income Tax	-	-	-	15,872,700	16,286,307	13,805,246	13,639,580	19,766,906	-	84,227,081
Taxable Income, less Tax	-	-	54,420,685	61,267,873	51,934,023	51,310,800	51,310,800	74,361,217	-	309,944,606
Cash Flow Calculation										
Adjustments for Non Cash Items										
Development Deduction	-	14,787,543	1,736,881	2,746,832	2,272,584	347,439	484,857	-	-	22,376,135
Amortization	-	633,752	708,190	825,911	923,307	938,198	958,977	3,642,460	-	9,589,772
Depreciation/Reclamation/Salvage	-	-	7,003,709	12,451,878	9,918,064	8,059,735	7,548,264	6,192,339	-	54,805,548
Net Operating Loss Adjustment	-	20,639,315	74,081,159	17,258,156	-	-	-	-	-	70,700,000
Depletion	-	-	-	26,882,075	24,859,248	21,590,950	21,295,755	24,601,368	-	125,834,977
Total Adjustments for Non Cash Items	-	5,216,020	83,529,939	60,164,852	37,973,202	30,936,321	30,287,852	31,752,684	-	283,306,432
Capital										
Investment - Mine	-	7,128,012	-	-	-	-	-	-	-	7,128,012
Investment - Primary Development	-	21,125,061	-	-	-	-	-	-	-	21,125,061
Investment - Exploration	-	714,583	-	-	-	-	-	-	-	714,583
Investment - Plant	-	16,316,125	-	-	-	-	-	-	-	16,316,125
Investment - G&A	-	1,410,300	-	-	-	-	-	-	-	1,410,300
Capital Indirects	-	3,768,100	-	-	-	-	-	-	-	3,768,100
Capital Contingency	-	5,450,435	-	-	-	-	-	-	-	5,450,435
Total Capital	-	55,912,616	-	-	-	-	-	-	-	55,912,616
Sustaining Capital - Mine	-	-	6,812,986	1,094,110	927,556	782,953	709,804	-	-	10,327,405
Sustaining Capital - Primary Development	-	-	2,481,259	3,924,046	3,246,548	496,342	692,652	-	-	10,840,846
Sustaining Capital - Exploration	-	-	5,322,914	2,712,503	-	-	-	-	-	8,035,417
Sustaining Capital - Plant	-	-	2,940,264	33,036	1,900,000	-	-	-	-	4,873,300
Sustaining Capital - G&A	-	-	-	-	-	-	-	-	-	-
Sustaining Capital - Indirects	-	-	571,356	792,107	584,337	-	-	-	-	1,947,800
Sustaining Capital - Contingency	-	-	2,663,540	559,968	181,427	93,954	85,176	-	-	3,584,066
Reclamation Closure Costs	-	-	-	-	-	-	-	-	5,000,000	5,000,000
Salvage Value	-	-	-	-	-	-	-	-	-	-
Total Capital & Sustaining	-	55,912,616	20,792,318	9,115,771	6,839,868	1,373,250	1,487,632	-	5,000,000	100,521,4

22.6 Sensitivity Analysis

22.6.1 Price

The Project, like almost all precious metals projects, is very responsive to changes in the price of its chief commodity, gold. From the base case, a change in the average gold price of US\$200/oz Au are reflected in Figure 22-1. Table 22-5 also shows the economic sensitivities, due to the change in gold price, Net Cash Flow, Net Present Value at 5%, the Internal Rate of Return, the Payback Period, and the Payback Multiple.

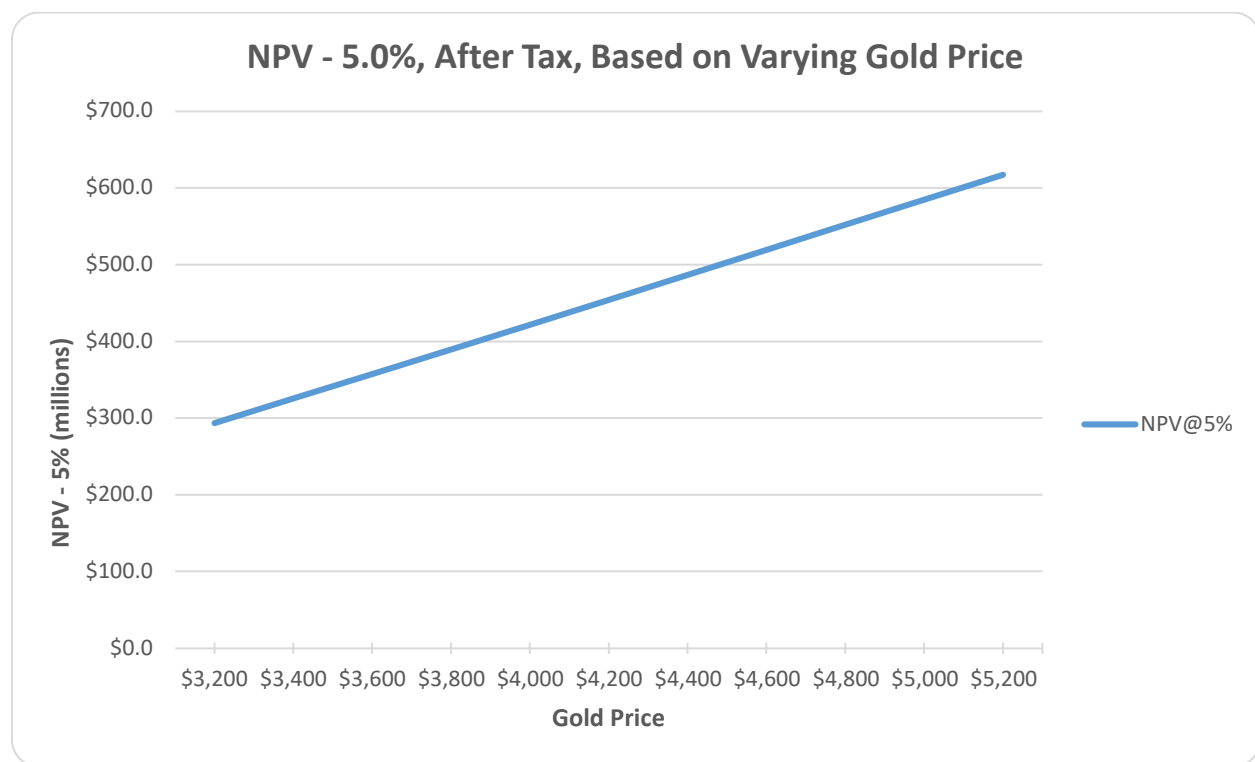


Figure 22-1 NPV Sensitivity

Table 22-5 Gold Price Sensitivity Economic Results

Au Price	Net Cash Flow, millions	NPV-5.0%, millions	IRR	Payback Period, Years	Payback Multiple
\$3,500	\$466.26	\$341.48	110.5%	1.13	7.87
\$3,200	\$403.5	\$293.6	95.8%	1.3	6.9
\$3,400	\$445.3	\$325.5	105.6%	1.2	7.5
\$3,600	\$487.2	\$357.4	115.5%	1.1	8.2
\$3,800	\$529.0	\$389.3	125.5%	1.0	8.8
\$4,000	\$571.1	\$421.3	135.5%	0.9	9.5
\$4,200	\$613.9	\$454.0	145.4%	0.8	10.2
\$4,400	\$656.8	\$486.7	155.4%	0.8	10.9
\$4,600	\$699.6	\$519.3	165.3%	0.7	11.6
\$4,800	\$742.5	\$552.0	175.3%	0.7	12.3
\$5,000	\$785.4	\$584.7	185.3%	0.6	13.0
\$5,200	\$828.2	\$617.4	195.3%	0.6	13.7

22.6.2 Operating and Capital Costs Sensitivity

The Project is somewhat sensitive to the cost of operations, incurring an approximately 3% decline in the NPV-5 for each increase of 10% in the operating costs. The Project is less sensitive to variances in the cost of capital, experiencing about 1.3% in decline in the NPV-5 for each increase of 10% in the capital costs, as shown in Figure 22-2.

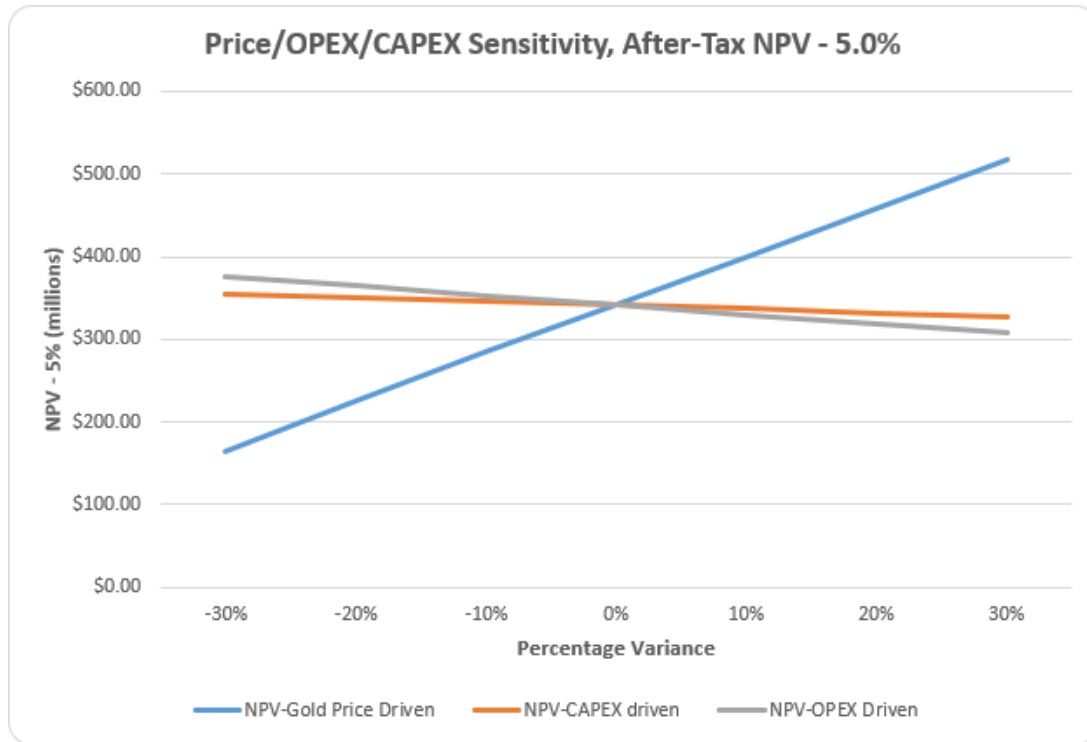


Figure 22-2 Project Operating Cost & Capital Cost Sensitivity Analysis

22.7 Conclusion

The Project would be economically viable based on the parameters considered in this study. The base case scenario produces approximately 291,161 salable ounces of gold over a 6.5-year production period. The Project is most sensitive to the gold price, but sensitive to a lesser extent to capital and operating costs.

The base case economic analysis of the Project at a gold price of US\$3,500/oz shows positive economics with a \$341.48 million post-tax NPV (5%) and 110.5% post-tax IRR. The PFS supports a decision to carry out additional detailed studies.

23 ADJACENT PROPERTIES

The QP knows of no adjacent properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Copperstone Project.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Star Royalties Gold Purchase and Sale Agreement

On October 31, 2023, Mining Americas Inc., through its wholly owned subsidiary, entered into an amended and restated Gold Purchase and Sale Agreement with Star Royalties Ltd². (the "Star Stream Agreement"). Under the terms of the agreement, Star Royalties is entitled to purchase 4.0% of the payable gold produced from the Copperstone Project for the life of the mine.

The agreement provided Mining Americas with an upfront payment to support the advancement and development of the Copperstone Project. Until the upfront deposit has been fully credited through gold deliveries, Star Royalties pays the prevailing market price for streamed gold, with a portion of the purchase price applied against the outstanding deposit balance in accordance with the terms of the agreement. Following repayment of the deposit, Star Royalties is entitled to purchase its attributable gold at a price equal to 25% of the prevailing market price.

The streaming obligation constitutes a material commercial agreement affecting revenues generated by the Copperstone Project. The financial effects of the Star Stream Agreement, including the reduced realized revenue associated with streamed ounces, have been incorporated into the economic analysis presented in Section 22 of this Technical Report.

24.2 Patch Royalty

Other than the Star Stream Agreement and the royalties described elsewhere in this report, the Qualified Persons are not aware of any additional material commercial agreements that would reasonably be expected to materially affect the conclusions of this Technical Report.

² As of July 3, 2026 Star Royalties has been acquired by Summit Royalties Ltd. ("Summit") (TSXV: SUM, OTCQX: SUMMF)

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

Results of the PFS indicate that the Project shows positive economics, benefitting from significant in-place infrastructure, underground development and permitting. There are numerous targets to potentially increase mineral resources, and economic grade mill feed appears to be readily available for mining and processing. The base case scenario produces approximately 292,000 salable ounces of gold over a 6.5-year period. The Project is most sensitive to the gold price but sensitive to a lesser extent to operating and capital costs. The base case economic analysis of the Project at a gold price of US\$3,500/oz shows positive economics with a \$341 million post-tax NPV (5%) and 110.5% post-tax IRR.

25.2 Geology and Mineral Resource Estimate

The Copperstone deposit is a Cretaceous to mid-Tertiary, low-angle, multi-stage fault-related gold deposit. Mineralization is predominantly controlled by the northwest trending shallow angle Copperstone fault and shear zone. These structures are not confined to any lithologic unit, although the majority of mineralization is hosted in quartz latite porphyry. Breccia textures as well as chloritization, silicification, and hematite and specularite flooding are reliable indications of gold mineralization.

The independent data verification program completed by the Qualified Person demonstrates that the Copperstone Project assay database is of sufficient quality and integrity to support Mineral Resource estimation. Verification of 4,871 assay intervals, representing approximately 5.35% of the total database, exceeded the generally accepted industry standard for independent spot-check verification and included data from multiple drilling campaigns, operators, and analytical laboratories spanning more than three decades.

The majority of discrepancies identified during the audit were determined to result from historical database management practices rather than analytical or transcription errors. These included the appropriate replacement of primary analytical results with gravimetric or metallic screen assays for overlimit samples, the calculation of weighted averages for multiple high-grade re-assays, and the standardization of below-detection-limit values for database consistency. Following recognition of these accepted practices, the verified database demonstrated a high level of agreement with the original laboratory records.

The audit also identified a limited number of genuine database issues, including isolated transcription and interval alignment errors within certain historical drillholes and a more recent database export error that resulted in the replacement of numerous assay values with a common value of 0.103 g/t Au. These issues were identified, quantified, and appropriately addressed prior to Mineral Resource estimation by removing affected values from the estimation database. While this approach may have resulted in the exclusion of some localized high-grade mineralization, it represents a conservative treatment of the affected data and is not considered to materially affect the overall Mineral Resource estimate.

The Qualified Person concludes that the remaining assay database is reliable, internally consistent, and suitable for Mineral Resource estimation. The data verification program provides confidence that the geological interpretations and resulting Mineral Resource estimate are supported by an adequate and representative analytical database. Nevertheless, a comprehensive review and recompilation of the historical assay database is recommended as part of future project development to recover omitted high-grade assay values and further improve the completeness of the database for subsequent Mineral Resource updates.

25.3 Mining Methods

Underground mining methods were evaluated to identify the approach that minimizes dilution, capital and operating costs, while maximizing mineral resource recovery and maintaining the planned mill throughput. The Copperstone orebody is relatively flat lying, with an average dip of approximately 38°. Although localized areas exceed 45 dip and may be amenable to gravity-assisted ore flow, most of the deposit is too shallow dipping for efficient longhole stoping.

Accordingly, mechanized cut-and-fill mining using both Rock Fill (RF) and Cemented Rock Fill (CRF) has been selected as the preferred mining method for the Copperstone Project. Cut-and-fill mining provides the flexibility required to efficiently extract low-angle mineralization while maintaining good ground control and minimizing dilution through selective mining and rigorous geological control.

Mineable stopes were generated using Movable Stope Optimizer (MSO) based on a gold price of US\$3,000/oz and a cut-off grade of 0.064 oz/ton Au. This approach produced stope geometries that optimize recoverable mineralization while supporting the planned mining sequence and production rate.

Cut-and-fill stoping is conducted by accessing the mineralized zone from the main ramp. Initial stope access is developed down-dip to the waste-ore contact and then advanced through the orebody to the hanging wall contact. Once the hanging wall is established, longitudinal stopes are mined perpendicular to the access drift, following the strike of the orebody to the stope limits.

The design is based on 10-ft-high cut-and-fill stopes. This stope height was selected to minimize dilution, improve ground stability, and accommodate the proposed mining equipment. In narrow portions of the orebody dipping at approximately 38°, additional waste must be excavated from both the hanging wall and footwall to achieve full recovery. Consequently, dilution is greatest in these narrow zones and increases with stope height. As the width of the mineralization increases, the proportion of waste dilution decreases significantly, improving overall mining efficiency.

25.3.1 Potential Risks

The QP identified the following potential risks that may impact the mining method:

- Variations in the forecast commodity price.
- Variations to the assumptions used in the constraining stope shapes, including mining loss/dilution, metallurgical recoveries, geotechnical assumptions and operating costs.

- Variations in permitting, environmental, and social license to operate assumptions that could affect project development and execution.

Other than the above, the QP is unaware of any legal, political, environmental or other risks that could materially affect the mine plan.

25.3.2 Potential Opportunities

The QP has identified the following potential opportunities for mining and mine planning:

- Optimize the mine design, including number of access points, stope height and width;
- Review the use of a lower cut-off grade in the operational mining plan to take advantage of the high gold price to increase amount of gold mined from the resource;
- Undertake a detailed evaluation of the orebody to determine whether a sufficient percentage of stopes are suitable for longhole stoping to justify the deployment of a longhole drill. Where feasible, this mining method has the potential to reduce development requirements and overall operating costs.
- Revisit studies on utilizing Cemented Hydraulic Fill (CHF) from the mill tailings, the Phase 2 lift of the tailings dam will not be required providing a future capital cost savings for CHF, especially if the mine life is expanded with additional mineralized material identification from exploration.

25.4 Recovery Methods

The Qualified Person (QP) concludes that the proposed process flowsheet for the Copperstone Project is technically appropriate and represents a practical and economically attractive processing strategy for the Mineral Resources considered within this Preliminary Feasibility Study. The selected hybrid flowsheet, consisting of conventional crushing, grinding, flotation concentration, and cyanide leaching of flotation tailings, is supported by historical metallurgical investigations, previous operations and subsequent engineering evaluations. Collectively, these studies demonstrate that the proposed process provides a robust balance between metallurgical performance, capital efficiency, and operational flexibility.

The process design incorporates conventional, well-established technologies that are widely used throughout the gold mining industry and therefore present a relatively low level of technical implementation risk. The use of flotation to recover the majority of the contained gold, followed by carbon-in-leach treatment of flotation tailings, provides a high overall projected gold recovery of approximately 96%, while reducing reagent consumption and minimizing the volume of material requiring cyanide treatment. The hybrid flowsheet also provides operational flexibility to accommodate anticipated variability in ore mineralogy throughout the life of the mine.

A significant advantage of the selected development strategy is the ability to utilize substantial portions of the existing Copperstone process infrastructure, including the crushing circuit and supporting facilities, while replacing or installing only those components necessary to achieve the planned production rate and metallurgical performance. This approach is expected to reduce initial capital expenditures, shorten the

construction schedule, and decrease overall project execution risk when compared to construction of a completely new processing facility.

The decision to produce both a flotation concentrate and loaded carbon for off-site processing is considered technically and economically appropriate. By leveraging existing processing infrastructure within the Company's Nevada operations, the project avoids the capital and operating costs associated with constructing and operating on-site stripping, electrowinning, refining, and smelting facilities. This strategy simplifies plant operations while maintaining anticipated metallurgical recoveries and reducing project capital intensity.

The process plant will process ore with an initial operating capacity of approximately 600 stpd while preserving the ability to increase throughput to approximately 1,000 stpd in the future. This staged development strategy is considered appropriate for a PFS-level project, allowing operational experience to be gained during initial production while providing flexibility for future growth. The proposed plant design incorporates accepted industry practice for process control, metallurgical accounting, water management, reagent handling, and cyanide destruction. The planned sampling and metallurgical accounting systems are considered sufficient to monitor plant performance, reconcile production, and support ongoing process optimization throughout operations.

Based on the available metallurgical testwork, engineering design, process simulations, and the incorporation of proven processing technologies, the QP concludes that the proposed recovery process is suitable for supporting the Preliminary Feasibility Study and provides a sound basis for advancement of the Copperstone Project toward detailed engineering and ultimately commercial production.

25.5 Infrastructure and Permitting

The Copperstone Mine benefits from substantial existing infrastructure, including approximately 10,000 ft of underground development accessed via two portals from the base of the historic open pit. Supporting infrastructure includes electrical distribution, compressors, and a mine ventilation system. The site also contains a crushing and grinding circuit and a tailings storage facility that, within existing permits, can be expanded to accommodate the life-of-mine tailings identified in the current mine plan. Additional surface infrastructure includes grid power, administration offices, a maintenance workshop, fuel bay, wash rack, assay laboratory, warehouse, and mine dry are also available to the project. The site has a compact layout, with the underground mine, processing plant, tailings storage facility, and support buildings located in close proximity, providing operational efficiencies and minimizing material handling distances.

All permits are in place for operations with the infrastructure as described above. The following modifications to existing permits have been approved under a new MPO for the Copperstone mine:

- Gold production at 600 tons per day
- Use of flotation and cyanide for recovery of gold from mineralized material
- Storage of stabilized tailings produced from the processing facility

- Construction and use of a water evaporation and infiltration basin to be used to manage surplus water generated from underground operations

25.6 Capital and Operating Cost Estimate

The capital and operating cost estimates developed for the Copperstone Project are considered appropriate for a Preliminary Feasibility Study and provide a reasonable basis for the economic evaluation presented in this Technical Report. The estimates were prepared using industry-standard methodologies consistent with the current level of engineering definition and are supported by a combination of quantity take-offs, vendor budget quotations, contractor pricing, first-principles estimating, and benchmarking against comparable underground gold mining operations.

The estimated initial capital cost of US\$57.2 million reflects the significant advantage provided by the existing Copperstone Mine infrastructure. The availability of existing underground workings, site infrastructure, electrical distribution, surface facilities, and portions of the crushing circuit substantially reduces the initial capital requirements relative to those of a comparable greenfield development. Sustaining capital requirements of approximately US\$69.8 million over the life of mine primarily reflect ongoing underground development, the planned transition from contract mining to an owner-operated mining fleet during the first year of production, staged tailings storage facility expansions, and closure obligations.

Operating cost estimates have been developed from detailed production schedules, equipment productivity assumptions, staffing requirements, consumable usage, and current market pricing. During the initial operating period, the use of a mining contractor reduces upfront capital requirements and execution risk while allowing the Project to transition to owner-operated mining following completion of fleet acquisition. The transition to owner mining is expected to reduce long-term unit mining costs and improve operating margins over the remaining life of the mine.

The capital and operating cost estimates incorporate appropriate indirect costs, owner's costs, and contingency allowances consistent with a PFS-level estimate. The overall estimate has been assigned an expected accuracy range of approximately $\pm 25\%$, which is consistent with the engineering maturity of the Project and accepted industry practice for studies at this level.

The Qualified Person considers the capital and operating cost estimates to be sufficiently comprehensive, internally consistent, and appropriate for supporting Mineral Reserve estimation and the economic analysis presented in this Technical Report. While future engineering, procurement activities, and market conditions may result in refinement of individual cost components, no material omissions or deficiencies have been identified that would reasonably be expected to affect the overall conclusions of the Preliminary Feasibility Study. The estimates therefore provide a suitable basis for evaluating the technical and economic viability of the Copperstone Project and support advancement of the Project toward a Definitive Feasibility Study.

25.7 Economic Analysis

The economic evaluation indicates that the Copperstone Project has the potential to be a technically and economically viable underground gold mining operation under the assumptions adopted for this Preliminary

Feasibility Study. The discounted cash flow analysis demonstrates strong economic performance, supported by the combination of relatively modest initial capital requirements, favourable operating costs, rapid payback of invested capital, and projected cash generation over the life of mine.

A significant contributor to the Project's economics is the existing brownfield infrastructure, which substantially reduces the initial capital investment required to return the operation to production. The proposed phased development strategy, including the use of contract mining during the initial operating period followed by a transition to an owner-operated mining fleet, provides a practical approach to reducing execution risk while improving operating margins over the remainder of the mine life.

The economic analysis incorporates all material project costs, including capital expenditures, sustaining capital, operating costs, royalties, taxes, closure obligations, and the existing gold streaming agreement. The resulting financial model indicates positive after-tax economic returns and a rapid recovery of the initial capital investment, supporting the technical feasibility of the proposed mining operation.

Sensitivity analyses demonstrate that the Project is most sensitive to fluctuations in the gold price, which is consistent with the economics of most single-commodity precious metal operations. Variations in operating costs and capital costs have a comparatively smaller effect on Project value, indicating that the Project maintains a reasonable degree of resilience to potential cost escalation within the ranges evaluated. The Project remains economically attractive across the range of capital and operating cost sensitivities considered in this study.

The economic evaluation is based on long-term assumptions regarding gold price, operating performance, capital costs, recoveries, taxation, and other financial parameters. As with all Preliminary Feasibility Studies, actual project performance will depend on future market conditions, final engineering, permitting, financing, and operational execution. Continued engineering optimization, procurement activities, and refinement of mine planning may further improve Project economics as the Project advances.

In the opinion of the Qualified Person, the economic analysis has been prepared using methodologies, assumptions, and cost estimates appropriate for a Preliminary Feasibility Study and provides a reasonable basis for evaluating the economic viability of the Copperstone Project. The results support advancement of the Project to the next stage of engineering and project development.

26 RECOMMENDATIONS

The following tasks are recommended to be completed to advance the Project and to prepare for Project development and operation.

26.1 Structural Understanding

The current structural interpretation of the Footwall and South zones assumes that mineralization is broadly parallel to the Copperstone Shear Zone. This interpretation should be validated through a systematic review of existing diamond drill core to identify and measure the orientation of mineralized structures relative to the core axis.

Future drilling programs should incorporate downhole acoustic or optical televiewer surveys to obtain accurate in-situ structural measurements that can be integrated with geological logging and assay data to improve the geological and resource models. If televiewer surveys are not available, oriented core drilling designed to intersect the Footwall and South zones at high angles to both strike and dip should be considered to provide reliable structural information.

Confirming the geometry and continuity of mineralization within the Footwall and South zones would improve confidence in the geological model and could support the reclassification of portions of the current Inferred Mineral Resource to the Indicated category, subject to the results of additional drilling and compliance with applicable reporting standards.

26.2 Additional Drilling

26.2.1 Infill Drilling

Infill drilling in the current known mineralized zones should be continued in future drilling programs with the intent of converting Inferred mineral resources to Indicated or Measured, expanding mineral resources, and improving the understanding of geometry and orientation of mineralized structures

26.2.2 Step Out Drilling

It is recommended that MAI undertake a future surface drilling program Drillhole spacing is recommended to be approximately 100 to 150 ft in order to target the down-dip and down-plunge extensions of known mineralized zones to evaluate its exploration potential. Execution of this program may require drilling through historic waste dump material and a substantial thickness of overburden before reaching the target horizons.

26.2.3 Near Surface Mineralization

It is recommended that the Company undertake a targeted drill program focused on improving the understanding and continuity of near-surface mineralization surrounding the existing and historical open

pits. The program should be designed to delineate the lateral and vertical extent of mineralization, confirm geological continuity, and provide sufficient drilling density to support future Mineral Resource estimation.

The primary objective of this program is to evaluate the potential for expanding the existing mineralized envelope and defining additional mineral resources that are amenable to open-pit mining methods. Particular emphasis should be placed on areas immediately adjacent to the historical pit limits where previous drilling indicates the potential for extensions of mineralization or where drilling density is insufficient to support higher-confidence resource classification.

The results of the drilling program should be incorporated into an updated geological and resource model to assess the potential for increasing the open-pit Mineral Resource inventory. If successful, the expanded resource could enhance project economics by increasing mineable inventory, improving mine scheduling flexibility, and extending potential mine life. The outcomes of this work would provide a stronger technical basis for future engineering studies and mine planning.

26.3 Mineral Processing and Recovery Methods

As expected for a Preliminary Feasibility Study, opportunities remain for further optimization during detailed engineering and future project development. A significant amount of metallurgical information was reviewed as part of the current project flowsheet design. Additional metallurgical testwork is recommended to optimize parameters of the current flowsheet.

Some recommended actions are:

- Conduct mineralogical evaluations on samples from various areas of the deposit in order to confirm variations in the minerology of the deposit
- Perform additional flotation studies on various zones to examine the optimal reagent schemes.
- Additional filtration and thickening studies
- Optimize optimal grind size and retention times

26.4 Mining

Based on the favorable results of the PFS it is recommended that further study during the next phase of work include the following:

- Continue to optimize the mine design, including number of access points, stope height and width
- Review the use of a lower cut-off grade in the operational mining plan to take advantage of the high gold price
- Develop grade control procedures
- Complete a comparative analysis of contract and owner-operated mining options to identify the preferred approach

- Recruit experienced underground technical and management personnel as a priority to advance the Pre-Feasibility Study and project development
- Optimize the ventilation, water management, and electrical power systems
- Using existing drill data, model an open pit resource based on a new set of open pit parameters that would outline the potential for a new maiden open pit resource

MAI should continue developing its geotechnical database alongside the geological database by systematically logging drill core for geotechnical parameters, including core recovery, RQD, structural features, alteration, and core photography prior to sampling. Existing core should be re-evaluated to improve geological and geotechnical models before additional geotechnical drilling is considered. Underground excavations should be mapped using Q and RMR rock mass classification systems to support ground support design and stope layout optimization. Geotechnical observations, including rock falls, ground instability, and deformation, should be documented and monitored to refine mine design and ground control practices.

26.5 Market Studies

It is recommended that MAI conduct a review of suitable buyers for concentration on Copperstone flotation circuit. As part of this review include transportation and terms of sale.

The estimated cost to complete the recommended scope of work is broken out by task in Table 26-1.

Table 26-1 Recommended Scope of Work Cost for the Copperstone Project

Recommendation	Estimate
Structural Understanding	\$30,000
Infill and Step Out Drilling	\$3,000,000
Mineral Processing & Recovery Optimization	\$200,000
Mining	
Optimize short term mine design	\$50,000
Review use of lower cut-off grades	\$10,000
Grade Control Program	\$10,000
Contract Mining vs Owner Mining analysis	\$10,000
Key underground mining staff	\$150,000
Optimize Ventilation, Water, and Power systems	\$55,000
Total Mining	\$285,000
Detailed Engineering	\$1,200,000
Near Surface Resource Evaluation	\$50,000
Total Budget	\$ 4,765,000

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28 CERTIFICATE OF QUALIFIED PERSON

28.1 Alex Duggan, P.Eng.

I, Alex Duggan, P.Eng., as an author of this report titled ***National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*** with an effective date of May 27, 2026 (the “Technical Report”) prepared for Mining Americas Inc. (formerly Minera Alamos Inc.) and dated 2nd day of September, 2026, do hereby certify that:

1. I am the Principal of Kristal Font Inc. with an office at 8045 Wyandotte Street, East, Windsor, Ontario, Canada.
2. I am a graduate of University of Aston, Birmingham, England with a B.Sc(Hons), M.Sc, in Civil Engineering.
3. I am a registered P.Eng. in good standing of Professional Engineers Ontario (Registration Number 100103898 and a member of Association for the Advancement of Cost Engineering International (AACEI).
4. I have over 38 years of direct experience in the mining industry.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I have not visited the Copperstone project site.
7. I am responsible for Sections, 22 and part of Sections 1, 25 and 26 of this Technical Report.
8. I have been engaged previously as a consultant with the Issuer; I have not had prior experience with the Property that is the subject of this Technical Report.
9. As of the date of this certificate, to the best of my knowledge, information, and belief, the parts of this Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.
10. I am independent of the Issuer, and the Property applying all the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with this instrument.
12. I consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of this Technical Report.

Dated this 2nd day of September, 2026 in London, Ontario

Original Signed and Sealed

Alex Duggan, M.Sc., P.Eng.

28.2 Peter Szkilnyk, P.Eng.

I, Peter Szkilnyk, P.Eng., as an author of the report titled ***National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*** with an effective date of May 27, 2026 (the “Technical Report”), prepared for Mining Americas Inc. (formerly Minera Alamos Inc.) and dated 2nd day of September, 2026, do hereby certify that:

1. I am the Principal Mining Engineer and owner of Rockpoint Advisory, 847 Arnold Crescent, Newmarket, Ontario L3Y 2E4, Canada.
2. I am a graduate of Queen’s University, Kingston, Ontario, with a B.A.Sc. in Mining Engineering (2008), and of the Smith School of Business, Queen’s University, with an M.B.A. (2018).
3. I am a licensed Professional Engineer in the Province of Ontario (Professional Engineers Ontario license#100182869) and a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum. By virtue of my education, professional registration, and relevant work experience, I fulfill the requirements of a Qualified Person as defined in NI 43-101.
4. I have over 18 years of progressive experience in the mining industry, including roles in mine engineering, technical due diligence, M&A evaluation, mineral reserve estimates, and strategic planning for both open-pit and underground operations. During my career, I have participated in multiple scoping, prefeasibility, and feasibility studies across various commodities, led or supported due diligence reviews for corporate development activities, and served as a technical consultant on projects in Canada and internationally. My relevant experience for the purpose of this Technical Report includes the preparation of underground mine designs and mine production schedules, and the estimation of mineral reserves, for underground precious metal projects.
5. I have not completed a personal inspection of the Copperstone Project property that is the subject of this Technical Report.
6. I am responsible for Sections 15 and 16, and for those portions of Sections 1, 25, and 26 of this Technical Report that relate to mine design, mine production scheduling, and Mineral Reserves.
7. I am independent of Mining Americas Inc. (formerly Minera Alamos Inc.) and its related entities, as independence is described in Section 1.5 of NI 43-101.
8. My prior involvement with the Copperstone Project consists of the preparation, on behalf of the issuer, of the underground mine design, mine production schedule, and mineral reserve estimate that support this Technical Report. I have had no other prior involvement with the property that is the subject of this Technical Report.
9. I have read NI 43-101 and the Sections of this Technical Report for which I am responsible have been prepared in compliance with that Instrument.
10. As at the effective date of this Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 2nd day of September, 2026, at Newmarket, Ontario.

Original Signed and Sealed

Peter Szkilnyk, P.Eng.

28.3 Garth Wilcox, P.Eng.

I, Garth Wilcox, P.Eng., as an author of this report titled ***National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*** with an effective date of May 27, 2026 (the “Technical Report”) prepared for Mining Americas Inc. (formerly Minera Alamos Inc.) and dated 2nd day of September, 2026, do hereby certify that:

1. I am an Independent Consulting Engineer located at 2 Clarinet Lane, Stouffville, Ontario, Canada, L4A 4N7.
2. I am a graduate of Technical University of Nova Scotia and received a Bachelor of Metallurgical Engineering 1996, (Dalhousie University).
3. I am a registered Professional Engineer in good standing of Professional Engineers Ontario (License #100022852) and a member of The Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
4. I have 30 years experience of practice in my profession since graduating. Working for mine development and mining projects in Canada, USA, and Mexico. Experience developing gold mines through metallurgical study work for bench scale testwork, pilot plant testing, project engineering feasibility study, process design and engineering for mineral processing projects. Other experience includes plant cost estimating, project management, mine operations and mine restart management. Consulting experience included working as a senior process engineer for multiple engineering companies on detailed plant studies for gold, phosphate, nickel, molybdenum, copper, and zinc project development.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I have visited the Copperstone Project site.
7. I am responsible for Section 13, 17 and parts of Sections 1, 21, 25 and 26 of this Technical Report.
8. I have been engaged previously as a metallurgical engineer with the Issuer; I have had prior experience with the Property that is the subject of this report.
9. As of the Effective Date of this Technical Report May 27, 2026, to the best of my knowledge, and belief, the parts of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.
10. I am independent of the Issuer, and the Property, applying all the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with this instrument.
12. I consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public.

Dated this 2nd day of September 2026 in Toronto, Ontario

Original Signed and Sealed

Garth Wilcox P.Eng.

28.4 Lawrence Segerstrom, CPG

I Lawrence Segerstrom, CPG, as an author of this report titled ***National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*** with an effective date of May 27, 2026 (the “Technical Report”) prepared for Mining Americas Inc. (formerly Minera Alamos Inc.) and dated 2nd day of September, 2026, do hereby certify that:

1. I am currently owner-operator of Segerstrom Consulting LLC, 190 W. Continental Rd, Suite 216-409, Green Valley, Arizona 85622.
2. I graduated with a Bachelor of Science degree in Geology from Colorado State University in Fort Collins in 1978, earned a Master of Science degree in Geosciences from the University of Arizona in Tucson in 1986, and a Master of Business Administration in international management from the Thunderbird School of Global Management in Glendale, Arizona in 2005.
3. I am a Certified Professional Geologist (#11557) in good standing with the American Institute of Professional Geologists. I am also a fellow in the Society of Economic Geologists and a member of several other geologic and mining societies.
4. Since 1983 I have worked as a geologist in the mining industry, predominantly in exploration and development, but also in mine geology and operations. I have worked in diverse metallic mineral deposit types and geologic settings, with emphasis on porphyry gold-copper, copper-gold, and copper-molybdenum deposits and associated skarns, as well as on low-, medium-, and high-sulfidation epithermal deposits. My work locations have been mainly in South America, USA, Indonesia, and Serbia, as an employee or consultant for small, medium, and large companies, both public and private.
5. I have read the definition of “qualified person” set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of this NI 43-101.
6. I visited the Copperstone project site January 14 – 16.
7. I am responsible for Sections 1-12, 18, 20, 23, 24 and parts of Sections 25 and 26 of this Report.
8. I have been previously engaged by the Issuer as a consulting geologist; I have not had prior experience with the Property that is the subject of this Report.
9. As of the Effective Date of this Technical Report May 27, 2026, to the best of my knowledge, and belief, the parts of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.
10. I am independent of the Issuer, and the Property, applying all the tests in section 1.5 of NI 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and this Report has been prepared in compliance with this instrument.
12. I consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 2nd day of September, 2026 in Guaymas, Sonora, Mexico

Original Signed and Sealed

Lawrence Segerstrom CPG.

28.5 Scott Zelligan, P.Geo.

I Scott Zelligan, P.Geo., as an author of this report titled ***National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*** with an effective date of May 27, 2026 (the “Technical Report”) prepared for Mining Americas Inc. (formerly Minera Alamos Inc.) and dated 2nd day of September, 2026, do hereby certify that:

1. I am an independent Consulting Geologist residing at 358 Ardagh Rd., Barrie, Ontario, L4N 9C2.
2. I graduated with a Bachelor of Science (Honours), Earth Sciences, from Carleton University (Ottawa, Ontario) in 2008.
3. I am a Professional Geoscientist (P.Geo.), registered with the Professional Geoscientists Ontario (#2078).
4. I have practiced my profession as a geologist for a total of over eighteen years since my graduation from university, as an employee of major and junior mining companies, as an employee of engineering consulting firms, and as an independent consultant, including: five months working underground in a producing gold mine; fourteen years working in exploration for numerous commodities (including base, precious, and other minerals); and fifteen years of resource estimation work including modelling, estimating, and evaluating mineral properties of all types (including base, precious, and other minerals) throughout North America, and occasionally globally (exploration and resource estimation work has overlapped significantly). I have previously been a primary author on ten NI 43-101 technical reports as well as secondary author or contributor on several others. I have worked on numerous properties with similar or comparative mineralization styles to the Project.
5. I have read the definition of “qualified person” set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of this NI 43-101.
6. I have visited the Copperstone Project site on July 21-22, 2026.
7. I am responsible for Sections 12, 14 and parts of Sections 1, 25 and 26 of this Technical Report.
8. I have been engaged previously as a resource geologist with the Issuer; I have had prior experience with the Property that is the subject of this Report.
9. As of the Effective Date of this Technical Report May 27, 2026, to the best of my knowledge, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.
10. I am independent of the Issuer and the Property, applying all the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with this instrument.
12. I consent to the filing of this Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 2nd day of September, 2026 in Barrie, Ontario

Original Signed and Sealed

Scott Zelligan P.Geol.