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

Report Date: August 2, 2023
Effective Date: June 26, 2023

Prepared for:

Sabre Gold Mines Corp.
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Prepared by:

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HRC Project Number: 23-SGM-01

Endorsed by QP(s):

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Sabre Gold Mines Corporation ("SGLD") by Hard Rock Consulting, LLC ("HRC"). The quality of information, conclusions, and estimates contained herein is consistent with the scope of HRC's services based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by SGLD subject to the terms and conditions of their contract with HRC, which permits SGLD to file this report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

CERTIFICATES OF QUALIFIED PERSONS

I, Richard A. Schwering, P.G., SME-RM, do hereby certify that:

1. I am currently employed as Principal Resource Geologist by:

   Hard Rock Consulting, LLC
   7114 W. Jefferson Ave., Ste. 313
   Lakewood, Colorado 80235 U.S.A.

2. I am a graduate of the University of Colorado, Boulder with a Bachelor of Arts in Geology, in 2009 and have practiced my profession continuously since 2013.

   2.1 I am a Registered member of the Society of Mining and Metallurgy and Exploration (No. 4223152RM) and a Licensed Professional Geologist in the State of Wyoming (PG-4086)

3. I have worked as a Geologist for 11 years and as a Resource Geologist for a total of 7 years since my graduation from university; as an employee of a junior exploration company, as an independent consultant, and as an employee of various consulting firms with experience in structurally controlled precious and base metal deposits.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I am responsible for the preparation of this report, titled "*National Instrument 43-101 Technical Report, Preliminary Economic Assessment for the Copperstone Project, La Paz County, Arizona, USA*", dated August 2, 2023 with an effective date of June 26, 2023, and I take specific responsibility for report Sections 1.4, 1.5, 6, 9 through 12, 14, 25.1 and 25.2.

6. I personally inspected the Copperstone Project on March 1 through 5, 2021 and was previously involved in preparation of the *NI 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*, with an effective date of April 1, 2018, as well as preparation of the *National Instrument 43-101 Technical Report, Updated Mineral Resource Estimate for the Copperstone Project, La Paz County, Arizona, USA*, dated September 21, 2021, with an effective date of September 3, 2021.

7. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

8. I am independent of the Issuer, vendor, and property applying all of the tests in section 1.5 of NI 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and submit that this Technical Report has been prepared in accordance with that instrument and form.

Dated this 2nd day of August 2023
Richard A. Schwering

Signature of Qualified Person

Richard A. Schwering; SME-RM
Printed name of Qualified Person

CERTIFICATES OF QUALIFIED PERSONS

I, Jennifer J. Brown, P.G., do hereby certify that:

1. I am currently employed as Principal Geologist by:

   Hard Rock Consulting, LLC
   7114 W. Jefferson Ave., Ste. 313
   Lakewood, Colorado 80235 U.S.A.

2. I am a graduate of the University of Montana and received a Bachelor of Arts degree in Geology in 1996.

3. I am a Licensed Professional Geologist in the State of Wyoming (PG-3719), a Registered Professional Geologist in the State of Idaho (PGL-1414), and a Registered Member in good standing of the Society for Mining, Metallurgy, and Exploration, Inc. (4168244RM).

4. I have worked as a geologist for a total of 25 years since graduation from the University of Montana, as an employee of various engineering and consulting firms and the U.S.D.A. Forest Service. I have more than 10 collective years of experience directly related to mining and or economic and saleable minerals exploration and resource development, including geotechnical exploration, geologic analysis and interpretation, resource evaluation, and technical reporting.

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 personally inspected the Copperstone Project on October 31 through November 2, 2017, and was previously involved in preparation of the *NI 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*, with an effective date of April 1, 2018, as well as preparation of the *National Instrument 43-101 Technical Report, Updated Mineral Resource Estimate for the Copperstone Project, La Paz County, Arizona, USA*, dated September 21, 2021, with an effective date of September 3, 2021.

7. I am responsible for the preparation of this report, titled "*National Instrument 43-101 Technical Report, Preliminary Economic Assessment for the Copperstone Project, La Paz County, Arizona, USA*," dated August 2, 2023, with an effective date of June 26, 2023, and I take specific responsibility for report Sections 1.1 through 1.3, 2 through 5, 7, 8, 20, 23, 24, and 27.

8. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and submit that this Technical Report has been prepared in accordance with that instrument and form.

Dated this 2nd day of August 2023

"Signed" Jennifer J. (J.J.) Brown

JJ Brown

Jennifer J. (J.J.) Brown, SME-RM
Printed name of Qualified Person

CERTIFICATES OF QUALIFIED PERSONS

I, Jeffery W. Choquette, P.E., do hereby certify that:

1. I am currently employed as Principal Engineer by:

   Hard Rock Consulting, LLC
   7114 W. Jefferson Ave., Ste. 313
   Lakewood, Colorado 80235 U.S.A.

2. I am a graduate of Montana College of Mineral Science and Technology and received a Bachelor of Science degree in Mining Engineering in 1995

3. I am a Registered Professional Engineer in the State of Montana (No. 12265) and a QP Member in Mining and Ore Reserves in good standing of the Mining and Metallurgical Society of America (No. 01425QP).

4. I have practiced my profession continuously since 1996. I have experience in project development, resource and reserve modeling, mine operations, mine engineering, project evaluation, and financial analysis. I have worked for mining and exploration companies for 15 years and as a consulting engineer for twelve years. I have been involved in industrial minerals, base metals and precious metal mining projects in the United States, Canada, Mexico, Asia and South America.

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 personally inspected the Copperstone Project on July 10 and 11, 2018 and I was previously involved in preparation of the NI 43-101 Technical Report: *Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA, with an effective date of April 1, 2018*, as well as preparation of the National Instrument 43-101 Technical Report, *Updated Mineral Resource Estimate for the Copperstone Project, La Paz County, Arizona, USA, dated September 21, 2021,* with an effective date of September 3, 2021.

7. I am responsible for the preparation of this report, titled "*National Instrument 43-101 Technical Report, Preliminary Economic Assessment for the Copperstone Project, La Paz County, Arizona, USA*," dated August 2, 2023, with an effective date of June 26, 2023, and I take specific responsibility for report Sections 1.6, 1.8 through 1.10, 15, 16, 18, 19, 21, 22, 25.4, 25.6, 25.7, and 26.

8. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

Dated this 2nd day of August 2023

"Signed" Jeffery W. Choquette

Jeffery W. Choquette, P.E.
Printed name of Qualified Person

MONTANA
JEFFERY
CHOQUETTE
No. 12265 PE
LICENSED
PROFESSIONAL ENGINEER

CERTIFICATES OF QUALIFIED PERSONS

I, Dr. Deepak Malhotra, Ph.D. do hereby certify that:

1. I am Director of Metallurgy for Forte Dynamics located at 120 Commerce Drive, Unit 3, Fort Collins, CO 80524 ,USA.
2. This certificate applies to the technical report titled "*National Instrument 43-101 Technical Report, Preliminary Economic Assessment for the Copperstone Project, La Paz County, Arizona, USA*," dated August 2, 2023, with an effective date of June 26, 2023 (the "Technical Report").
3. I graduated with a Master of Science in Metallurgical Engineering from Colorado School of Mines in 1973. In addition, I have obtained a Ph.D in Mineral Economics in 1977 from Colorado School of Mines. I am a Registered Member in good standing of the Society of Mining, Metallurgy and Exploration Inc. (SME) (License # 2006420) and a member of Canadian Institute of Mining, Metallurgy and Petroleum (CIM). I have worked as a Metallurgist/Mineral Economist for a total of 50 years since my graduation from university. My relevant experience includes metallurgical testwork, plant design and troubleshooting of several dozen operations worldwide.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am specifically responsible for Sections 1.7, 13, 17, 25.3, 25.5 of the Technical Report titled "*National Instrument 43-101 Technical Report, Preliminary Economic Assessment for the Copperstone Project, La Paz County, Arizona, USA*," dated August 2, 2023, with an effective date of June 26, 2023.
6. I personally inspected the Copperstone Project on February 6 through February 8, 2018.
7. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
8. I was previously involved in preparation of the NI 43-101 Technical Report: *Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA, with an effective date of April 1, 2018.*
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 2nd Day of August 2023.

"Signed" Dr. Deepak Malholtra, PhD

Deepak Malhotra
Printed name of Qualified Person

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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	tpd	Tones per day
GCMP	Ground Control Management Plan	tph	Tones 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

Sabre Gold Mines Corp. ("SGLD") (TSX:SGLD) is a North American gold exploration company headquartered in Vancouver, Canada. 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 was previously mined by Cyprus Minerals Corporation, producing in excess of 500,000 ounces of gold (from 5.6 million tons of mill feed grading 0.089 oz/ton Au) from the open pit between 1987 and 1993. Cyprus successfully recovered the gold through a combination of whole ore and heap leaching, with no known associated environmental liabilities resulting from the process. The results of ongoing exploration at the Project indicate that gold and copper mineralization with mineable potential exists at the site. In 2011, American Bonanza constructed a 450 tons per day ("tpd") flotation mill on site and in 2012 started underground mining from two declines which 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 Cyprus 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 Au. In late 2017, Kerr completed a combined surface and underground drilling exploration program including the addition of 800 ft of new underground access. Kerr completed an underground drilling program in 2019, and AZG completed combined surface and underground drilling programs in 2020 and 2021. In late 2021, SGLD completed an underground infill drilling program.

SGLD retained Hard Rock Consulting ("HRC") to update the mineral resource estimate and prepare a subsequent Preliminary Economic Assessment ("PEA") for the Copperstone Project. This report presents the updated mineral resource statement and documents the results of the PEA in fulfillment of the Standards of Disclosure for Mineral Projects according to Canadian National Instrument 43-101 ("NI 43-101").

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 February 15, 2023. The effective date of this report in full is June 26, 2023.

1.2 Property Description and Ownership

The Copperstone Project encompasses approximately 13.8 square miles of surface area and mineral rights in La Paz County, Arizona, roughly 19 miles north of the town of Quartzsite. The Project is wholly owned by SGLD, which controls the 546 federal unpatented mining claims and two Arizona state mineral leases which together comprise the Copperstone Project area. The Project area covers all or portions of Sections 6 through 10 and 15 through 23, T6NR19W; Sections 1, 2, 10 through 14 and 22 through 27, T6NR20W; and Section 19, T7NR19W, Gila and Salt River Meridian. The federal claims cover approximately 10,920 acres (4,419 hectares) while the state mineral leases total approximately 1,338 acres (542 hectares). The approximate geographic center of the Project area lies at 33°52'6" N latitude, 114°17'42" W longitude.

The Copperstone Project PEA is based on an underground gold mining and onsite milling concept. Drilling exploration has identified approximately 1.22 million tons of mill feed material, with an estimated project life of approximately 6.9 years and 5.7 years of mill production. This schedule estimates a mill through-put of approximately 600 tons per day, which translates to an annual mill through-put of approximately 219,000 tons per year. Most construction personnel are expected to be sourced locally depending on skill levels available in the local communities.

All facilities will be located on currently disturbed ground. No new surface disturbance is contemplated or expected. All non-mineralized rock excavated in the underground mine is planned to be used as back fill or to be stored in the open pit waste rock stockpile. The mine was designed as a zero-discharge facility with all mine water created from pumping of the underground workings either used in the mill, for dust suppression on roads, or directed to an evaporation/infiltration gallery. Existing wells supply make-up water and non-potable water for buildings.

A significant portion of the existing on-site infrastructure is in good repair and useful for the Project. Compared to similar-sized projects with no existing infrastructure, only a small amount of capital will be required to upgrade the existing infrastructure, which is a benefit to the economics of the Project. To process the underground run-of-mine ("ROM") material, the existing crushing circuit will be refurbished, and crushed material will feed directly into the grinding circuit which is also planned to be refurbished. The processing plant beyond the grinding circuit will be rebuilt to facilitate whole ore leaching followed by Merril Crowe for gold recovery.

Tailings generated by the new mining activities will be impounded in the existing, lined tailings storage facility ("TSF"). Water from the TSF is allowed to flow to a lined water collection facility to be re-used in the processing of gold bearing material at the mill.

1.3 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 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.

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 km^2 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 occurs in the hanging wall of the Moon Mountain detachment fault, which has not been penetrated in drilling to date. Gold mineralization is largely restricted to the immediate vicinity of the Copperstone fault (also referred to as the Copperstone shear or the Copperstone structure), a moderately northeast-dipping, semi-planar zone of shear which is interpreted as a listric splay of the Moon Mountain detachment, and which has hosted the bulk of the gold historically produced from the Copperstone mine. 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 180 ft in width with characteristic fault gouge, multi-phase breccia textures, shear fabric, and intense fracture sets across this width.

SGLD's current conceptual geologic model interprets the Copperstone structure as part of a detachment fault system related to regional mid-Miocene extension. More recently, Strickland et al. (2017) have recognized late Laramide detachment related to magmatism and the denudation of a Cretaceous subduction complex found across southern Arizona and California. Regardless of the age of the deformation, detachment faulting with an upper-plate-to-the-east sense of motion is presently considered the primary control/conduit for mineralization.

1.4 History and 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. 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. 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, 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 8,609 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 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 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 tpd floatation 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 Au. 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.). The purposes of the 2019 program were to test the margins of gold mineralization as understood at that time, and to obtain sufficient data to support converting Inferred mineral resources to Measured and Indicated mineral resources in the D zone and portions of the C zone. 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. The primary purpose of the drilling was to expand gold mineral resources in the D and C zones. 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.

1.5 Mineral Resource Estimate

HRC's Richard Schwering, P.G., SME-RM, is the Qualified Person ("QP") responsible for the mineral resource estimate presented herein. Mr. Schwering is a QP as defined by NI 43-101 and is independent of SGLD. Mr. Schwering estimated the mineral resources for the Project based on drillhole data constrained by geologic boundaries with an Ordinary Krige ("OK") algorithm. Gold is the metal of interest at the Project. The mineral resource estimate reported herein was prepared in a manner consistent with the Committee of Mineral Reserves International Reporting Standards ("CRIRSCO"), of which both the CIM and the Petroleum and Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, are members. The mineral resources are classified as Measured, Indicated, and Inferred in accordance with "CIM Definition

Standards for Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014, and Best Practices Guidelines (November 29, 2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council. Classification of the resources reflects the relative confidence of the grade estimates.

In total, 1,118 drillholes totaling 524,762 ft were incorporated into the geologic model and resource estimate. Eighty-five RC drillholes totaling 9,855 ft completed by SGLD in October and December of 2021 were not incorporated into the mineral resource estimate. These holes were drilled from underground into targets of expected gold mineralization. Since these drillholes targeted and largely confirmed expected gold mineralization, they are not material to the mineral resource estimate and are more appropriate for use in short term mine planning studies.

The Copperstone deposit is a mid-Tertiary, detachment 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, though most mineralization is hosted in quartz latite porphyry. Breccia textures as well as chloritization, silicification, and hematite and specularite flooding are typically reliable indicators of gold mineralization.

Gold grades were constrained within estimation domains modelled with 3D wireframe solids. Estimation domains follow the overall northwest, shallowly dipping structural trends, and were defined by drillhole interval selections of gold grades greater than or equal to 0.100 oz/ton. Domains were reviewed in 3D to ensure the models agree with the overall geologic interpretation and maintained continuity along strike and down dip. Samples were composited inside estimation domains to a target length of 5 ft. Composite gold grades within each domain were reviewed for statistically high outliers, which were then constrained and capped. The capping analysis considered each domain separately and a global gold cap was not used. Semi-variograms from composites were used to inform the search ellipse. Densities were determined inside and outside estimation domains by lithology from drill core. The strike length of the deposit is approximately 4,000 ft and mineralization has been encountered by drillholes to a depth of -330 ft below mean sea level ("bmsl") (approximately 1,200 ft below surface). The geologic model was created using Leapfrog, and is comprised of four structural domains, six stratigraphic units, and 48 estimation domains.

The undiluted Copperstone project mineral resource statement is presented in Table 1-1. The results are rounded to reflect the approximation of grade and quantity which can be achieved at this level of resource estimation. Rounding may result in apparent differences when summing tons, grade and contained metal content. Tonnage and grade measurements are in U.S. Customary units and Metric units. All costs are in 2023 US dollar denominations. 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. Inferred mineral resources are that portion of a mineral resource for which the grade or quality are estimated on the basis of limited geological evidence and sampling. Inferred mineral resources do not have demonstrated economic viability and may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

The mineral resources are confined to material exceeding the gold cut-off grade of 0.092 oz/ton within coherent wireframe models. After the block grade estimations were complete, the estimated blocks at and above the cut-off grade for each domain were reviewed in long section by the QP. The majority of estimated blocks demonstrate grade continuity and meet the criteria of a minable shape. Small, and isolated blocks that did not meet the QP's opinion of a minable shape were excluded from the mineral resource statement. The application of a cut-off grade to estimated blocks which meet the criteria of a minable shape within coherent wireframe models meet the test of reasonable prospect for economic extraction. The cut-off is calculated based on the operating costs, royalties, recoveries and metal prices as presented in note four of Table 1-1. A gold price of $1,800/oz was chosen which is the 36-month moving average price as of January 31, 2023. The effective date of the mineral resource estimate is February 15, 2023.

Table 1-1 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

1. The effective date of the mineral resource estimate is February 15, 2023. The QP for the estimate, Mr. Richard A. Schwering, P.G., SME-RM of HRC, is independent of Sabre Gold Mine Corp.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Inferred mineral resources are that part of a mineral resource for which the grade or quality are estimated on the basis of limited geological evidence and sampling. Inferred mineral resources do not have demonstrated economic viability and may not be converted to mineral reserves. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.
4. The mineral resource is reported at an underground mining cut-off of 0.092 oz/ton (3.15 g/t) Au beneath the historic open pit and within coherent wireframe models, and for estimated blocks which meet the criteria of a minable shape. The cut-off is based on the following assumptions: a gold price of $1,800/oz; assumed mining cost of $90/ton ($99.21/tonne), process costs of $47/ton ($51.81/tonne), general and administrative and property/severance tax costs of $15.00/ton ($16.53/tonne), refining and shipping costs of $12.00/oz, a metallurgical recovery for gold of 95%, and a 3.0% gross royalty.
5. Rounding may result in apparent differences when summing tons, grade and contained metal content. Tonnage and grade measurements are in U.S. Customary and Metric units. Grades are reported in troy ounces per short ton (oz/ton) and grams per tonne (g/t). Contained metal is reported as troy ounces.

1.6 Mining Methods

The PEA underground mine plan for the Copperstone Project includes approximately 1,222,300 tons of mineralized mill feed to be extracted by underground mining in 6 years. The underground mine designs and schedule utilize Inferred mineral resources as part of the analysis. Mineral resources that are not mineral reserves do not have demonstrated economic viability. This PEA is preliminary in nature in that it includes Inferred mineral resources that are considered too speculative to have economic considerations applied to them and should not be relied upon for that purpose. The mine production schedule calls for the production of 600 tpd for an annual production of 219,000 tons through the milling circuit. Mining recoveries of 95% were applied and overall dilution factors averaged 32%. 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 three eight-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 and Cemented Rock Fill;
- Cut-off grade of 0.107 oz/ton gold for underground mining;
- For planning purposes, the stopes have been separated into six zones: The A, B, C, and D zones. and the Footwall ("FW") zone, South ("S") zone, and Upper Fracture ("UPFX") zone;
- A production rate ramped up to 600 tpd;
- The underground design allows for 28.9% planned dilution, 10% unplanned dilution, and a mining recovery of 95%;
- Development drifting and raising of approximately 61,416 ft for the life of mine;
- Four operating crews with an average of 21 workers/crew working 10hr shifts, four days on and four days off.

The mining method proposed for the Copperstone Project is a mechanized cut and fill using cemented rock fill ("CRF"). The cut and fill method was chosen for its flexibility in effectively mining low vein dip angles. This method also minimizes the amount of dilution during mining by careful geological and management control of the mining.

Underground mining methods that will minimize dilution, capital, and operating costs, and that will maximize recovery of the mineral resources while maintaining the design production throughput at the mill were reviewed. The mineralized zone at Copperstone is relatively flat with an average dip of 38º degrees. Though there are some areas where mineralization is steeper (>45-degree dip), the majority of the deposit is too flat to facilitate a long hole mining method. Historic underground mine workings and a 2017 exploration drift provide approximately 12,800 ft of existing access development across 500 ft of strike, including two declines from the bottom of the pit.

The primary ramp development is planned in the mineralized footwall to access the cut and fill stopes. The main haulage drifts and ramps are planned to be developed at a 15-ft height x 15-ft width, which is similar to the size of the existing development. The main ramp is designed to limit curves and turns to promote efficient truck haulage and reduce ventilation constraints. Muck bays 30 ft deep are planned near the stope access points or every 500 ft along the ramp to facilitate the development mucking process. As the development progresses, these muck bays will be converted for use as sumps, transformer bays, storage areas and exploration drill bays.

The total length of the new main haulage ramps is 26,441 ft. One new portal in the pit bottom is planned, the portal will be installed towards the end of year one and will provide access to the A zone. There is also a historic decline that was put in by Cyprus towards the end of the operations of the open pit in the mid 1990's. The PEA includes costs to rehab 3,475 ft of this decline which will allow for an alternative haulage route to the mill from the Footwall zone. The portal for this decline is presently buried by open pit waste material, and approximately 100,000 to 150,000 tons of material will need to be moved on surface to uncover the portal. Re-establishing this decline will also improve the ventilation circuit by providing an additional exhaust route for the air flow, should the decline be found to be deteriorated beyond re-pair, a ventilation raise could be established in this area as an alternative option.

The main haulage ramps are developed approximately 160 ft beyond the mineralized zone in the footwall. The stope access ramps are planned at 10-ft height x 10-ft width to allow sufficient access height for highly productive mining equipment. A nominal level spacing of 60 ft was selected, providing access to six 10-ft high drifts and fill cuts from a single access point. The first access ramp is driven at -15% to access the first of six lifts of the stope. The remaining five lifts are developed by backslashing and ramping up at +15% to access subsequent lifts. Each stope access point also includes a 30-ft muck bay. The total length of stope access ramps is planned at 11,407 ft and the total length of stope access backslash ramps is planned at 20,248 ft.

The mine operations schedule is based on 365 days/year, 7 days/week, with two 10-hour shifts each working day. There are four crews scheduled working a four-on, four-off schedule. The production rate at full production is 600 tpd with a 3-month ramp-up period. Each stope is calculated to be able to produce 176 tpd. Based on that assumption, 3.5 active faces are required to meet production requirements. Due to inefficiencies in developing new stopes, backfill placement and unplanned delays a total of six active areas are scheduled in the mine plan.

Table 1-2Table 16-4 presents the annual mining schedule. Stoping 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 1-2 Annual Mining Schedule

Production Schedule	Life-of-Mine	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
MINE PRODUCTION								
Tons Mill Feed Mined	1,222,317	18,148	199,678	240,468	232,153	200,989	222,630	108,251
Au, oz/ton	0.197	0.184	0.256	0.208	0.195	0.184	0.167	0.155
Development Feet	61,416	3,330	14,378	14,721	9,947	6,916	8,153	3,972
Development Waste	815,686	50,864	233,713	237,774	135,301	57,400	67,668	32,965
Total Tons Mined	**2,038,003**	**69,013**	**433,391**	**478,242**	**367,454**	**258,389**	**290,298**	**141,216**

1.7 Mineral Processing

Several metallurgical studies have been undertaken since 1986 to evaluate the best processing option for ore from Copperstone mine. The mine was operated for a short period in 2011 by American Bonanza who produced a flotation concentrate.

The following options have been evaluated by SGLD:

- Production of flotation concentrate for sale to a smelter.
- Production of flotation concentrate which would be leached at site to produce doré.
- Whole ore leach with/without recovery of copper.

Whole ore leach exhibited the highest operating costs while reducing the CAPEX for the project. However, it produced the highest recovery of gold while eliminating flotation concentrate smelter charges.

A conceptual process flowsheet for recovering gold and copper was developed and consists of crushing and grinding the ore followed by cyanidation, CCD, and MC to produce a gold/silver/copper sludge. The sludge will be batch leached with sulfuric acid and filtered to produce a gold/silver rich sludge. The filtrate, which will have copper can be subjected to cementation for recovery of copper. The gold/silver rich sludge will be smelted to produce doré.

1.8 Economic Analysis

The Project has been evaluated using a constant US dollar, after-tax discounted cashflow methodology. Information contained and certain statements made herein are considered forward-looking within the meaning of applicable Canadian securities laws. These statements address future events and conditions and so involve inherent risks and uncertainties. Actual results could differ from those currently projected. This PEA is preliminary in nature, and includes inferred mineral resources that are considered too speculative geologically to have economic consideration applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

The Project is planned to be an underground mining operation with milling and WOL followed by a Merril Crowe plant for gold recovery. The life of mine PEA plan includes 1,222,000 tons of mill feed with an average grade of 0.197 oz/ton Au. The process operations are planned to run at a rate of 600 tpd with a metallurgical gold recovery of 95%.

The economic analysis of the base case scenario for the Project uses a price of US$1,800/oz for gold, which is the 3-year trailing average price as of the end of January 2023. The economic model shows an After-Tax Net Present Value @ 5% ("NPV-5") of $61.78 million using a 0.107 oz/ton Au mining cut-off grade, as well as an After-Tax Internal Rate of Return ("IRR") of 50.5%. Table 1-3 summarizes the projected Cashflow, Net Present Value at varying rates, Internal Rate of Return ("IRR"), years of positive cash flows to repay the negative cash flow ("payback period"), and multiple of positive cash flows compared to the maximum negative cash flow ("payback multiple") for the Project on both After-Tax and Before-Tax bases.

Table 1-3 Summary of Copperstone Economic Results

Project Valuation Overview	After Tax	Before Tax
Net Cashflow (millions)	$86.77	$89.78
NPV @ 5.0%; (millions)	$61.78	$63.97
NPV @ 7.5%; (millions)	$52.21	$54.09
NPV @ 10.0%; (millions)	$44.15	$45.76
Internal Rate of Return	50.5%	51.2%
Payback Period, Years	1.81	1.81
Payback Multiple	2.90	3.53
Total Initial Capital (millions)	-$36.27	-$36.27
Max Neg. Cashflow (millions)	-$45.66	-$35.54

Table 1-4 summarizes the projected gold production schedule and cash flows. Readers are cautioned that the PEA is preliminary in nature. The mill feed includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and there is no certainty that the PEA will be realized.

Table 1-4 Cashflow Summary

Note: All Dollars are in US	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Life-of-Mine
MINE PRODUCTION									
Tons Ore Mined		18,148	199,678	240,468	232,153	200,989	222,630	108,251	1,222,317
Au, oz/tn		0.184	0.256	0.208	0.195	0.184	0.167	0.155	0.197
PROCESS PRODUCTION									
Tons Ore Processed			214,350	219,000	219,000	219,600	219,000	131,367	**1,222,317**
Au, oz/t			0.250	0.208	0.196	0.185	0.170	0.156	0.197
Income Statement									
Contained Oz Au to Mill			53,542	45,583	42,879	40,732	37,298	20,504	240,538
Saleable Oz Au, post 99.9% Refinery credit			50,814	43,260	40,694	38,657	35,398	19,460	228,283
Gross Revenue			$91,465,920	$77,868,360	$73,249,020	$69,582,780	$63,715,680	$35,027,460	$410,909,220
Transportation and Refinery Charges			($637,788)	($569,732)	($546,611)	($528,262)	($498,897)	($149,190)	($2,930,480)
Net Refined Revenue			$90,828,132	$77,298,628	$72,702,409	$69,054,518	$63,216,783	$34,878,270	$407,978,740
Royalties			($2,743,978)	($2,336,051)	($2,197,471)	($2,087,483)	($1,911,470)	($1,050,824)	($12,327,277)
Star Royalties- Stream			($4,527,563)	($3,854,484)	($3,625,826)	($3,444,348)	($3,153,926)	($773,855)	($19,380,002)
Net Revenue			**$83,556,591**	**$71,108,093**	**$66,879,112**	**$63,522,687**	**$58,151,386**	**$33,053,592**	**$376,271,462**
OPERATING EXPENSES									
Total Mining		$0	($24,299,197)	($19,786,236)	($20,268,849)	($19,879,071)	($20,802,585)	($12,046,623)	($117,082,561)
Total Processing	$0	$0	($10,155,903)	($10,376,220)	($10,376,220)	($10,404,648)	($10,376,220)	($6,224,178)	($57,913,389)
Total G&A		$0	($3,366,926)	($3,366,926)	($3,366,926)	($3,251,327)	($3,366,926)	($2,114,672)	($18,833,703)
Property Tax	$0	($64,528)	($106,918)	($101,808)	($101,583)	($73,605)	($69,433)	($65,235)	($583,110)
Mine Severance Tax	$0	$0	($559,439)	($387,519)	($359,576)	($344,163)	($263,471)	($86,181)	($2,000,349)
Total Operating Costs	**$0**	**($64,528)**	**($38,488,383)**	**($34,018,709)**	**($34,473,153)**	**($33,952,815)**	**($34,878,634)**	**($20,536,890)**	**($196,413,112)**
Operating Margin (EBITDA)	**$0**	**($64,528)**	**$45,068,208**	**$37,089,385**	**$32,405,959**	**$29,569,872**	**$23,272,752**	**$12,516,702**	**$179,858,349**
Development Deduction	$0	($6,416,052)	($9,733,061)	($4,996,587)	($2,370,698)	$0	$0	$0	($23,516,398)
Amortization	$0	($274,974)	($692,105)	($906,244)	($1,007,846)	($1,007,846)	($1,007,846)	($5,181,596)	($10,078,457)
Depreciation	$0	$0	($8,144,092)	($12,665,940)	($9,822,740)	($7,912,836)	($7,523,916)	($7,533,057)	($53,602,581)
Reclamation Deduction	$0	$0	$0	$0	$0	$0	$0	($1,200,000)	($1,200,000)
Interest Expense	$0	($28,916)	($526,937)	($709,976)	($364,111)	($50,789)	$0	$0	($1,680,730)
Income - before NOL & Perc Depletio	**$0**	**($6,784,470)**	**$25,972,014**	**$17,810,638**	**$18,840,563**	**$20,598,401**	**$14,740,990**	**($1,397,951)**	**$89,780,184**
Net Operating Loss Adjustment	$0	$6,784,470	($25,972,014)	($17,810,638)	($18,840,563)	($11,861,255)	$0	$1,397,951	($66,302,049)
Depletion	$0	$0	$0	$0	$0	($4,215,673)	($7,112,527)	$674,512	($10,653,688)
State Income Tax	$0	$0	$0	$0	$0	($305,800)	($515,935)	$48,928	($772,807)
Federal Income Tax	$0	$0	$0	$0	$0	($885,291)	($1,493,631)	$141,647	($2,237,275)
Taxable Income, less Tax	**$0**	**$0**	**$0**	**$0**	**$0**	**$3,330,382**	**$5,618,897**	**$865,087**	**$9,814,366**
Cash Flow Calculation									
Adjustments for Non Cash Items									
Development Deduction	$0	$6,416,052	$9,733,061	$4,996,587	$2,370,698	$0	$0	$0	$23,516,398
Amortization	$0	$274,974	$692,105	$906,244	$1,007,846	$1,007,846	$1,007,846	$5,181,596	$10,078,457
Depreciation/Reclamation/Salvage	$0	$0	$8,144,092	$12,665,940	$9,822,740	$7,912,836	$7,523,916	$8,733,057	$54,802,581
Net Operating Loss Adjustment	$0	($6,784,470)	$25,972,014	$17,810,638	$18,840,563	$11,861,255	$0	($1,397,951)	$66,302,049
Depletion	$0	$0	$0	$0	$0	$4,215,673	$7,112,527	($674,512)	$10,653,688
Total Adjustments for Non Cash Item	**$0**	**($93,444)**	**$44,541,271**	**$36,379,409**	**$32,041,847**	**$24,997,610**	**$15,644,289**	**$11,842,190**	**$165,353,172**
Capital									
Investment - Mine	$0	($7,747,290)							($7,747,290)
Investment - Primary Development	$0	($9,165,788)							($9,165,788)
Investment - Plant	($1,338,840)	($10,284,860)	$0						($11,623,700)
Investment - G&A	$0	($1,132,200)	$0						($1,132,200)
Capital Indirects & Contingency	($600,913)	($5,999,687)	$0						($6,600,600)
Total Capital	**($1,939,753)**	**($34,329,825)**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**($36,269,578)**
Sustaining Capital - Mine			($14,269,608)	($994,878)	($900,320)	($591,245)	($609,843)	($374,759)	($17,740,654)
Sustaining - Primary Development			($13,904,373)	($7,137,982)	($3,386,712)	$0	$0	$0	($24,429,067)
Sustaining Capital - Plant			($2,500,000)	$0	($1,900,000)	$0	$0	$0	($4,400,000)
Sustaining Capital - Indirects& Contingency			($3,322,935)	($191,750)	($539,727)	($113,955)	($117,540)	($72,230)	($4,358,137)
Reclamation Closure Costs	$0	$0	$0	$0	$0	$0	$0	($1,200,000)	($1,200,000)
Total Capital & Sustaining	**($1,939,753)**	**($34,329,825)**	**($33,996,915)**	**($8,324,611)**	**($6,726,759)**	**($705,200)**	**($727,383)**	**($1,646,989)**	**($88,397,436)**
Working capital		$0	($2,835,000)	$0	$0	$0	$0	$2,835,000	$0
Equipment Financing	$0	$911,850	$9,487,500	$0	$0	$0	$0	$0	$10,399,350
Principal Payments	$0	($87,183)	($1,719,576)	($3,316,717)	($3,546,483)	($1,729,391)	$0	$0	($10,399,350)
Total Capital & Working Capital	**($1,939,753)**	**($33,505,158)**	**($29,063,991)**	**($11,641,328)**	**($10,273,242)**	**($2,434,592)**	**($727,383)**	**$1,188,011**	**($88,397,436)**
Beginning Cash	**$0**	**($1,939,753)**	**($35,538,355)**	**($20,061,075)**	**$4,677,006**	**$26,445,611**	**$52,339,012**	**$72,874,814**	
Period Net Cash Flow	**($1,939,753)**	**($33,598,602)**	**$15,477,280**	**$24,738,080**	**$21,768,605**	**$25,893,400**	**$20,535,803**	**$13,895,287**	**$86,770,102**
Ending Cash	**($1,939,753)**	**($35,538,355)**	**($20,061,075)**	**$4,677,006**	**$26,445,611**	**$52,339,012**	**$72,874,814**	**$86,770,102**	**$86,770,102**

The projected total lifespan of the Project is 6.9 years: 1.2 years of pre-production and construction and 5.7 years of operations. Approximately 240,500 oz Au is projected to be mined, with 228,200 oz recovered and produced for sale. An initial capital investment of $36.2 million, including contingency/working capital is projected. Following the All-In-Sustaining-Cost ("AISC") guidelines, life-of-mine average base case Cash Operating Cost is projected to be $864/oz of gold sold. The AISC LOM average base case Total Operating Cost (including royalties and production taxes) is expected to be $1,012/oz. The total AISC summary per ton of mill feed and per ounce of gold is expected to be $1,286/oz, as presented in Table 1-5.

Table 1-5 Copperstone Project Total Operating Cost per oz Au & per ton Mill Feed

Operating Costs	$/oz Au	$/ton mill feed
Mining	$512.88	$95.79
Processing	$253.69	$47.38
Site G&A	$85.05	$15.89
Transportation & Refining	$12.84	$2.40
CASH OPERATING COSTS	**$864.46**	**$161.46**
Royalties and Stream	$138.89	$25.95
Production Taxes	$8.76	$1.64
TOTAL CASH COSTS	**$1,012.11**	**$189.05**
Reclamation	$5.26	$0.98
Sustaining Capital	$268.65	$50.17
ALL-IN SUSTAINING COSTS	**$1,286.02**	**$240.20**

1.9 Conclusions

The total Measured and Indicated mineral resources for the Copperstone Project are estimated at 1.33 million tons grading 0.226 oz/ton Au. Additional Inferred resources are estimated to be 1.07 million tons grading 0.184 oz/ton Au. Factors that may affect the mineral resource estimate include changes to geological or grade interpretations, including grade shell considerations; changes to the modelling method or approach; changes to metallurgical recovery assumptions; and changes to any of the social, political, economic, permitting, and environmental assumptions considered when evaluating reasonable prospects for eventual economic extraction.

Based on the assumptions and parameters presented in this report, the PEA shows positive economics with a $61.78 million post-tax NPV (5%) and 50.5% post-tax IRR. The PEA supports a decision to carry out additional detailed studies.

1.10 Recommendations

The following paragraphs summarize recommended tasks that should be completed in order to advance the Project and to prepare for development and operations.

1.10.1 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, and it is recommended that these efforts are continued. All drillhole information should be migrated into a formal database software such as AcQuire or Access.

1.10.2 Structural Understanding

Structural understanding in the Footwall zone and South zone is reasonably assumed to be parallel to the Copperstone shear zone. Support for this structural interpretation of the Footwall and South zones can be accomplished by systematically reviewing intercepts from core drilling and attempting to measure the orientation of mineralization relative to the core axis. Future drilling could incorporate the use of down-hole Televiewer imagery to produce accurate in-situ structural measurements which can be linked to the geological and assay logs for modeling purposes. If a Televiewer is not used or available core drilling with oriented across the strike and dip of the Footwall and South zones could also be considered. Confirmation of the structural understanding of the Footwall and South zones could upgrade portions of the mineral resource estimate currently classified as Inferred to Indicated.

1.10.3 Additional Drilling

1.10.3.1 Step Out Drilling

SGLD intends to complete a surface core drillhole program totaling approximately 14,000 ft (2,270 m) at an expected cost of approximately 1.5 million $US. The drilling plan will utilize wedges to cover more area from a single initial collar location. The drillhole spacing is planned to be between 100 and 150 feet and targets down dip and down plunge mineralization in the C and D zones, down dip extension of the South zone, as well as an exploration drillhole testing for the furthest down dip expression of the Footwall zone.

Anomalous gold grades have been intersected by drilling northeast of the C zone. Drillhole C96-15 has two intercepts greater than 0.100 oz/ton gold (810-815 ft grading 0.185 oz/ton Au, 855-860 ft grading 0.112 oz/ton Au). Additionally, hole C96-16 encountered two intervals of greater than 0.100 oz/ton Au (1135-1140 ft grading 0.182 oz/ton Au; 1205-1210 ft grading 0.192 oz/ton Au). C96-16 is 283 ft northwest from C96-15. Drillhole C96-14 is 287 ft southeast of C96-15 but returned no significant gold values. No drilling has been completed to the northeast or southwest of C96-15. The intercepts could represent either the down dip extension of the C zone, or the expression of a new mineralization zone. Two drillholes are recommended to test between C96-15 & C96-14, and C96-15 & C96-16; 2 drillholes testing up dip between C96-16 & H4-63 on approximately 200-ft centers; and a step out drillhole 100 ft northeast of C96-15 to test down dip. Drilling these targets will require drilling through previously mined dump material as well as a significant amount of overburden.

Drillhole H5-108 encountered significant gold grade intercepts at depth (1084-1088 ft grading 0.30 oz/ton Au; 1121-1124 ft grading 0.64 oz/ton Au; 1124-1129 ft grading 0.35 oz/ton Au). The hole ended in low grade gold. These intercepts could be the expression of the Footwall zone beneath the A/B zone. Follow up drilling is recommended surrounding this intercept to test the extent and continuity of mineralization, as well as its relation to the South zone.

1.10.3.2 Infill Drilling

Infill drilling in the current Footwall zone and South zone to convert Inferred mineral resources to Indicated or Measured, expand mineral resources, and improve the understanding of geometry and orientation of mineralized structures should be continued in future drilling programs.

1.10.3.3 Exploration Drilling

The following targets are recommended for exploration drilling:

- Test for expression of Footwall zone mineralization at depth below D zone;
- Historic drillhole 06CS-20 intercepted 5 feet of gold mineralization grading 0.6 oz/ton from 1035-1040 ft approximately 3,000 ft southwest of the Copperstone pit and has not been followed up on. No drilling is within 500 ft of the drillhole;
- Testing around CS-266 which had a 10-ft intercept of 0.1 oz/ton Au from 780-790 ft. The intercept is approximately 650 ft southwest from the Copperstone pit, and is not currently incorporated into any known zone of mineralization; and
- Expansion of the Southwest target located approximately 2,500 ft southwest of the mine, to determine mineralization extent.

1.10.4 Byproduct Estimates

The mineral resource estimation was performed only for gold. Silver and copper may be by-products of mining for gold. Silver assays are limited but should be collected in the future to allow for grade estimations. Further exploration drilling, assaying and modelling work of copper bearing gold mineralized material is also required. The amount of cyanide soluble copper in metallurgical test samples had a significant effect on the cyanide consumption. In addition, cyanide consumption generally increased slightly as the particle size decreased. The amount of cyanide soluble copper should be incorporated into the mine plan and economic analysis. Currently, cyanide soluble copper assays make up 0.05% of the total gold assay database. When compared to gold assays greater than 0.05 oz/ton, cyanide soluble copper assays make up 13% of the database. There is reasonable spatial distribution of cyanide copper soluble assays along strike and down dip of available mineral resources. This information will be useful in determining the amount of cyanide that will be consumed.

1.10.5 Metallurgy

The Copperstone Project has undergone significant value engineering during the course of the significant engineering work completed to date. The opportunities identified by SGLD, Hanlon Engineering and Associates (HEA), and Forte Dynamics are:

- Purchase of a new tire driven ball mill;
- Purchase of a new cone crusher;
- Rental of portable crushers;

- Sourcing used equipment in lieu of new equipment where feasible, and
- Process re-design to utilize filtration technologies as an alternative to CCD's.

Utilizing a new tire-driven ball mill and a new cone crusher offer operation and maintenance opportunities with comparative capital cost. Rental of portable crushers will shift expenditures from sustaining capital to operating costs. Purchasing used equipment will be reviewed in more detail in the next phase to evaluate potential capital cost and schedule benefits. Finally, evaluating the use of filtration for pregnant solution recovery will be studied at a preliminary level under a separate trade-off proposal.

1.10.6 Mining

Based on the favorable results of the PEA, a mine design and mine plan is recommended to be advanced to a pre-feasibility level prior to a production decision. The following areas are recommended for further study during the next phase of work:

- 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 to amount of gold recovered from the resource;
- Develop grade control procedures based on the recent stope infill drilling programs that have been completed;
- Further investigate contract mining versus owner mining;
- Hire key underground technical and management staff on a priority basis to facilitate the feasibility design phase; and
- Optimize the ventilation, water management, and electrical power systems.

Estimated costs for the recommendations are shown in Table 1-6:

Table 1-6 Recommended Scope of Work Cost for the Copperstone Project

Recommendation	Estimate
Drillhole Database	**$10,000**
Structural Understanding	**$30,000**
Step Out Drilling	**$1,500,000**
Mineral Processing & Recovery Methods Trade Off Study	**$10,000**
Mining	
Optimize short term mine design	$25,000
Review use of lower cut-off grades	$5,000
Grade Control Program	$5,000
Contract Mining vs Owner Mining analysis	$10,000
Key underground mining staff	$100,000
Optimize Ventilation, Water, and Power systems	$25,000
Total Mining	**$170,000**
Update PEA or Pre-Feasibility Study	**$150,000**
Total Budget	**$1,870,000**

2. INTRODUCTION

2.1 Issuer and Terms of Reference

Sabre Gold Mines Corp. ("SGLD") (TSX:SGLD) is a North American gold exploration company headquartered in Vancouver, Canada. 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.

SGLD, the Issuer of this report, has retained Hard Rock Consulting, LLC ("HRC") to prepare a Preliminary Economic Assessment ("PEA") for the Copperstone Project. This report presents the results of the PEA and is intended to fulfill the Standards of Disclosure for Mineral Projects according to Canadian National Instrument 43-101 ("NI 43-101").

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 29, 2019. The mineral resource estimate reported herein is based on all available technical data and information as of February 15, 2023. The effective date of this report in full is June 26, 2023.

2.2 Sources of Information

A portion of the information and technical data for this study was obtained from the following previously filed NI 43-101 Technical Reports:

AMEC Americas Limited, 2006. *NI 43-101 Technical Report, Copperstone Property, La Paz, Arizona*; prepared for American Bonanza Gold Corp., March 27, 2006.

Continental Metallurgical Services, 2010. *NI 43-101 Technical Feasibility Report, Copperstone Project, La Paz County, Arizona*; prepared for American Bonanza Gold Corp., January 10, 2011.

Hard Rock Consulting LLC, 2018. *National Instrument 43-101 Technical Report: Preliminary Feasibility Study for the Copperstone Project, La Paz County, Arizona, USA*; prepared for Kerr Mines Inc., May 18, 2018.

Hard Rock Consulting LLC, 2021. *National Instrument 43-101 Technical Report: Updated Mineral Resource Estimate for the Copperstone Project, La Paz County, Arizona, USA*; prepared for Sabre Gold Mines Corp., September 21, 2021.

Mine Development Associates, 2000. *Geological Report for the Copperstone Gold Property, La Paz County, Arizona U.S.A.*; prepared for American Bonanza Gold Corp., October 26, 2000.

Telesto Nevada, Inc., 2010. *NI 43-101 Technical Report for the Copperstone Project, La Paz County, Arizona*; prepared for American Bonanza Gold Corp., February 11, 2010.

The information contained in current report Sections 4 through 8 was largely previously presented in, and in some cases is excerpted directly from, the technical reports listed above. HRC has reviewed this material in detail, and finds the information contained herein to be factual and appropriate with regard to guidance provided by NI 43-101 and associated Form NI 43-101F1.

Additional information was requested from and provided by SGLD. With respect to Sections 9 through 13 of this report, the authors have sourced information in part from historical documents, including exploration reports, technical papers, sample descriptions, assay results, and maps and drill logs generated by previous operators and associated third party consultants. Historical documents and data sources used during the preparation of this report are cited in the text, as appropriate, and are summarized in report Section 27.

2.3 Qualified Persons and Personal Inspection

This report is endorsed by the following Qualified Persons, as defined by NI 43-101: Ms. J.J. Brown, P.G., SME-RM, Mr. Richard Schwering, P.G., SMR-RM, and Mr. Jeff Choquette, P.E., all of HRC and Dr. Deepak Mulhotra Ph.D., of Forte Dynamics.

Ms. Brown, P.G., SME-RM, has 25 years of professional experience as a consulting geologist and has contributed to numerous mineral resource projects, including more than twenty gold, silver, and polymetallic resources throughout the southwestern United States and South America over the past five years. Ms. Brown is specifically responsible for report Sections 1.1 through 1.3, 2 through 5, 7, 8, 20, 23, 24, and 27.

Mr. Schwering has over 10 years of combined experience in mineral exploration and geologic consulting, including a variety of project work specifically related to structurally controlled gold and silver resources and reserves. Mr. Schwering is specifically responsible for report Sections 1.4, 1.5, 6, 9 through 12, 14, 25.1, and 25.2.

Mr. Choquette, P.E., is a professional mining engineer with more than 25 years of domestic and international experience in mine operations, mine engineering, project evaluation and financial analysis. Mr. Choquette has been involved in industrial minerals, base metals and precious metal mining projects around the world and is responsible for the current report Sections 1.6, 1.8 through 1.10, 15, 16, 18, 19, 21, 22, 25.4, 25.6, 25.7, and 26.

Dr. Malhotra, Ph.D., is Director of Metallurgy for Forte Dynamics and has worked as a mineral process economist and metallurgical engineer for over 50 years. Dr. Deepak Malhotra visited the Copperstone Property on February 6 through February 8, 2018 and reviewed the plant flowsheet and equipment in the plant. Dr. Malhotra is responsible for Sections 1.7, 13, 17, 25.3, and 25.5 of this Technical Report.

HRC representative and QP J.J. Brown, P.G., SME-RM, conducted an on-site inspection of the Copperstone Project on October 31 through November 2, 2017. Ms. Brown spent three full days at Project site accompanied by Kerr Mines Director of Exploration and Geology Brad Atkinson. While on site, Ms. Brown conducted general site and geologic field reconnaissance, including inspection of on-site facilities, examination of surface and underground bedrock exposures, and ground-truthing of reported drill collar locations. Ms. Brown also examined select core intervals from historic and recent drilling, obtained a variety of duplicate samples for independent check sampling, and reviewed with SGLD geology staff the conceptual geologic model, data entry and document management protocols, and drilling and sampling procedures and the associated quality assurance and quality control ("QA/QC") methods presently employed.

Richard A. Schwering, P.G., SME-RM, of HRC conducted a follow-up site visit of the Copperstone Project between March 1st and March 5th, 2021. During the site visit, Mr. Schwering reviewed the following information with Mr. Mike Smith and Mr. Andrew Eubanks of AZG:

- Core and RC sample preparation during the 2020/2021 drilling campaign;
- Review of selected core and RC chips;
- Site tour of underground developments, open pit; and
- Review of records, particularly downhole surveys, for the Project.

Jeff Choquette, P.E., conducted an on-site inspection of the Copperstone Project on July 10 and 11, 2018. While on site, Mr. Choquette conducted general site investigations, including inspection of the office facilities, crushing plant area and the mine laboratory. Mr. Choquette also toured the open pit and underground workings to become familiar with the geology and the general conditions of the underground workings. Although there are a few areas where instabilities of the roof have occurred underground, the majority of the workings are in good shape. Dewatering and ventilation systems are adequately maintained and continue to operate for care and maintenance.

2.4 Units of Measure

Unless otherwise stated, all measurements reported herein are U.S. customary, and currencies are expressed in constant 2023 U.S. dollars. Gold grades are presented in troy ounces per short ton ("oz/ton", "oz/T", or "opt"), unless otherwise indicated.

3. RELIANCE ON OTHER EXPERTS

The QP's have fully relied upon SGLD for information regarding property ownership, mineral tenure, and royalties or other agreements and encumbrances. Such information is presented in report Section 4, and was provided to HRC via the following source documents:

- *Amended and Restated Copperstone Mining Lease Between and Among Angie Patch Survivor's Trust, Daniel L. Patch Credit Trust and Bonanza Explorations, Inc.*, dated January 4, 2017
- *Memorandum of Royalty Interest Between Bonanza Explorations, Inc. and Trans Oceanic Mineral Co.*, dated January 4, 2017
- *Kerr Mines, Inc., Annual Information Form,* dated October 6, 2017
- *Bonanza Explorations Inc. and Trans Oceanic Mineral Company Ltd., Amended and Restated Royalty Purchase Agreement Relating to Copperstone Property,* dated February 23, 2021
- *Bonanza Explorations Inc. and Trans Oceanic Mineral Company Ltd., Royalty Purchase Agreement Relating to Copperstone Property,* dated March 1, 2023
- *Resolution of the Board of Directors of Sabre Gold Mines Corp.,* dated March 1, 2023

The QP's have not reviewed the permitting requirements nor independently verified the permitting status or environmental liabilities associated with the Project and disclaims responsibility for that information. Environmental and permitting information presented in report Sections 4 and 20 was provided to HRC via the following documents:

- *Draft Biological Evaluation for Proposed Exploratory Drilling Activities at Copperstone Mine*; Internal report prepared for Kerr Mines, Inc. by Logan Simpson, September 2017.
- *Mine Plan of Operations, Copperstone Mine, La Paz County, Arizona*; BLM submittal prepared for Kerr Mines, Inc., and Bonanza Explorations, Inc., by Karen Johnson, November 2017.
- *Mine Plan of Operations, AZA 3502, Copperstone Mine, La Paz County, Arizona, Revision MPO 08/12/2019,* prepared for Kerr Mines, Inc., by David Thomas, Copperstone Mine General Manager, dated August 8, 2019.
- *Environmental Permit Review, Copperstone Mine, La Paz County, Arizona*; Internal report prepared for Kerr Mines, Inc., by David Abranovic of Environmental Resources Management, June 2017.

Additional environmental information presented in Section 20 of this report was provided by Mr. Sid Tolbert, Vice President and General Manager of SGLD, in written format on June 6, 2023.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Copperstone Project encompasses approximately 13.8 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 6 through 10 and 15 through 23, T6NR19W; Sections 1, 2, 10 through 14 and 22 through 27, T6NR20W; and Section 19, T7NR19W, 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.

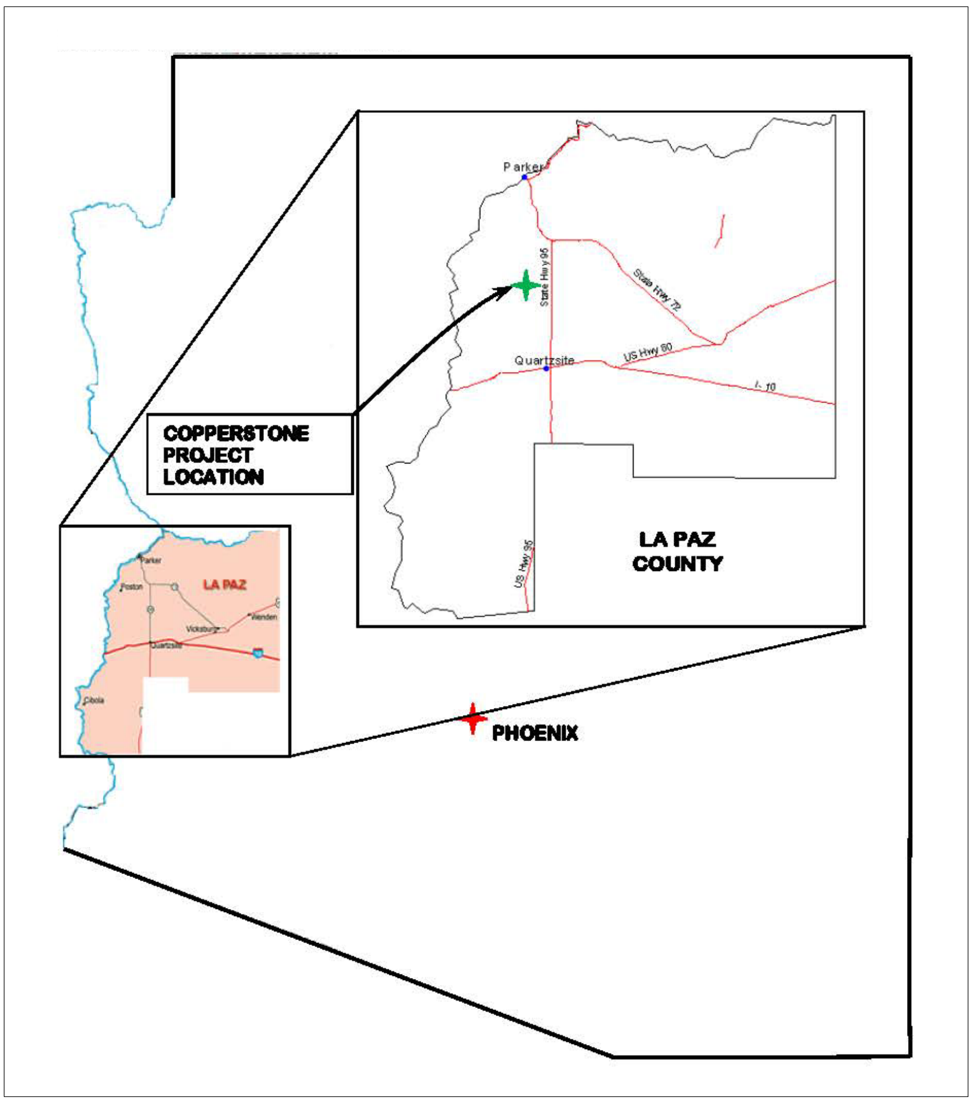

Figure 4-1 Copperstone Project Location

SGLD controls 546 federal unpatented mining claims and two Arizona state mineral leases which together comprise the Copperstone Project area. The federal claims cover approximately 10,920 acres (4,419 hectares) while the state mineral leases total approximately 1,338 acres (542 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.

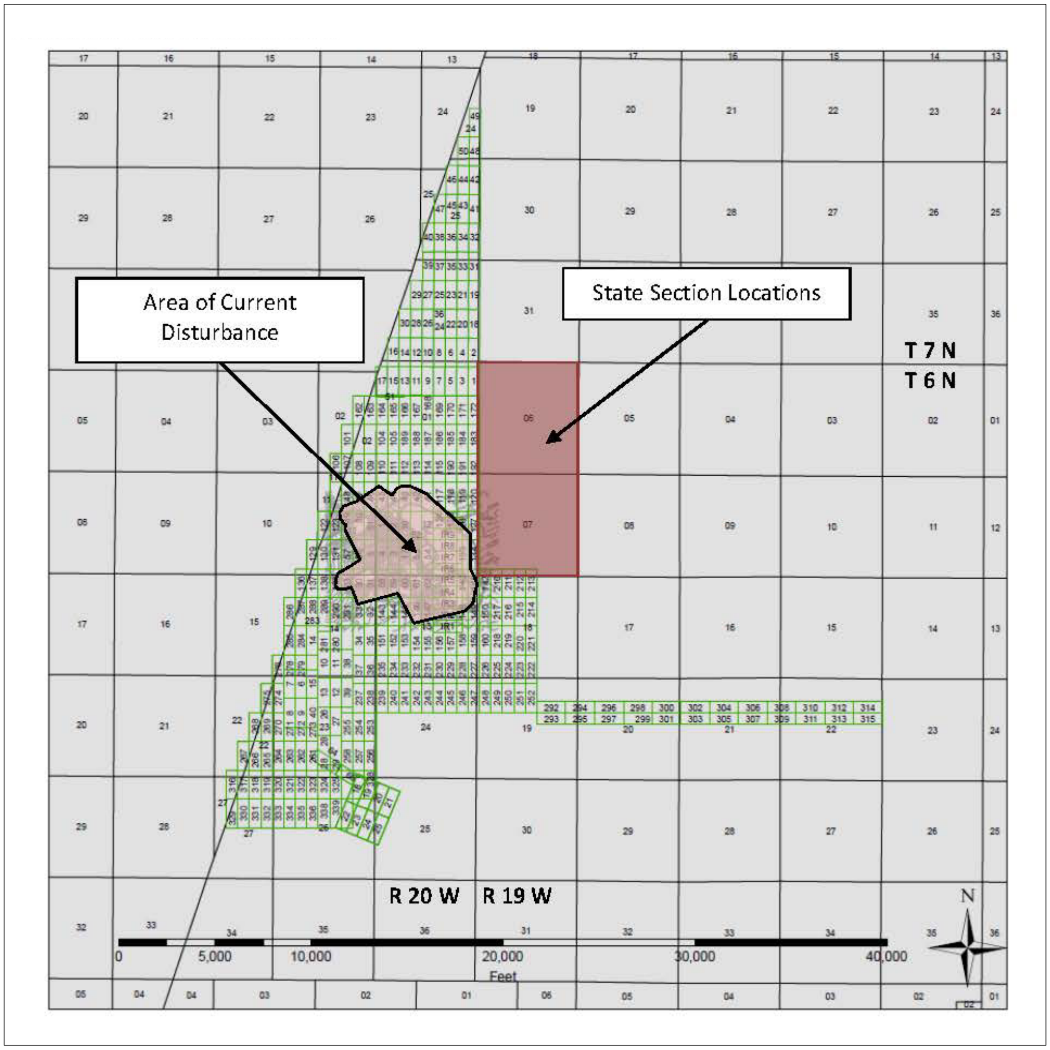

Figure 4-2 Copperstone Project Claim Area

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.

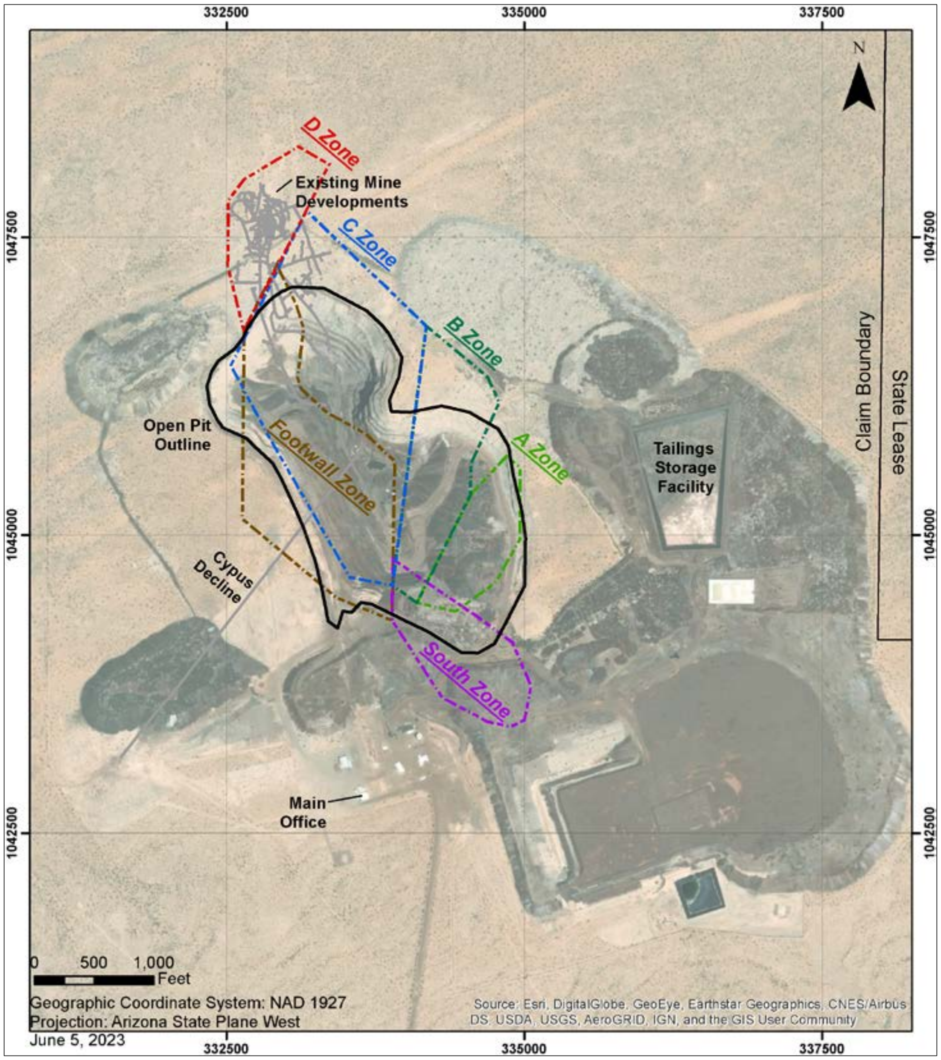

Figure 4-3 Zone Subdivisions within the Existing Disturbance Area

4.2 Property Ownership, Mineral Tenure, Agreements and Encumbrances

The Copperstone Project is wholly owned by SGLD via SGLD's 100% ownership of American Bonanza. On June 27, 2014, Kerr announced the acquisition of all issued and outstanding common shares of American Bonanza by way of plan of arrangement under the *Business Corporations Act* (British Columbia). The arrangement was approved by Kerr shareholders by written consent, by American Bonanza shareholders at its annual general and special meeting of shareholder meeting held on June 20, 2014, and by the Supreme Court of British Columbia on June 25, 2014.

SGLD 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 lease is renewable for one or more ten-year terms at the option of SGLD under the same terms and conditions. SGLD 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 with a minimum advance royalty per year of US$40,000.

In addition to the 1.5 percent royalty held by Angie Patch Survivor's Trust, a 1.5 percent gross production royalty is payable to Trans Oceanic Mineral Company Ltd. ("TOMCL"). The total annual gross production royalty on Copperstone is 3.0 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.

4.3 Permits and Environmental Liabilities

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 on the Copperstone Mine as required by the National Environmental Policy Act. This process involved a number of 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 approved under 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 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.

On February 5, 2019 the Company also announced the Arizona Department of Environmental Quality ("ADEQ") issued approval for the modification of the existing Air Permit ("Air Permit) governing air quality. On September 19, 2019 the Company announced the ADEQ issued approval for the modification of the existing Aquifer Protection Permit ("APP"). The APP is effective for the life of mine and the Air Permit is valid for five years. Table 4-1 provides a summary of the state and federal environmental permits currently in place at the Copperstone mine.

In 2009 American Bonanza posted a reclamation bond of $1.6 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 4-1 Copperstone Mine Permit Summary

Permit	Approval/Permit #	Granting Agency
Mine Plan of Operations	Latest Revision Approved September 2019 to include cyanide usage, increased plant throughput to 600 tpd, 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
ATF Explosives Permit	Permit No. 9-AZ-012-20-0M-00394; Expires December 1, 2020	Alcohol, Tobacco and Firearms (ATF)
AZPDES 2010 Multi-Sector General Permit	Permit No. AZMSG2010-003	Arizona Department of Environmental Quality (ADEQ)
Air Quality Control Permit	Registration No. 73215 as amended December 18, 2018.	ADEQ – Air Quality
Aquifer Protection Permit (APP)	Permit No. P106172 as amended September 18, 2019.	ADEQ – Groundwater Protection
Wastewater Treatment (Type IV APP – Septic)	Permit No. AZMSG2010-00	La Paz County
Well permits	Issued for Life of Mine	Arizona Department of Water Resources (ADWR)
International Cyanide Code Pre-Certification	Issued for Life of Mine	International Cyanide Management Institute
Tire disposal area	Authorized for Life of Facility	ADEQ – Waste Programs Division
Exploration Permit	April 2018/5 years/008-119806; 008- 119807	Arizona State Land Department

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

5.1 Access and Climate

General access to the vicinity of the Copperstone Project is provided by Interstate I-10 out of Phoenix, Arizona, approximately 125 miles west to the city of Quartzsite. The primary Project access road, Cyprus Mine Road, is located roughly 13 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.

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.

5.2 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, warehouse, equipment maintenance shop and assay laboratory buildings, a change house, 10 trailer house hook-ups, a septic system, and a variety of shipping containers which provide for secure core storage. Incoming commercial 69 kV overhead electrical power is delivered to an on-site power substation. Water is currently delivered from three 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 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 tpd floatation 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").

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 Magnetics;
- 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 magnetics 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.

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. RC drilling completed by SGLD, totaling 9,855 ft, is discussed under Item 10 of this report.

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

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, 746 RC holes totaling 342,679.8 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
	1985	Core		55	17,209.40
		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
	2013	Percussion	520-, 650W-, 654-, 654cc-, 690-,726-, 730-, 750-, 810-	154	13,447.00
Arizona Gold Corp. (Kerr Mining Inc.)	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
Total Historic Drilling				1,401	624,163.20
Total Historic Core				475	251,191.40
Total Historic RC				746	342,679.80
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 analyses. 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 minus150 mesh fraction was fire assayed;
- The entire plus150 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%. These limits are represented by the red dashed lines on the figures.

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, ± 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.

6.3.5.1 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 A00-10, with continued success. Additionally, gold intercepts in H5-141, H5-147, H5-148, H5-163, and 06CS-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.

6.3.5.2 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.

6.3.5.3 Surface Exploration Drilling

Thirty-seven core drillholes totaling 37,488.5 ft and 11 RC drillholes totaling 10,511 ft constitute American Bonanzas 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.

6.3.5.4 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.

6.3.5.5 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.

6.3.6 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.

6.3.6.1 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 vertical, 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			

6.3.6.2 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.

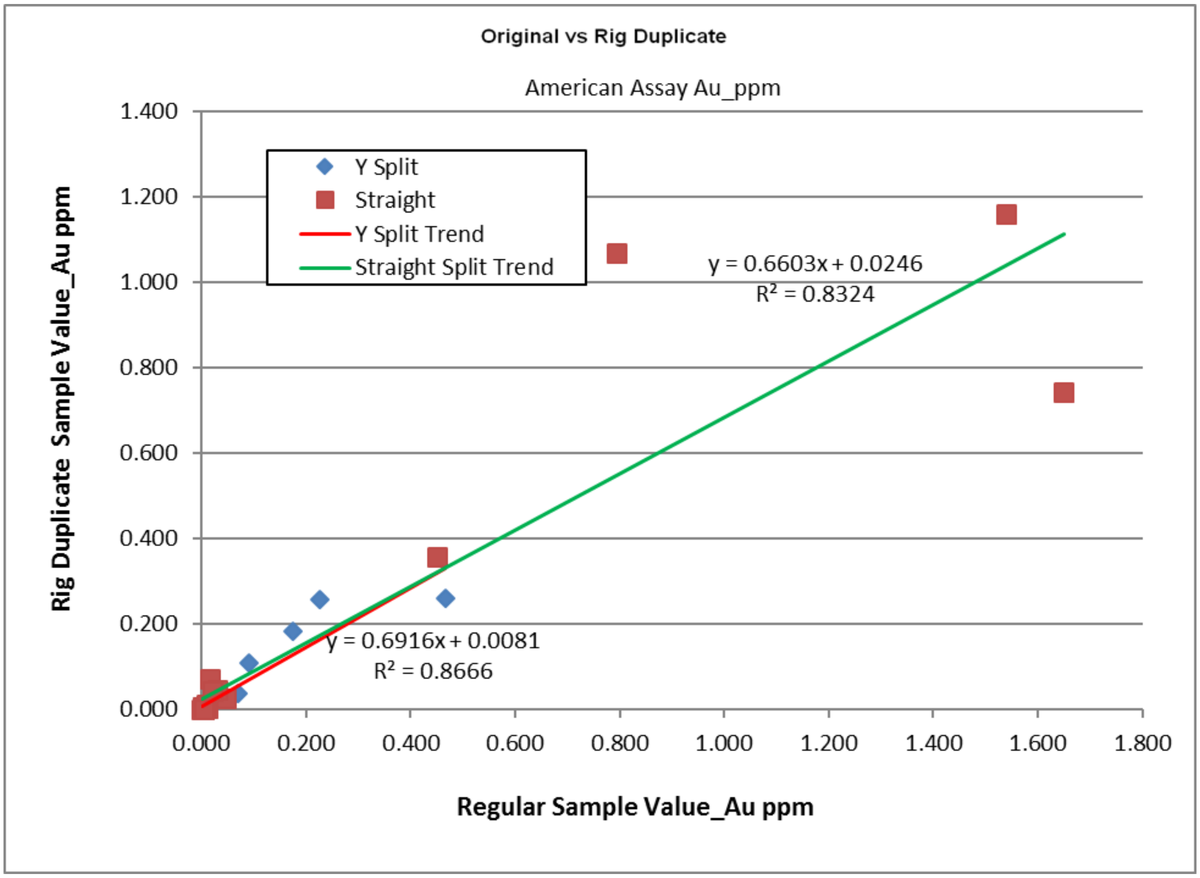

Figure 6-1 Comparison of Y-Split and Straight Split Rig Duplicate Methods

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.

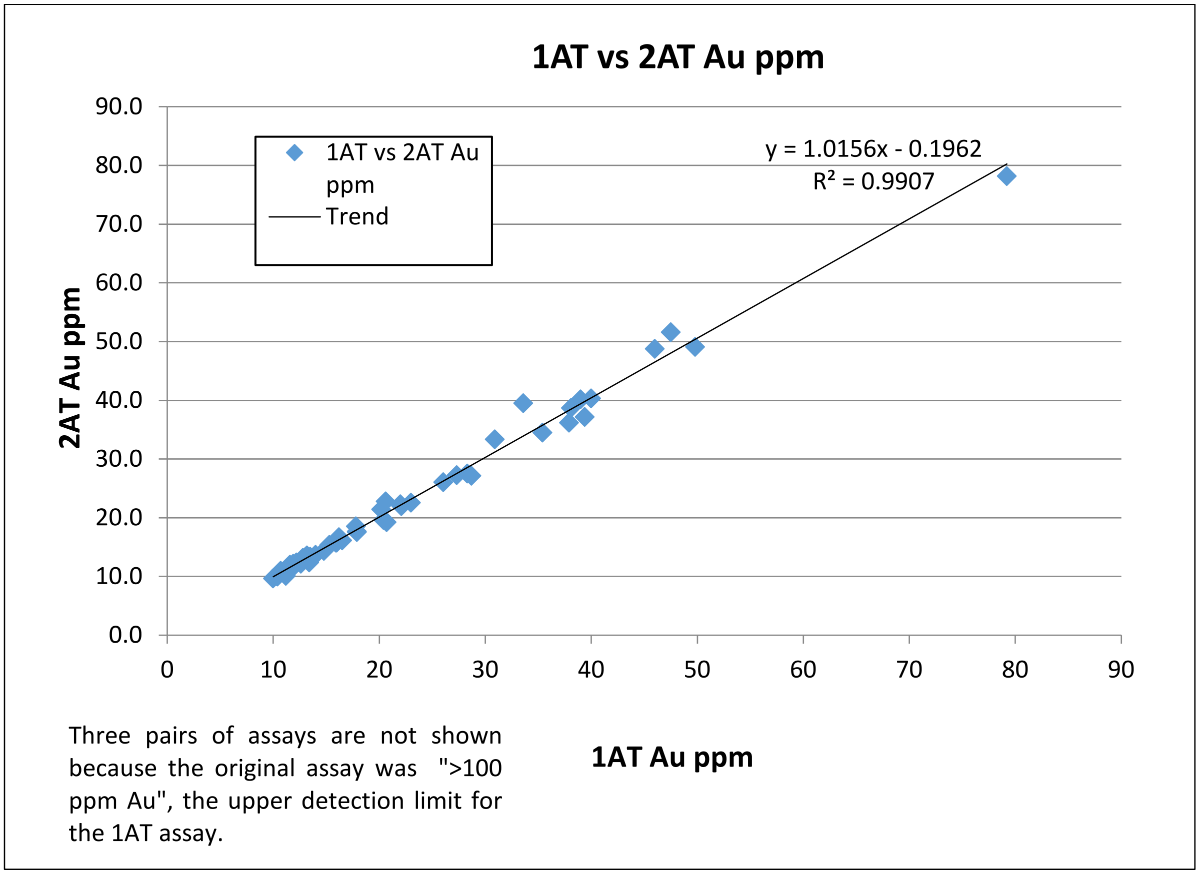

Figure 6-2 Comparison of 1AT with 2AT Charges

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 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).

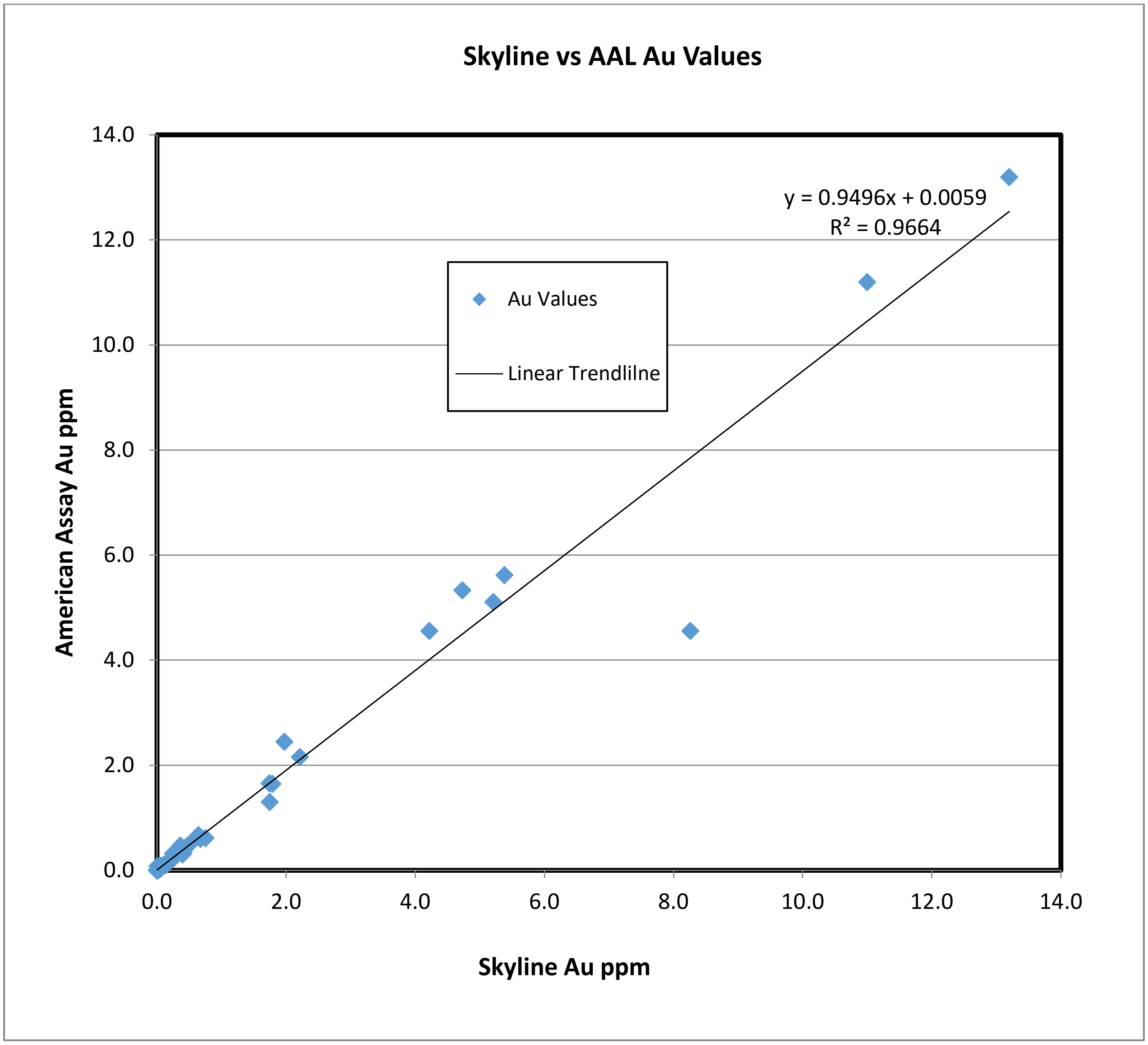

Figure 6-3 Comparison of Skyline with American 1AT Assays, for 2017 Program Pulps

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).

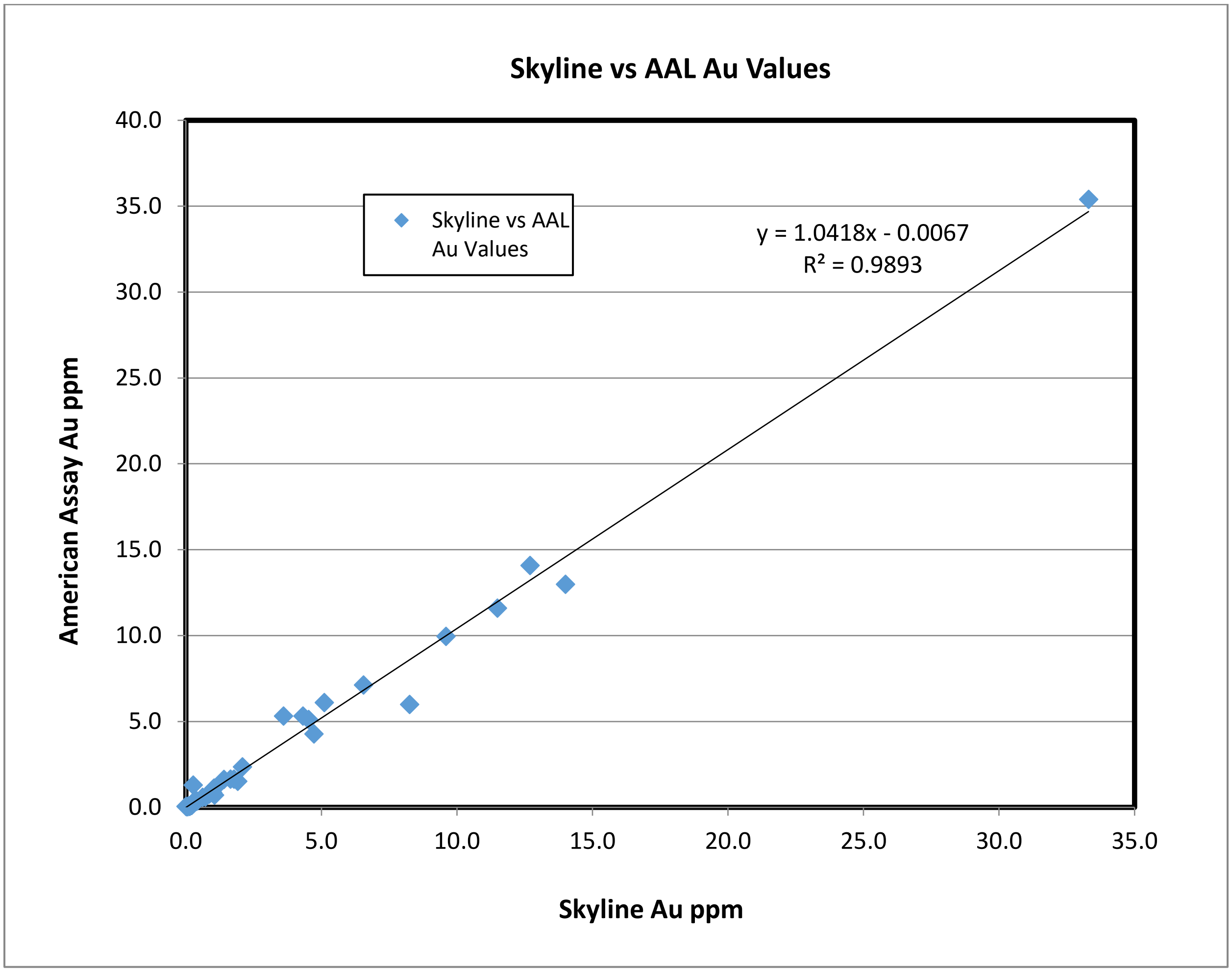

Figure 6-4 Comparison of Original Pulp Analysis by AAL and Skyline on AAL Coarse Rejects

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.

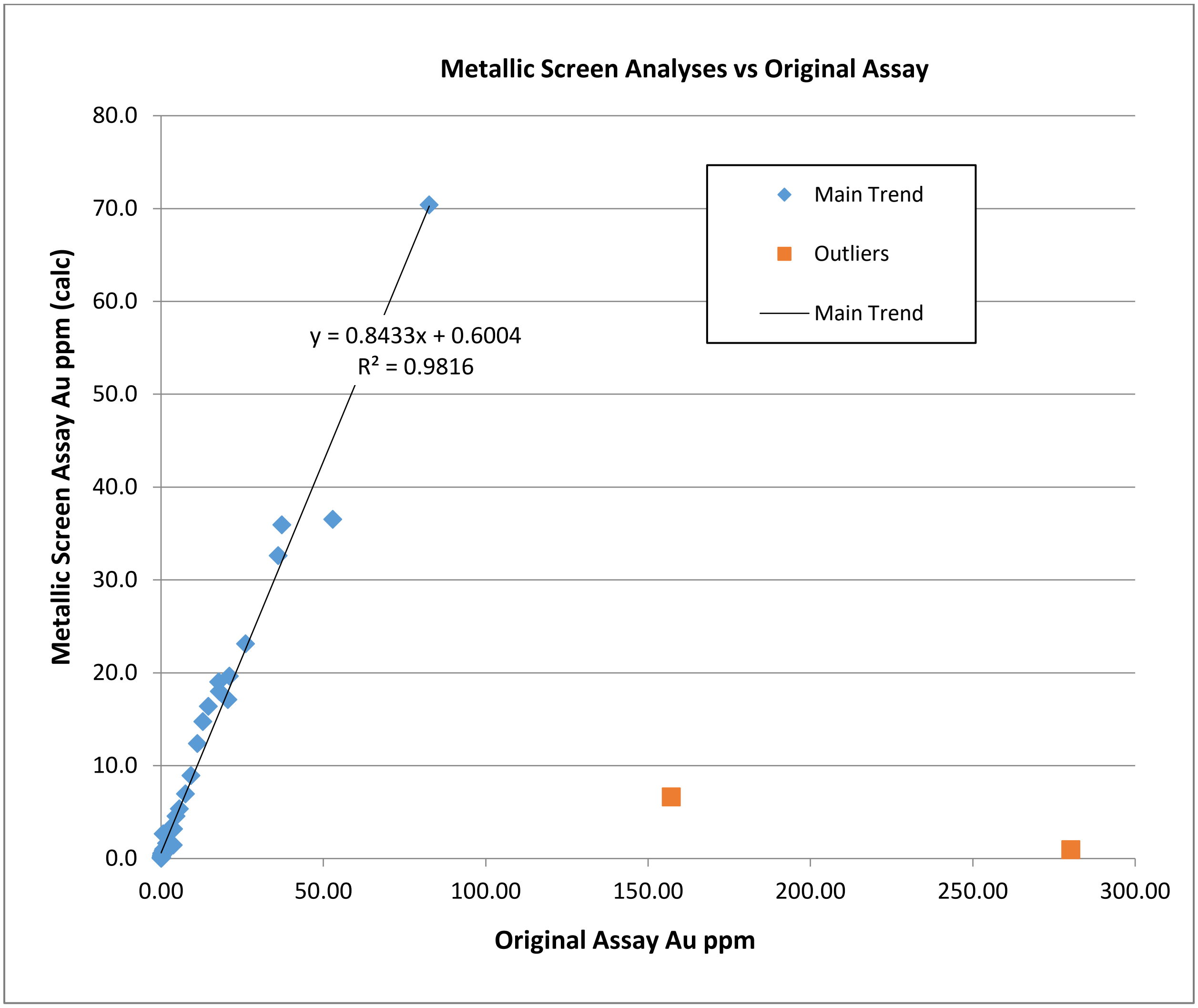

Figure 6-5 Comparison of Original Assay with Metallic Screen Analysis of Coarse Reject, both by AAL

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.

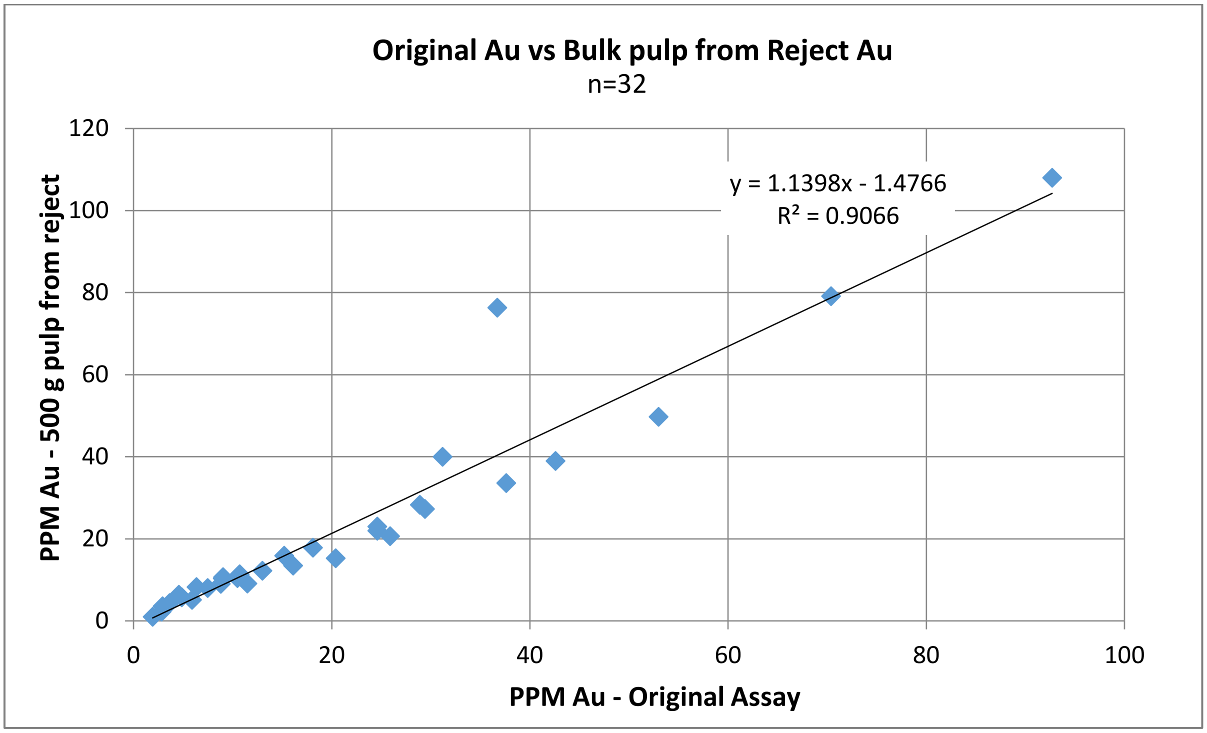

Figure 6-6 Comparison of Original Assay with Assay of 500 g Pulp from Coarse Reject

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

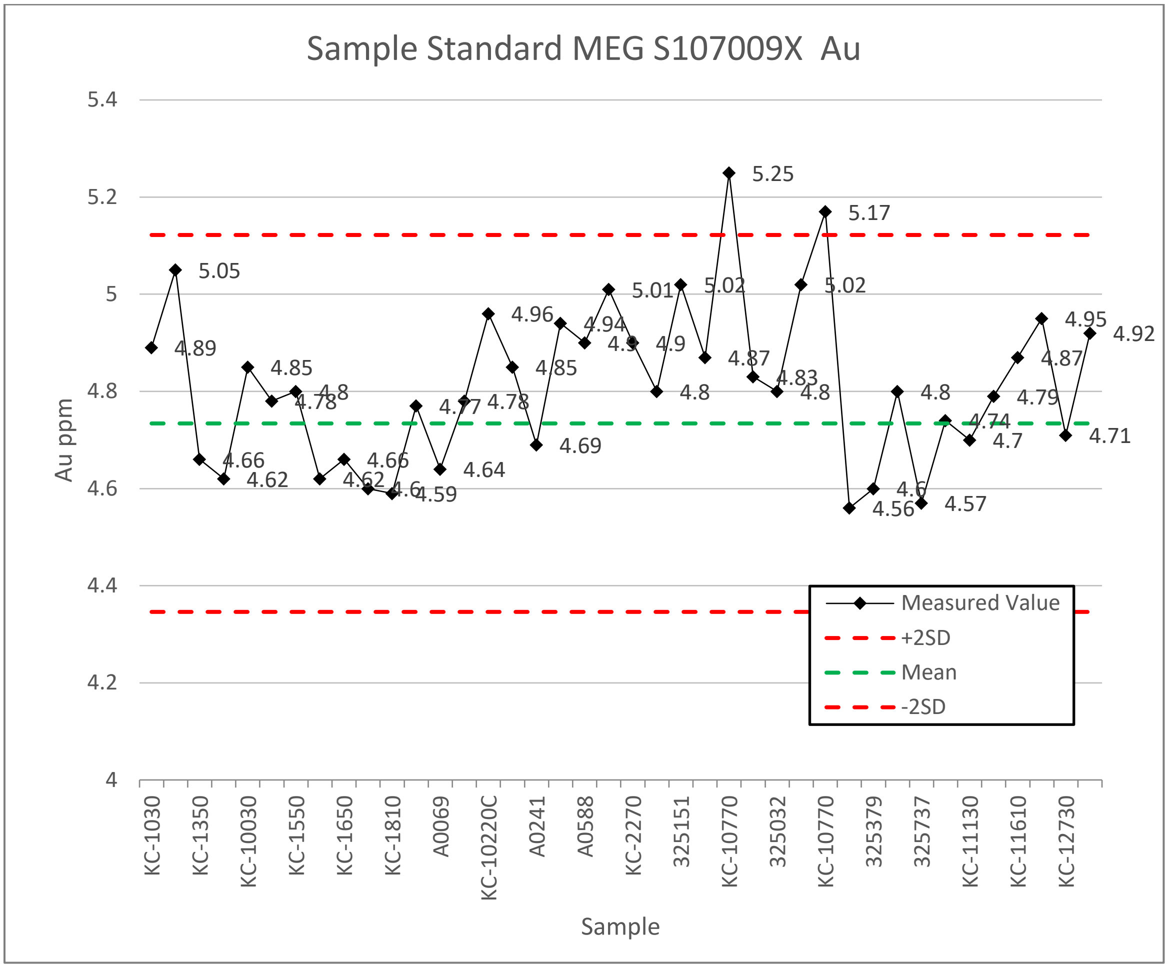

Figure 6-7 Analytical Results for Control Sample MSL, All 2017 Drilling

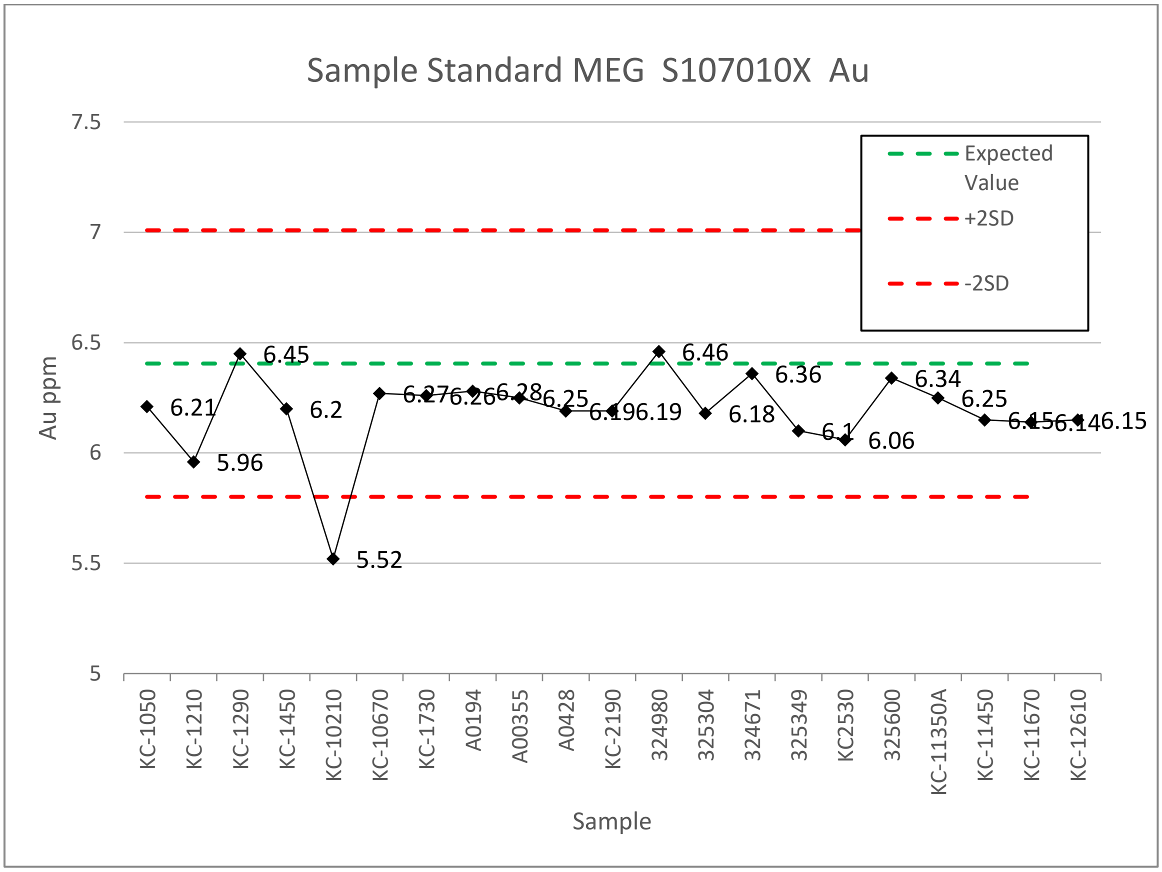

Figure 6-8 Analytical Results for Control Sample MSM, All 2017 Drilling

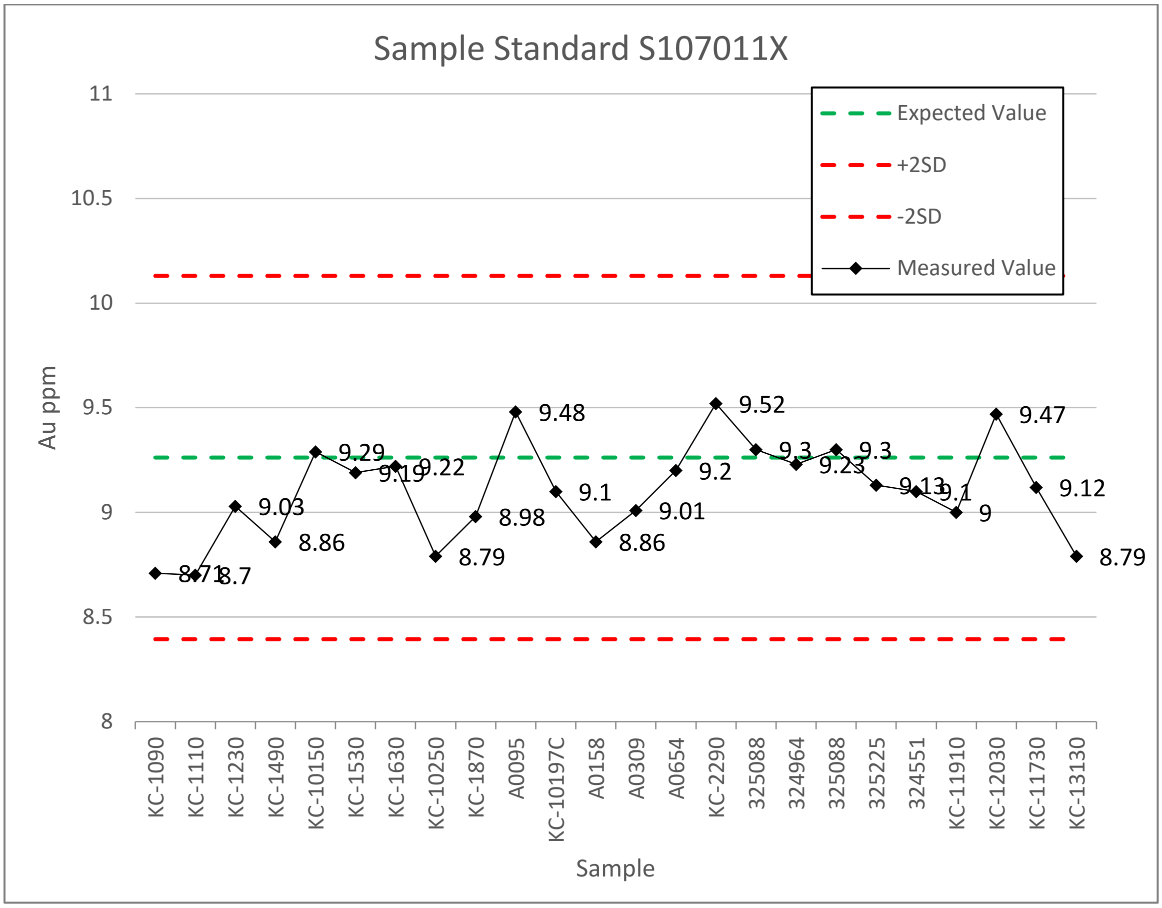

Figure 6-9 Analytical Results for Control Sample MS, All 2017 Drilling

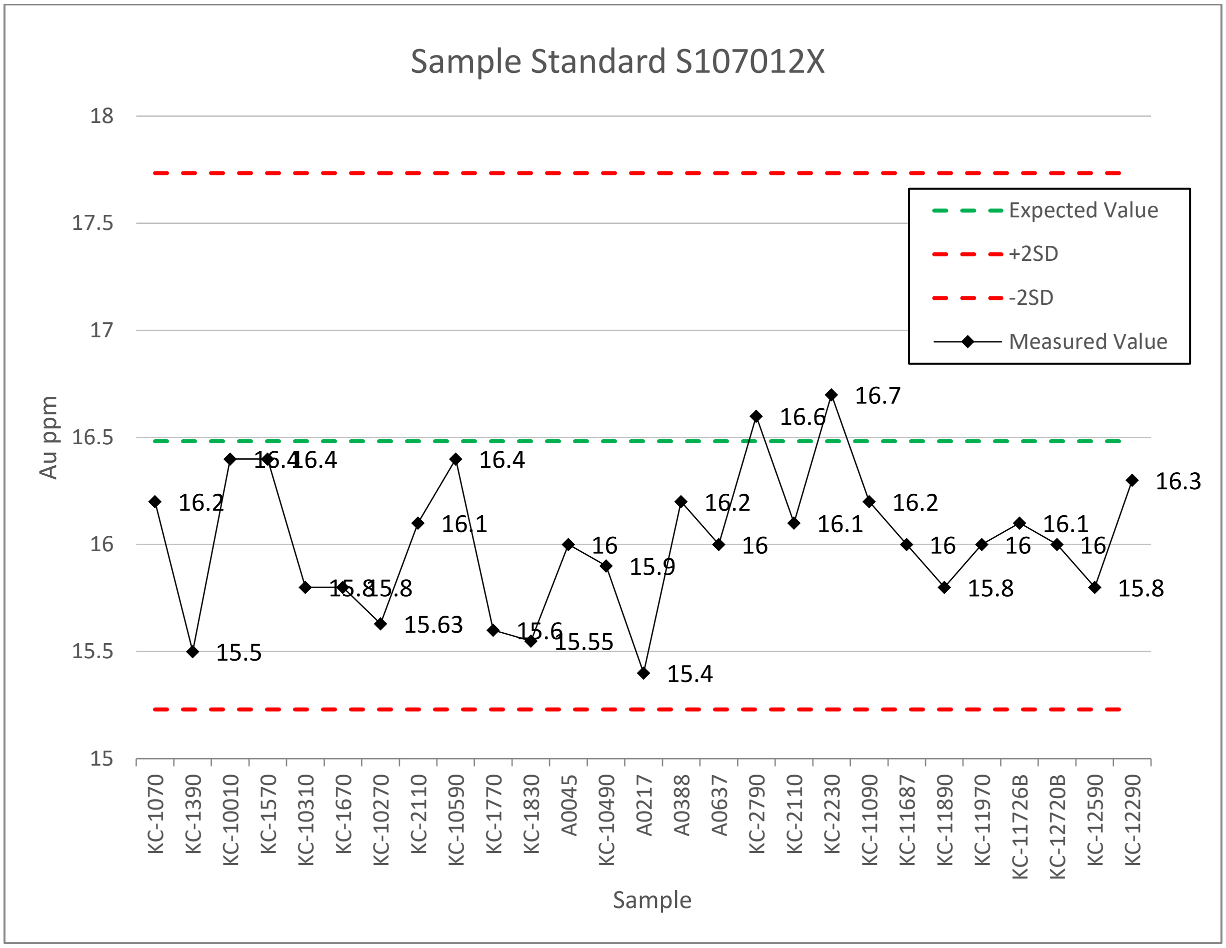

Figure 6-10 Analytical Results for Control Sample MSH, All 2017 Drilling

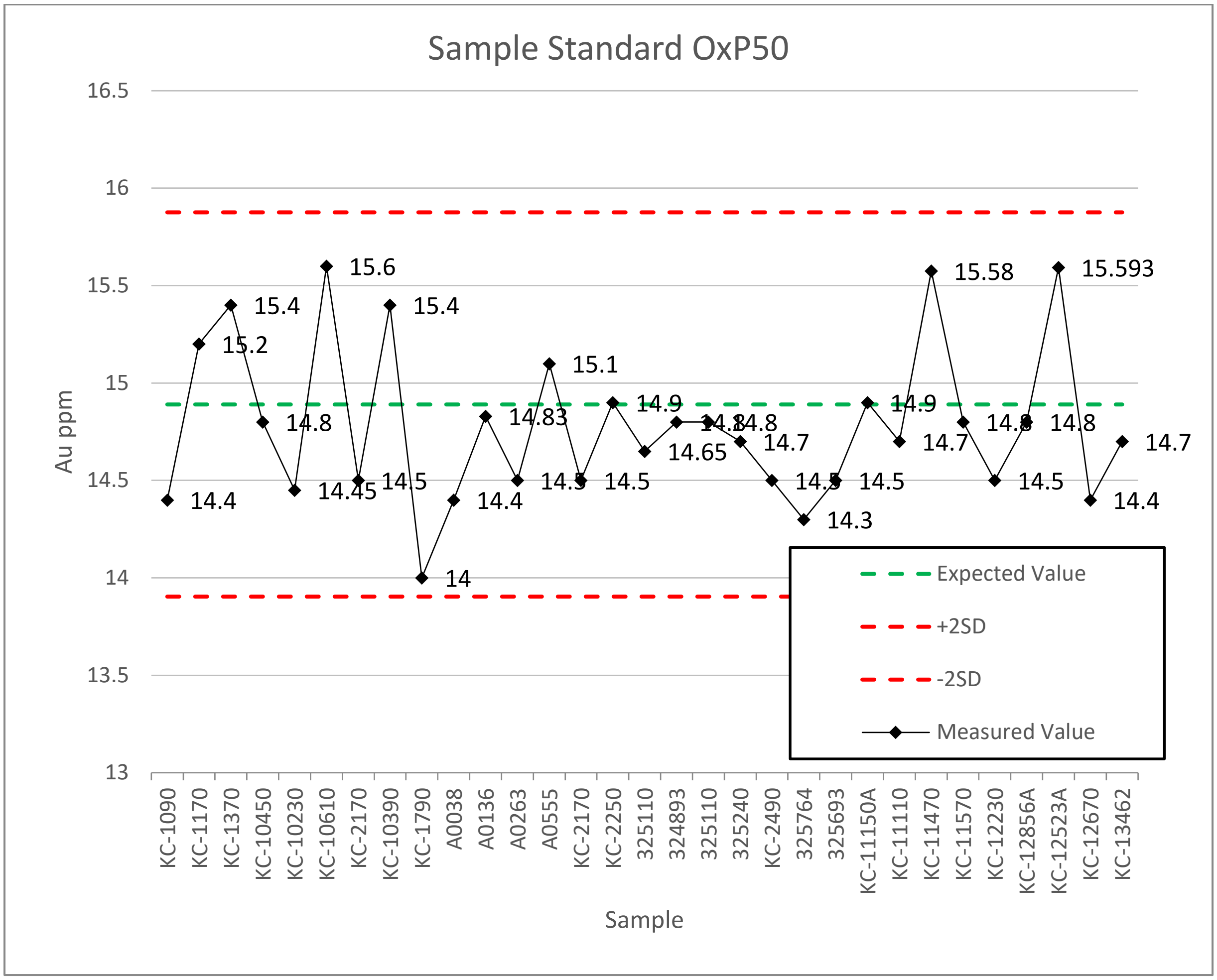

Figure 6-11 Analytical Results for Control Sample MH, All 2017 Drilling

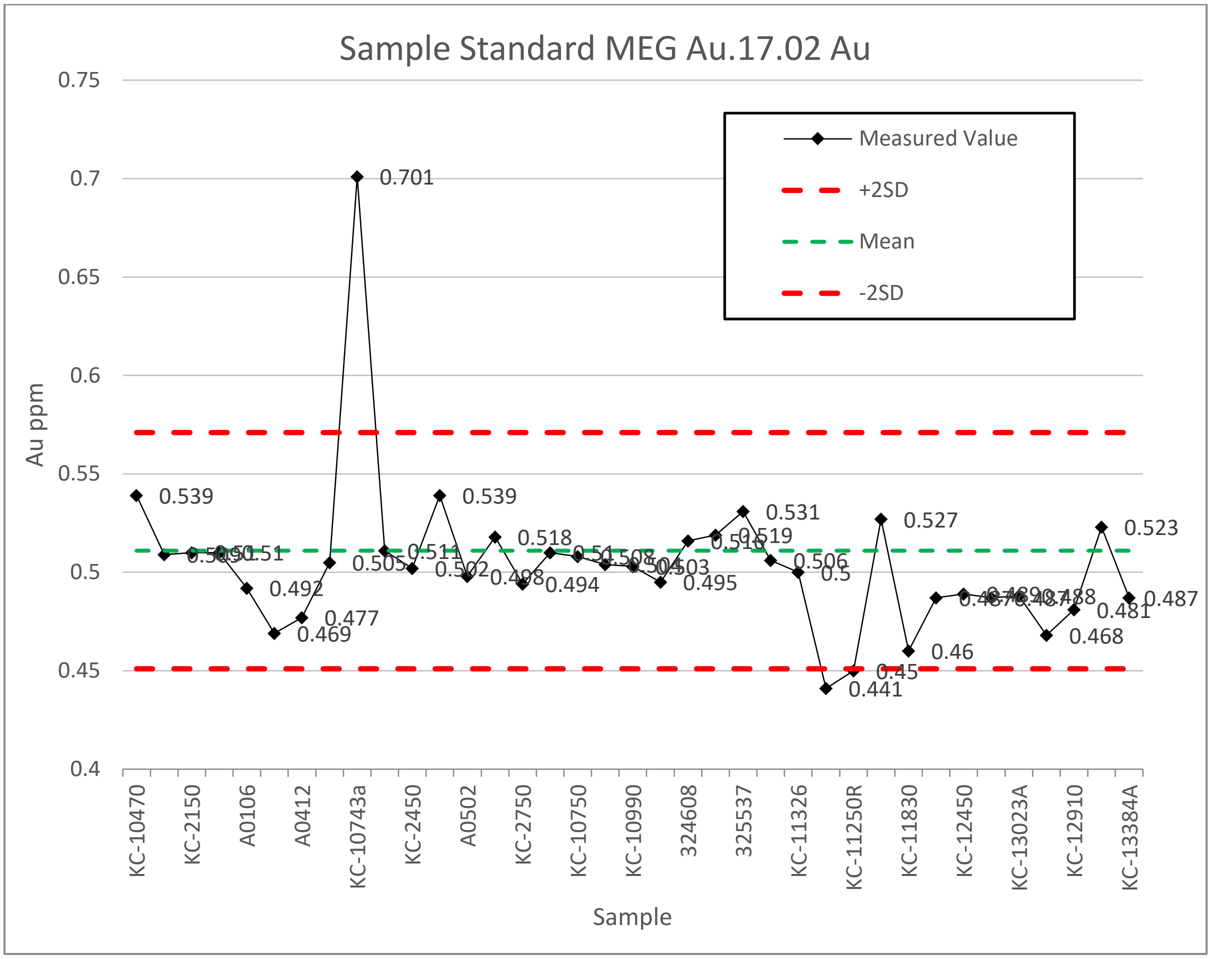

Figure 6-12 Analytical Results for Control Sample L, All 2017 Drilling

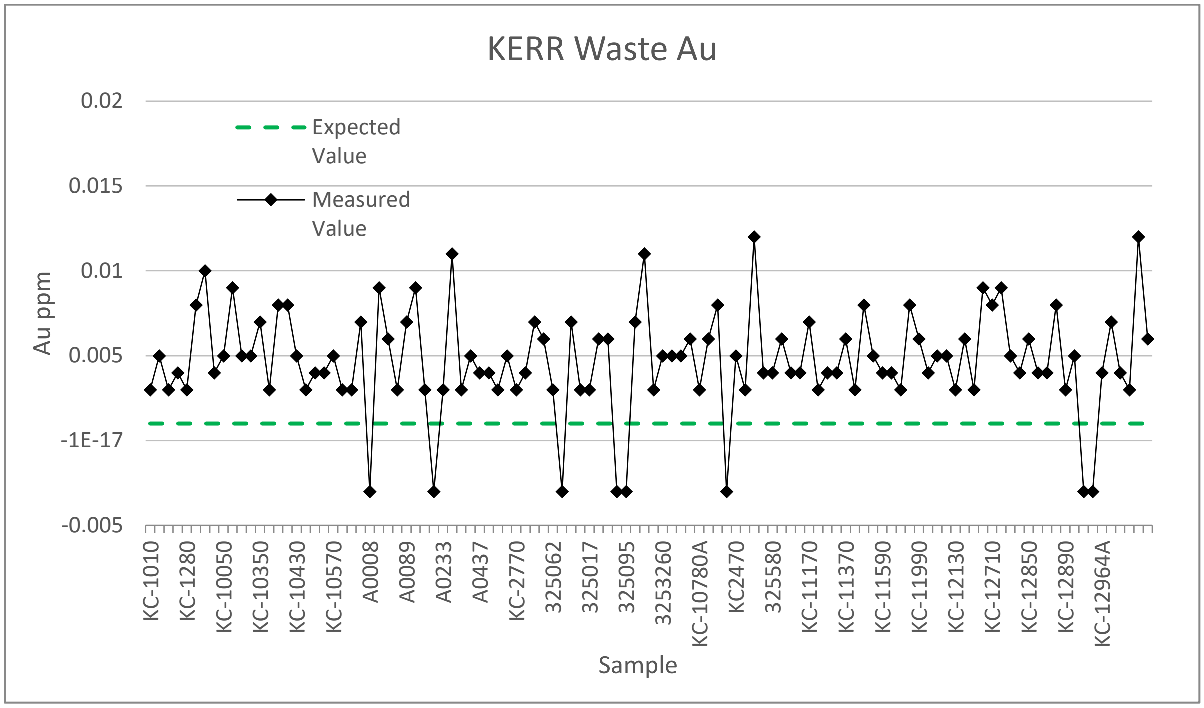

Figure 6-13 Analytical Results for Control Sample W, All 2017 Drilling

All but one assay of MSL, by AAL, have been within +/-2STDEV 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 +/-2STDEV 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

6.3.6.3 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.

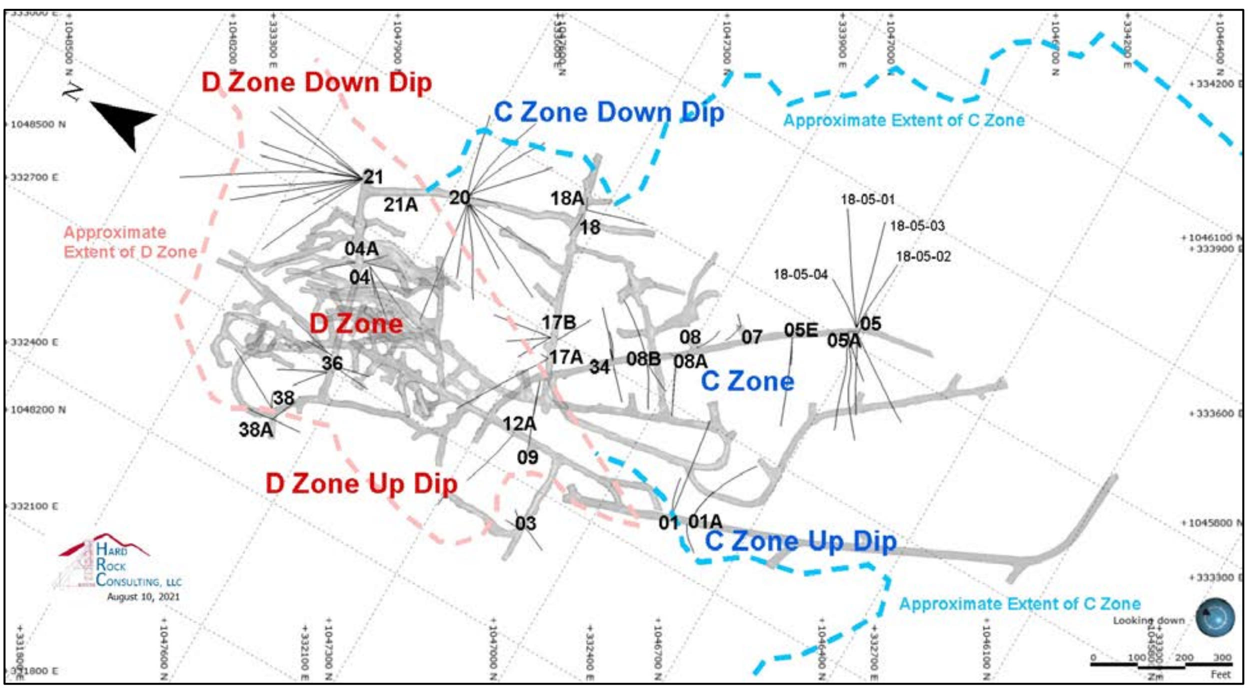

Figure 6-14 2019 Underground Drill Station Locations

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 $\pm$ 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 $\pm$ 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.

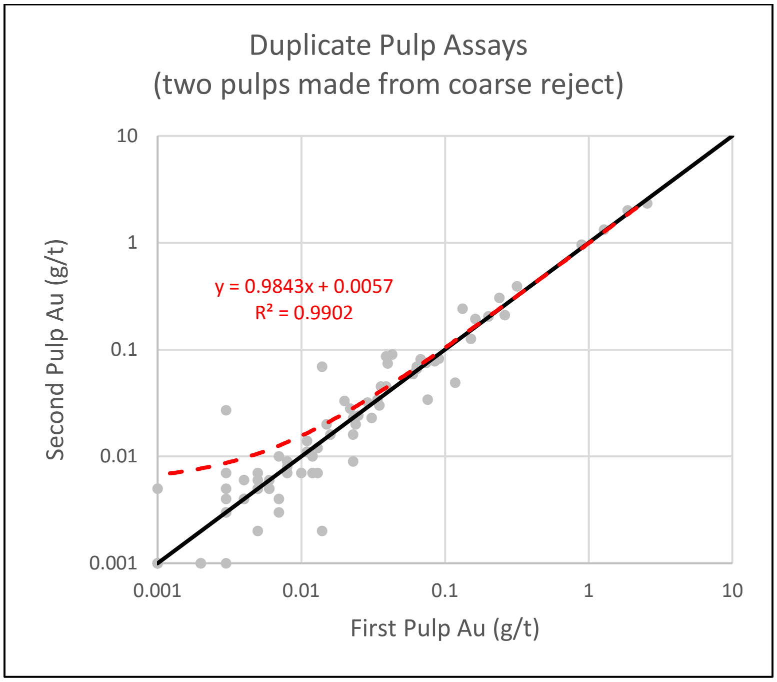

Figure 6-15 Analysis of Crusher Duplicates

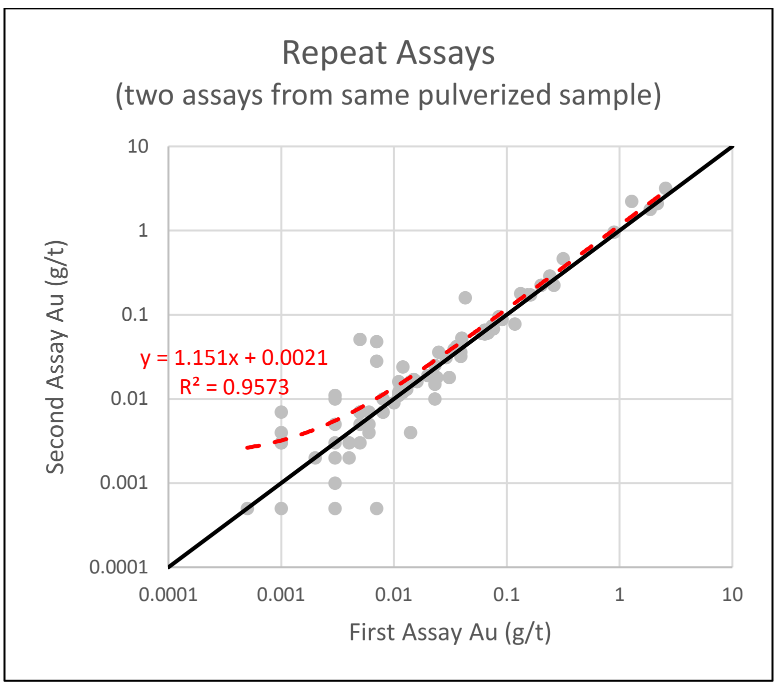

Figure 6-16 Analysis of Pulp Duplicate

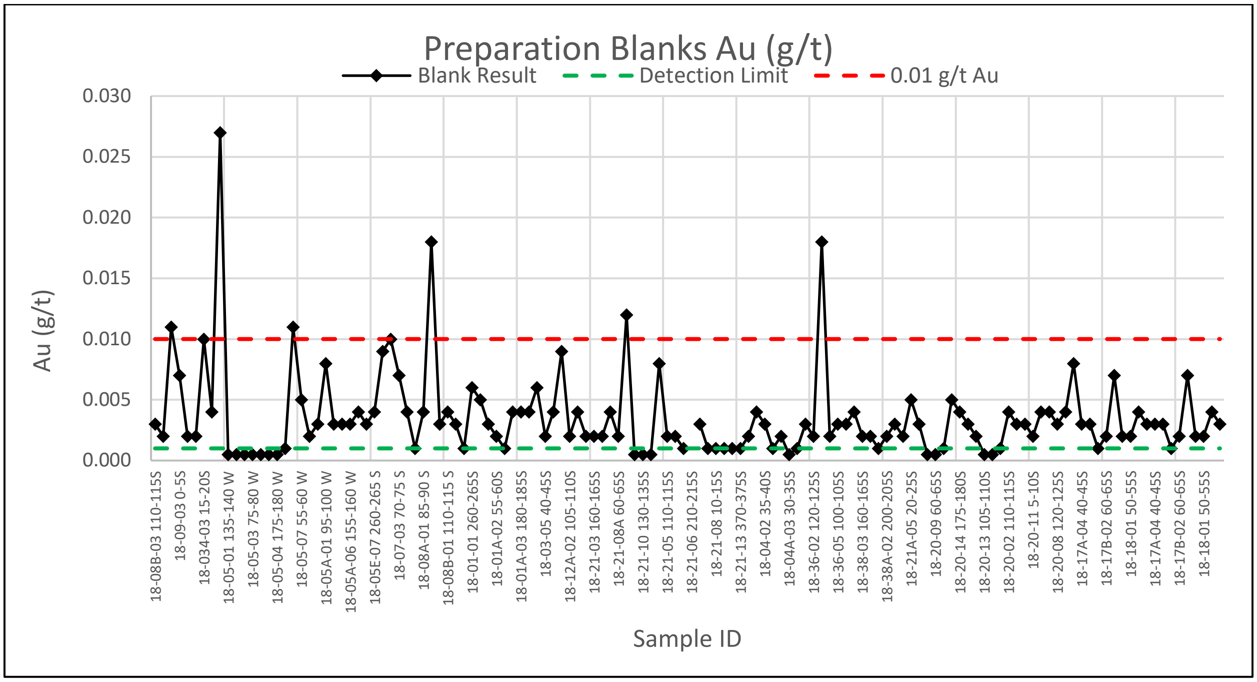

Figure 6-17 Analysis of Blank Material

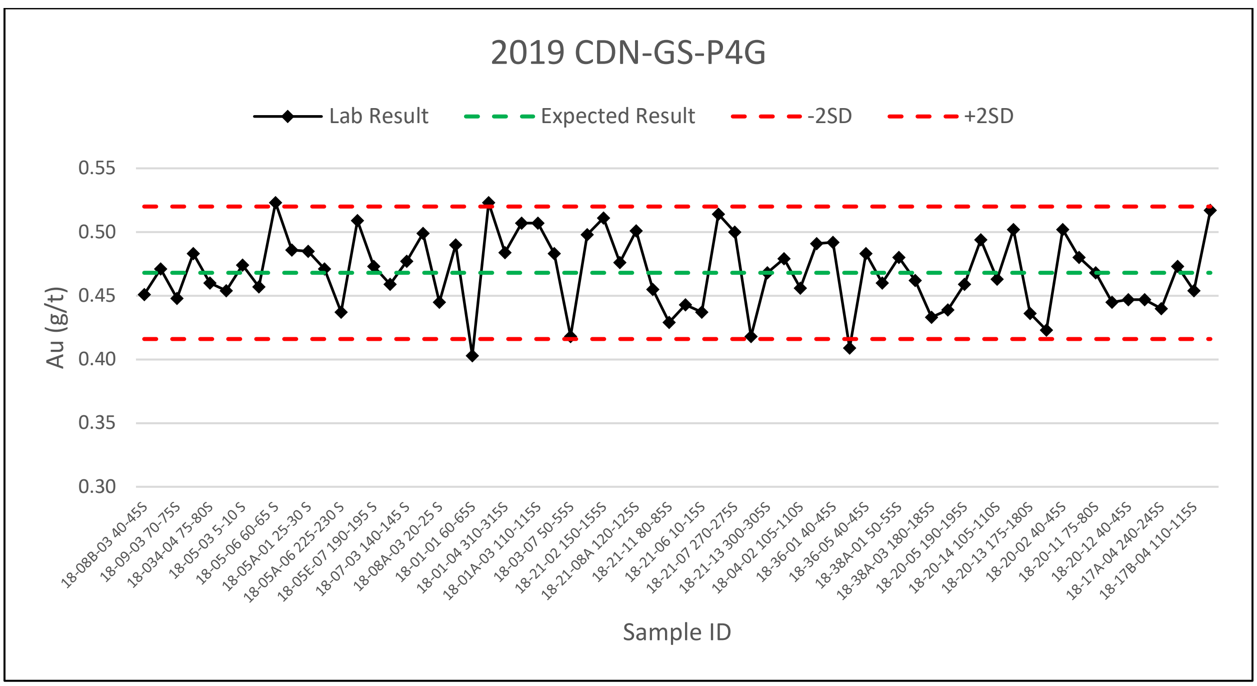

Figure 6-18 Analysis of CDN-GS-P4G

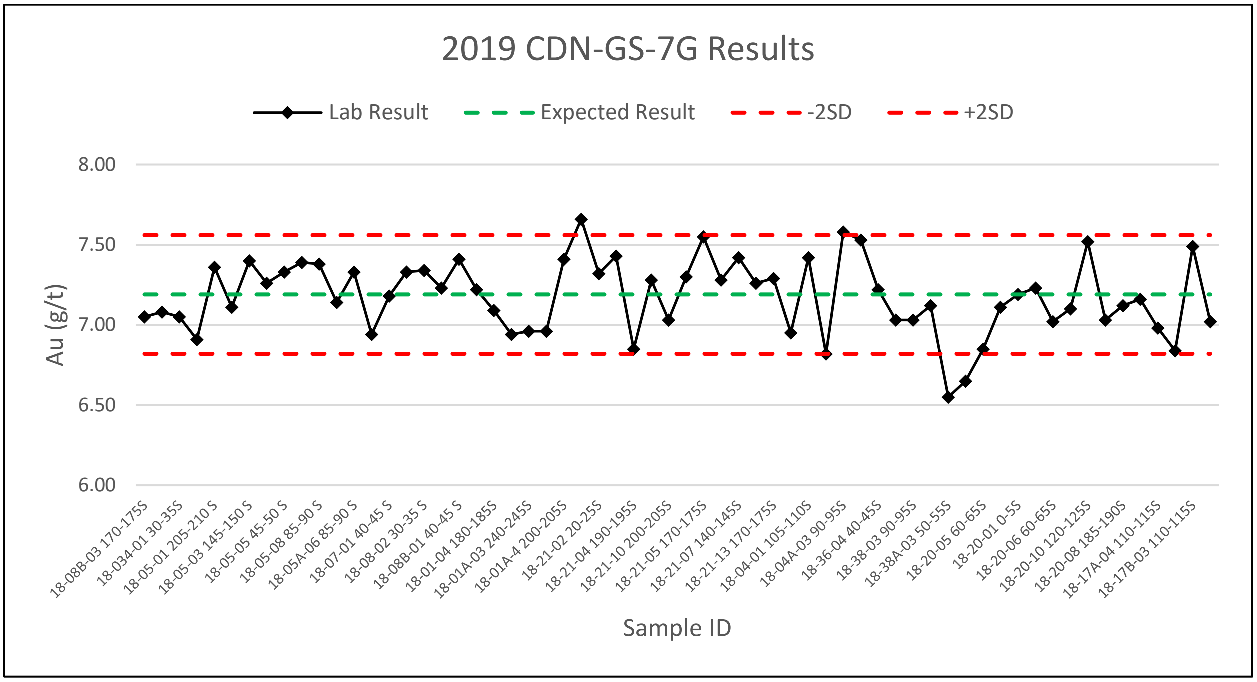

Figure 6-19 Analysis of CDN-GS-7G

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

6.3.6.4 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 $\pm$ 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.

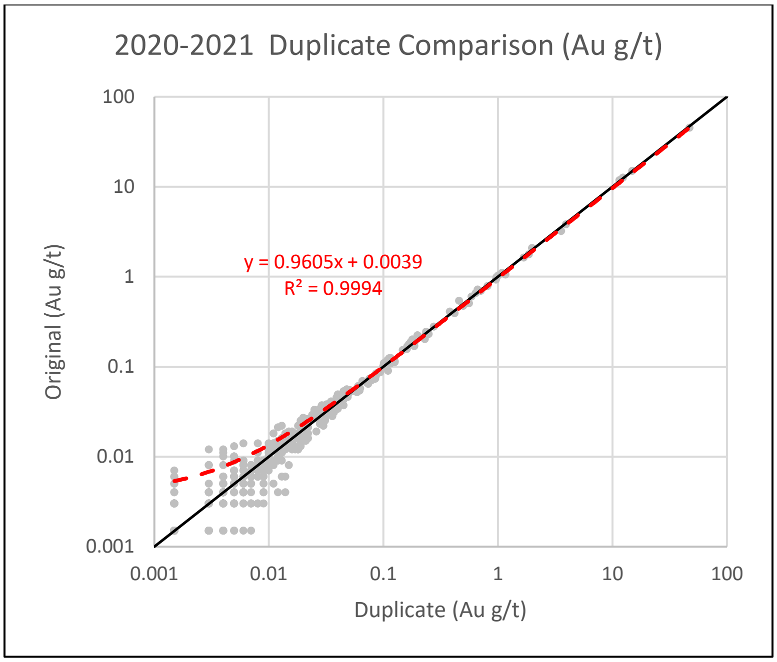

Figure 6-20 Analysis of Duplicates

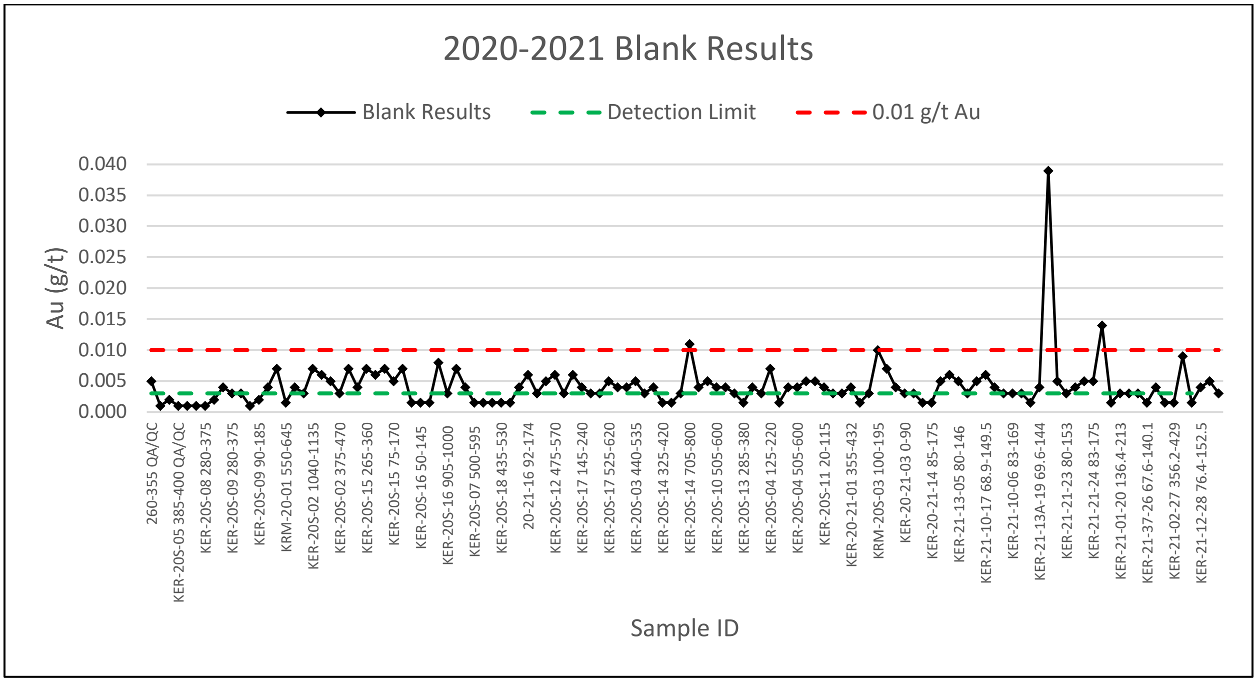

Figure 6-21 Analysis of Blank Results

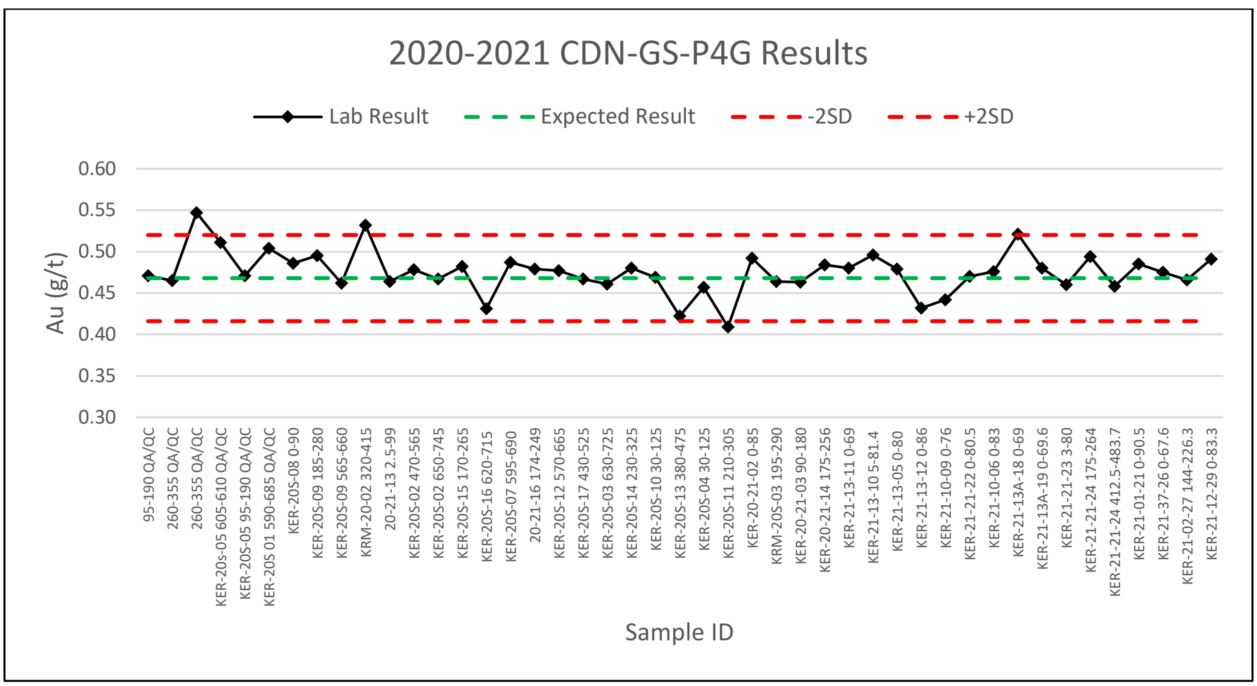

Figure 6-22 Analysis of CDN-GS-P4

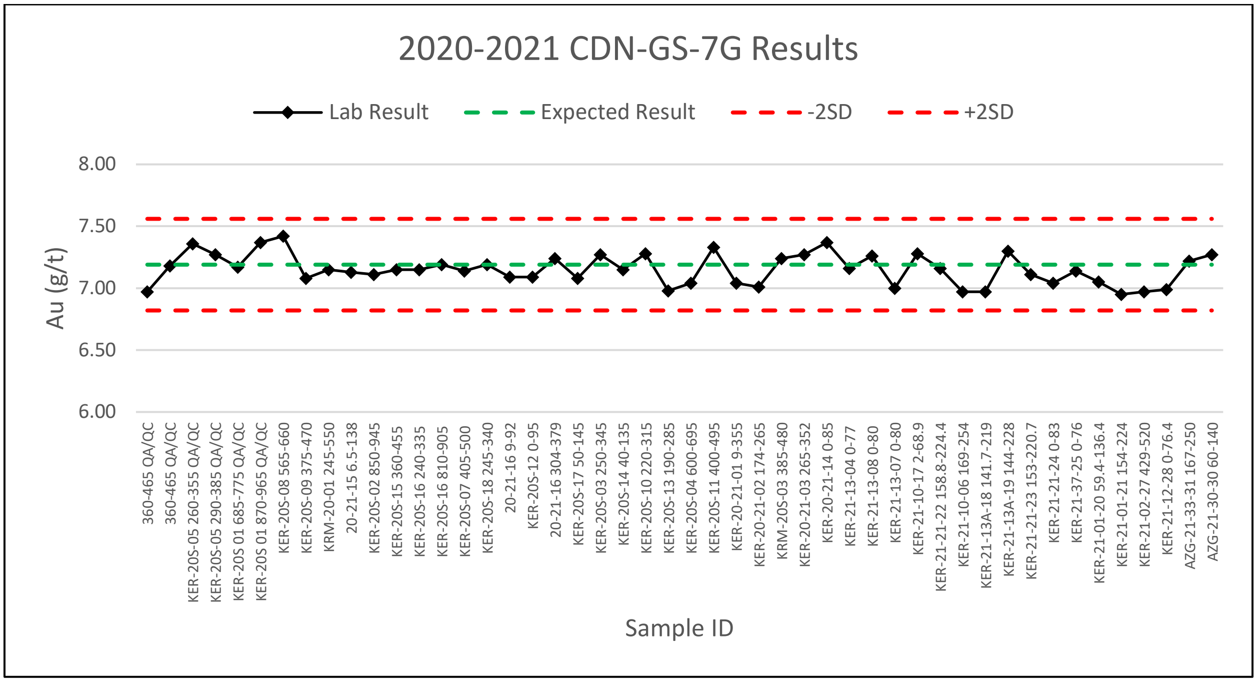

Figure 6-23 Analysis of CDN-GS-7G

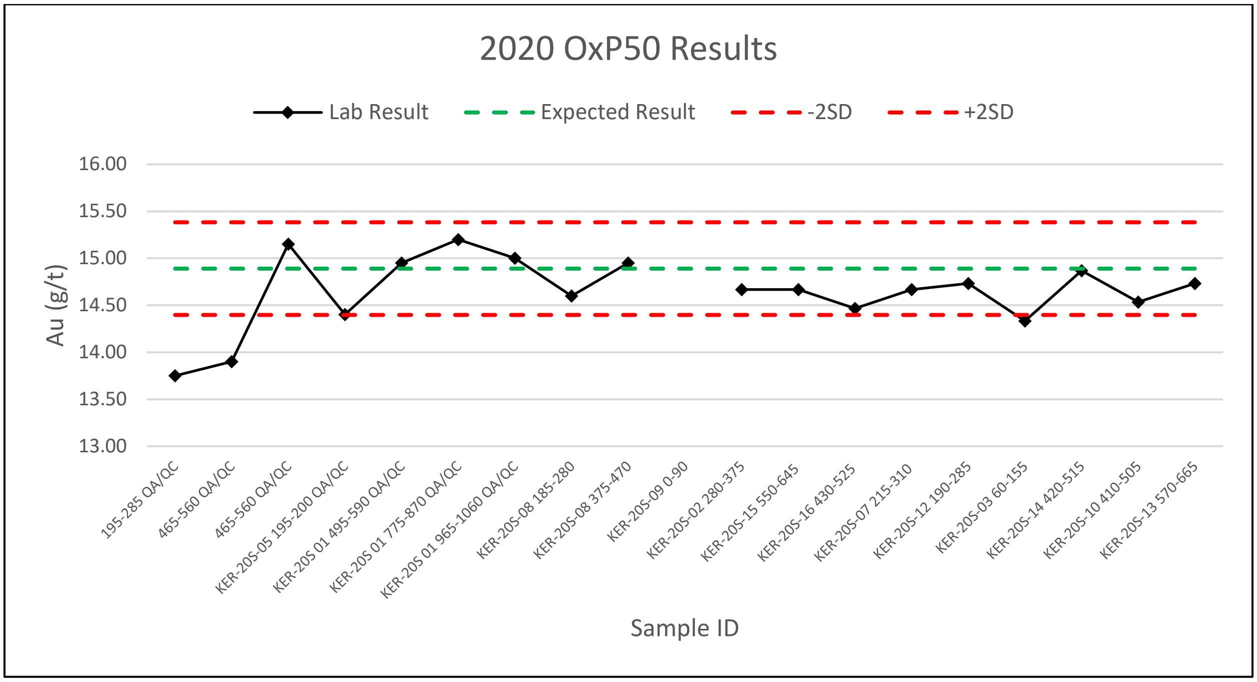

Figure 6-24 Analysis of OxP50

Drillhole results are summarized in Table 6-18. Four drillholes targeted the C zone. AZG-20S-01 successfully intersected the gold mineralization where projected down dip. 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.

6.3.6.5 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.

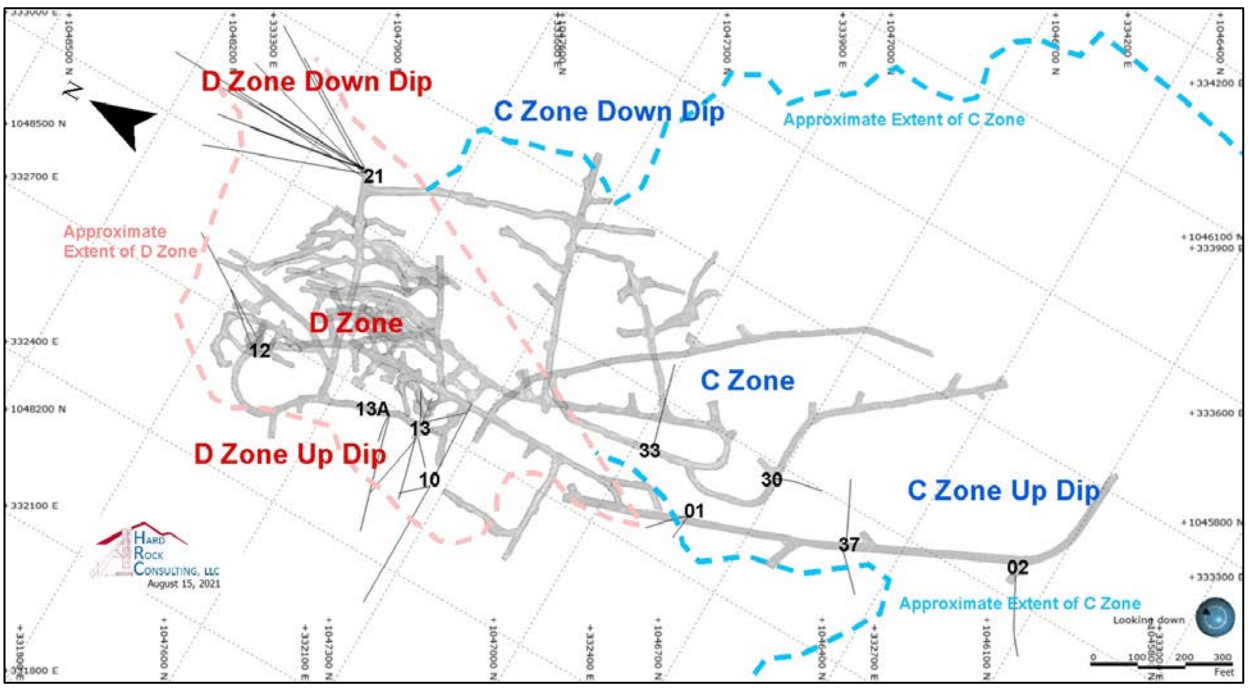

Figure 6-25 2020, 2021 Underground Drill Station Locations

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.

6.3.6.6 *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%.

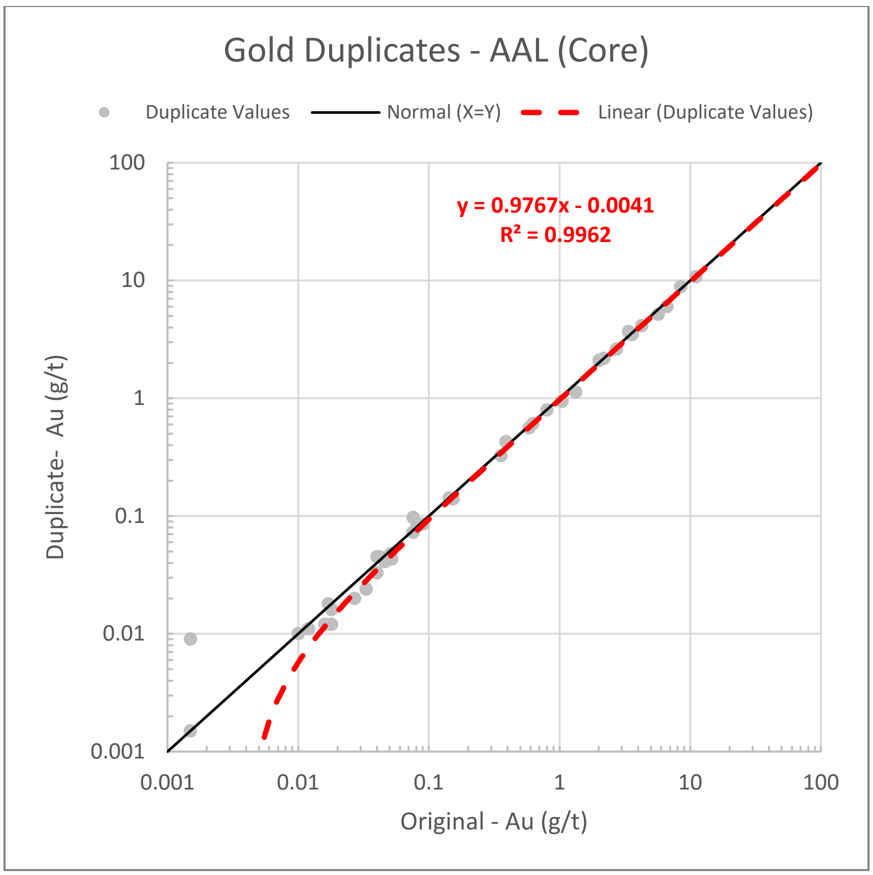

Figure 6-26 Gold Assay Results from Infill Core Duplicates

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%.

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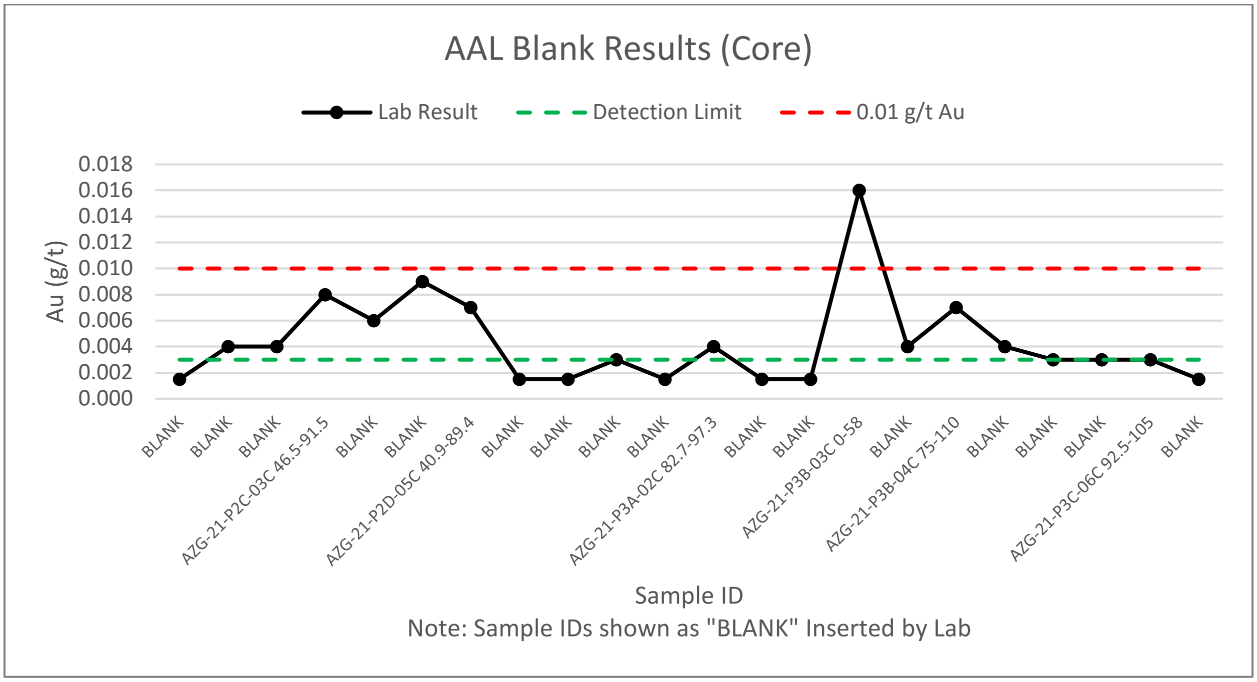

Figure 6-27 Gold Assay Results from Infill Core Blanks

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) were within acceptable limits and all seven results from CDN-GS-7G reported within acceptable limits as shown in Figure 6-29.

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.

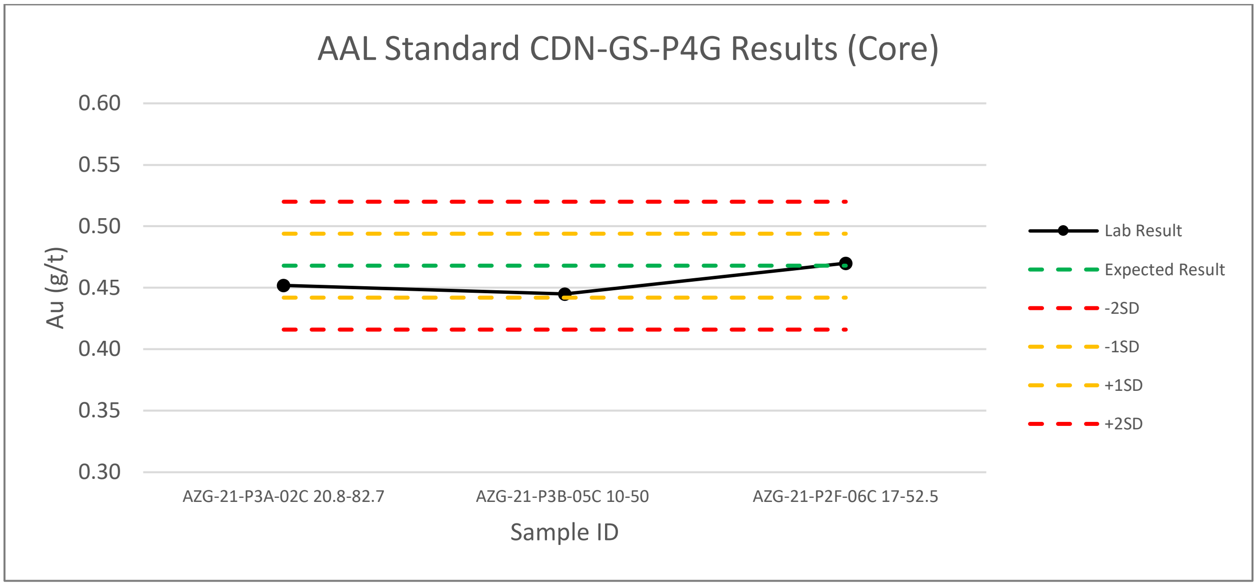

Figure 6-28 Gold Assay Results from CDN-GS-P4G

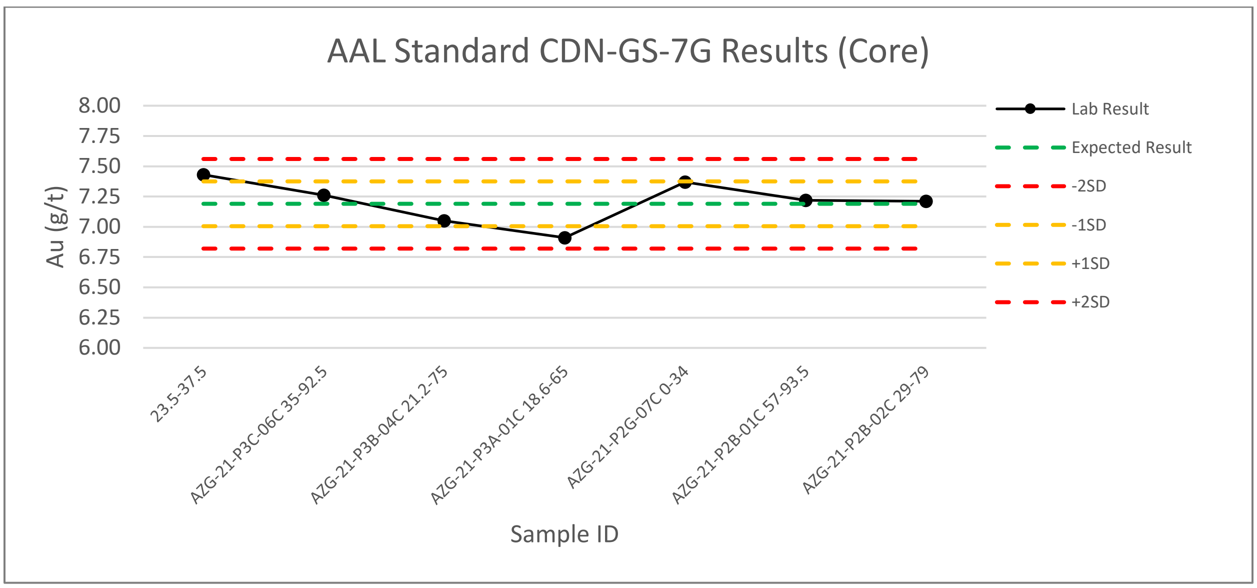

Figure 6-29 Gold Assay Results from CDN-GS-7G

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.

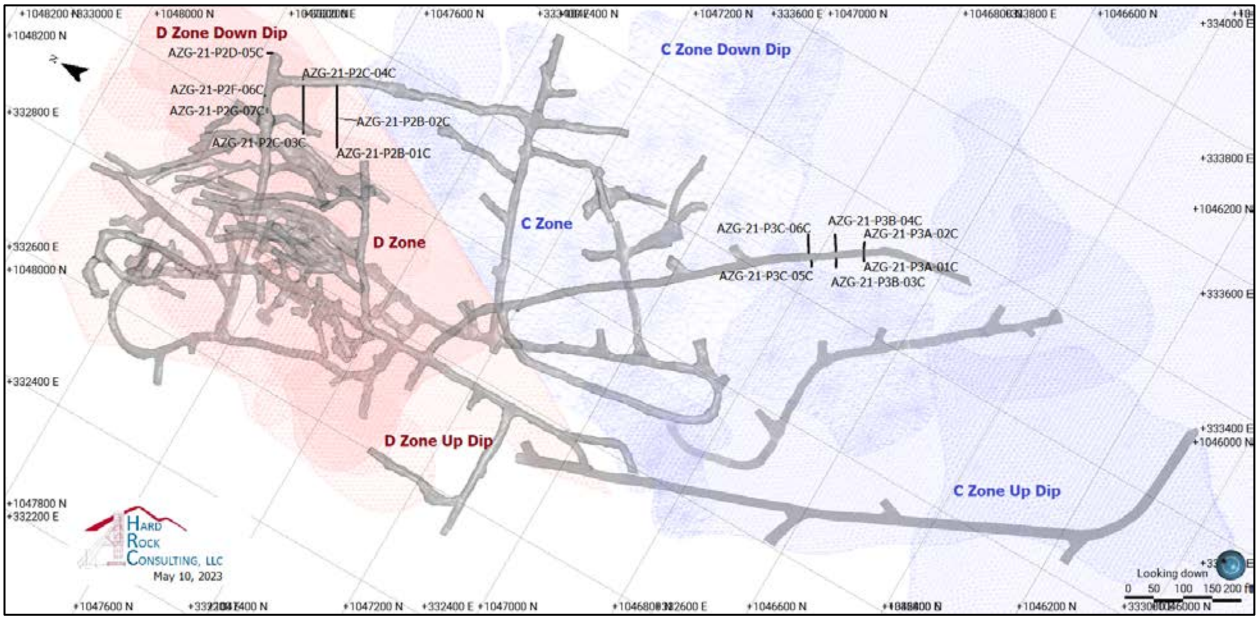

Figure 6-30 Location of 2021 Core Infill Drillholes

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.

6.4 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 tpd floatation 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.

6.5 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 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.

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). The QP has reviewed this information and available supporting data and documentation in detail, and finds the discussions and interpretations presented here 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.

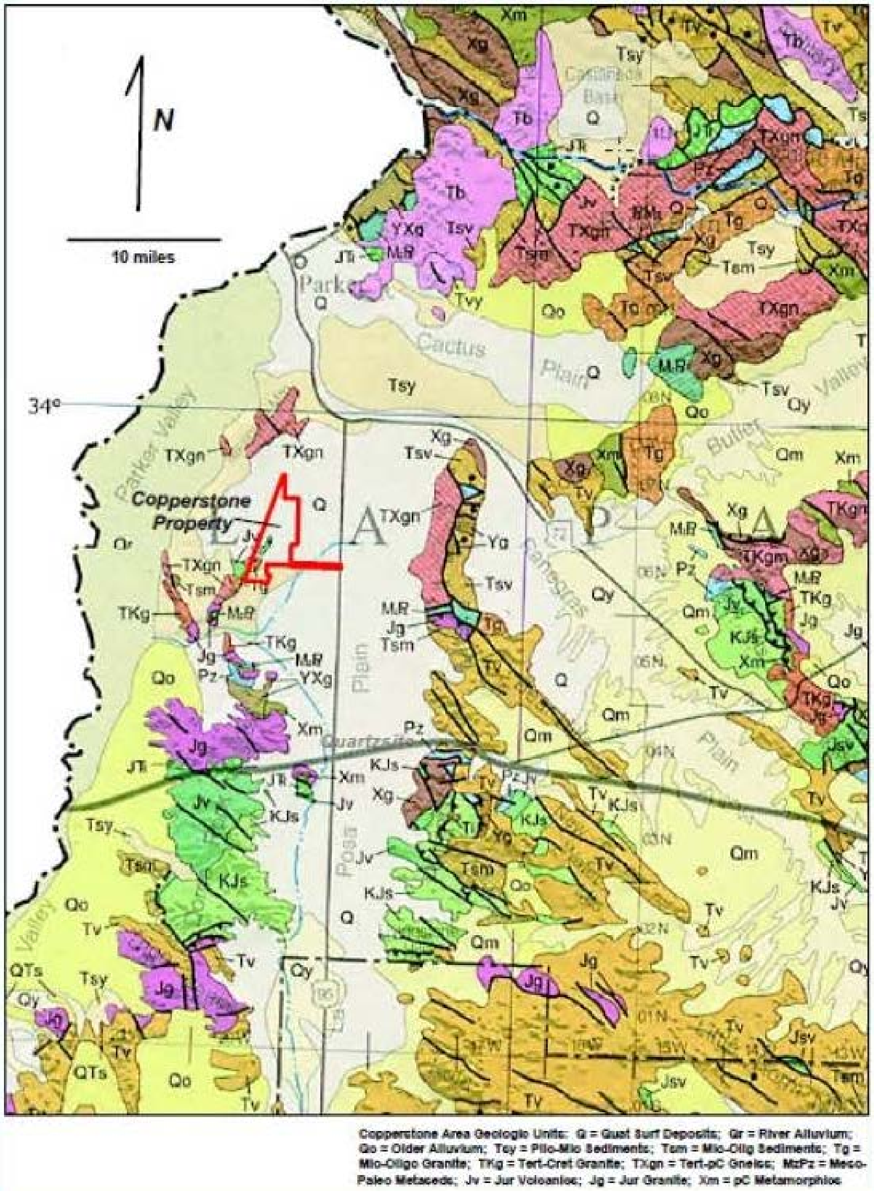

Figure 7-1 Regional Geologic Setting of the Copperstone Project (Pawlowski, 2005)

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 km^2 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. Brecciated granite along the plane of the low-angle detachment separates the lower plate mid-Tertiary granitic rocks from upper plate rocks, which consists (from bottom to top) of Triassic phyllites and metasediments, Jurassic quartz latite porphyry, and 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).

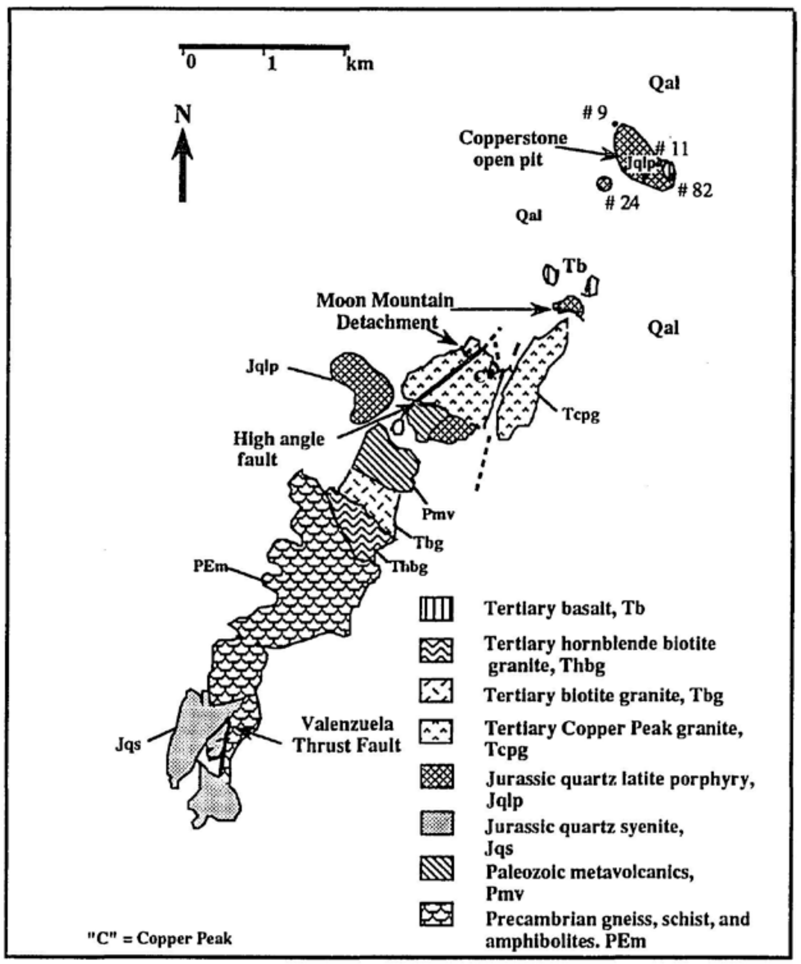

Figure 7-2 Geologic Map of the Copperstone Project (Salem, 1983)

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, clinochlore 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% SiO_2 , 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 at Copperstone occurs in the hanging wall of the Moon Mountain detachment fault, which has not been penetrated in drilling to date. Gold mineralization is largely restricted to the immediate vicinity of the Copperstone fault (also referred to as the Copperstone shear or the Copperstone structure), a moderately northeast-dipping, semi-planar zone of shear which is interpreted as a listric splay of the Moon Mountain detachment, and which has hosted the bulk of the gold historically produced from the Copperstone mine. 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 180 ft. in width with characteristic fault gouge, multi-phase breccia textures, shear fabric, and intense fracture sets across this width (MDA, 2000).

The Copperstone fault appears to be a brittle deformation feature situated within the extremely deformed upper plate volcanic sequence. The fault presents strong evidence of shearing, with schistose textures and conjugate sets of planar and curved faults indicated by 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 listric fault 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 structure has remained a dependable target for exploration and mining.

SGLD's current conceptual geologic model interprets the Copperstone structure as part of a detachment fault system related to regional mid-Miocene extension. More recently, Strickland et al. (2017) have recognized late Laramide detachment related to magmatism and the denudation of a Cretaceous subduction complex found across southern Arizona and California. Regardless of the age of the deformation, detachment faulting with an upper-plate-to-the-east sense of motion is presently considered the primary control/conduit for mineralization. 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.

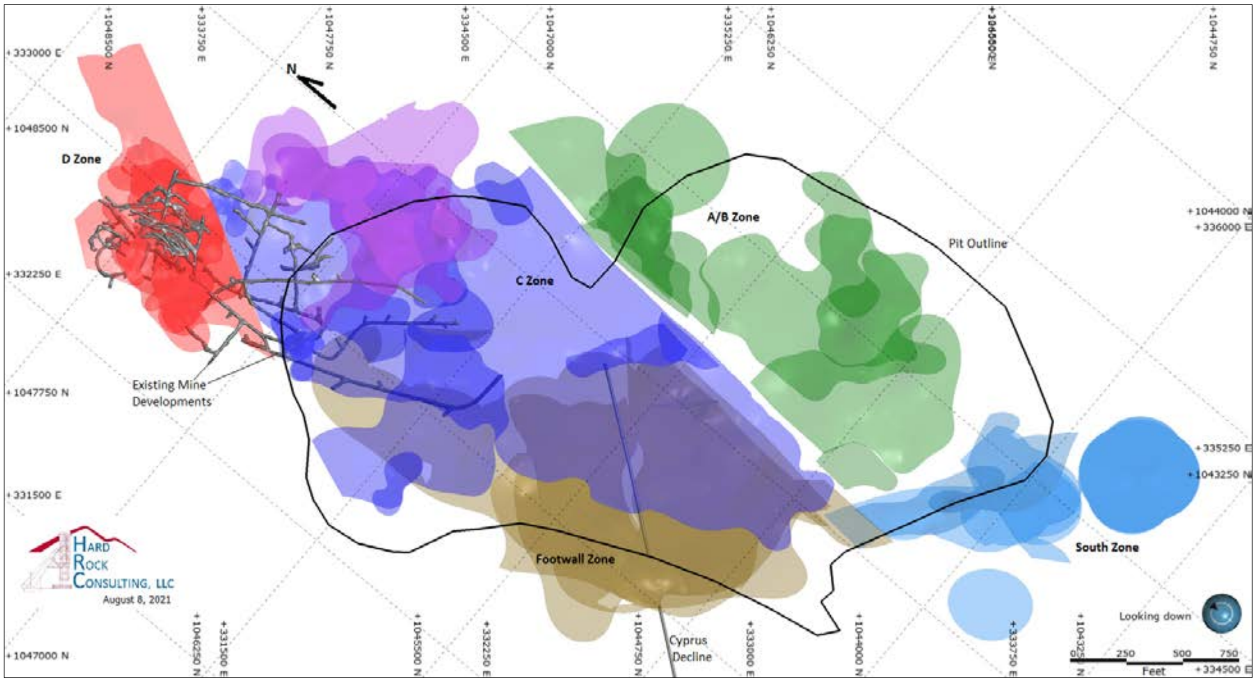

Figure 7-3 Mineralized Zones within the Copperstone Project Area

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 secondary structures within the zone of shear. 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 listric fault 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 the 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. 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.

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. Predominate 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 domanial basis by the recognized zones.

8. DEPOSIT TYPES

The Copperstone deposit is presently best described as a mid-Tertiary, detachment-fault-related gold deposit. Detachment faults are low-angle (up to 30°) normal faults of regional extent that have accommodated significant regional extension by upward movement of the footwall (lower-plate) producing horizontal displacements on the order of tens of kilometers. Common features of these faults are supracrustal rocks in the upper-plate on top of lower-plate rocks that were once at middle and lower crustal depths, mylonitization in lower-plate rocks that are cut by the brittle detachment fault, and listric and planar normal faults bounding half-graben basins in the upper plate (Davis and Lister, 1988).

The term 'detachment-fault-related' intentionally implies that mineralization is strongly controlled by detachment-fault structures, but also that it is apparently related to the formation of detachment faults themselves (Roddy et al., 1988). Early chloritic alteration and associated sulfide mineralization appears to result from retrograde metamorphism as hot lower-plate rocks are brought up to shallower depths. Potassium feldspar alteration and oxide mineralization appear to be related to the upward circulation of saline brines derived from syntectonic basins along the detachment fault into more steeply dipping upper-plate normal fault (Long, 1992). This fluid movement may have been driven by heat derived either from lower-plate rocks or from syntectonic microdiorite to rhyolite intrusives (Reynolds and Lister, 1987).

Features of detachment-fault-related mineralization that distinguish it from other deposit types, as first presented by Long (1992), are listed below. Further details are available in Spencer and Welty (1986), Roddy et al. (1988), and Spencer and Reynolds (1989).

- Deposits are controlled by structures formed during detachment faulting. These include the low-angle, detachment-fault system, high-angle faults in the lower-plate just below the detachment fault, and low- to high-angle normal faults in the upper-plate.
- Deposits are often brecciated or deformed by movement along or above the detachment fault.
- Chlorite-epidote-calcite alteration occurs along and below the detachment fault. These altered zones sometimes contain base-metal sulfides and barite.
- There is massive potassium feldspar replacement of upper-plate rocks. This alteration appears to generally precede gold formation and is not always spatially associated with mineralization.
- Weak sericite-silica alteration of wall rock is sometimes present around barite-fluorite veins.
- Most mineralization consists of iron and copper oxides, principally specular to earthy hematite and chrysocolla. Common gangue minerals are chalcedonic to amethystine quartz, ferrous to manganiferous calcite, barite, fluorite and manganese oxides. Distal barite-fluorite veins consist of variable proportions of barite, fluorite, and manganese oxides. Common gangue minerals are quartz and manganiferous calcite.
- Fluid inclusions have moderate homogenization temperatures (150 to 350 °C) and salinities (10 to 23 equivalent weight percent NaCl), compatible with precipitation from connate brines. Fluid inclusions from barite-fluorite veins have lower homogenization temperatures (90 to 200 °C) and are somewhat less saline (6 to 20 equivalent weight percent NaCl), compatible with precipitation from variably cooled and diluted connate brines.

- Host rocks are enriched in Cu. Pb, Zn, Au, Ag, and Ba and are depleted in Mn, Sr, Ni, and Rb. Elements characteristic of epithermal environments, such as As, Sb, Hg, and Tl, occur in very low, background-level concentrations.

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).

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% NaCI equivalent, ending with 190°C and 25.5 wt% NaCI equivalent (Salem, 1993).

9. EXPLORATION

9.1 Exploration Conducted on behalf of SGLD

The QP is not aware of any exploration activity, other than drilling, that has been carried out by or on behalf of SGLD.

10. DRILLING

10.1 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 10-1 shows the location of the RC infill drilling with station IDs labeled.

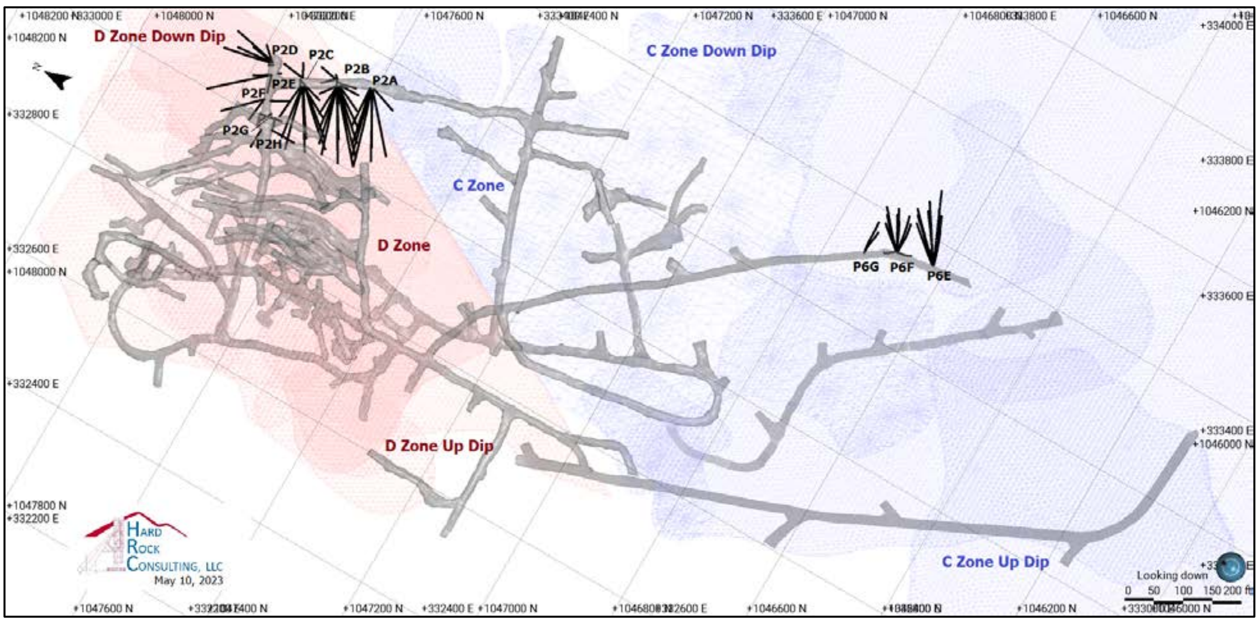

Figure 10-1 Location of 2021 RC Infill Drillholes

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 10-1. A more detailed view of the D zone drilling with assays filtered to greater than 1 g/t is shown in Figure 10-2.

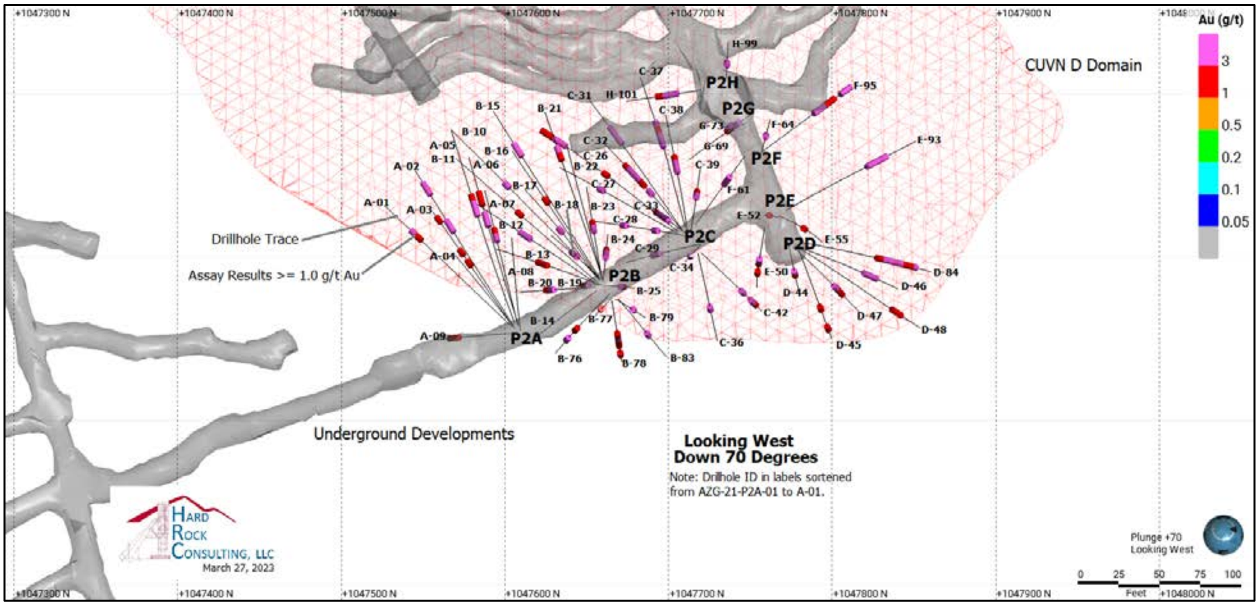

Figure 10-2 D Zone RC Infill Drilling

Table 10-1 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
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

Hole ID	From (ft)	To (ft)	Interval Length		Gold		True Thickness*	
			ft	m	oz/ton	g/t	ft	m
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.

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 10-2. A more detailed view of the C zone drilling is shown in Figure 10-3.

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.

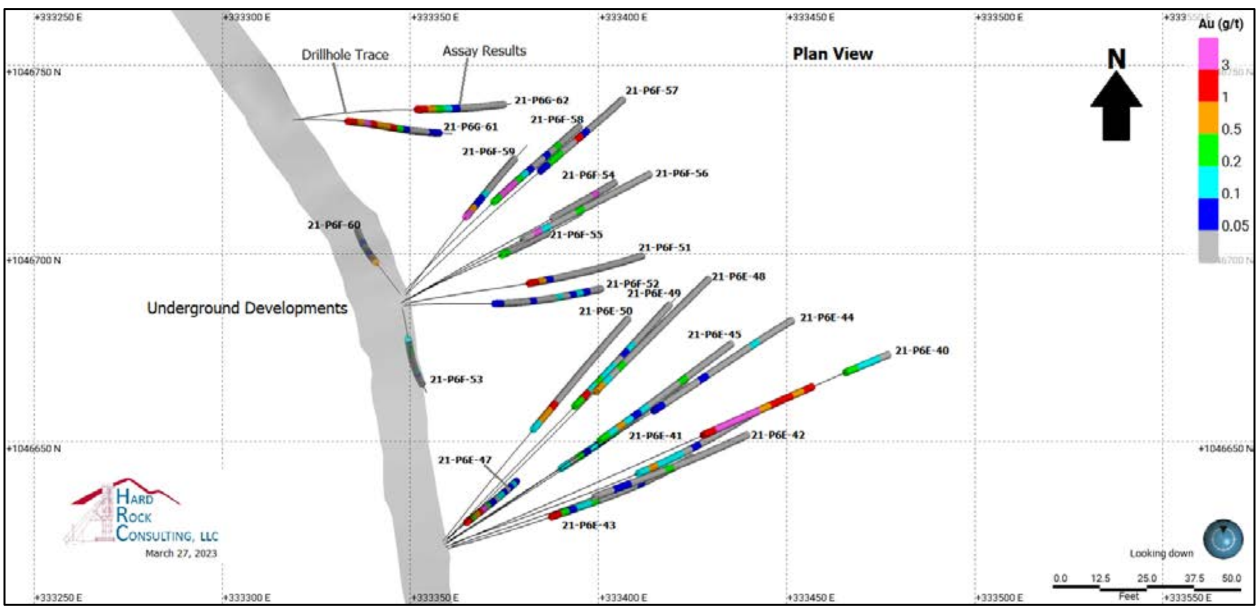

Figure 10-3 C Zone RC Infill Drilling

Table 10-2 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.

Figures 10-4 through 10-6 show drillhole collar locations for all drilling programs except Newmont and Southwest Silver. Figure 10-4 shows surface drillhole collar locations for the entire property. Figure 10-5 shows surface drillhole collar locations for drilling in the area of the open pit. Figure 10-6 shows underground drillhole collar locations. A complete list of drillhole collar locations, orientations, type, year, and operator are available in Appendix B.

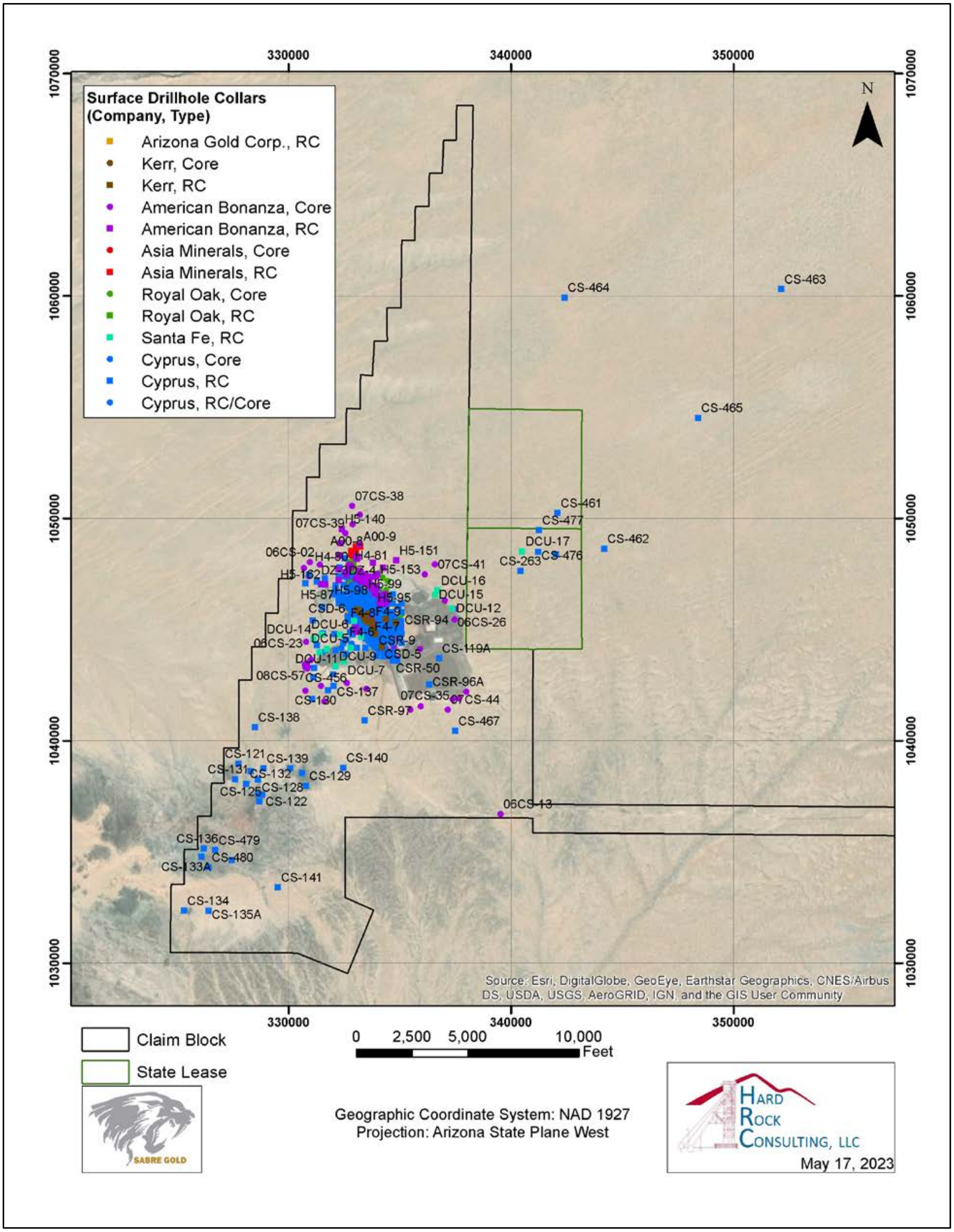

Figure 10-4 Plan View of Surface Drilling for the Property

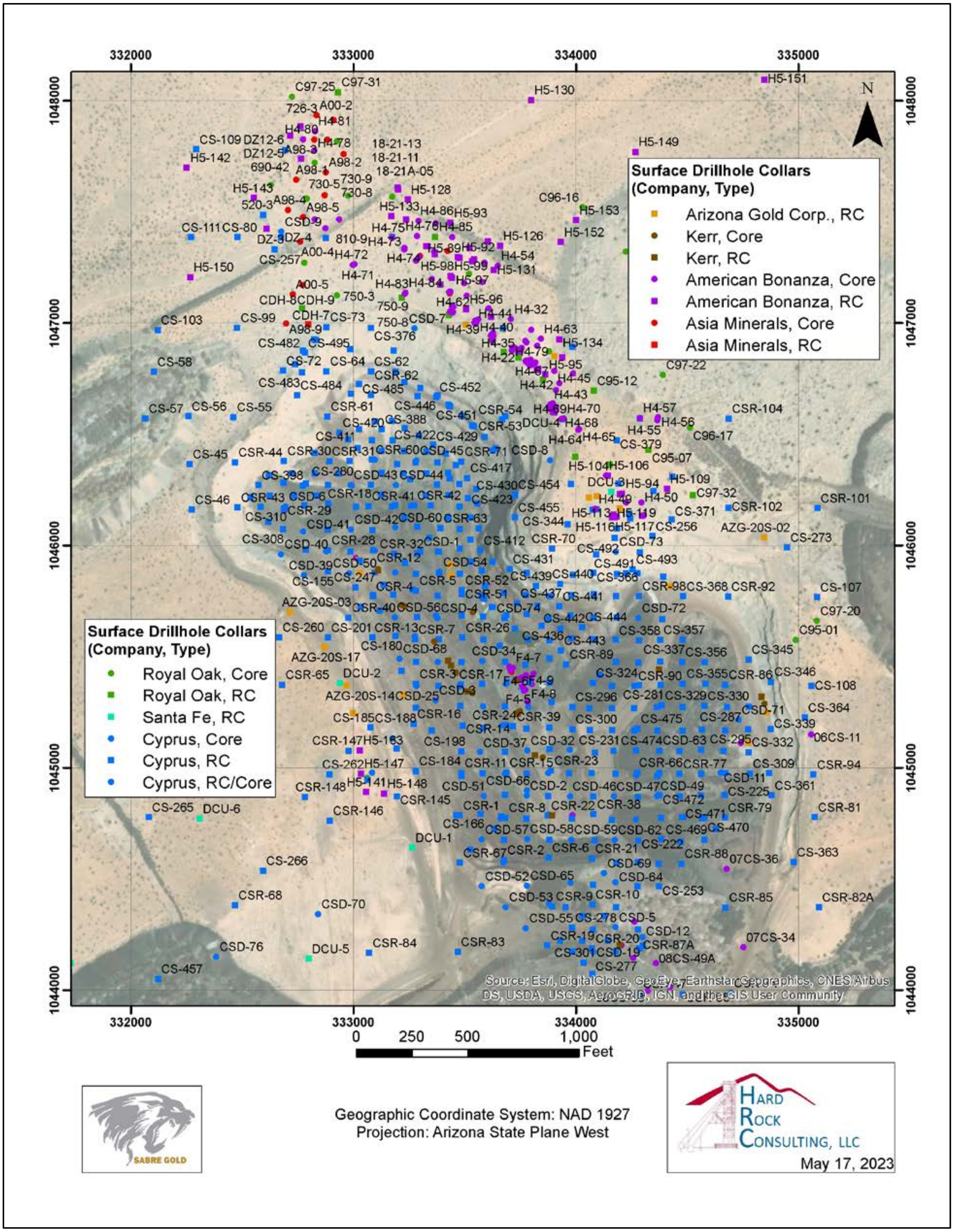

Figure 10-5 Surface Drillhole Locations in the Area of the Open Pit

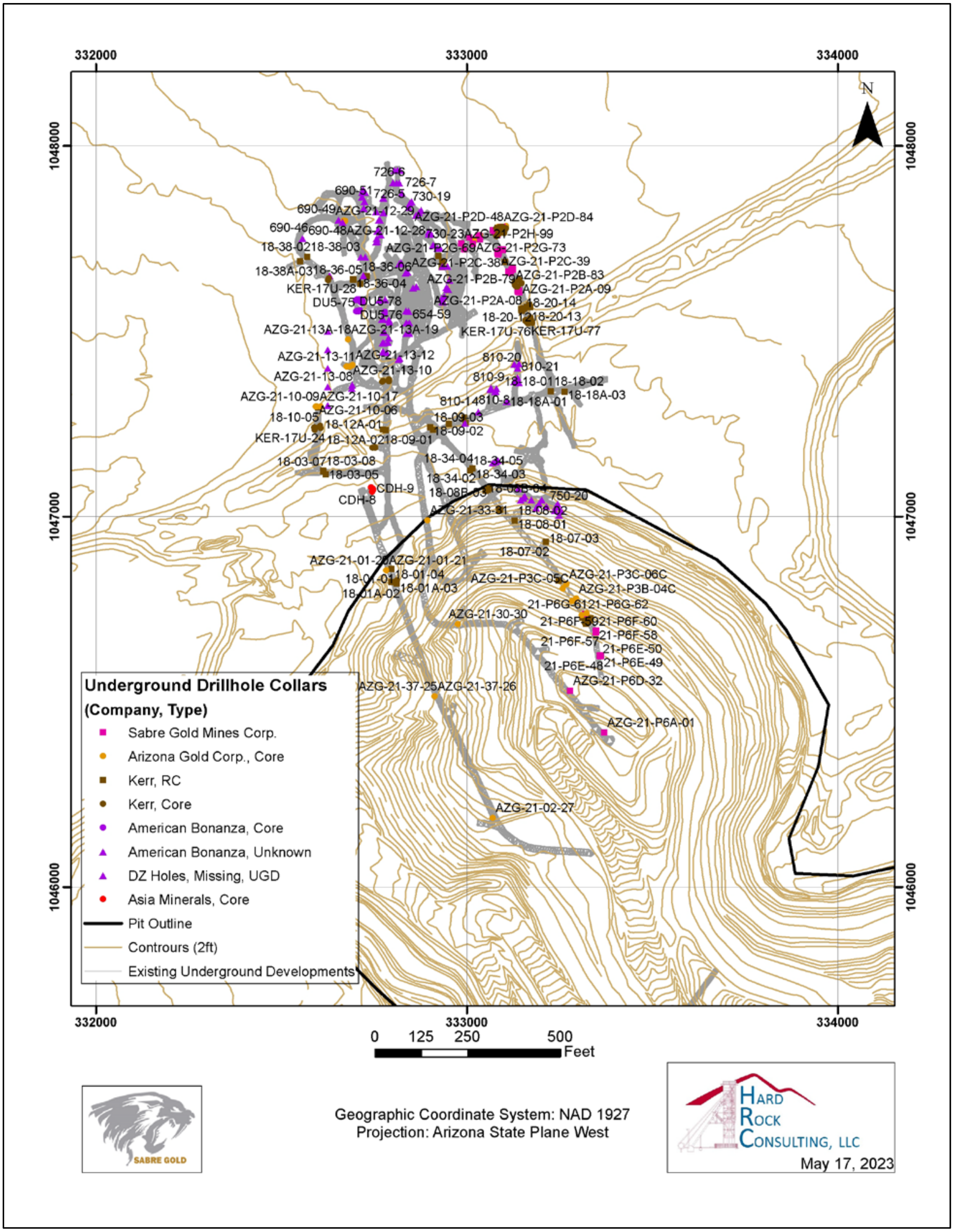

Figure 10-6 Underground Drillhole Locations

11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 SGLD Sample Handling, Analysis, and Security

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 11-1 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.

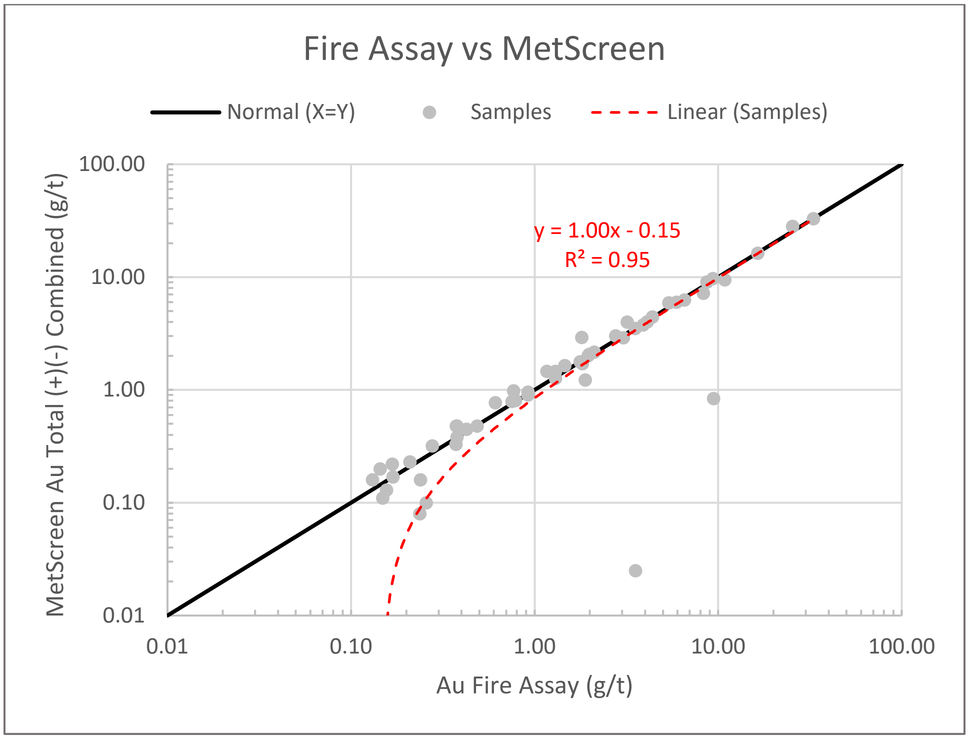

Figure 11-1 Metallic Screen Assay Comparison - 2021

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 11-2 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 11-3 and 11-4 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%.

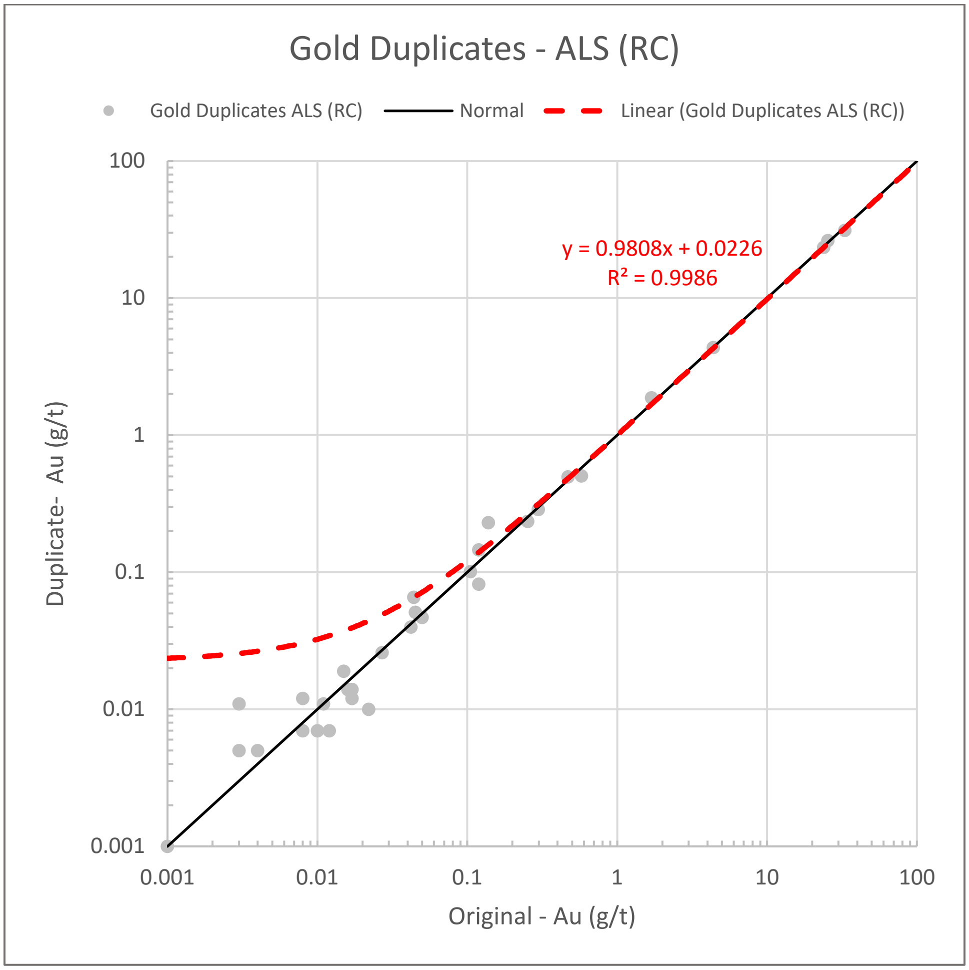

Figure 11-2 Gold by Fire Assay Results from RC Duplicates

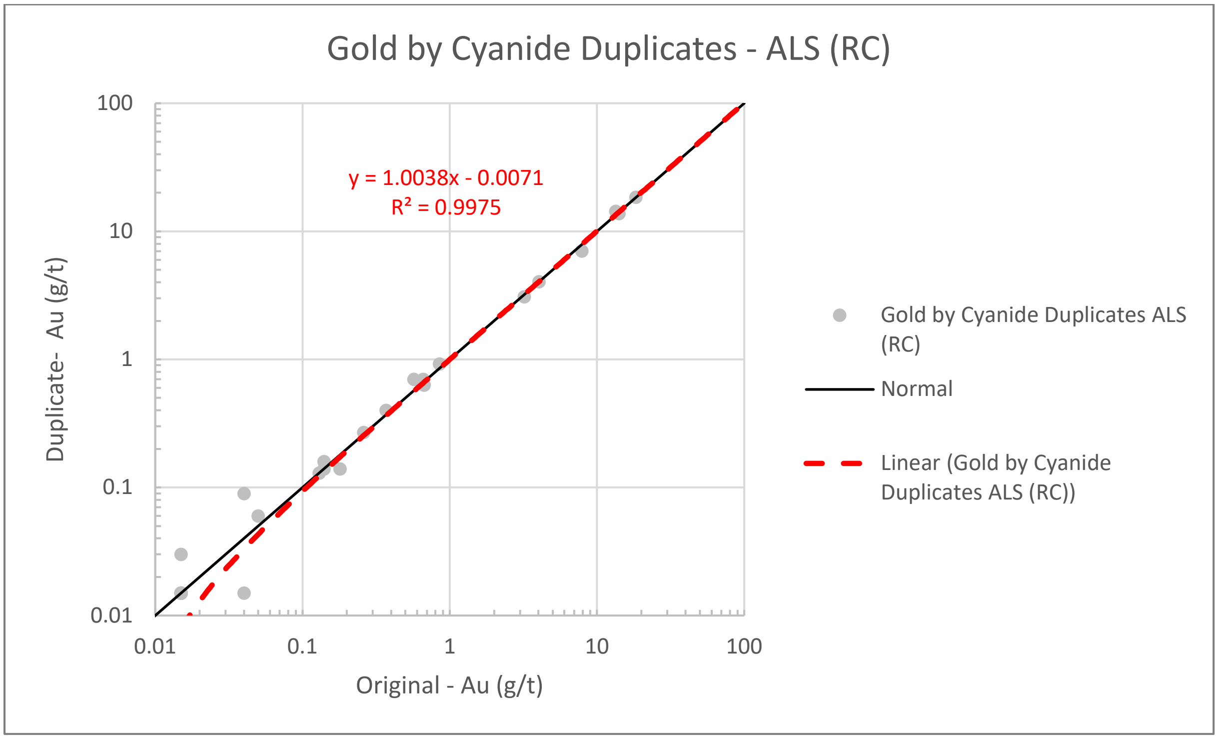

Figure 11-3 Gold by Cyanide Leach Assay Results from RC Duplicates

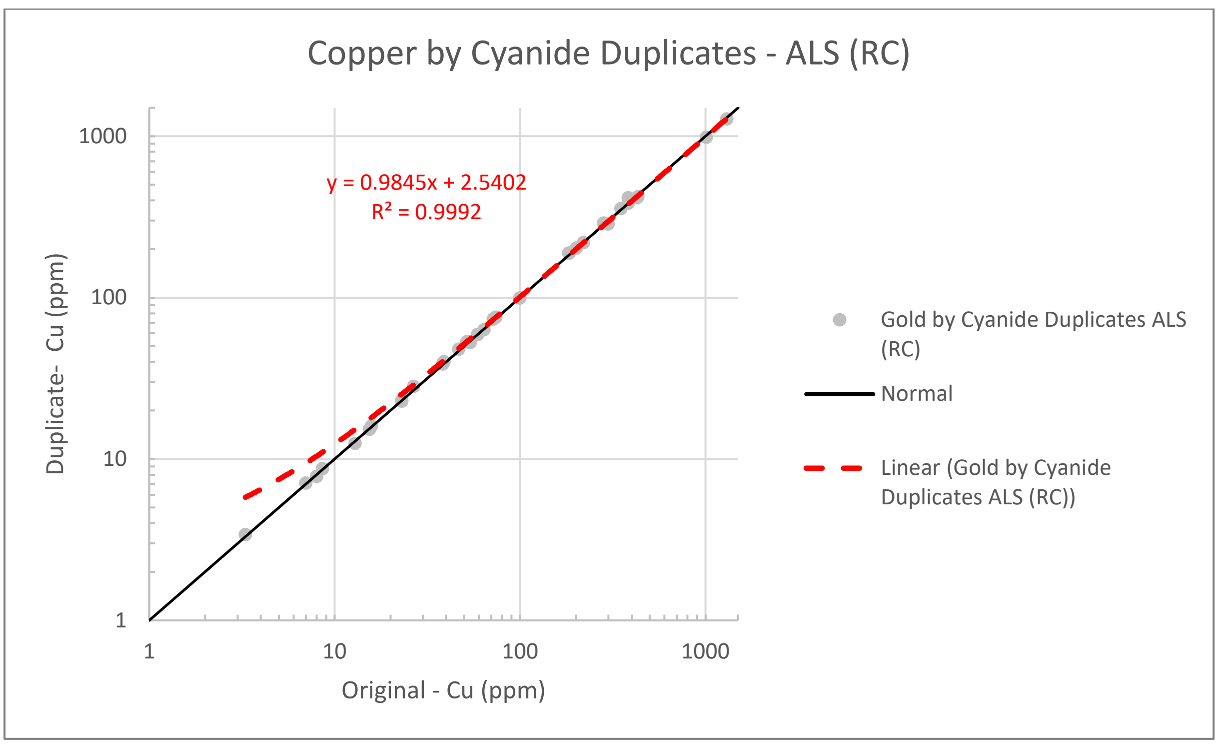

Figure 11-4 Copper by Cyanide Leach Assay Results from RC Duplicates

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 11-5. 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%.

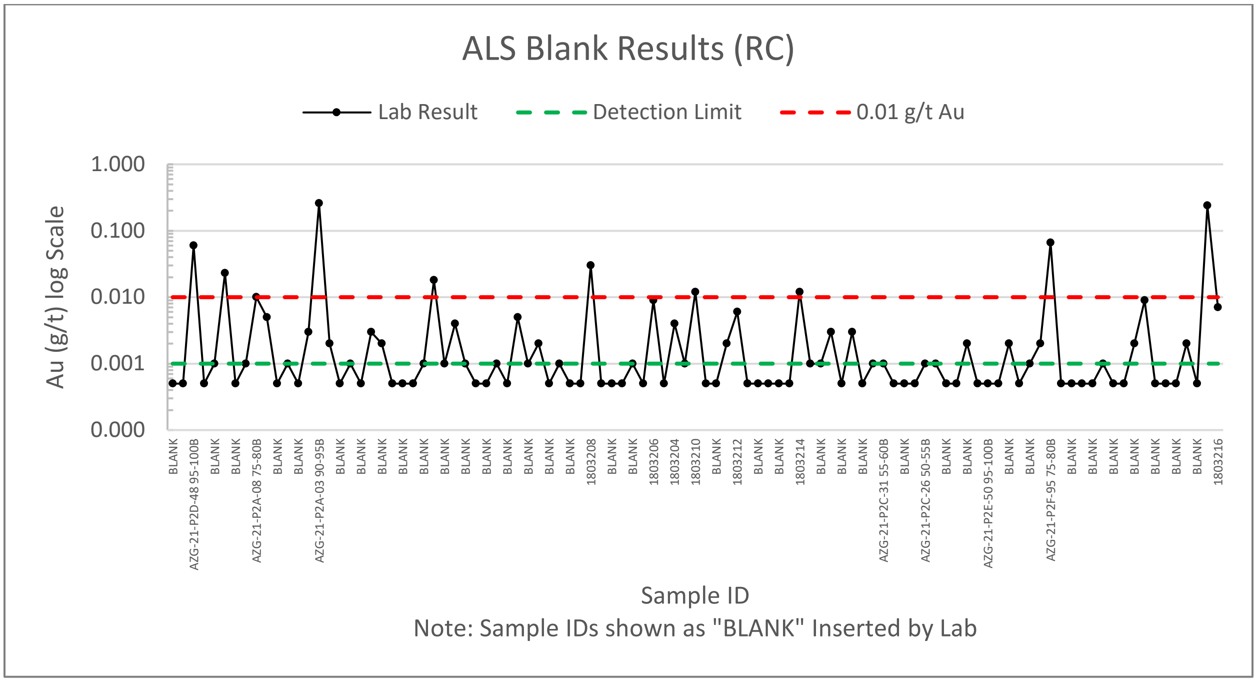

Figure 11-5 Gold Assay Results from RC Blanks

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 11-6 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 11-7 shows the eleven results from that standard, and all but one result reported within acceptable limits.

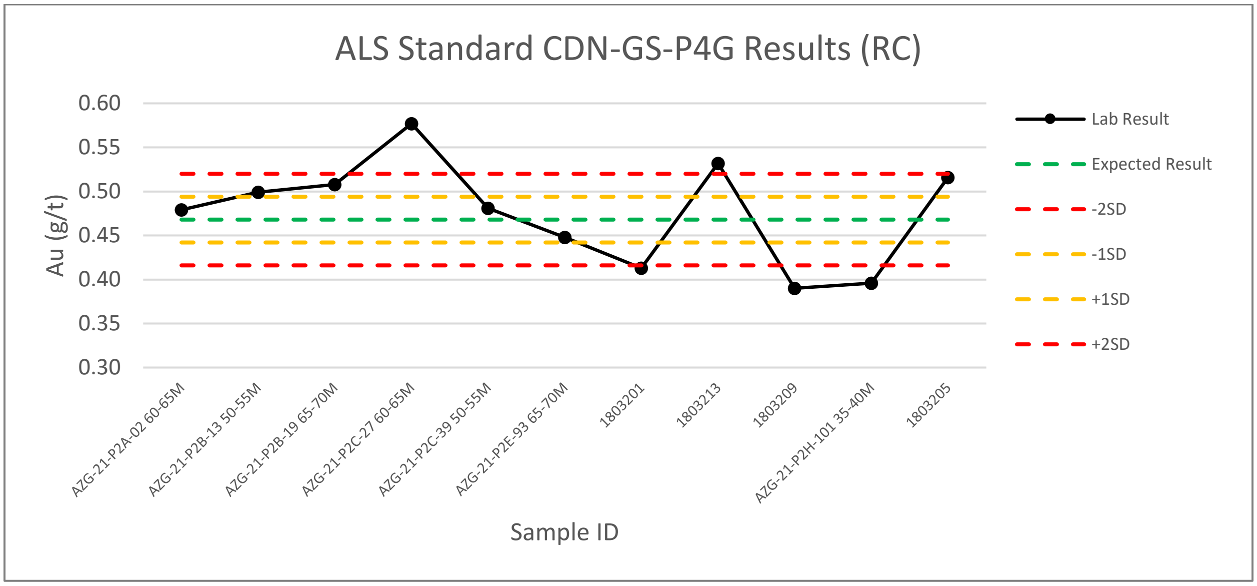

Figure 11-6 Gold Assay Results from CDN-GS-P4G

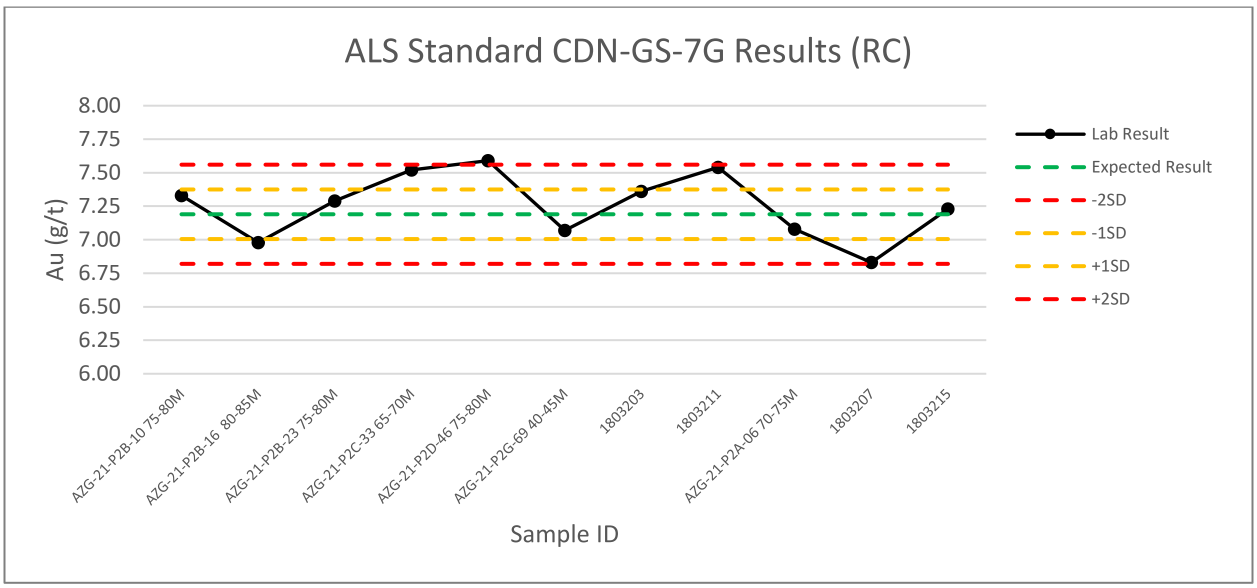

Figure 11-7 Gold Assay Results from CDN-GS-7G

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.

12. DATA VERIFICATION

Data verification efforts carried out by HRC include:

- Discussions with AZG (Kerr) and SGLD personnel;
- Personal investigation of the Project and field office;
- Manual and mechanical auditing of the drillhole database received from previously from AZG and subsequently from SGLD;
- A limited audit of exploration work conducted;
- Review and evaluation of additional information obtained from historical reports and internal company reports.

12.1 Site Investigation

HRC representative and QP J.J. Brown, P.G., SME-RM, conducted an on-site inspection of the Copperstone Project on October 31 through November 2, 2017. Ms. Brown spent three full days at Project site accompanied by Kerr Mines Director of Exploration and Geology Brad Atkinson. While on site, Ms. Brown conducted general site and geologic field reconnaissance, including inspection of on-site facilities, examination of surface and underground bedrock exposures, and ground-truthing of reported drill collar locations. Ms. Brown also examined select core intervals from historic and recent drilling and reviewed with Kerr geology staff the conceptual geologic model, data entry and document management protocols, and drilling and sampling procedures and the associated QA/QC methods presently employed.

Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretations regarding local geology and mineralization (as described in Section 7 of this report).

Specific core intervals from a variety of drillholes (both historic and modern) were selected for visual inspection based on a preliminary review of the drillhole logs and associated assay values. The core intervals were selected prior to the site visit based on a preliminary review of the drillhole logs. Not all core intervals requested were available, and efforts to locate those boxes were unsuccessful subsequent to the site visit. While the lack of select core intervals represents a limitation to the data validation effort, the QP considers this limitation negligible based on the cumulative positive results of all other data validation measures. In all cases, the core samples that were available for inspection accurately reflect the lithologies recorded on the logs, and the degree of visible alteration and evidence of mineralization observed is consistent with the grade range indicated by reported assay values.

Jeff Choquette, P.E., of HRC conducted an on-site inspection of the Copperstone Project on July 10 and 11, 2018. While on site, Mr. Choquette conducted general site investigations, including inspection of the office facilities, crushing plant area and the mine laboratory. Mr. Choquette also toured the open pit and underground workings to become familiar with the geology and the general conditions of the underground workings. Although there are a few areas where instabilities of the roof have occurred underground, the majority of the workings are in good shape. Dewatering and ventilation systems are adequately maintained and continue to operate for care and maintenance.

Richard A. Schwering, P.G., SME-RM, of HRC conducted a follow-up site visit of the Copperstone mine property between March 1st and March 5th, 2021. During the site visit, Mr. Schwering reviewed the following information with Mr. R. Mike Smith and Mr. Andrew Eubanks of AZG:

- Core and RC sample preparation during the 2020/2021 drilling campaign;
- Review of selected core and RC chips;
- Site Tour of underground, open pit; and
- Review of records, particularly downhole surveys, for the project.

Andrew Eubanks verbally described the logging and sampling procedure for the core and RC chips. Overall, the procedure is comparable to industry standards and best practices as described in Section 11. Mr. Eubanks also showed the area where core is cut, and verbally went over the cutting procedure which also conformed to industry standards and best practices. Documentation of the core logging and RC sampling procedure was provided to Mr. Schwering and confirmed the verbal procedure described by Mr. Eubanks.

Review of core drillholes reveal the complexity of mineralization within the D zone. Visual indicators are reliable in defining where gold can occur, but not necessarily grade. Review of the drillhole logs shows the geologists on site are accurately describing lithology, alteration, and mineralization in a format that could be used for modeling. Review of the RC chips showed similar characteristics to the core. During the tour of the underground workings, Mr. Schwering observed several detachment fault related structures, sub-horizontal shallow dipping, indicating the overall trend of the mineralized domains in the model are oriented properly. The exposure of the CUVND domain southwest of Station 21, was particularly useful for ground truthing the model.

Significant progress on the organization of assay and drilling documents had been accomplished by AZG during the time between HRC's site visits. As a result, Mr. Schwering was able to complete a detailed audit of the underground drilling collars and down-hole surveys. Mr. Schwering did find some underground drillholes in the database were oriented below horizontal when the surveys and drillhole logs indicate they were actually angled above horizontal. Mr. Schwering also found some downhole surveys did not correctly apply the correction for magnetic declination. Additionally, downhole surveys from 59 surface drillholes were found to not be recorded in the database. The database used for the mineral resource estimate was corrected to reflect the records from the site visit.

12.2 Database Audit

The following tasks were completed as part of the QP's database audit:

- Mechanical audit of the database;
- Validation of the geologic information as compared to the paper logs; and
- Validation of the assay values contained in the exploration database as compared to assay certificates from records found on file in formally AZG's Copperstone Project field office.

The database provided to HRC contained the rock-type, alteration, geotechnical, specific gravity, assay, drillhole collar, and survey data.

12.2.1 Mechanical Audit

A mechanical audit of the drillhole database for drilling conducted prior to 2017 was completed using Leapfrog Geo software version 4.2.3 in 2017. The database was checked for missing values, duplicate records, interval overlap errors, from-to data exceeding maximum collar depth, and special (i.e. non-numeric or less than zero) values. All mechanical audit errors were reviewed with Kerr staff and resolved prior to modeling and calculation of the mineral resource estimate.

The mechanical audit identified eight drillholes (Table 12-1) with duplicate collar coordinates and surveys which could not be resolved and were not included in the mineral resource estimate. Two of those drillholes, CS238A and CS-291, are outside the model extents and did not impact the modeling or mineral resource estimate. The remaining eight "DZ" drillholes are located in the D zone, but only one 2-ft interval had significant gold grade. Tables 12-2 through 12-7 summarize the count and error types identified by the mechanical audit for the survey, assay, lithology, alteration, geotechnical, and specific gravity tables respectively.

Table 12-1 Drillholes Excluded from the Model by the Mechanical Audit

CS-238A	DZ12-2
CS-291	DZ12-3
DZ-3	DZ12-4
DZ-4	DZ12-5
DZ12-1	DZ12-6

Table 12-2 Issues within the Survey Table Identified by the Mechanical Audit.

Survey Table		
Issue	**Type**	**Count**
Duplicate collar and surveys	Warning	17
No surveys for collar	Warning	28
Wedge found (possible duplicate collar)	Warning	2

Table 12-3 Issues within the Assay Table Identified by the Mechanical Audit.

Assay Table		
Issue	**Type**	**Count**
To value exceeds max depth in collar table	Error	6
Interval overlaps an interval in a wedge hole	Warning	8
No samples for collar	Warning	83
Re-drilled hole has conflicting data in 'au oz/ton'	Warning	79

Table 12-4 Issues within the Lithology Table Identified by the Mechanical Audit.

Lithology Table		
Issue	**Type**	**Count**
From depth >= to depth	Error	10
Interval overlaps another interval	Error	23
To value exceeds max depth in collar table	Error	1
No samples for collar	Warning	324
Interval overlaps an interval in a wedge hole	Warning	3

Table 12-5 Issues within the Alteration Table Identified by the Mechanical Audit.

Alteration Table		
Issue	**Type**	**Count**
From depth >= to depth	Error	20
Interval overlaps another interval	Error	27
To value exceeds max depth in collar table	Error	1
No samples for collar	Warning	609

Table 12-6 Issues within the Geotechnical Table Identified by the Mechanical Audit.

Geotechnical Table		
Issue	**Type**	**Count**
From depth >= to depth	Error	20
Interval overlaps another interval	Error	27
To value exceeds max depth in collar table	Error	1
No samples for collar	Warning	605

Table 12-7 Issues within the Specific Gravity Table Identified by the Mechanical Audit.

Specific Gravity Table		
Issue	**Type**	**Count**
No samples for collar	Warning	1148

The QP mechanically audited interval information from drilling conducted in 2017 with Kerr staff. Issues identified by the mechanical audit were resolved.

Review of downhole surveys from the 2017 drilling campaign identified 25 readings as inaccurate. The inaccurate downhole surveys caused unrealistic drillhole deflections. The inaccurate downhole survey readings were ignored from the database. Based on survey readings above and below the ignored surveys in the drillhole, the impact of these readings on the geologic model and mineral resource estimate is negligible.

Gold assay samples below detection limit and un-sampled intervals were assigned values of 0.0001 oz/ton. Zero values are assumed to be un-mineralized and are set to 0.0001 oz/ton for the purpose of mineral resource estimation.

Mechanical audits of the 2019, 2020-2021 drillhole information, including core and RC infill drilling, was conducted as information was being supplied to the QP by Kerr, AZG, and SGLD staff. Any errors were reported and corrected prior to the completion of the current mineral resource estimate.

12.2.2 Manual Audit

The QP conducted a manual audit of the historical database using certificates, or the best information available to ensure the accuracy of gold values contained within the database from all operators prior to Kerr. The audit focused on drillholes with assays beyond the maximum pit extent and therefore, have the most significant impact on the resource. 3,436 assays (4.4%) were compared to the certificate values and found 22 assays (0.6%) had significant incorrect gold entries (differences greater than 0.0035 oz/ton). 123 assays (3.6%) had incorrect gold entries with a difference less than 0.0035 oz/ton gold. Other inconsistencies in the database include rounding errors, below detection limit and “zero” values, inconsistencies to how intervals with multiple gold assays were handled, and some interval errors. These issues are inherent and may be inevitable in large historic databases inherited through multiple operators.

The QP verified all the gold assay results against their certificates for the 2017 drillholes.

The QP compared all 3,353 of the 2019 gold assays to certificates and found eight assays (0.2%) were incorrectly entered into the database and corrected prior to the estimation of mineral resources.

The assay certificates, and digital drillhole logs from the 2020 and 2021 drilling, including core and RC infill drilling, were received by the QP to construct the drillhole database for those programs.

12.3 Adequacy of Data

Based on the results of the site investigations by the QP’s and data validation efforts, the QP considers SGLD’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. For historic data contained in the Project database, it is recommended that SGLD conduct scans of the certificates and all associated drillhole data to be stored in a digital format. The acquisition of database software and the entry of drillhole information as recorded, so systematic rules for how assay values are handled can be implemented is also recommended.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Several metallurgical studies have been completed to select an appropriate processing method and to optimize process operating parameters since 1986. Samples of various types of mineralized material from the four main mineralized zones at variable grades have been 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.

13.1 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.

13.2 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.

13.3 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

13.4 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 A98-2, 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 Floatation 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				

13.5 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 Floatation 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

13.6 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 work and abrasion 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 work and abrasion tests, thickening and pulp rheology tests.

13.6.1 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.

13.6.2 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 m^2/tonnes per day to 0.150 m^2/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 Work Index. Bond 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 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

					Grade				Parameter		Recovery %			CN Consumption
Zone	Lab	Date	Test	Wt Kg	Au oz/T	Au (Calc)	Ag oz/T	Cu % Total	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			
			Grav/Float						200M		91			
D zone+HW	McClelland	2005	Conc Cyanidation			11.23			200M	48/72	89/99			0.477

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

This testing was undertaken to review the previous metallurgical work completed on Copperstone deposit and verify previous testing, 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 and grind requirements and

optimize mass pulls. Other reagent schemes should be evaluated to maximize final recovery for flotation.

- *Grinding.* Grinding of the D zone and C zone samples identified a 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

13.8 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 tpd floatation 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 mineralogy represented primarily by 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	*	*
Leucoxene	*	*
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 Floatation 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 tpd of mineralized material to produce a product of P_{80} 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 P_{80} 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.

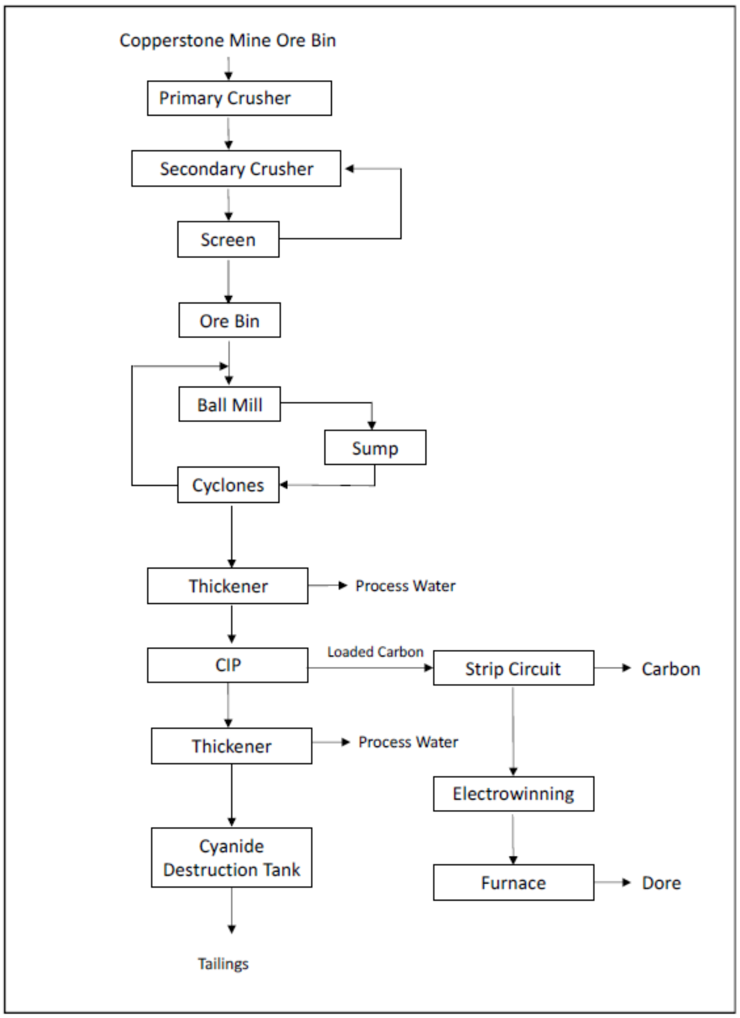

Figure 13-1 Whole Ore Leach Process Flow diagram

13.9 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 m^2/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/ft^2/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.

13.10 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.

13.10.1 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
$Cu_{CN\ Sol}$, ppm	72.47	491.6	1763	776.8
C_{Total}, %	0.02	0.61	0.35	1.06
$C_{Organic}$, %	<0.01	0.15	0.06	0.17
$C_{Inorganic}$, %	0.02	0.47	0.29	0.89
S_{Total}, %	0.07	0.06	0.08	0.28
$S_{Sulfide}$, %	<0.01	<0.01	<0.01	0.09
$S_{Sulfate}$, %	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

13.10.1.1 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/ton)	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

13.10.1.2 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.

13.10.2 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 P_{80} 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%.

13.10.3 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 P_{80} 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 (P_{80} 200 mesh) 7.0 g/L NaCN Maintained			BR 42 - CIL (P_{80} 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		

13.10.4 Thickening and Rheology Testing

Thickening test work was conducted with each composite sample at a particle size of P_{80} 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 m^2/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 m^2/tonne/day, while the other zones needed +/- 2.4 m^2/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 P_{80} 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.

13.11 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.

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 P_{80} 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%

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 (1.75 g/L NaCN)			BR44 (2.5 g/L NaCN)			BR45 (1.75 g/L NaCN)			BR46 (2.5 g/L NaCN)			BR47 (1.75 g/L NaCN)			BR48 (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	36.0	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	97.0	9.0	22.2	98.3	25.0	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	99.0	82.9	20.7	95.7	9.0	23.2	100.3	26.0	15.9	97.9	29.4	18.1
48 hr	97.7	23.1	22.6	98.0	32.7	25.3	97.2	81.7	21.4	97.3	9.2	22.6	98.6	25.5	16.4	98.2	30.0	17.5
Residue, ppm	0.13	1.5	3200	0.10	0.8	3200	0.14	0.1	2800	0.14	3.4	2700	0.08	1.0	2100	0.10	0.8	1900
Cal. Feed, ppm	5.48	2.0	4136	5.23	1.2	4285	5.06	0.4	3563	5.30	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			3.460			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	61.0	12.8	87.5	64.0	14.4
8 hr	94.1	51.4	20.3	94.6	65.6	21.7	94.0	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	22.0	98.1	64.1	16.2	98.8	67.0	17.6
24 hr	98.0	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	0.10	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	6.30	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 ≥ 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 m^2/tonne/day for Composite 1, 0.019-0.026 m^2/tonne/day for Composite 2, and 0.018-0.037 m^2/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

13.12 Conceptual Process Flowsheet

Based on the recent test work in 2019 – 2022 at RDI, a conceptual flowsheet presented in Figure 13-1 for the whole ore leach was modified to include direct cyanidation, CCD, and Merill Crowe (MC) for recovery of gold and copper. The gold-copper precipitate will be leached with sulfuric acid in a small tank followed by cementation to recover copper. The acid leach residue will consist of Au/Ag and will be smelted to produce gold doré bar.

The modified process flowsheet is presented in Figure 13-2

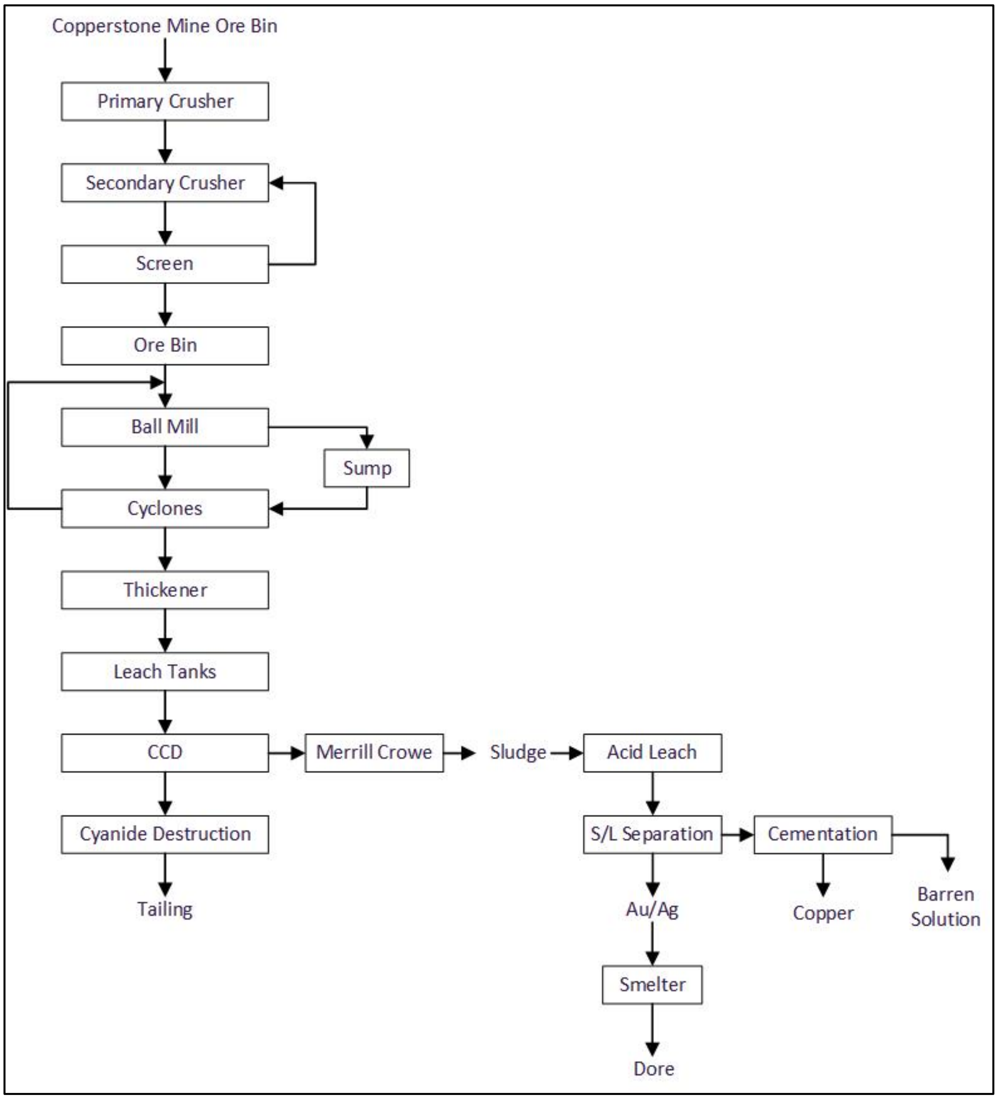

Figure 13-2 Conceptual Process Flowsheet

14. MINERAL RESOURCE ESTIMATE

Richard A. Schwering, PG, SME-RM, of Hard Rock Consulting, LLC, is responsible for the mineral resource estimate presented herein. Mr. Schwering is a Qualified Person as defined by NI 43-101 and is independent of Sabre Gold Mine Group. Mr. Schwering estimated the mineral resource for the Project based on drillhole data constrained by geologic boundaries with an Ordinary Krige ("OK") algorithm. Gold is the metal of interest at the Project. All units are U.S. Customary, and all costs are reported in U.S. Dollars unless otherwise specified. All coordinates are in presented using Arizona State Place West North American Datum 1927 U.S. feet. Elevation is in feet relative to mean sea level.

The geologic model, estimation domains, and resource estimate were all completed using Leapfrog Geo® software version 2022.1.2 (Leapfrog).

The mineral resource estimate reported herein was prepared in a manner consistent with the Committee of Mineral Reserves International Reporting Standards, of which both the Canadian Institute of Mining, Metallurgy and Petroleum and Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, are members. The mineral resources are classified as Measured, Indicated and Inferred in accordance with "CIM Definition Standards for Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014 and Best Practices Guidelines (November 29, 2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council. Classification of the resources reflects the relative confidence of the grade estimates.

14.1 Drillhole Database

In total, 1,118 drillholes totaling 524,762 ft were incorporated into the geologic model and mineral resource estimate. The mechanical audit identified eight drillholes (Table 14-1) with duplicate collar coordinates and surveys which were not included in the model. Two of those drillholes, CS-238A and CS-291, are outside the model extents and did not impact the modeling or mineral resource estimate. The remaining eight "DZ" drillholes are located in the D zone, but only one 2-ft interval had significant gold grade. Additionally, 154 underground drillholes (Table 14-2) drilled by American Bonanza in 2013 were excluded from the model because the drilling methods, and sampling procedures were not well documented. Three underground drillholes from 2017 and two underground drillholes from 2019 (KER-17U-23, KER-17U-30, KER-17U-69, 18-09-01, and 18-09-01B) were removed from the mineral resource estimate because they were not sampled. These drillholes are located in the D zone and are reasonable to exclude because they are not coincident with any of the modeled domains. An additional 85 RC drillholes completed by SGLD between October and December of 2021 were excluded from the mineral resource database. The holes were drilled from underground into targets of expected gold mineralization. Since these drillholes targeted and largely confirmed expected gold mineralization, they are not material to the mineral resource estimate, and are more appropriate for use in short term mine planning studies. Finally, 114 drillholes (Table 14-3) were outside the geologic model extent. Appendix B summarizes operator, year, type and location of the drillholes as well as their incorporation or exclusion from the mineral resource estimate.

Based on examination of drill logs and the spatial relationship of unsampled intervals, it was determined these areas were not sampled due to lack of mineralized indicators. As a result, missing intervals, zero and negative values, and non-numeric values were replaced with a below detection limit value of 0.0001 oz/ton for gold assays.

For the 2020-2021 drilling, gold assays with a gravimetric finish replaced gold assays by fire assay with ICP finish. Additionally, AZG-20S-05 and AZG-20S-06 had two assay submittals, one with a smaller sample weight, another with a larger sample weight. The assays from the larger sample weight were used for these drillholes unless no sample was present from the larger sample weight. In those thirteen cases, the assays from smaller sample weight were included in the database. Of those thirteen, only one sample was included in a domain.

Table 14-1 Drillholes Excluded from the Model by the Mechanical Audit

CS-238A	DZ12-2
CS-291	DZ12-3
DZ-3	DZ12-4
DZ-4	DZ12-5
DZ12-1	DZ12-6

Table 14-2 American Bonanza 2013 Underground Drilling Excluded from the Model

520-1	650W-15	654-60	690-29	690-7	730-20	750-5
520-10	650W-16	654-61	690-3	690-8	730-21	750-6
520-11	650W-17	654cc-1	690-31	726-1	730-22	750-7
520-12	650W-2	654cc-2	690-32	726-10	730-23	750-8
520-13	650W-3	654cc-3	690-33	726-11	730-3	750-9
520-14	650W-4	690-1	690-34	726-13	730-4	810-10
520-15	650W-5	690-10	690-35	726-14	730-5	810-12
520-16	650W-6	690-11	690-36	726-2	730-7	810-13
520-17	650W-8	690-12	690-38	726-3	730-8	810-14
520-2	650W-9	690-13	690-39	726-4	730-9	810-15
520-3	654-1	690-14	690-40	726-5	750-1	810-16
520-4	654-2	690-15	690-41	726-6	750-10	810-16a
520-5	654-50	690-19	690-42	726-7	750-11	810-17
520-6	654-51	690-2	690-43	730-11	750-12	810-18
520-7	654-52	690-20	690-44	730-13	750-13	810-19
520-8	654-53	690-21	690-45	730-14	750-14	810-2
520-9	654-54	690-22	690-46	730-15	750-17	810-20
650W-1	654-55	690-23	690-47	730-16	750-18	810-21
650W-10	654-56	690-24	690-48	730-17	750-2	810-3
650W-12	654-57	690-25	690-49	730-18	750-20	810-4
650W-13	654-58	690-26	690-50	730-19	750-3	810-8
650W-14	654-59	690-28	690-51	730-2	750-4	810-9

Table 14-3 Drillholes Outside the Geologic Model Extent

06CS-01	06CS-26	08CS-56	CS-132	CS-462	CSR-97
06CS-02	06CS-27	08CS-57	CS-133A	CS-463	DCU-1
06CS-03	07CS-28	CS-112	CS-134	CS-464	DCU-10
06CS-04	07CS-29	CS-113	CS-135A	CS-465	DCU-11
06CS-05	07CS-30	CS-116	CS-136	CS-466	DCU-12
06CS-06	07CS-31	CS-117	CS-137	CS-467	DCU-13
06CS-07	07CS-35	CS-119A	CS-137A	CS-476	DCU-14
06CS-08	07CS-37	CS-120	CS-138	CS-477	DCU-15
06CS-09	07CS-38	CS-121	CS-139	CS-478	DCU-16
06CS-10	07CS-39	CS-122	CS-140	CS-479	DCU-17
06CS-13	07CS-40	CS-123	CS-141	CS-480	DCU-6
06CS-14	07CS-41	CS-124	CS-263	CS-491	DCU-7
06CS-19	07CS-42	CS-125	CS-265	CSD-10	DCU-9
06CS-20	07CS-43	CS-126	CS-456	CSR-75A	H5-130
06CS-21	07CS-44	CS-127	CS-457	CSR-76	H5-139
06CS-22	08CS-45	CS-128	CS-458	CSR-91	H5-140
06CS-23	08CS-46	CS-129	CS-459	CSR-93	H5-151
06CS-24	08CS-47	CS-130	CS-460	CSR-95	H5-162
06CS-25	08CS-48	CS-131	CS-461	CSR-96A	H5-87

14.2 Construction of Original Topographic Surface

Detailed 2-ft contours of the current topography, including mined out pit, was provided to HRC by Kerr in 2017. In order to construct the original topographic surface before open pit mining activity, surface drillhole collars within the pit boundary were used to interpolate the original surface in conjunction with the detailed topographic contours in Leapfrog. The use of surface drillhole collars is acceptable practice due to the low topographic relief in the property area.

14.3 Geologic Model

The Copperstone deposit is presently best described as a mid-Tertiary, detachment fault related gold deposit. Detachment faults are low-angle (up to 30°) normal faults of regional extent that have accommodated significant regional extension by upward movement of the footwall (lower-plate) producing horizontal displacements on the order of tens of kilometers. Common features of these faults are supracrustal rocks in the upper-plate on top of lower-plate rocks that were once at middle and lower crustal depths, mylonitization in lower-plate rocks that are cut by the brittle detachment fault, and listric and planar normal faults bounding half-graben basins in the upper plate (Davis and Lister, 1988).

The strike length of the deposit is approximately 4,000 ft and mineralization has been encountered by drillholes to a depth of -330 ft bmsl (approximately 1,200 ft below surface). The geologic model was created using Leapfrog, and is comprised of four structural domains, six stratigraphic units, and 48 estimation domains. The extents of the geologic model are 3,800 ft east-west by 5,900 ft north-south, by 2,200 ft depth. Table 14-4 summarizes the minimum and maximum extents of the geologic model. Furthermore, the extents of the geologic model are limited to within 200 ft of a drillhole.

Table 14-4 Geologic Model Extents

Axis	Minimum	Maximum	Extent
X	332,000	335,800	3,800
Y	1,043,300	1,049,200	5,900
Z	-900	1,300	2,200

14.3.1 Structure

The Copperstone property consists of several structural regimes, which have a significant influence on mineralization.

14.3.1.1 Northwest Trending Faults/Shear:

Represented by the Copperstone shear, these structures are the primary control of mineralization. The Copperstone shear strikes approximately 320 degrees northwest, and dips range between 45 and 25 degrees to the northeast depending on the location in the property. Additional parallel structures to the Copperstone shear may exist at depth and mineralization encountered in the Footwall zone target, and South zone target may be related to a parallel structure(s).

14.3.1.2 Northwest Trending Listric Faults:

These secondary structures can be mineralized and are likely controlled by the primary northwest trending structures. Following similar strike orientations to the primary northwest trending structures, the dips can vary from 30 to 75 degrees. The strike lengths are not as consistent as the strike length for the primary northwest trending structures. These listric faults do not show much, if any offsets in the mineralization, and may be more accurately thought of as fractures, or shear zones related to the offsets in the primary northwest trending structures.

14.3.1.3 Northeast Trending Strike Slip Faults:

These structures bound the Copperstone mineralization in the southeast and northwest, as well as offset the mineralization. These faults are steeply dipping and demonstrate normal dip slip, left lateral strike slip, and in the case of the D zone, a rotational component.

Three faults were modeled to create four structural domains (Figure 14-1). These structural zones follow the northeast trending strike slip faults structural regime and are further constrained to a maximum extent of 200 ft from a drillhole. Data used to model these faults include, an existing regional geology map developed by Amaco (Cyprus), a structural map developed by MRDI, as well as visual inspection of offsets in grades, and changes in stratigraphy. NE fault 1, the most prominent northeast striking structure, shows normal dip slip as well as left lateral strike slip components offsetting the A/B zone from the C zone by as much as 200 ft. NE fault 2 separates the C zone from the D zone. The offsets in NE fault 2 are less dramatic than those observed in NE fault 1, however, the D zone mineralization is approximately 10 degrees shallower than mineralization in the C zone. The 10-degree rotation in addition to the presence of sedimentary and metasedimentary lithologies in the D zone which are less significant in the C zone indicate the presence of a significant northeast trending structure. The Terminator fault separates the D zone from the TERM zone and represents the northwestern extent of known mineralization at Copperstone.

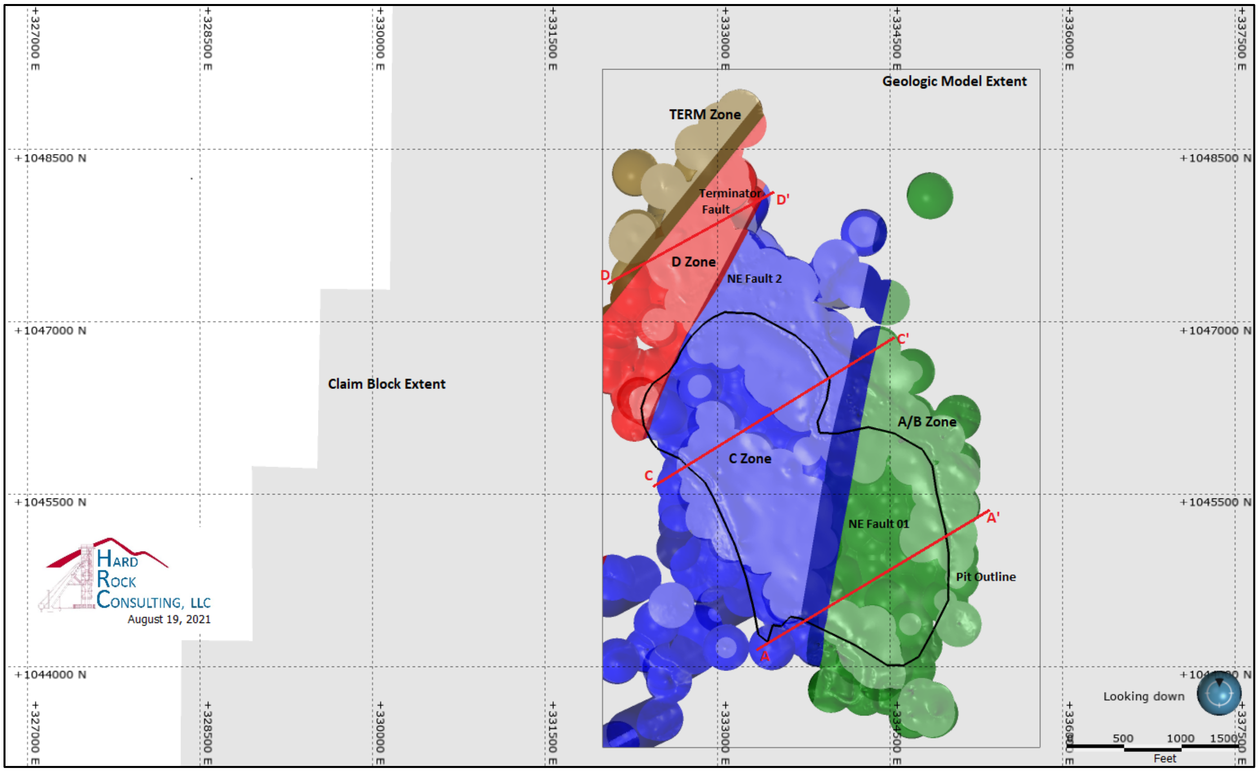

Figure 14-1 Plan View of Modeled Faults and Structural Domains

14.3.2 Stratigraphy

The host rock consists of six stratigraphic units:

- Quaternary Colorado River Sediments - Combined with overburden, the Colorado River sediments of the Bouse Formation are comprised of well developed, poorly consolidated, clay, sand, and silt beds in the Copperstone Project area (Wood D., 2013).
- Miocene Basalt - "Basaltic to andesitic stocks are cut by mineralized amethyst-quartz-specularite veins to the southeast of the pit where economic mineralization developed." (MDA, 2000).
- Ironstones - This rock unit may be more accurately described as an alteration package with extensive hemitization. Ironstones as a lithologic unit are restricted to the D zone and the northwestern extent part of the C zone.
- Jurassic Planet Volcanics - Consisting primarily of Quartz Latite Porphyry, both D. Wood and MDA characterize this unit as consisting of three-unit sub-types, however, these sub-units were unable to be modeled due to inconsistent logging in the drillholes.
    - Monolithic Breccia
    - Quartz Latite Porphyry (including other felsic intrusive such as dacite)
    - Quartz Latite Tuff
- Triassic Buckskin Formation - A meta-sediment unit comprised of marble/limestone, schist/siltstone, and quartzite. The Buckskin Formation is the principal host-rocks for D zone mineralization. The unit is identified by both D. Wood and MDA.
- Phyllite - Lowest stratigraphic unit, encountered at the base of drillholes largely on the northwest side of the pit beneath Triassic chlorite schist, quartzite, and marble. The phyllite is fine-grained quartz and chlorite with elongate oriented crystals or narrow aggregated lenses.

These stratigraphic units were modeled from logged lithologies in drillholes. Lithologies were grouped based on lithology type. Intervals were then selected by the QP based on the lithologic groups. Drillhole logs and rock type characterizations often differ from person to person, a problem which can be compounded in a property with many operators, as is the case with Copperstone. In some cases, logged drillhole lithologies differed drastically from surrounding drillholes. In these cases, the interval in the selection was changed to the lithology in surrounding drillholes. Intervals logged as structure (faults breccia alteration and veins), and missing lithologies (Missing lithology, no recovery, and backfill) were not included in the stratigraphic model.

The logged length percent from original logs were compared to the modeled volume percent in the stratigraphic model (Table 14-5). Comparison results suggest appropriate model representation of lithology relative to original drillhole logs. Differences in the basalt and phyllite stratigraphies can be explained by the limited amount of drilling intersecting these units compared to the extent of the geologic model. The majority of historic drilling within the D zone is poorly oriented and inconsistently logged, only volumes with demonstrated continuity between multiple drillholes were included in the final model, and account for the relatively low percentage in the modeled volume compared to the logged volume. Figures 14-2 through 14-4 show cross sections of the stratigraphic model throughout the property, starting at the southeast and stepping northwest. All sections are oriented southwest to northeast, looking northwest. The location of the cross sections in plan view are shown in Figure 14-1.

Table 14-5 Comparison of Modeled Stratigraphy Volumes to Logged Interval Lengths

ROCK TYPE	Description	Length (ft)	Length (%)	Group	Group Length %	Modeled Volume (ft^3)	Modeled Volume %
CGL	Conglomerate	354.0	0.06%	Quaternary Alluvium	18.54%	1,889,200,000	15.29%
OB	Overburden	26,493.7	4.65%				
OVB	Overburden	41,980.7	7.38%				
QAL	Quaternary Alluvium	36,569.8	6.42%				
US	Unconsolidated Sediments	145.0	0.03%				
BST	Basalt	8,363.8	1.47%	Basalt	1.51%	370,590,000	3.00%
BAS	Basalt	203.0	0.04%				
DAC	Dacite	2,326.5	0.41%	Quartz Latite Porphyry	65.85%	7,277,400,000	58.90%
DK	Dike	40.0	0.01%				
GNT	Granite	2,014.0	0.35%				
INT	Intrusive	840.2	0.15%				
QL	Quartz Latite	315.0	0.06%				
QLP	Quartz Latite Porphyry	367,977.2	64.65%				
QLP & Cslt	Quartz Latite Porphyry & Chloritized Siltstone	10.0	0.00%				
QLP-CSlt	Quartz Latite Porphyry & Chloritized Siltstone	5.0	0.00%				
Qlp/Phy	Quartz Latite Porphyry & Phyllite	10.0	0.00%				
QLT	Quartz Latite	400.7	0.07%				
REG	Regolith	4.2	0.00%				
TFF	Volcanic Tuff	488.8	0.09%				
VNC	Volcanics	357.0	0.06%				
CH	Chert	58.5	0.01%	Metasedimentary Unit	6.03%	544,260,000	4.40%
CSlt	Chloritized Siltstone	697.3	0.12%				
DOL	Dolomite	9.5	0.00%				
HF	Hornfels	425.5	0.07%				
JSP	Jasparoid	642.0	0.11%				
LMS	Limestone	8,363.7	1.47%				
LS	Limestone	6,880.7	1.21%				
Ls/Phy	Limestone/Phyllite	25.0	0.00%				
LS/QPL	Limestone/Quratz Latite Porphyry	9.5	0.00%				
MAR	Marble	2,525.9	0.44%				
Mar + Csch	Marble + Chloritized Schist	25.0	0.00%				
MBL	Marble	483.1	0.08%				
MD	Mudstone	75.0	0.01%				
metaLS	Metamorphosed Limestone	437.7	0.08%				
Metasediment	Metasediment	15.0	0.00%				
metaSLT	Metamorphosed Siltstone	43.6	0.01%				
MF		5.0	0.00%				
MS	Mudstone	35.0	0.01%				
MSB	Mudstone	1,501.3	0.26%				
MSK	Magnetite Skarn	551.2	0.10%				
MST	Mudstone	97.5	0.02%				
Quartzite	Quartzite	500.0	0.09%				
QZT	Quartz	5,993.3	1.05%				
QZTr	Quartz	34.0	0.01%				
SED	Sediments	3,401.7	0.60%				
SK	Skarn	20.4	0.00%				
SLT	Siltstone	1,005.8	0.18%				
SND	Sandstone	101.8	0.02%				
SS	Sandstone	371.0	0.07%				
FeSt	Ironstone	32.9	0.01%	Ironstone	1.00%	4,528,800	0.04%
FST	Ironstone	2,844.6	0.50%				
FSTK	Ironstone Stockwork	1,527.9	0.27%				
Ironstone	Ironstone	795.0	0.14%				
SiFe	Silicified Ironstone	26.2	0.00%				
Silicified Ironstone	Silicified Ironstone	460.0	0.08%				
CSch	Chloritized Schist	785.0	0.14%	Phyllite	7.07%	2,269,600,000	18.37%
PHY	Phyllite	29,076.3	5.11%				
SCH	Schist	10,387.6	1.82%				
SH	Schist	13.2	0.00%				

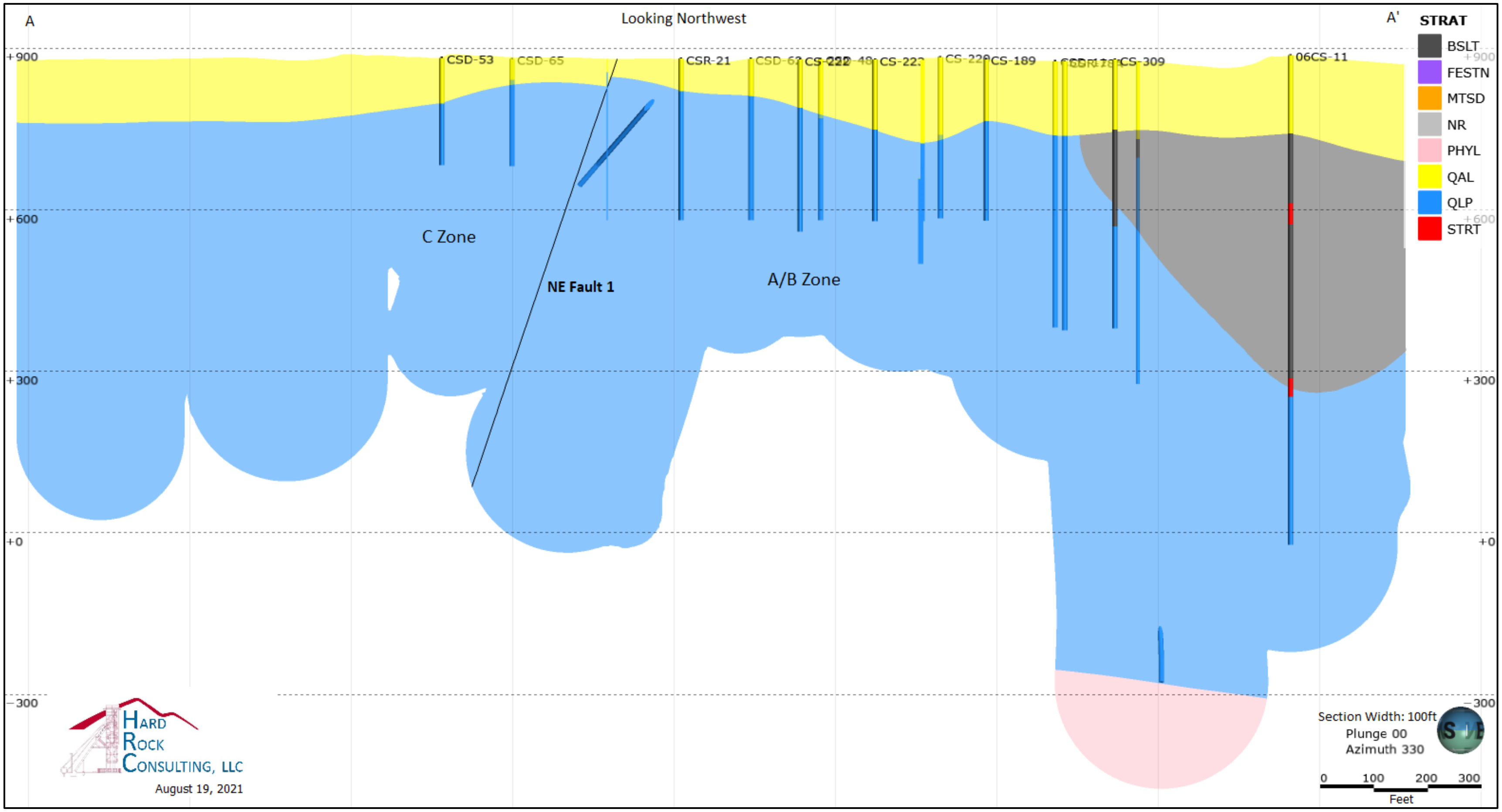

Figure 14-2 Cross Section A-A' of Modeled Stratigraphy through the A/B Zone

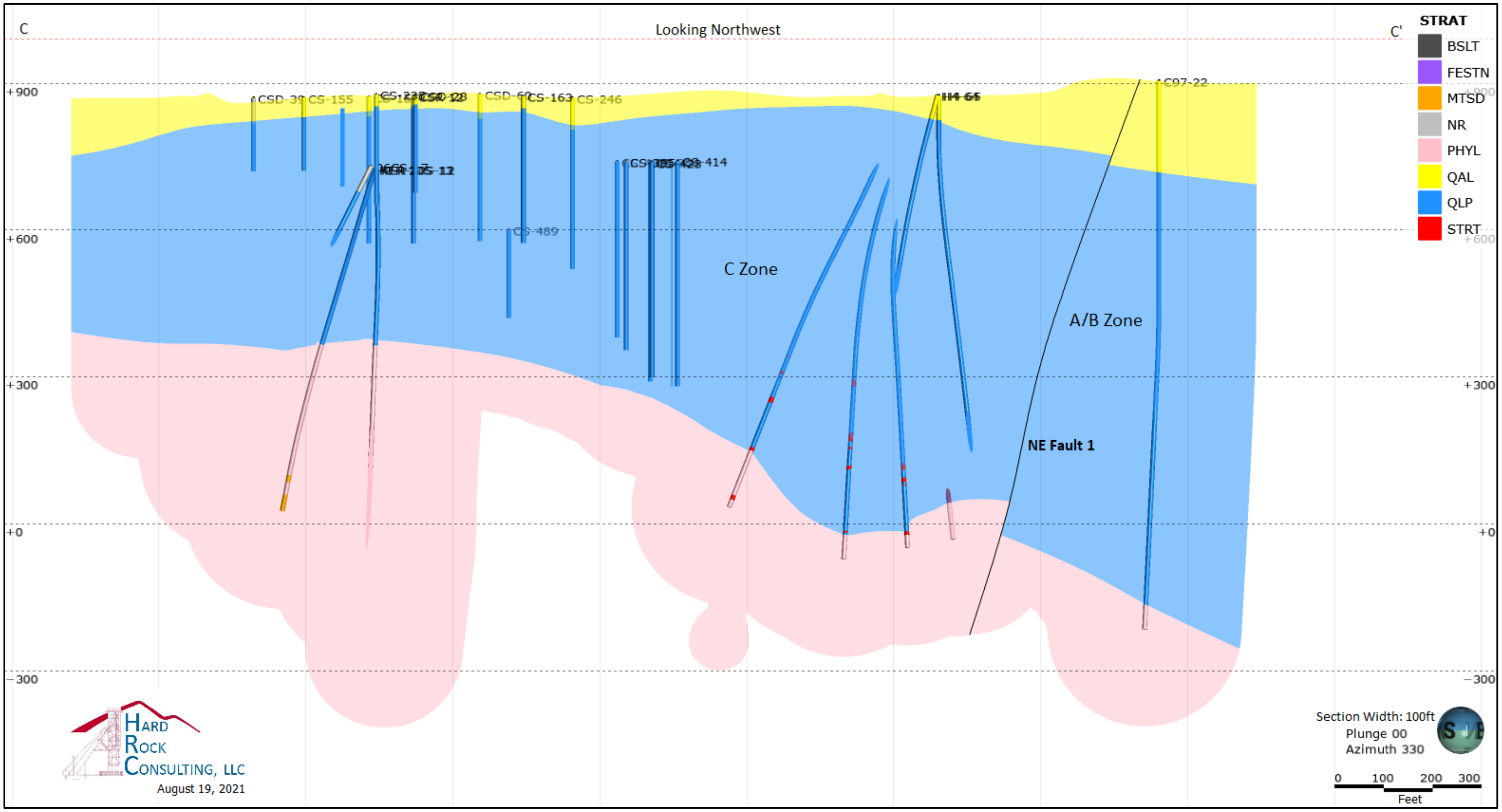

Figure 14-3 Cross Section C-C' of Modeled Stratigraphy through the C Zone

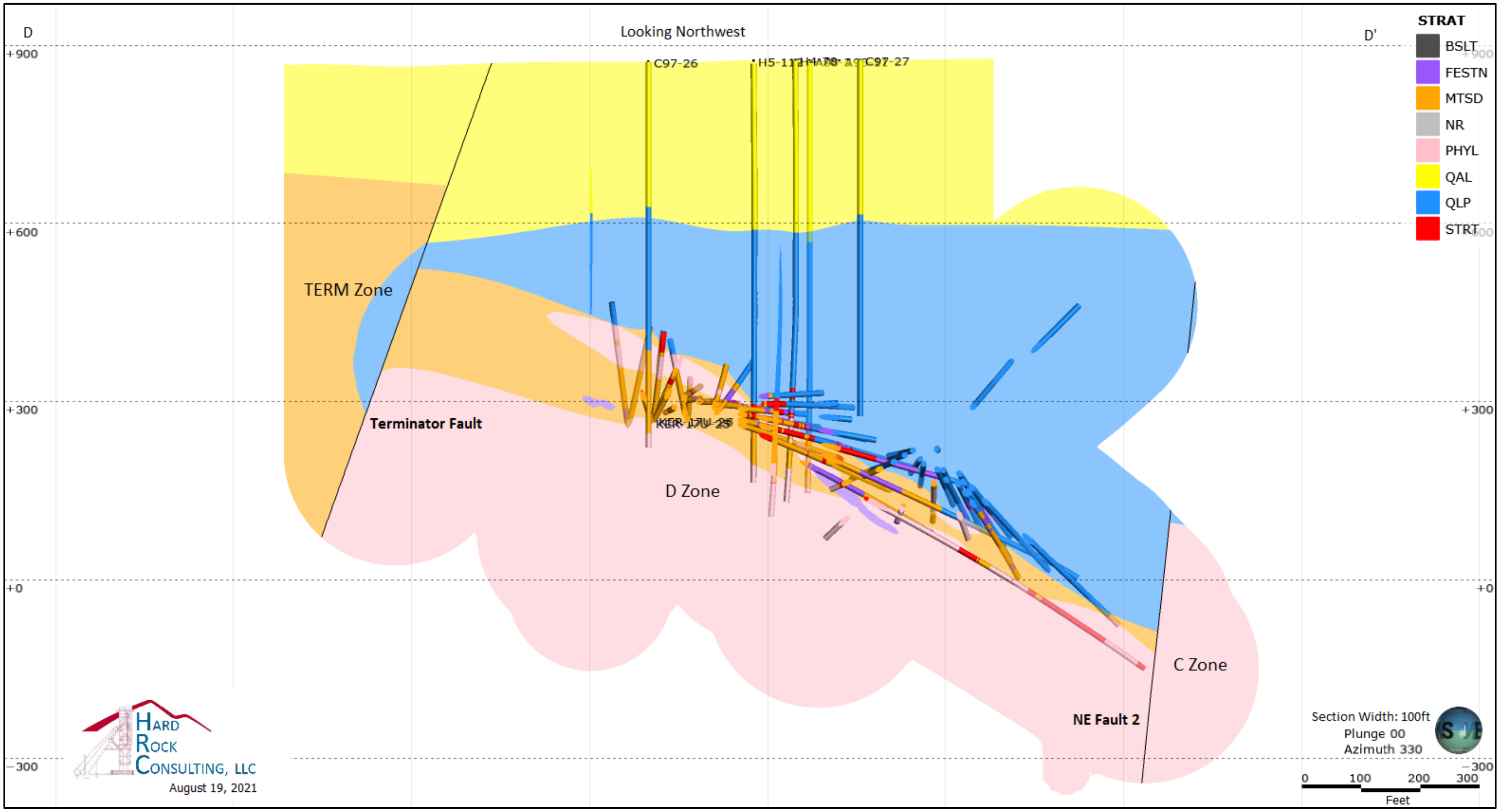

Figure 14-4 Cross Section D-D' of Modeled Stratigraphy through the D Zone

14.4 Mineral Resource Estimation

14.4.1 Estimation Domains

Gold mineralization at Copperstone is controlled by shallow angle northwest striking structures related to listric/detachment faults or shears, such as the Copperstone shear. Additional fractures/shear zones above and below the major structures can also be mineralized. These secondary fractures/shear zones follow the same overall strike but can have dips ranging from 30 to 75 degrees.

General structural trends within the Copperstone property can be visualized by displaying gold assay grades using maximum intensity projections ("MIP"). MIP is a method of interpreting and modeling structural controls from assay data. The method is applied by projecting all assay data onto a 2-dimensional ("2D") plane (e.g. computer screen) and allowing assays with higher grades (more intensity) to be projected in front of assays with lower grades. Figure 14-5 shows the MIP for gold grades oriented down dip. The black lines show the general trends of the major controlling structures observed at Copperstone.

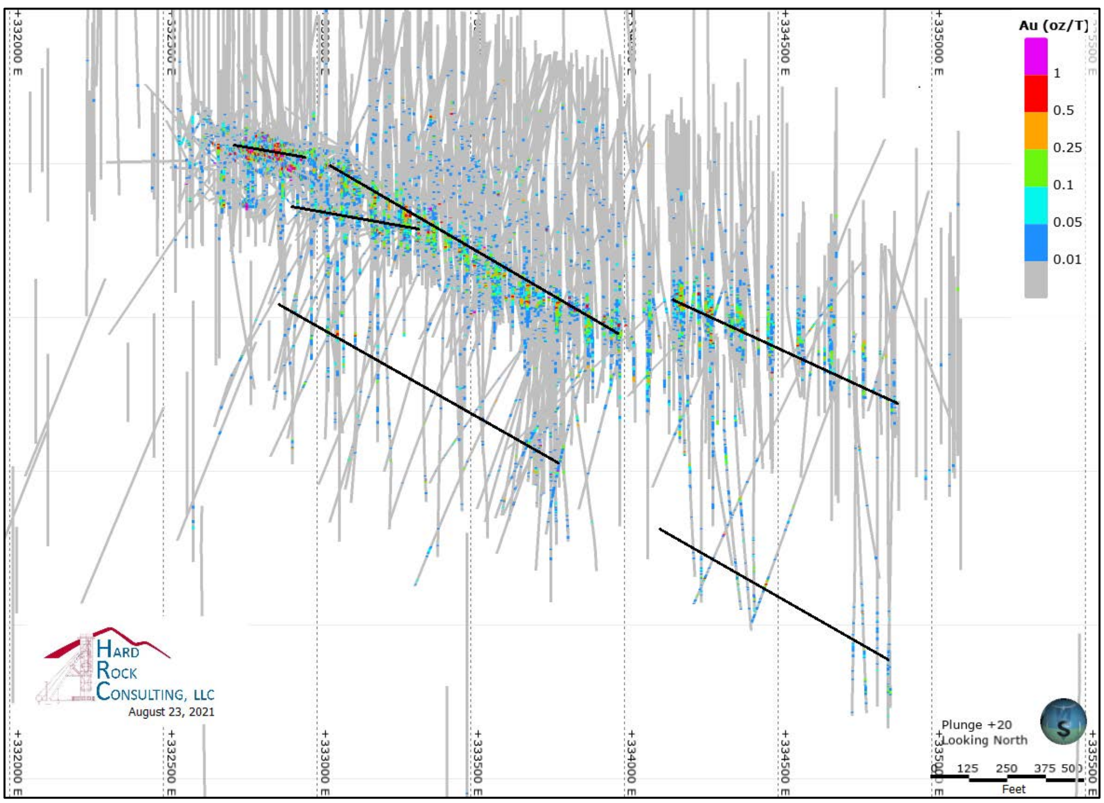

Figure 14-5 MIP of Gold Grades Showing Interpretations Major Structural Controls

MIP from gold grades confirmed the overall northwest structural trends described in previous reports. The QP determined modeling estimation domains from gold grades was appropriate for the Copperstone property. Domains were initially identified by reviewing gold grades greater than, or equal to, 0.10 oz/ton in cross section. The cut-off was selected based on initial assumptions that the underground mining cut-off would be near to 0.10 oz/ton gold for the deposit. A minimum interval length of 5 ft was selected based on assumptions about minimum mining width. Intervals greater than the cut-off, with a minimum length of 5 ft, and demonstrating continuity down dip were selected and grouped into domains. If a sample was above cut-off, but smaller than the 5-ft requirement, additional samples were added until the 5-ft minimum was met or exceeded. Domain thickness was allowed to expand as necessary to include gold grades above cut-off.

Volumes were modeled using the vein modeling method in Leapfrog after initial interval selections. Vein modeling in Leapfrog takes the selected intervals and bisects each interval into footwall and hanging wall based on the midpoint of the selection. Radial Basis Function ("RBF") interpolants are applied to fit a footwall and hanging wall surface to the selections. The interior between the two surfaces is filled to create a vein volume.

Selective inclusion of below cut-off material was utilized to preserve continuity along strike and dip as needed. The method was applied on a case-by-case basis using the same methodology as selecting intervals above cut-off. Figures 14-6 and 14-7 show the vein modeling methodology in Leapfrog.

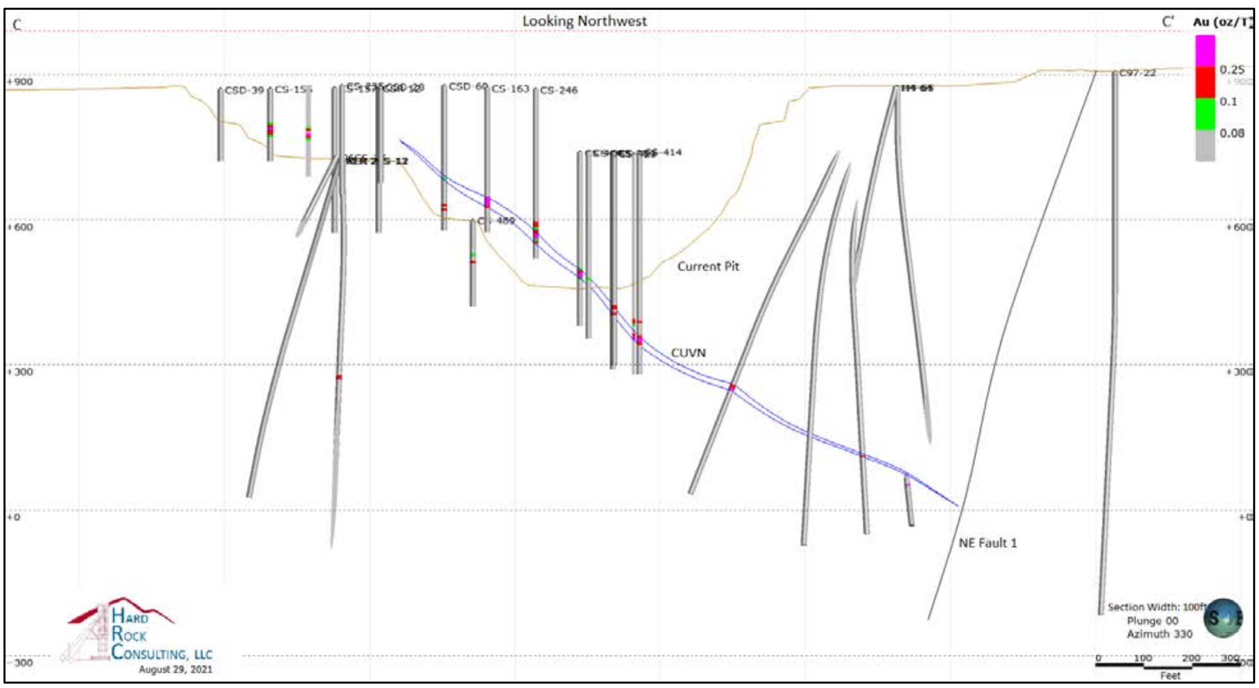

Figure 14-6 Cross Section C-C', Showing CUVN Domain and Gold Assay Intervals

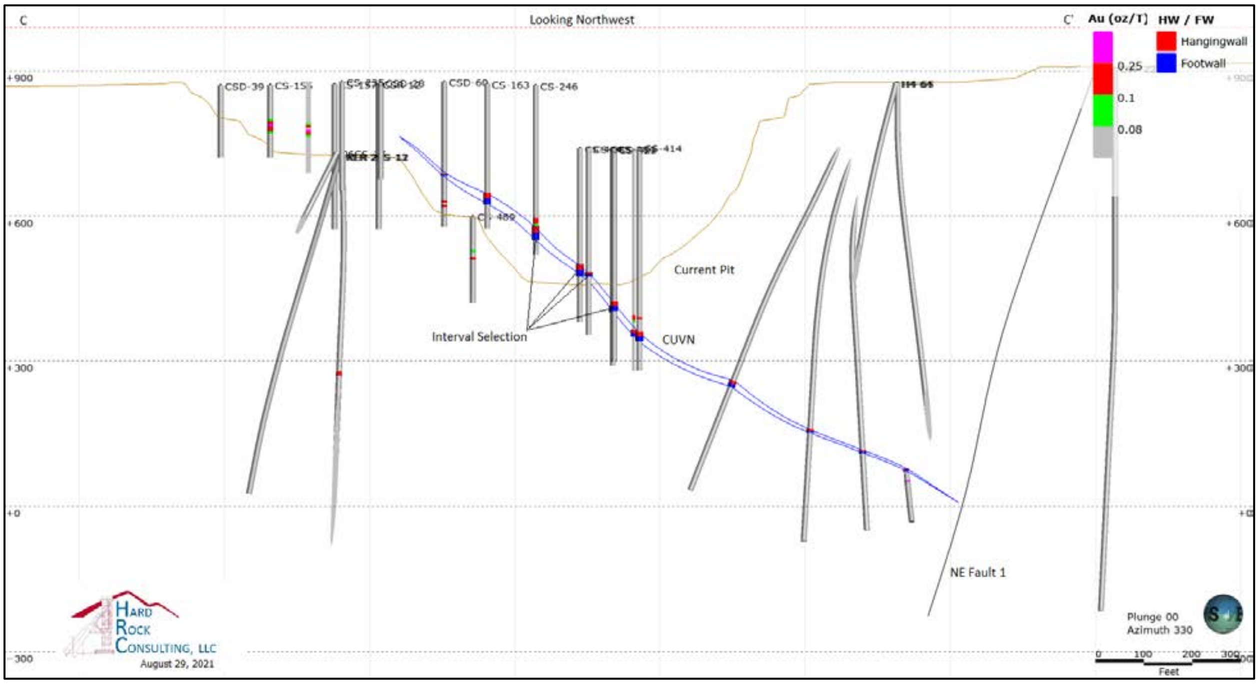

Figure 14-7 Cross Section C-C', Showing Interval Selections, and Footwall and Hanging Wall Determinations

A total of 48 mineralized domains were modeled using this method. The domains are grouped into six zones based on their spatial location. Table 14-6 summarizes the domain names and additional descriptions, and Figures 14-8 through 14-14 show the estimation domains viewed from multiple directions in 3D.

14.4.1.1 A/B Zone

The A/B zone (Figure 14-8) includes eleven domains in the southeast extent of the geologic model. The domains are controlled by the Copperstone shear and are bounded to the northwest by NE fault 1. Past reports have separated the A/B zone into two parts, the A zone, and B zone. The QP did not see any geologic, or geostatistical distinction between the A zone and B zone. The intercept from drillhole AZG-20S-02 (Table 6-18) supports the interpretation by the QP. Orientation of these domains strike between N35W and N65W, and dip between 30NE and 40NE. There are three domains, AUP4, AUP5, and ALW2, that strike more easterly/westerly and dip more shallowly.

14.4.1.2 C Zone

The C zone domains (Figures 14-9 and 14-10) located within the central portion of the geologic model are controlled by the Copperstone shear. The eleven C zone domains are bounded by NE fault 1 to the southeast, and NE fault 2 to the northwest. The orientation of these domains strike between N40W and N80W, and dip between 20NE and 45NE. The CUVN domain represents the largest, and most continuous domain, and is likely the mineralized expression of the Copperstone shear zone. CLW3, CLW7, and CLW8 strike more northerly. CLW9 strikes more easterly and dips more shallowly.

Table 14-6 Descriptive Information for Modeled Estimation Domains

Zone	Domain	ft³ x 1,000	Volume % Zone	Volume % Total	Strike	Dip	Pitch
A/B zone	CUVNA	6,467.8	34.60%	6.26%	N40W	40NE	25
	CUVNB	2,103.7	11.25%	2.04%	N40W	30NE	45
	ALW1	2,987.3	29.51%	2.89%	N45W	30NE	55
	ALW2	239.1	2.36%	0.23%	N80W	20NE	80
	ALW3	1,127.5	11.14%	1.09%	N65W	30NE	65
	ALW4	1,339.5	13.23%	1.30%	N35W	30NE	40
	AUP1	1,213.7	11.99%	1.18%	N40W	30NE	40
	AUP2	1,400.2	13.83%	1.36%	N55W	30NE	40
	AUP3	879.8	8.69%	0.85%	N35W	40NE	50
	AUP4	537.4	5.31%	0.52%	N85W	25NE	90
	AUP5	397.7	3.93%	0.39%	N75E	20SE	105
C zone	CUVN	25,226.0	75.20%	24.43%	N30W	35NE	40
	CLW1	3,255.3	9.70%	3.15%	N40W	30NE	45
	CLW2	3,496.1	10.42%	3.39%	N65W	30NE	65
	CLW3	205.5	0.61%	0.20%	N15W	35NE	15
	CLW4	173.9	0.52%	0.17%	N70W	20NE	70
	CLW5	149.6	0.45%	0.14%	N80W	25NE	80
	CLW6	181.3	0.54%	0.18%	N40W	30NE	45
	CLW7	238.4	0.71%	0.23%	N25W	45NE	40
	CLW8	132.3	0.39%	0.13%	N15W	20NE	20
	CLW9	127.1	0.38%	0.12%	N10E	15SE	180
	CUP6	361.8	1.08%	0.35%	N75W	20NE	80
D zone	CUVND	1,554.7	19.95%	1.51%	N10W	25NE	10
	CUVND2	1,092.0	14.01%	1.06%	N25W	25NE	30
	CUVND3	1,386.3	17.79%	1.34%	N25W	25NE	25
	CUVND4	2,921.7	37.50%	2.83%	N10W	30NE	10
	CUVND5	414.2	5.32%	0.40%	N20W	20NE	30
	CUVND6	423.2	5.43%	0.41%	N15W	25NE	15

Zone	Domain	ft³ x 1,000	Volume % Zone	Volume % Total	Strike	Dip	Pitch
Footwall zone	FW01	31.3	0.12%	0.03%	N45W	80NE	90
	FW02	11,287.0	44.26%	10.93%	N30W	40NE	35
	FW03	3,762.7	14.76%	3.64%	N20W	40NE	25
	FW04	2,752.6	10.79%	2.67%	N30W	40NE	30
	FW05	4,410.8	17.30%	4.27%	N30W	40NE	30
	FW06	1,244.7	4.88%	1.21%	N30W	25NE	30
	FW07	1,596.6	6.26%	1.55%	N35W	35NE	40
	FW08	413.7	1.62%	0.40%	N30W	45NE	40
S zone	S01	1,096.2	8.88%	1.06%	N55W	45NE	60
	S02	1,058.2	8.57%	1.02%	N50W	50NE	60
	S03	4,550.8	36.87%	4.41%	N25W	45NE	30
	S04	2,914.5	23.61%	2.82%	N30W	40NE	35
	S05	1,741.7	14.11%	1.69%	N30W	35NE	35
	S06	981.2	7.95%	0.95%	N45W	40NE	50
Upper Fracture zone	CUP1	41.9	0.78%	0.04%	N85W	10NE	85
	CUP2	515.3	9.55%	0.50%	N35W	20NE	40
	CUP3	675.3	12.52%	0.65%	N35W	30NE	40
	CUP4	2,271.4	42.10%	2.20%	N25W	40NE	30
	CUP5	1,649.2	30.57%	1.60%	N45W	40NE	50
	CUP7	242.4	4.49%	0.23%	N45W	30NE	50

Note: Domain naming convention reflects the Zone in which the domain is located and the relative position to the main Copperstone shear. Domains within the Copperstone shear are denoted with a *CUVN*. The UP or LW refers to whether the domain is above (UP) or below (LW) the Copperstone shear. The number is a unique identifier for the domain and is not a reflection of the relative distance from the Copperstone shear.

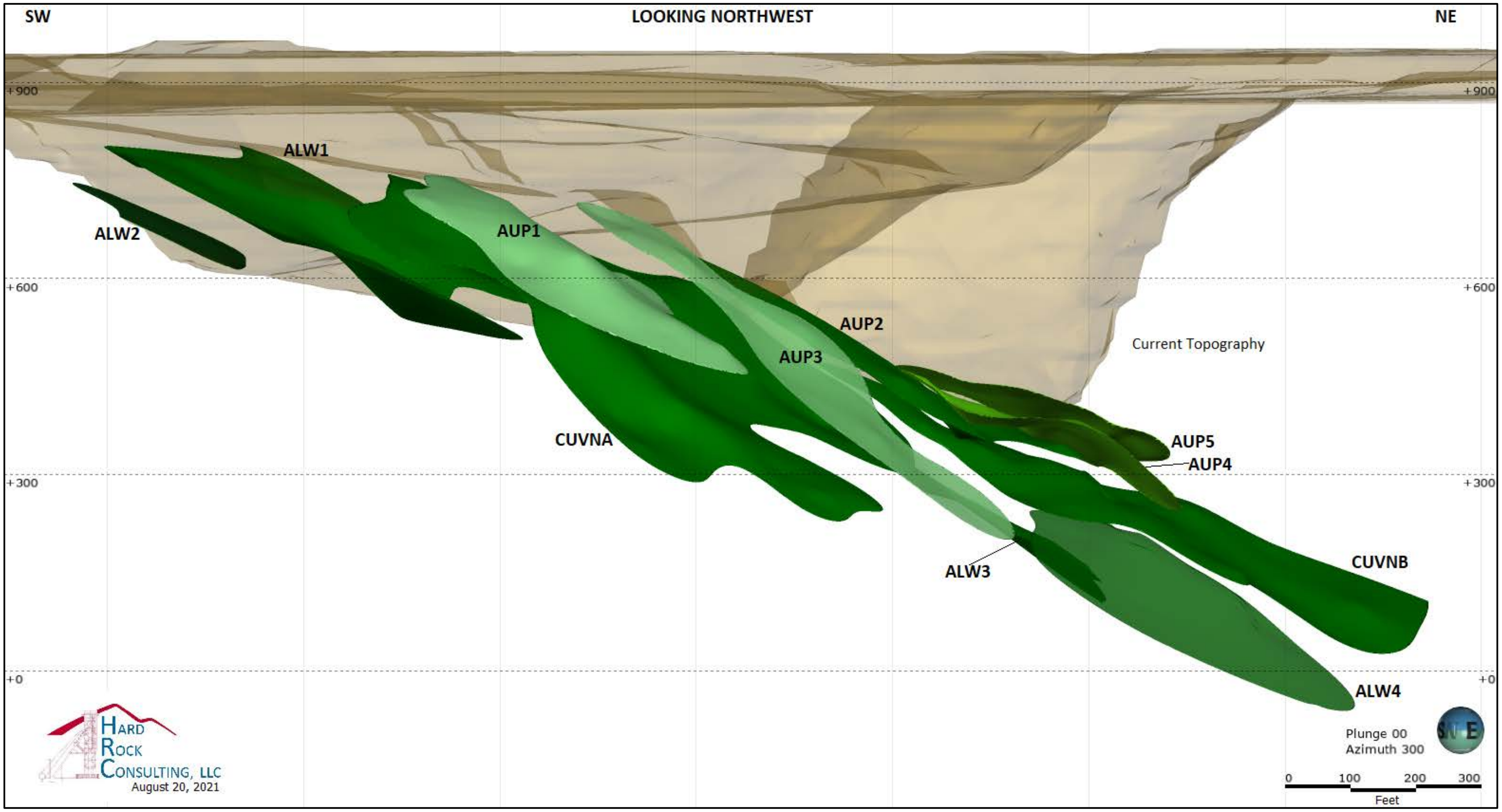

Figure 14-8 3D View of Modeled A/B Zone Estimation Domains

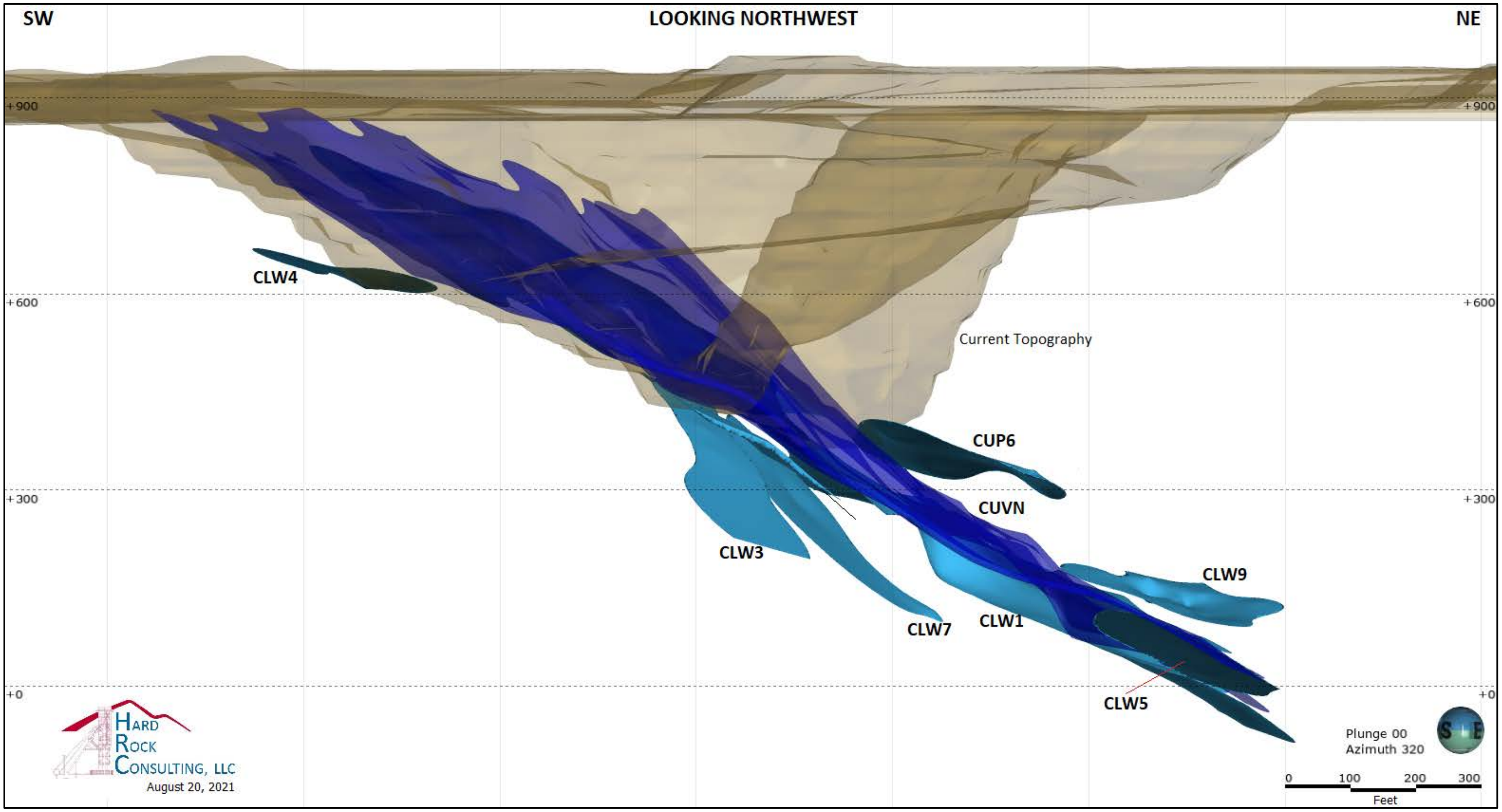

Figure 14-9 3D View of Modeled C Zone Estimation Domains

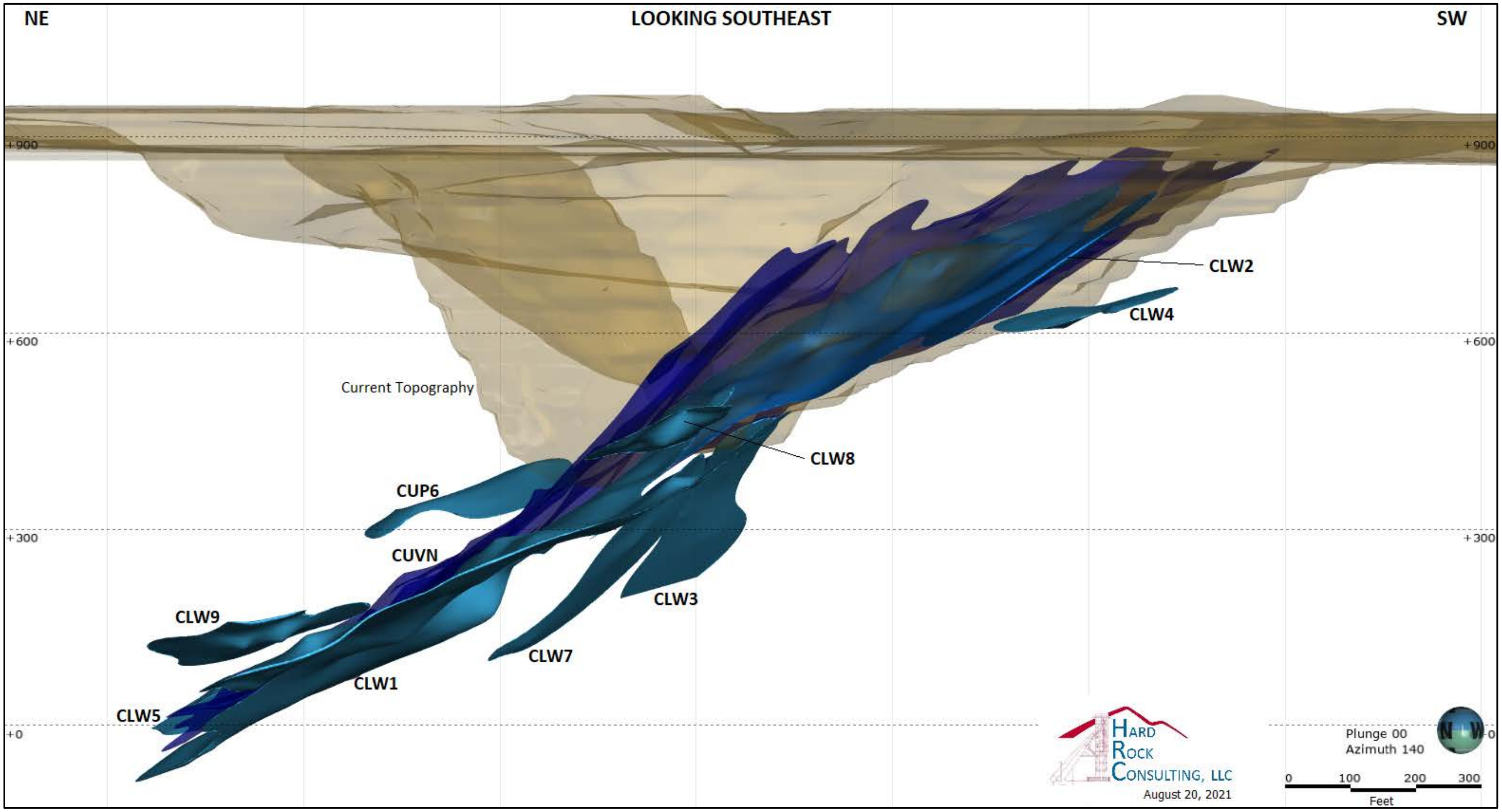

Figure 14-10 3D View of Modeled C Zone Estimation Domains

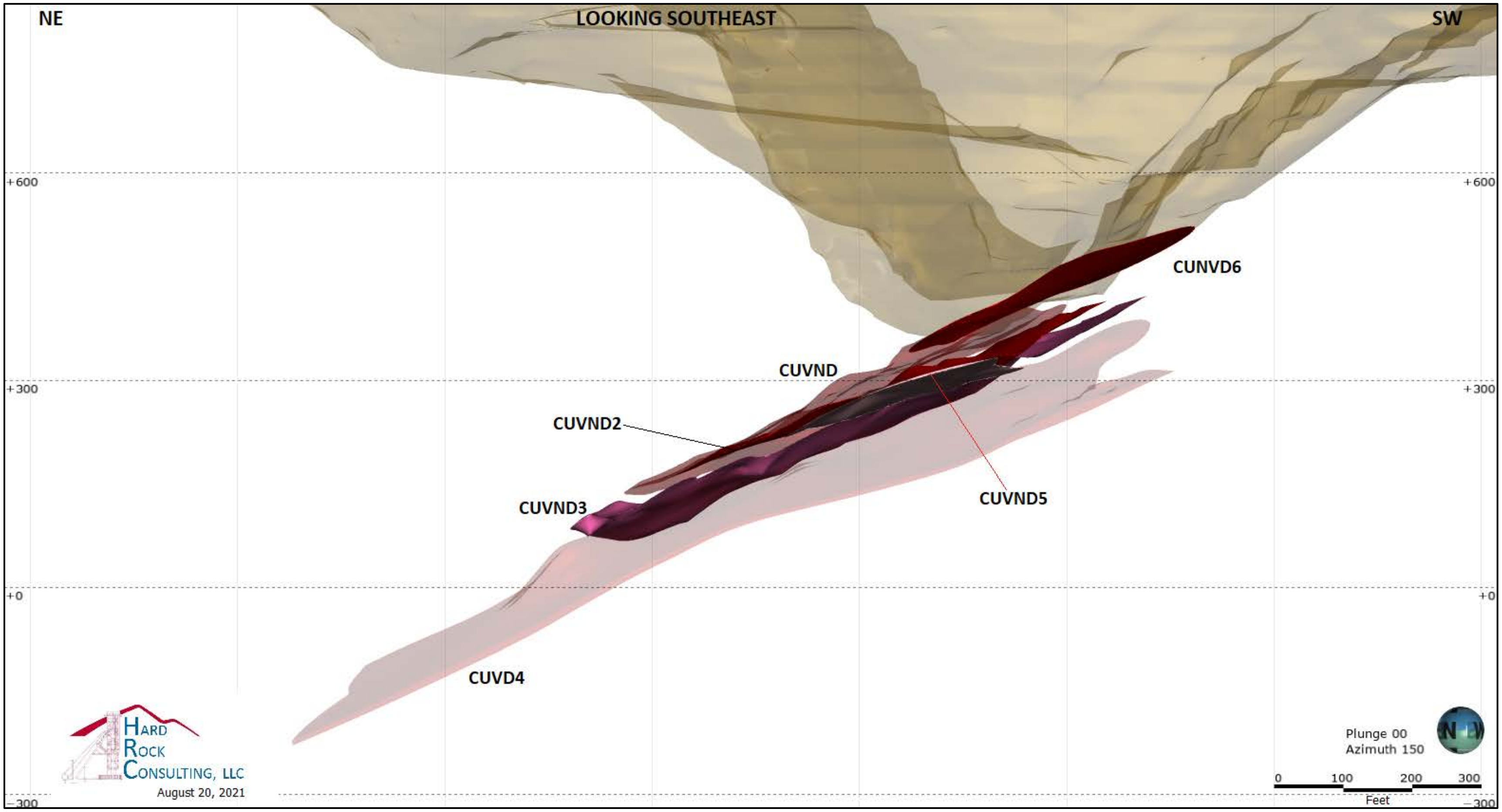

Figure 14-11 3D View of Modeled D Zone Estimation Domains

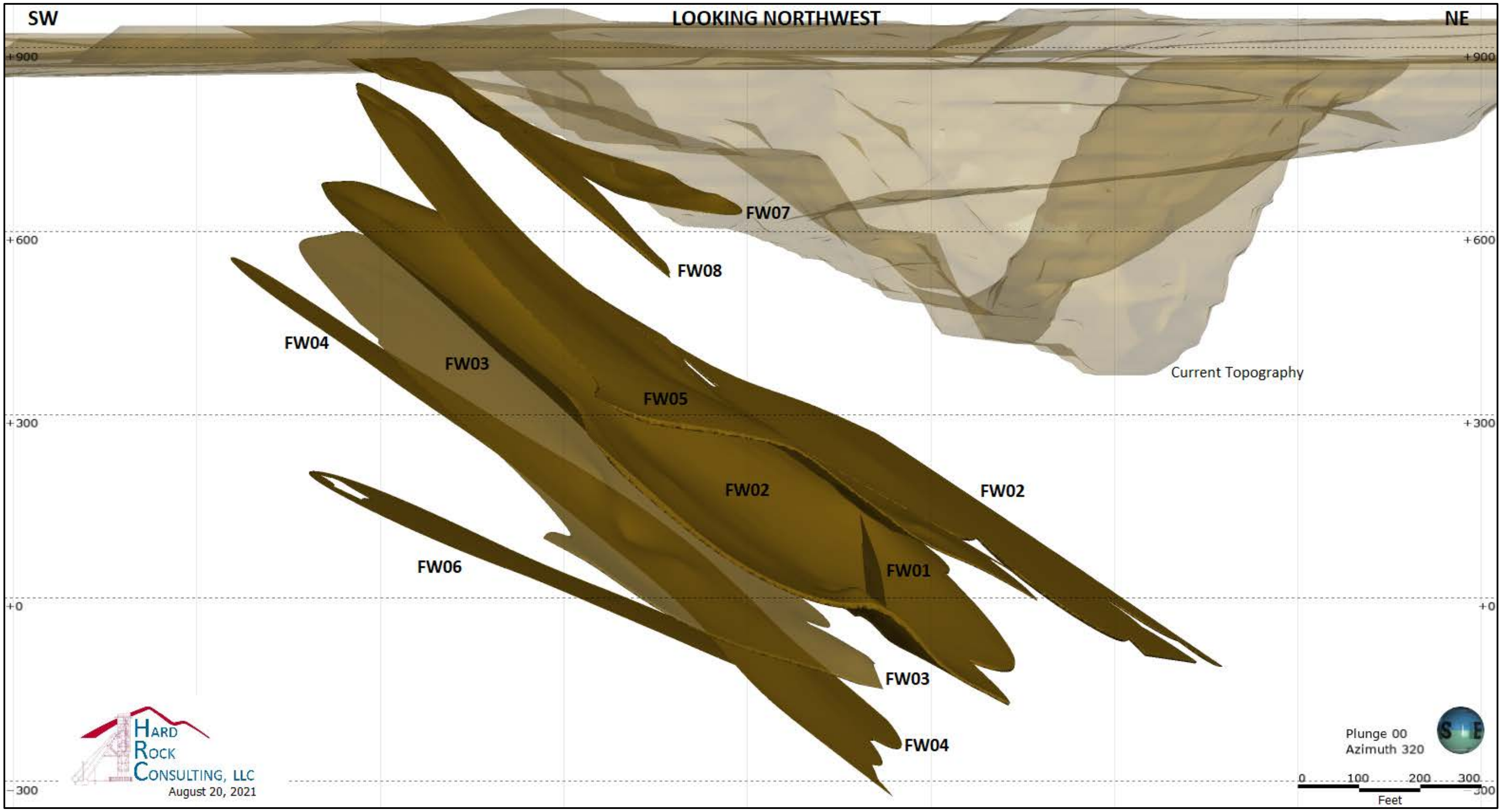

Figure 14-12 3D View of Modeled Footwall Zone Estimation Domains

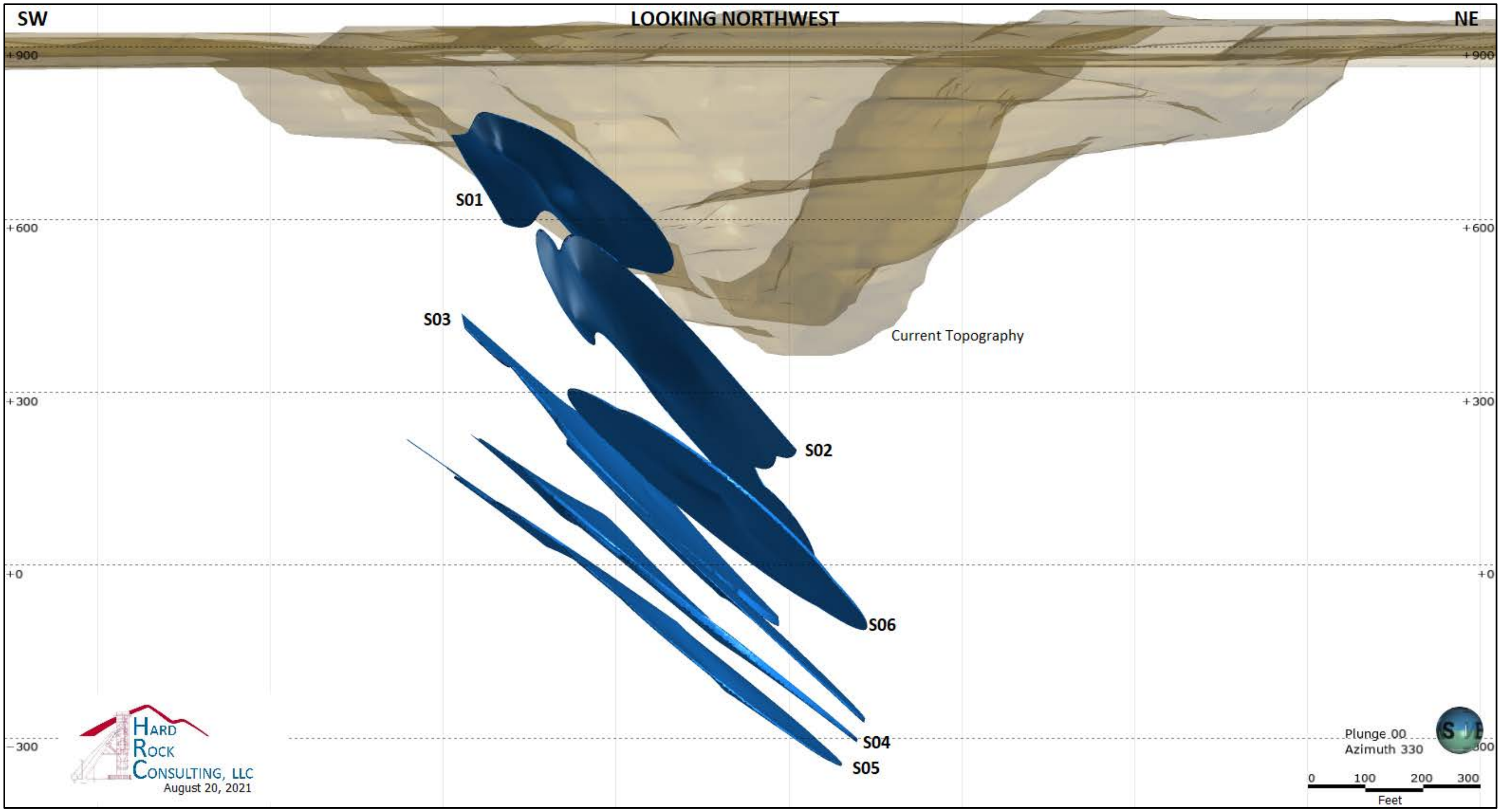

Figure 14-13 3D View of Modeled South Zone Estimation Domains

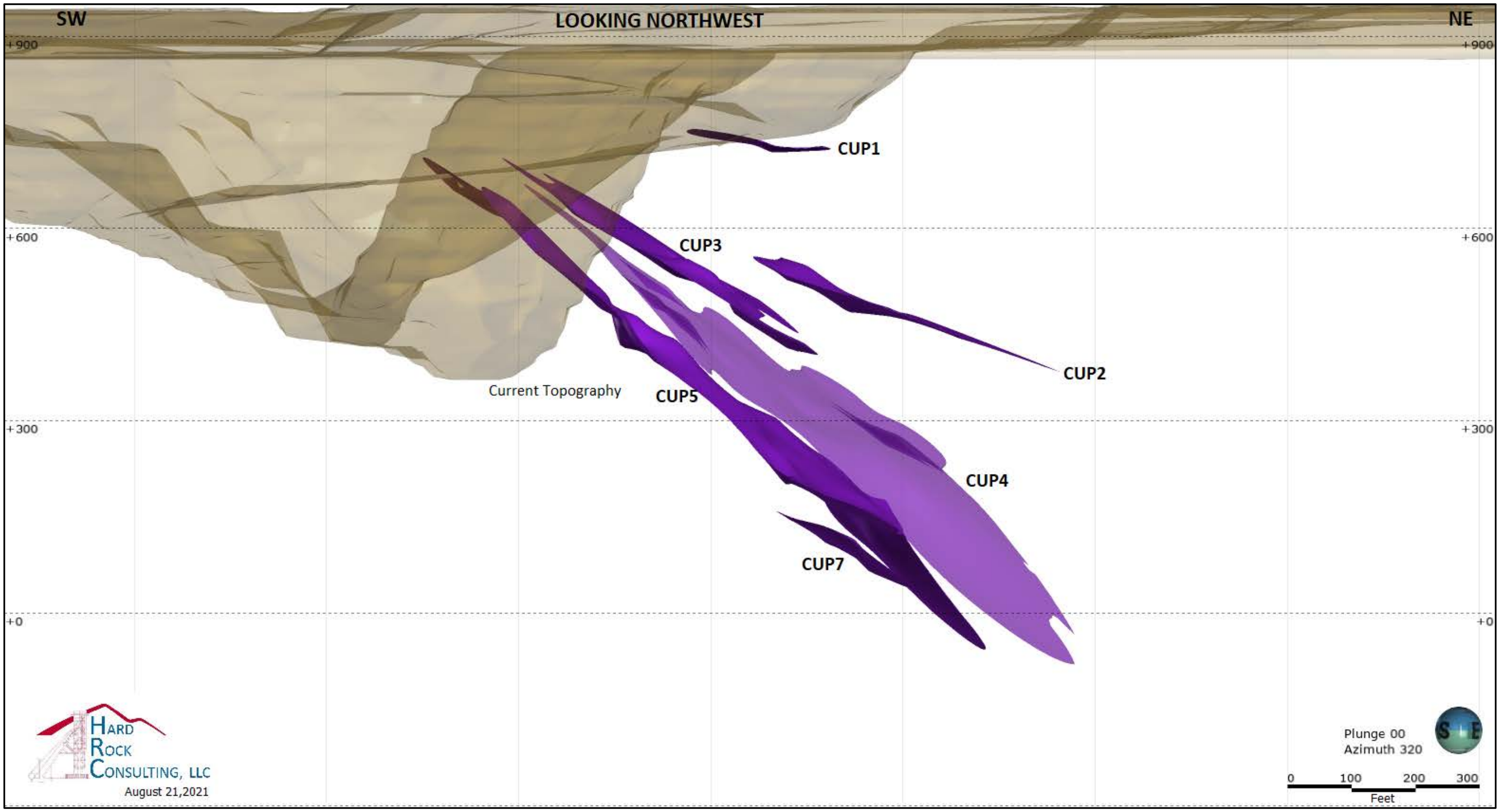

Figure 14-14 3D View of Modeled Upper Fracture Zone Estimation Domains

14.4.1.3 D Zone

Underground drilling in the D zone by American Bonanza was conducted from a single primary station, and two secondary stations. The drillholes were oriented to spread out radially from the stations and oriented down dip of the structure. Although the drilling did encounter significant gold grade intercepts over extensive interval lengths, the orientation was not conducive for the software to accurately identify the footwall and hanging wall from sample selections. In order to model the estimation domains in the D zone, drillholes from surface were initially used to model the domains using the same modeling method described above. Intervals from underground drilling were then selected based on the domain projection, gold grades, and until a minimum thickness of approximately 5 ft was reached for the domain. As a result, the underground drillholes in the D zone often include more grades below cut-off and show longer lengths than those drillholes in other parts of the Project.

Six domains modeled in the D zone (Figure 14-11) are bounded to the southeast by NE fault 2 and restricted to the northwest by the Terminator fault. These domains are still controlled by the Copperstone shear and dip at shallower angles relative to other Copperstone shear domains. The D zone domains strike between N10W and N25W, and dip between 20NE and 30NE.

14.4.1.4 Footwall Zone

The Footwall zone (Figure 14-12) is comprised of eight estimation domains, residing to the southwest and beneath the C zone domains. These domains appear to be controlled by parallel structures stratigraphically below the Copperstone shear. These domains strike between N20W and N35W, and dip between 35NE and 45NE. FW06 dips more shallowly to the northeast. FW01 dips 80NE and is bounded by FW02, and FW05. Vertical fractures related to listric faulting show potential mineralization and may be present in other parts of the property. These generally appear to exhibit shorter strike lengths relative to domains following the shear zone trends.

14.4.1.5 South Zone

Six estimation domains constitute the South zone (Figure 14-13). The South zone resides deeper and to the southwest relative to A/B zone domains. These domains are likely the Southeast continuation of the Footwall zone offset by NE fault 1. These domains strike between N25W and N55W, and dip between 35NE and 50NE.

14.4.1.6 Upper Fracture Zone

The Upper Fracture zone (Figure 14-14) is represented by six domains above the C zone. These domains can carry significant gold grades, though the continuity between high grade intercepts is low. These domains are interpreted to be the result of fractures/shear zones related to the displacement of country rock by the Copperstone shear. These domains strike between N25W and N45W, and dip between 20NE and 40NE. CUP1 strikes more westerly and dips more shallowly.

14.4.2 Validation

Visual inspection of the domains show agreement with the primary structural orientations identified from MIP (Figure 14-5) and coincide with the overall structural interpretation of the Copperstone property (Figure 14-15).

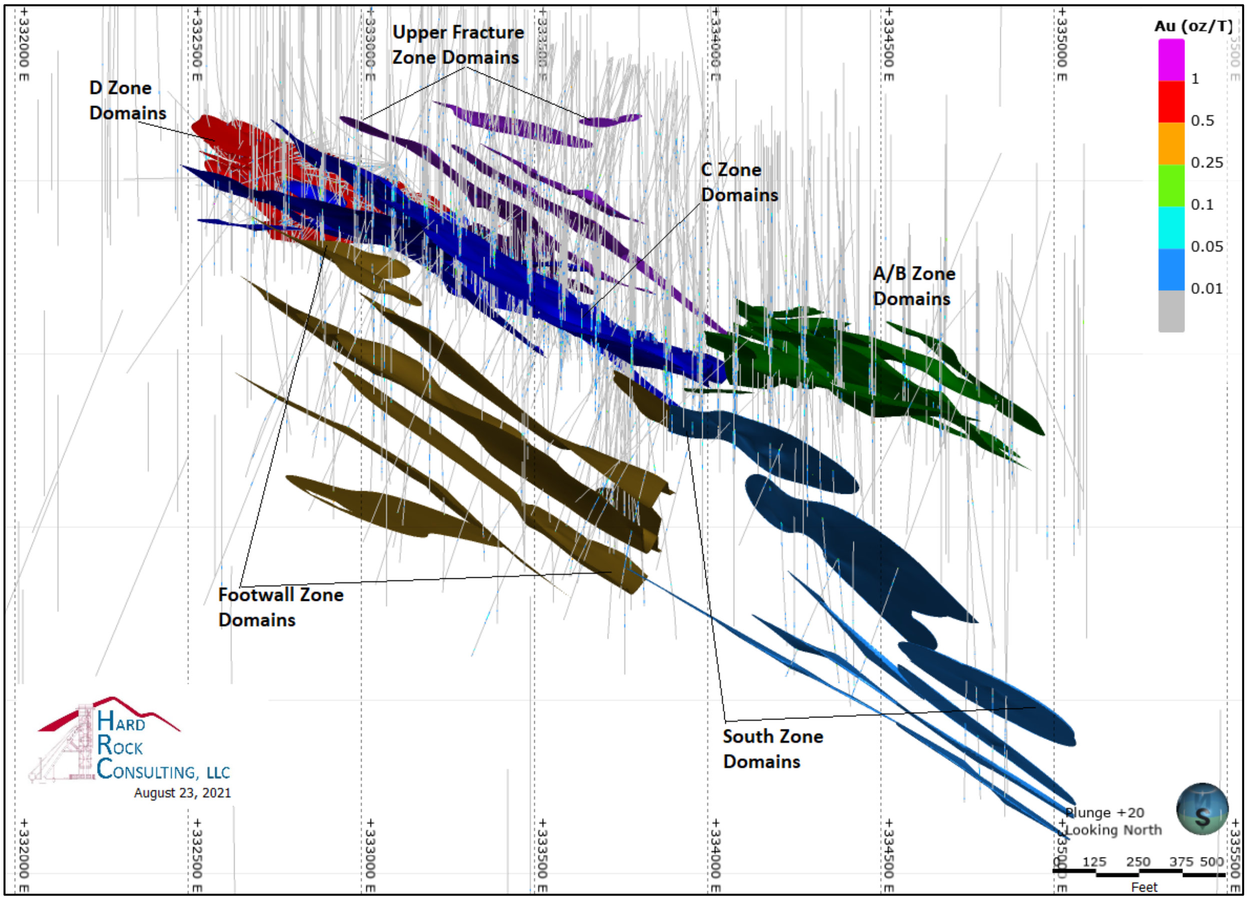

Figure 14-15 All Modeled Estimation Domains Shown in Same View Orientation as MIP Figure 14-5

Contact plots showing the average gold grade across the estimation domain boundary show the modeled estimation domains accurately represent the gold mineralization at Copperstone. Average grades report significantly higher within the modeled domains compared to average grade outside the domains. Figures 14-16 through 14-19 show contact plots for four domains.

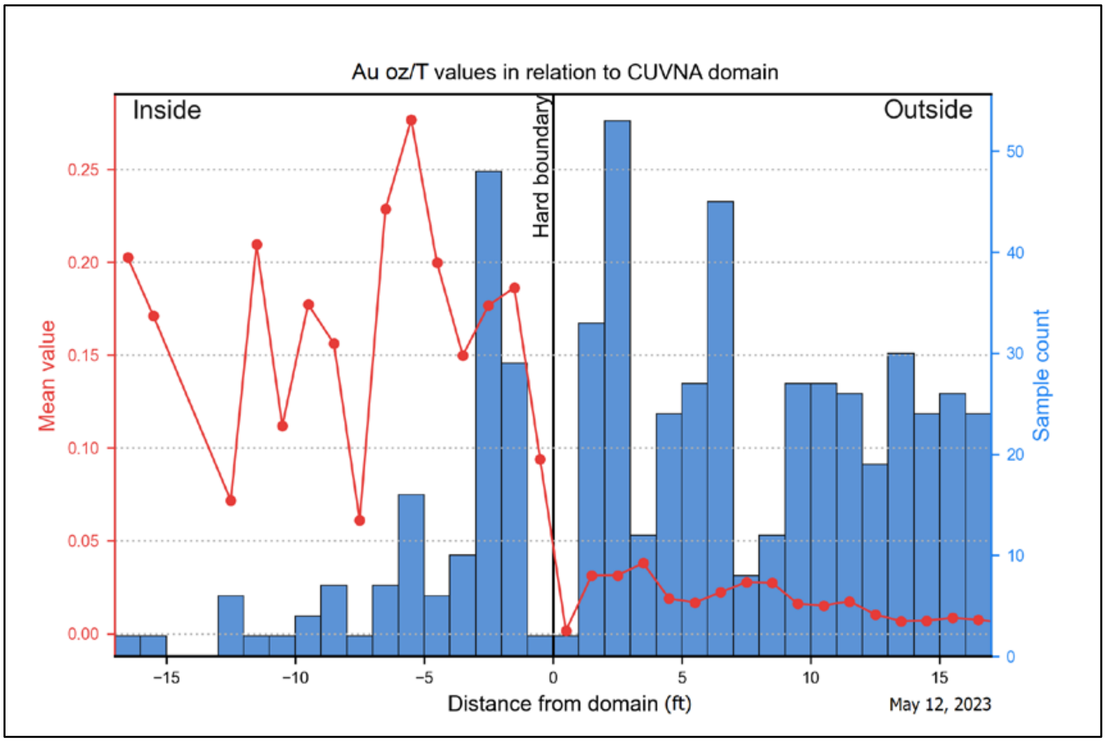

Figure 14-16 Contact Plot for the CUVNA Domain Showing Average Gold Grade

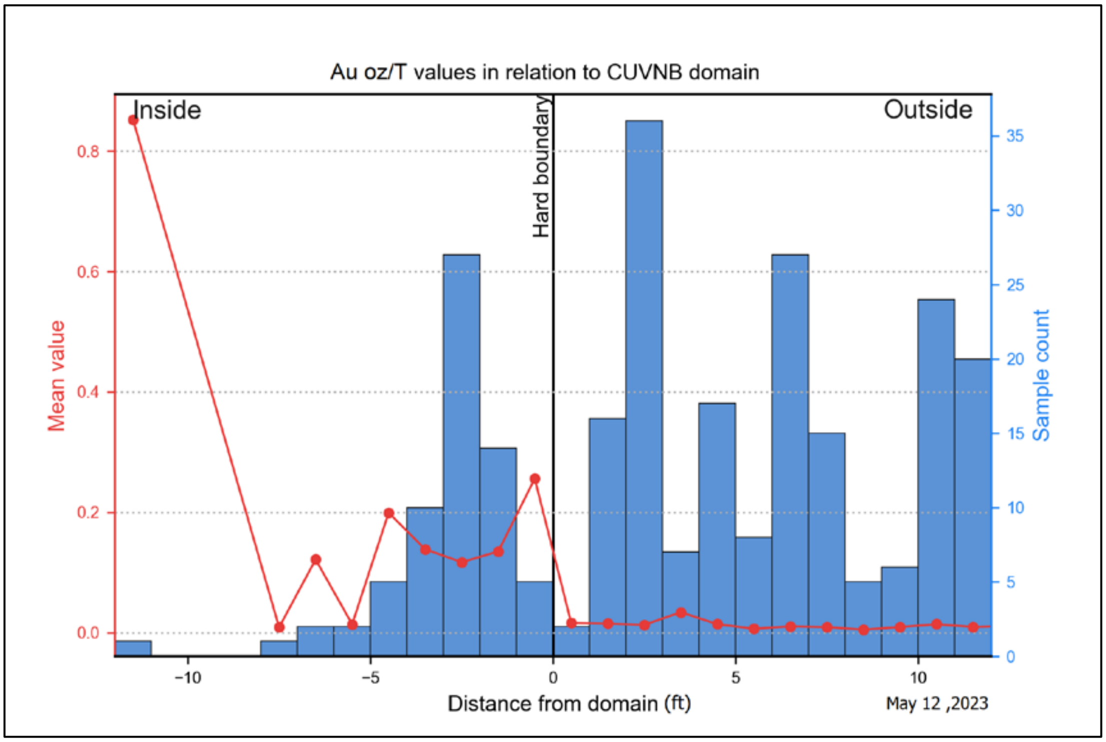

Figure 14-17 Contact Plot for the CUVNB Domain Showing Average Gold Grade

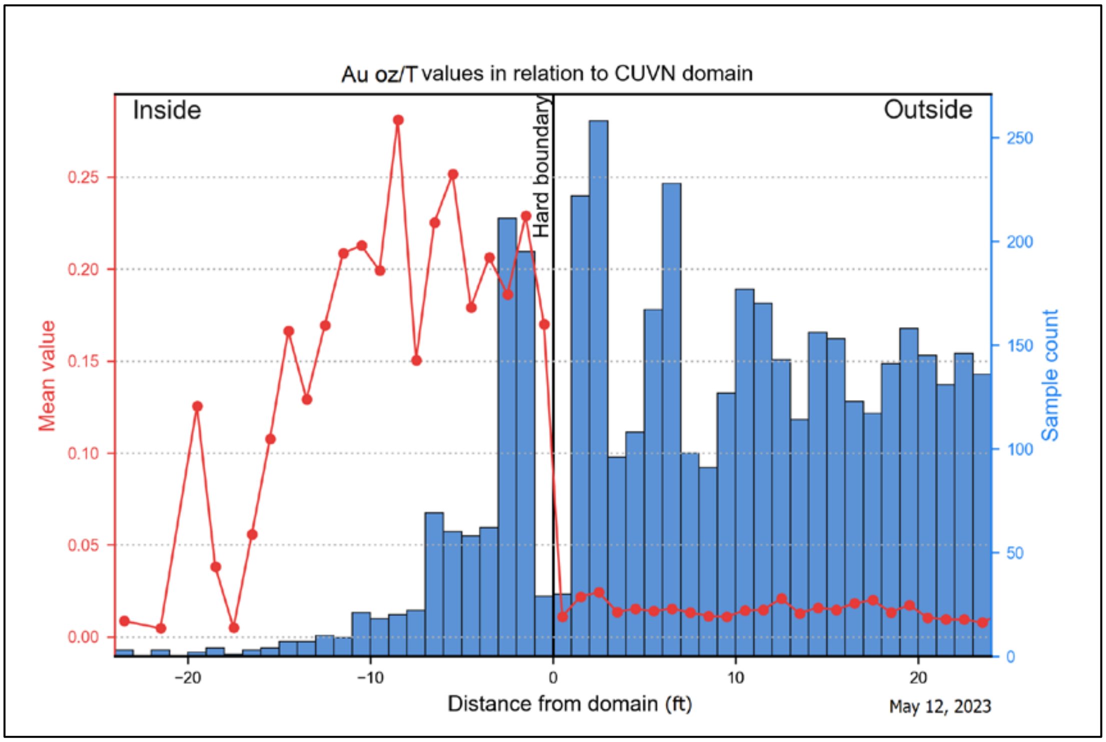

Figure 14-18 Contact Plot for the CUVN Domain Showing Average Gold Grade

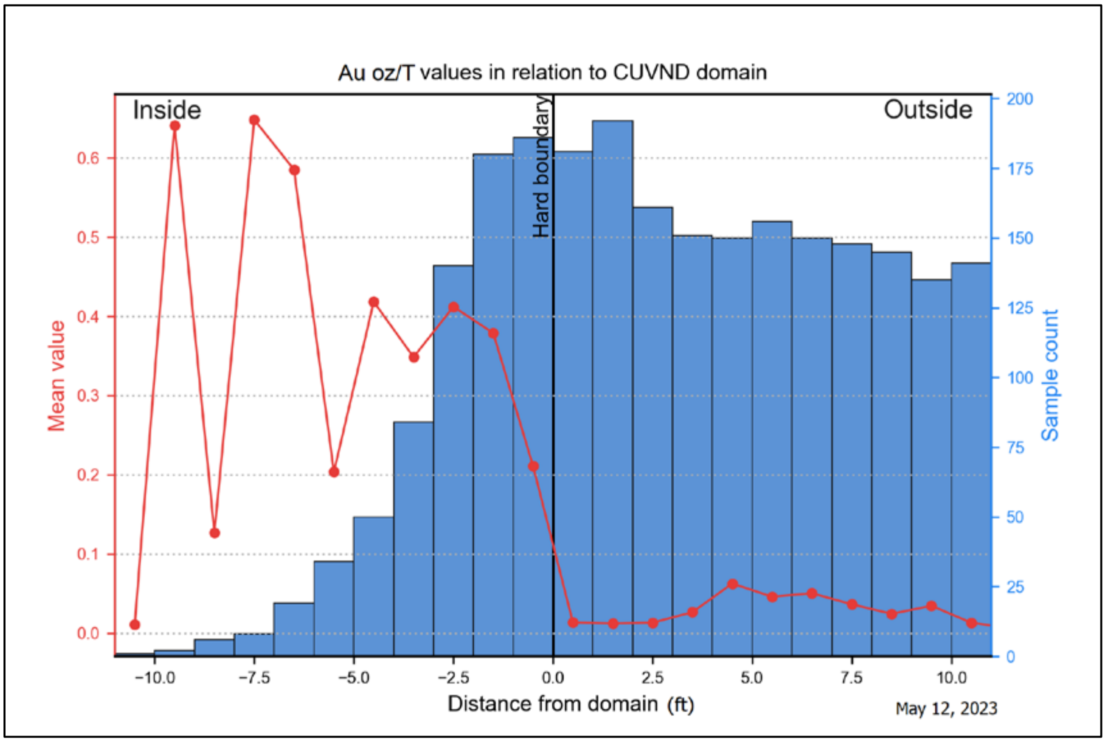

Figure 14-19 Contact Plot for the CUVND Domain Showing Average Gold Grade

14.4.3 Block Model Setup

Modeled stratigraphy and estimation domains were coded into a block model for the purpose of estimating gold mineral resources. The block model parameters defined in Table 14-7, encompass all modeled estimation domains. The origin is located on the upper left corner (ULC) of the block model boundary. The block model is rotated along strike 60 degrees west from north along the Z axis, and 35 degrees down around the X axis. Sub-blocking was applied in the Y direction to a minimum block size of 5 ft, and to a minimum of 0.625 ft in the Z direction to accurately capture estimation domain volumes. Comparison of volumes between estimation domain blocks and wireframe solids is reported in Table 14-8. Differences are within 1% for all domains.

Table 14-7 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	1	20	257	5140
Y	1047564.0	20	4	5	126	2520
Z	1710.0	5	8	0.625	306	1530
Dip Az	60					
Dip	35					

Table 14-8 Volume Comparison Between Block Model and Modeled Solids

Zone	Domain	Wireframe ft³ x 1,000	Block Model ft³ x 1,000	% Diff	Zone	Domain	Wireframe ft³ x 1,000	Block Model ft³ x 1,000	% Diff
A Zone	CUVNA	6,467.8	6,468.6	0.01%	Footwall Zone	FW01	31.3	31.0	-0.83%
	CUVNB	2,103.7	2,105.1	0.06%		FW02	11,287.0	11,286.5	0.00%
	ALW1	2,987.3	2,986.0	-0.04%		FW03	3,762.7	3,762.6	0.00%
	ALW2	239.1	239.0	-0.03%		FW04	2,752.6	2,752.4	-0.01%
	ALW3	1,127.5	1,126.3	-0.11%		FW05	4,410.8	4,411.1	0.01%
	ALW4	1,339.5	1,339.9	0.03%		FW06	1,244.7	1,246.4	0.13%
	AUP1	1,213.7	1,214.1	0.03%		FW07	1,596.6	1,595.8	-0.05%
	AUP2	1,400.2	1,401.0	0.06%		FW08	413.7	414.4	0.17%
	AUP3	879.8	880.3	0.05%	South Zone	S01	1,069.2	1,068.7	-0.05%
	AUP4	537.4	537.6	0.04%		S02	1,058.2	1,059.2	0.09%
	AUP5	397.7	398.0	0.07%		S03	4,550.8	4,534.4	-0.36%
C Zone	CUVN	25,226.0	25,224.0	-0.01%		S04	2,914.5	2,904.5	-0.34%
	CLW1	3,255.3	3,258.4	0.10%		S05	1,741.7	1,745.5	0.22%
	CLW2	3,496.1	3,497.4	0.04%		S06	981.2	976.7	-0.46%
	CLW3	205.5	204.9	-0.29%	Upper Fracture Zone	CUP1	41.9	41.9	0.11%
	CLW4	173.9	174.8	0.54%		CUP2	515.3	512.8	-0.49%
	CLW5	149.6	149.6	-0.04%		CUP3	675.3	673.8	-0.22%
	CLW6	181.3	182.3	0.53%		CUP4	2,271.4	2,270.9	-0.02%
	CLW7	238.4	238.1	-0.15%		CUP5	1,649.2	1,646.6	-0.16%
	CLW8	132.3	131.0	-0.97%		CUP7	242.4	242.8	0.17%
	CLW9	127.1	127.7	0.50%					
	CUP6	361.8	362.8	0.28%					
D Zone	CUVND	1,544.7	1,543.9	-0.05%					
	CUVND2	1,092.0	1,093.1	0.10%					
	CUVND3	1,386.3	1,384.2	-0.15%					
	CUVND4	2,921.7	2,893.6	-0.96%					
	CUVND5	414.2	412.3	-0.46%					
	CUVND6	423.2	422.0	-0.28%					

14.4.4 Compositing

Five-foot downhole composites were broken out by domain contact. Sample lengths less than 2.5 ft were distributed equally throughout the drillhole inside the domain. Composite descriptive statistics were compared to sample statistics to ensure populations within domains were not overly affected by composite length. Descriptive statistics of gold grades for composites by domain are presented in Table 14-9.

14.4.5 Outlier Composite Handling

Estimation of highly skewed grade distributions, as is present within the Copperstone estimation domains, can be sensitive to the presence of even a few extreme values resulting in an overestimation of the mean. To better estimate the true mean of the deposit, a detailed capping study was performed on a domain-by-domain basis. The coefficient of variation ("CV") is a statistical measure of the population's uniformity. A CV greater than two indicates the presence of statistical outliers which require to be treated before estimation of gold grades. A CV less than one can indicate no significant statistical outliers are present. The CV of each domain was reviewed. If the CV of the domain was less than one, no action was taken. If the CV of the domain was greater than two, outliers in the domain were capped. For domains with a CV between one and two, outliers were restricted by distance. Outliers and capping/restrictive limits were identified using a combination of histograms, cumulative frequency plots, and visual inspection of the composite population within the domain. Table 14-10 summarizes the methodology, capping limits, and number of composites impacted for each domain.

Table 14-9 Descriptive Statistics of Gold Grades (oz/ton) for Composites by Domain

Zone	Domain	Count	Length	Mean	Std. Dev.	CV	Minimum	Median	Maximum
A/B Zone	CUVNA	167	832.7	0.181	0.146	0.81	0.0004	0.130	0.87
	CUVNB	67	335.2	0.161	0.184	1.14	0.0001	0.105	0.85
	ALW1	64	320.0	0.139	0.202	1.46	0.0001	0.059	1.06
	ALW2	8	40.0	0.153	0.059	0.39	0.1050	0.132	0.26
	ALW3	38	186.1	0.125	0.171	1.37	0.0001	0.080	0.84
	ALW4	35	172.0	0.097	0.120	1.24	0.0001	0.089	0.60
	AUP1	39	195.0	0.120	0.127	1.06	0.0100	0.071	0.44
	AUP2	42	208.2	0.141	0.087	0.62	0.0095	0.101	0.52
	AUP3	30	144.0	0.130	0.139	1.07	0.0001	0.115	0.39
	AUP4	27	138.3	0.288	0.615	2.13	0.0030	0.090	2.69
	AUP5	25	125.2	0.054	0.095	1.77	0.0001	0.006	0.39
C Zone	CUVN	812	4,018.3	0.188	0.240	1.28	0.0001	0.128	3.68
	CLW1	154	769.4	0.164	0.331	2.02	0.0001	0.076	2.90
	CLW2	112	564.8	0.079	0.095	1.21	0.0001	0.045	0.47
	CLW3	9	45.0	0.087	0.070	0.80	0.0030	0.109	0.20
	CLW4	13	59.9	0.187	0.134	0.72	0.0001	0.168	0.47
	CLW5	7	30.4	0.100	0.178	1.78	0.0030	0.056	0.51
	CLW6	9	45.0	0.181	0.068	0.37	0.0950	0.185	0.26
	CLW7	8	41.0	0.314	0.399	1.27	0.0070	0.232	1.09
	CLW8	10	51.0	0.079	0.096	1.22	0.0013	0.064	0.28
	CLW9	25	120.7	0.104	0.142	1.37	0.0001	0.040	0.50
	CUP6	19	93.9	0.112	0.096	0.86	0.0001	0.105	0.31
D Zone	CUVND	407	2,028.7	0.314	0.864	2.76	0.0004	0.112	13.54
	CUVND2	359	1,782.3	0.176	0.727	4.14	0.0001	0.016	12.01
	CUVND3	368	1,820.3	0.176	0.461	2.61	0.0001	0.025	4.85
	CUVND4	208	1,032.6	0.080	0.395	4.91	0.0001	0.003	3.75
	CUVND5	29	139.1	0.164	0.569	3.47	0.0002	0.014	2.94
	CUVND6	21	102.0	0.034	0.072	2.13	0.0001	0.010	0.43
Footwall Zone	FW01	13	64.0	0.288	0.538	1.87	0.0020	0.009	1.50
	FW02	102	520.2	0.134	0.453	3.38	0.0001	0.011	4.23
	FW03	40	192.3	0.068	0.100	1.46	0.0001	0.045	0.52
	FW04	17	86.4	0.057	0.080	1.41	0.0001	0.018	0.22
	FW05	65	328.4	0.038	0.069	1.80	0.0001	0.005	0.32
	FW06	7	33.0	0.158	0.126	0.80	0.0070	0.148	0.40
	FW07	36	181.6	0.066	0.079	1.20	0.0001	0.042	0.38
	FW08	9	46.8	0.147	0.144	0.98	0.0001	0.202	0.37
South Zone	S01	58	289.4	0.099	0.099	1.00	0.0001	0.077	0.37
	S02	24	117.1	0.132	0.121	0.91	0.0001	0.114	0.45
	S03	22	112.9	0.209	0.447	2.14	0.0010	0.067	2.09
	S04	24	121.4	1.121	5.018	4.48	0.0001	0.064	24.81
	S05	7	31.4	0.056	0.051	0.91	0.0001	0.054	0.15
	S06	6	28.5	0.071	0.097	1.37	0.0020	0.044	0.26
Upper Fracture Zone	CUP1	9	45.0	0.067	0.141	2.12	0.0030	0.003	0.42
	CUP2	19	96.1	0.137	0.236	1.72	0.0001	0.027	0.88
	CUP3	37	184.0	0.032	0.047	1.44	0.0001	0.011	0.18
	CUP4	90	448.0	0.046	0.111	2.41	0.0001	0.003	0.77
	CUP5	78	385.9	0.609	3.453	5.67	0.0001	0.009	23.54
	CUP7	21	98.0	0.172	0.561	3.26	0.0020	0.010	2.50

Table 14-10 Capping Values Applied by Domain

Zone	Domain	Composite				Capping/Restriction		
		Count	Mean	Maximum	CV	Method	Value	Composites Capped/Restricted
A/B Zone	CUVNA	167	0.181	0.87	0.81	No Action	None	0
	CUVNB	67	0.161	0.85	1.14	Restriction	0.629	2
	ALW1	64	0.139	1.06	1.46	Restriction	0.825	1
	ALW2	8	0.153	0.26	0.39	No Action	None	0
	ALW3	38	0.125	0.84	1.37	Restriction	0.506	1
	ALW4	35	0.097	0.60	1.24	Restriction	0.238	1
	AUP1	39	0.120	0.44	1.06	Restriction	0.422	1
	AUP2	42	0.141	0.52	0.62	No Action	None	0
	AUP3	30	0.130	0.39	1.07	Restriction	0.390	2
	AUP4	27	0.288	2.69	2.13	Cap	0.822	1
	AUP5	25	0.054	0.39	1.77	Restriction	0.213	1
C Zone	CUVN	812	0.188	3.68	1.28	Restriction	2.119	1
	CLW1	154	0.164	2.90	2.02	Restriction	1.466	3
	CLW2	112	0.079	0.47	1.21	Restriction	0.446	1
	CLW3	9	0.087	0.20	0.80	No Action	None	0
	CLW4	13	0.187	0.47	0.72	No Action	None	0
	CLW5	7	0.100	0.51	1.78	Restriction	0.100	1
	CLW6	9	0.181	0.26	0.37	No Action	None	0
	CLW7	8	0.314	1.09	1.27	Restriction	0.613	1
	CLW8	10	0.079	0.28	1.22	Restriction	0.216	1
	CLW9	25	0.104	0.50	1.37	Restriction	0.339	2
	CUP6	19	0.112	0.31	0.86	No Action	None	0
D Zone	CUVND	407	0.314	13.54	2.76	Restriction	6.278	1
	CUVND2	359	0.176	12.01	4.14	Restriction	2.249	4
	CUVND3	368	0.176	4.85	2.61	Restriction	2.234	2
	CUVND4	208	0.080	3.75	4.91	Restriction	1.718	2
	CUVND5	29	0.164	2.94	3.47	Restriction	0.562	1
	CUVND6	21	0.034	0.43	2.13	Restriction	0.100	1
Footwall Zone	FW01	13	0.288	1.50	1.87	Restriction	0.382	2
	FW02	102	0.134	4.23	3.38	Cap	0.590	2
	FW03	40	0.068	0.52	1.46	Restriction	0.194	2
	FW04	17	0.057	0.22	1.41	Restriction	0.216	1
	FW05	65	0.038	0.32	1.80	Restriction	0.280	1
	FW06	7	0.158	0.40	0.80	No Action	None	0
	FW07	36	0.066	0.38	1.20	Restriction	0.224	1
	FW08	9	0.147	0.37	0.98	No Action	None	0
South Zone	S01	58	0.099	0.37	1.00	No Action	None	0
	S02	24	0.132	0.45	0.91	No Action	None	0
	S03	22	0.209	2.09	2.14	Cap	0.758	1
	S04	24	1.121	24.81	4.48	Cap	0.421	1
	S05	7	0.056	0.15	0.91	No Action	None	0
	S06	6	0.071	0.26	1.37	Restriction	0.100	1
Upper Fracture Zone	CUP1	9	0.067	0.42	2.12	Cap	0.156	1
	CUP2	19	0.137	0.88	1.72	Restriction	0.331	2
	CUP3	37	0.032	0.18	1.44	Restriction	0.149	1
	CUP4	90	0.046	0.77	2.41	Cap	0.268	2
	CUP5	78	0.609	23.54	5.67	Cap	0.618	2
	CUP7	21	0.172	2.50	3.26	Cap	0.345	1

14.4.6 Variography

Variography analysis of composites within each estimate domain was conducted. The variogram was rotated along strike and down dip of the modeled domains. Radial plots were used to determine the plunge. The total sill was set to the variance of the composite population which was normalized to 1.0. The nugget was set based on the combined evidence from the major (down dip), semi-major (along strike), and minor axis (thickness). The average nugget was approximately 20% for all estimation domains. A spherical model was used to assign the range in all three axis directions. In some cases, composite populations were too low to provide a reliable model fit using pair counts alone. In those cases, and for all variograms, a plot set to 1.5x moving average of the gamma was incorporated to fit the variogram model. Figures 14-20 through 14-23 show modeled variograms for four significant domains across the strike of the Project. Variogram parameters are summarized in Table 14-11.

Overall, the modeled variograms tend to show a low anisotropy along strike and down dip with an average of 1.5 and a maximum of 2.6. The maximum range for the variograms had an average of approximately 140 ft with the longest range being 250 ft. The variography study suggests an appropriate search ellipse for the purpose of estimating gold grades should have a low anisotropy down dip and along strike, and the maximum range should reside between 150 and 200 ft.

Table 14-11 Summary of Variogram Parameters

Zone	Domain	Direction			Nugget	Structure 1				Total Sill
		Dip	Dip Azi.	Pitch		Sill	Major	Semi-major	Minor	
A/B zone	ALW1	30	45	75	0.20	0.80	170	110	5	1.00
	ALW2	20	10	60	0.21	0.79	105	105	5	1.00
	ALW3	30	25	90	0.20	0.80	215	100	5	1.00
	ALW4	30	55	30	0.20	0.80	80	60	5	1.00
	AUP1	30	50	55	0.20	0.80	155	115	5	1.00
	AUP2	30	35	75	0.51	0.50	175	100	5	1.00
	AUP3	40	55	75	0.20	0.80	110	110	10	1.00
	AUP4	25	5	115	0.20	0.81	95	50	10	1.01
	AUP5	20	345	75	0.15	0.85	100	90	10	1.00
	CUVNA	40	50	45	0.10	0.90	180	130	5	1.00
	CUVNB	30	50	65	0.16	0.84	110	95	5	1.00
C zone	CLW1	30	50	105	0.10	0.90	120	80	20	1.00
	CLW2	30	25	155	0.20	0.80	225	200	10	1.01
	CLW3	35	75	135	0.36	0.64	70	70	10	1.00
	CLW4	20	20	105	0.20	0.80	85	65	25	1.00
	CLW5	25	10	120	0.20	0.80	80	85	10	1.00
	CLW6	30	50	70	0.43	0.57	90	70	10	1.00
	CLW7	45	65	60	0.20	0.80	120	80	5	1.00
	CLW8	20	75	15	0.20	0.80	75	45	5	1.00
	CLW9	15	100	100	0.20	0.80	100	50	5	1.00
	CUP6	20	15	90	0.20	0.80	100	60	10	1.00
	CUVN	35	60	45	0.06	0.94	225	205	30	1.00
D zone	CUVND	25	80	65	0.23	0.77	125	160	10	1.00
	CUVND2	25	65	45	0.20	0.80	165	120	10	1.00
	CUVND3	25	65	25	0.20	0.80	205	80	10	1.00
	CUVND4	35	80	55	0.31	0.69	80	70	20	1.00
	CUVND5	20	70	55	0.20	0.80	130	80	5	1.00
	CUVND6	25	75	25	0.20	0.80	105	60	5	1.00
Footwall zone	FW01	80	45	45	0.20	0.80	45	30	5	1.00
	FW02	40	60	110	0.20	0.80	110	75	10	1.00
	FW03	40	70	75	0.17	0.84	185	155	10	1.00
	FW04	40	60	75	0.20	0.80	235	220	10	1.00
	FW05	40	60	45	0.20	0.80	240	165	10	1.00
	FW06	25	60	90	0.20	0.80	235	225	10	1.00
	FW07	35	55	35	0.20	0.80	250	135	10	1.00
	FW08	45	60	75	0.20	0.80	160	125	10	1.00
South zone	S01	45	35	165	0.20	0.80	180	90	24	1.00
	S02	50	40	105	0.18	0.82	140	80	10	1.00
	S03	45	65	135	0.20	0.80	140	100	10	1.00
	S04	40	60	134	0.20	0.80	150	100	10	1.00
	S05	35	60	110	0.20	0.80	165	70	10	1.00
	S06	40	45	25	0.20	0.80	165	80	10	1.00
Upper Fracture zone	CUP1	10	5	160	0.10	0.90	80	80	10	1.00
	CUP2	20	55	105	0.20	0.80	100	100	10	1.00
	CUP3	30	55	70	0.20	0.80	130	90	10	1.00
	CUP4	40	65	65	0.20	0.80	175	105	10	1.00
	CUP5	40	45	115	0.20	0.80	105	70	10	1.00
	CUP7	30	45	20	0.20	0.80	95	65	10	1.00

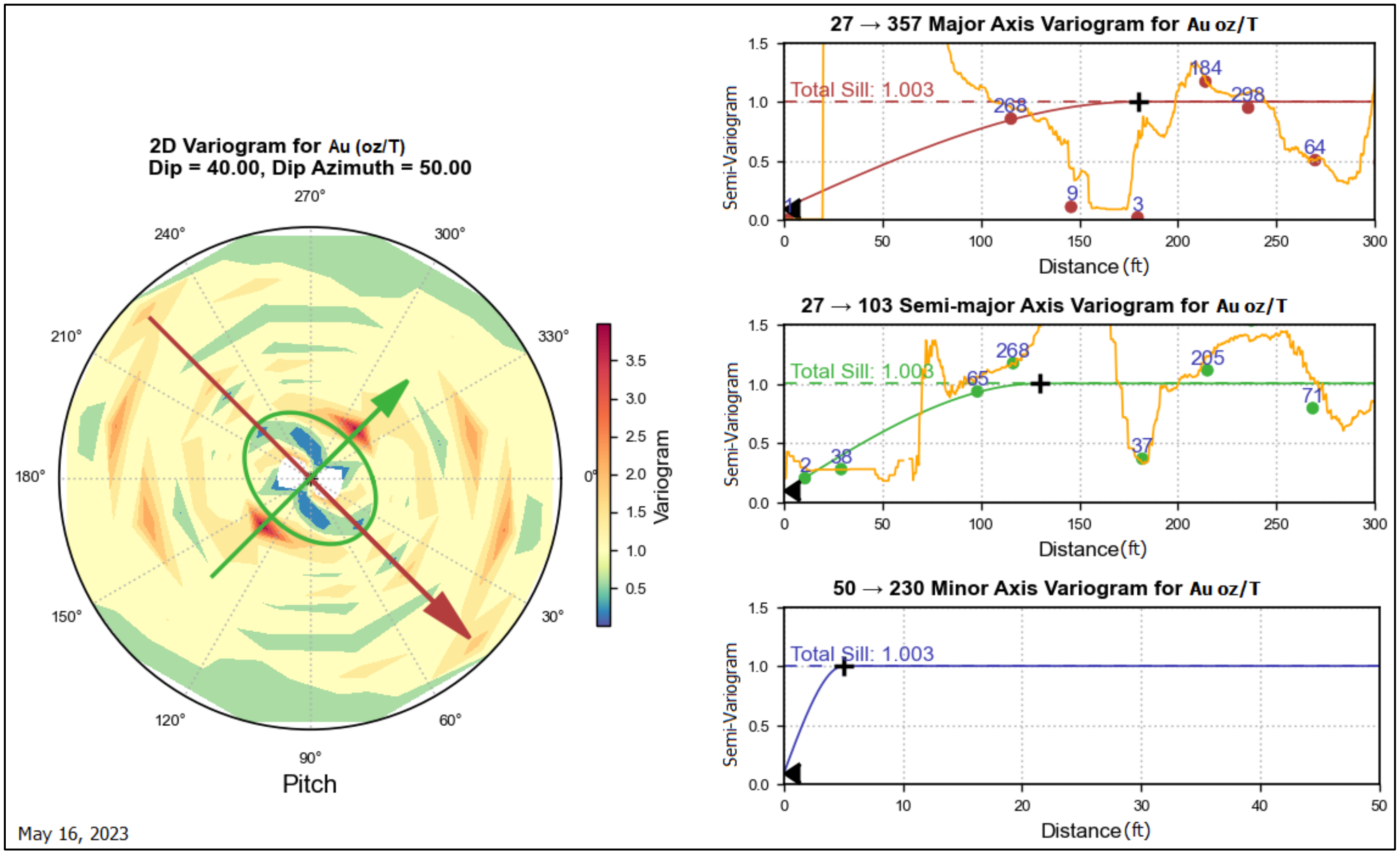

Figure 14-20 Major and Semi-Major Axis Variograms and Radial Plot for CUVNA Domain.

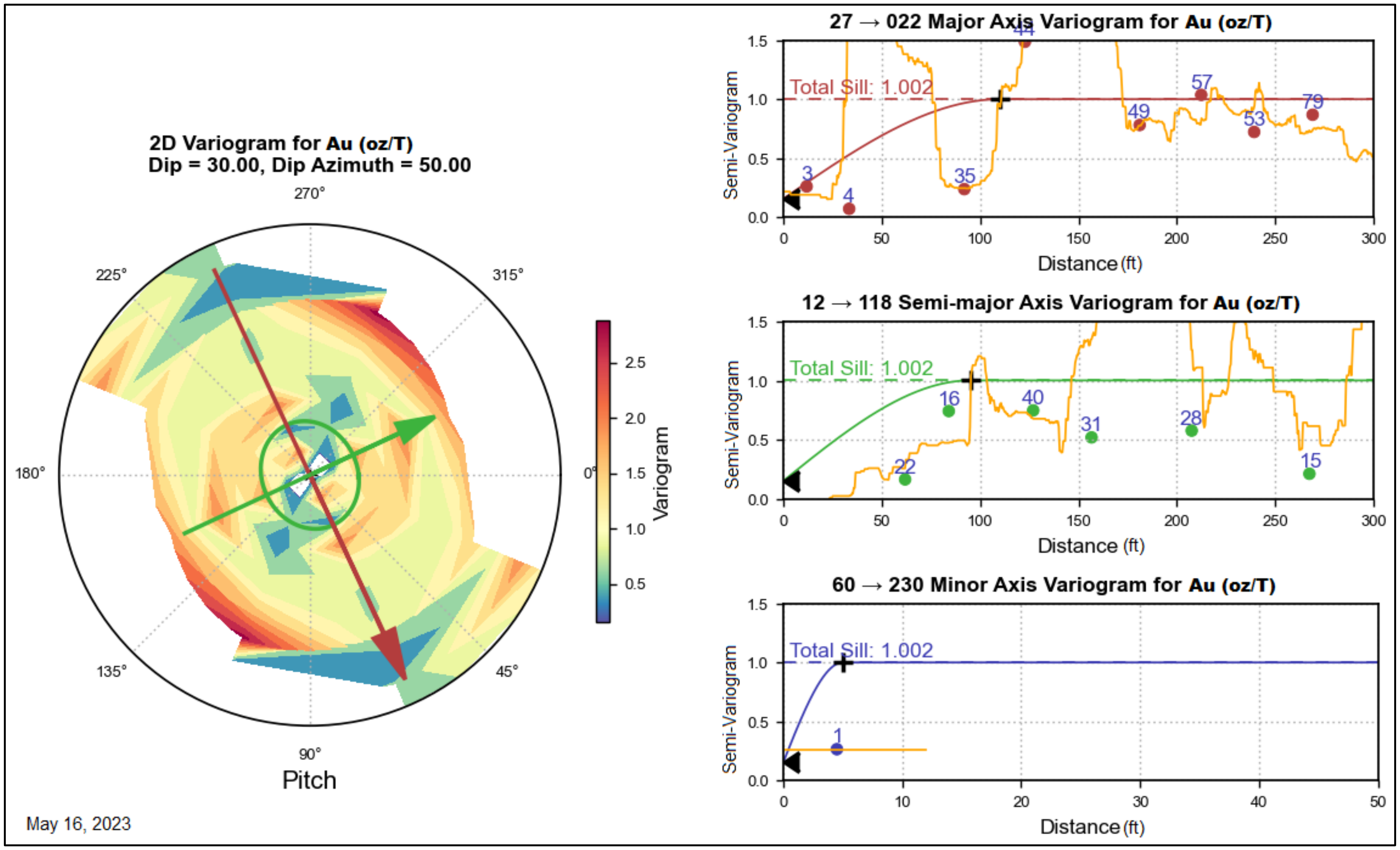

Figure 14-21 Major and Semi Major Axis Variograms and Radial Plot for CUVNB Domain.

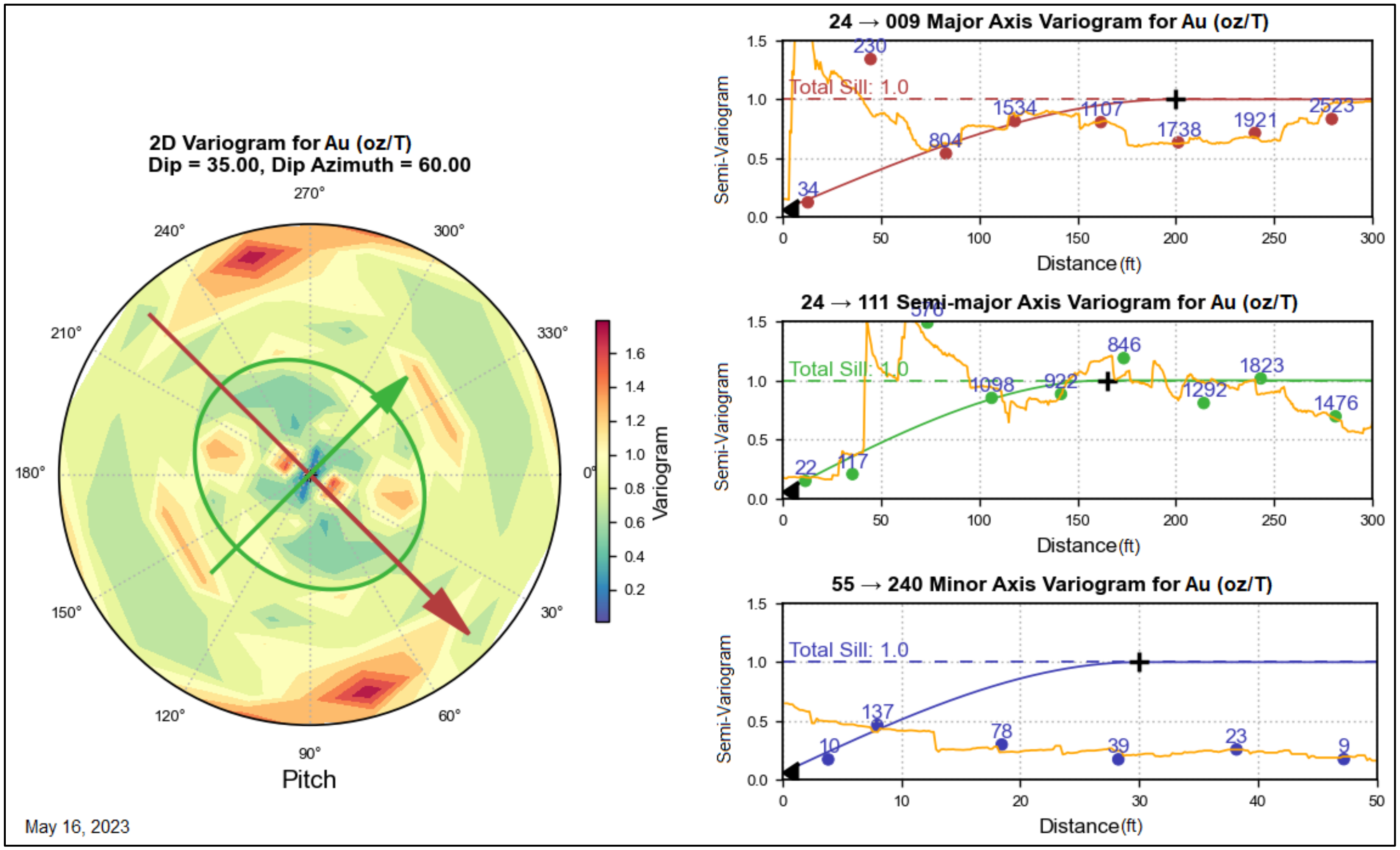

Figure 14-22 Major and Semi Major Axis Variograms and Radial Plot for CUVN Domain.

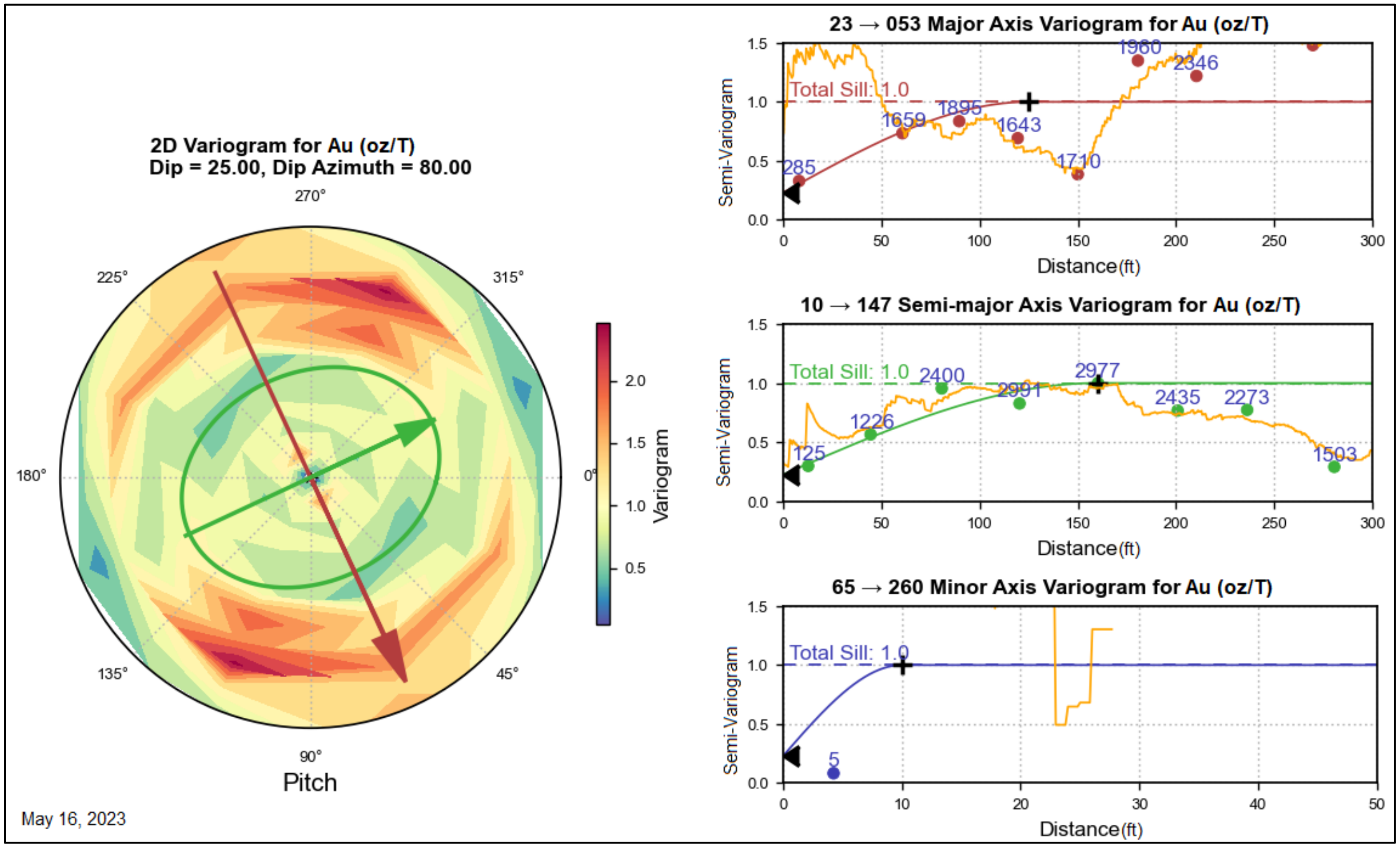

Figure 14-23 Major and Semi Major Axis Variograms and Radial Plot for CUVND Domain.

14.4.7 Estimation Parameters

Gold mineralization for the Project was estimated using a single pass and an ordinary kriging algorithm ("OK"). A variable orientation was selected as the best method to model gold grades allowing for the search ellipse to follow the curvature of modeled domains. The major, and semi-major axis ranges were based on the volume weighted average range of the modeled variograms by zone. Search ellipses could be expanded in order to ensure the majority of the modeled domains received a gold estimate. The anisotropy from the volume weighted average ranges was always preserved. Once the search ellipse ranges were established, the distance limits for the restricted outliers could be determined. Based on visual inspections of gold grade within the model, as well as observations from drilling conducted to date, a restrictive distance of 50% of the volume weighted average variogram range was applied to all zones except the D zone, which restricted the distance of outliers to 33% of the volume weighted average variogram range. The composite selection methodology used a minimum of 1 composite, and a maximum of 5 composites with no more than 2 composites coming from a single drillhole. Due to the clustered drilling by American Bonanza, the methodology was modified in the D zone where an octant search methodology was used ensuring the estimated block incorporated composites from multiple sectors of the search ellipse. The composite selection methodology within the D zone used a minimum of 1 composite, a maximum of 10 composites, with no more than 2 composites coming from a single drillhole, and with no more than 3 composites coming from a single sector. The estimation parameters are summarized in Table 14-12.

Table 14-12 Summary of Estimation Parameters used to Estimate Gold Grades at Copperstone

Zone	Estimate Search Ellipse*			
	Major	Semi-Major	Minor	Anisotropy
A/B zone	200	140	15	1.43
C zone	210	185	15	1.14
D zone	150	120	15	1.25
Footwall zone	200	150	15	1.33
South zone	200	120	15	1.67
Upper Fracture zone	200	130	15	1.54
Zone	**Volume Weighted Average Variogram Range**			
	Major	**Semi-Major**	**Minor**	**Anisotropy**
A/B zone	155	108	5	1.43
C zone	209	186	26	1.12
D zone	126	96	14	1.32
Footwall zone	176	132	10	1.33
South zone	152	91	11	1.68
Upper Fracture zone	137	90	10	1.52
Zone	**Restrictive Distance**			
	% of Variogram	**Major**	**Semi-Major**	**Minor**
A/B zone	50%	77	54	3
C zone	50%	104	93	13
D zone	33%	42	32	4
Footwall zone	50%	88	66	5
South zone	50%	76	45	6
Upper Fracture zone	50%	68	45	5
Zone	**Composite Selection**			
	Min	**Max**	**Max/DH**	**Max/Octant**
A/B zone	1	5	2	N/A
C zone	1	5	2	N/A
D zone	1	10	2	3
Footwall zone	1	5	2	N/A
South zone	1	5	2	N/A
Upper Fracture zone	1	5	2	N/A

* A variable orientation was applied to all domains for the estimate

14.4.8 Validation

Visual and statistical methods were employed to validate the estimate of gold grades within the Project.

14.4.8.1 Visual Inspection

Modeled gold grades were compared against composites in 3D and in cross section. The example sections displaying estimated gold grades are shown in Figures 14-25 through 14-28. The figures show agreement between modeled and composite grades. Modeled blocks display grade continuity along strike and down dip. Figure 14-24 shows an orthographic view of the Copperstone gold estimate and approximate section locations.

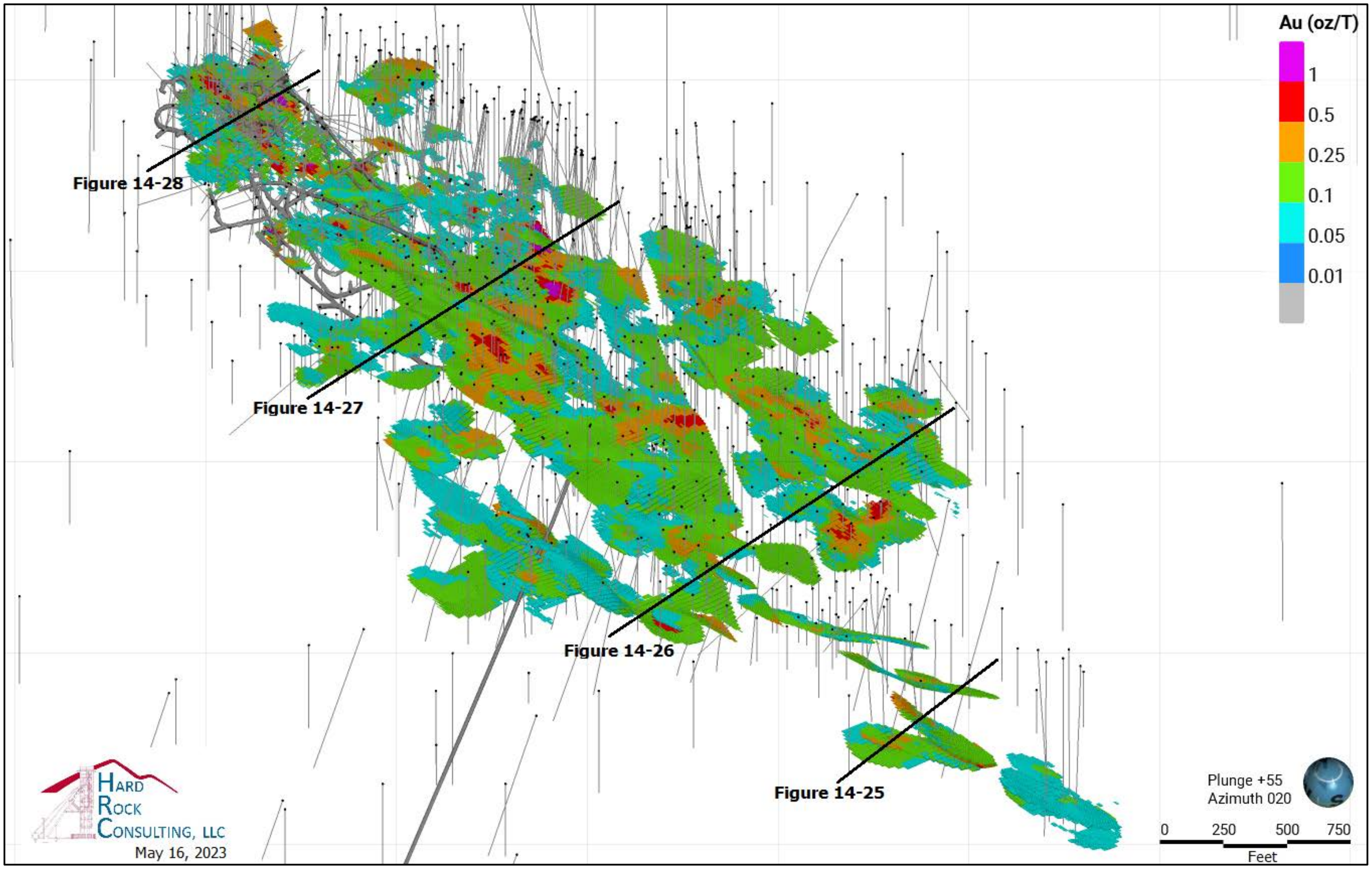

Figure 14-24 Orthographic 3D View of the OK Gold Estimate for the Copperstone Property Showing Gold Grades Above 0.05 oz/ton.

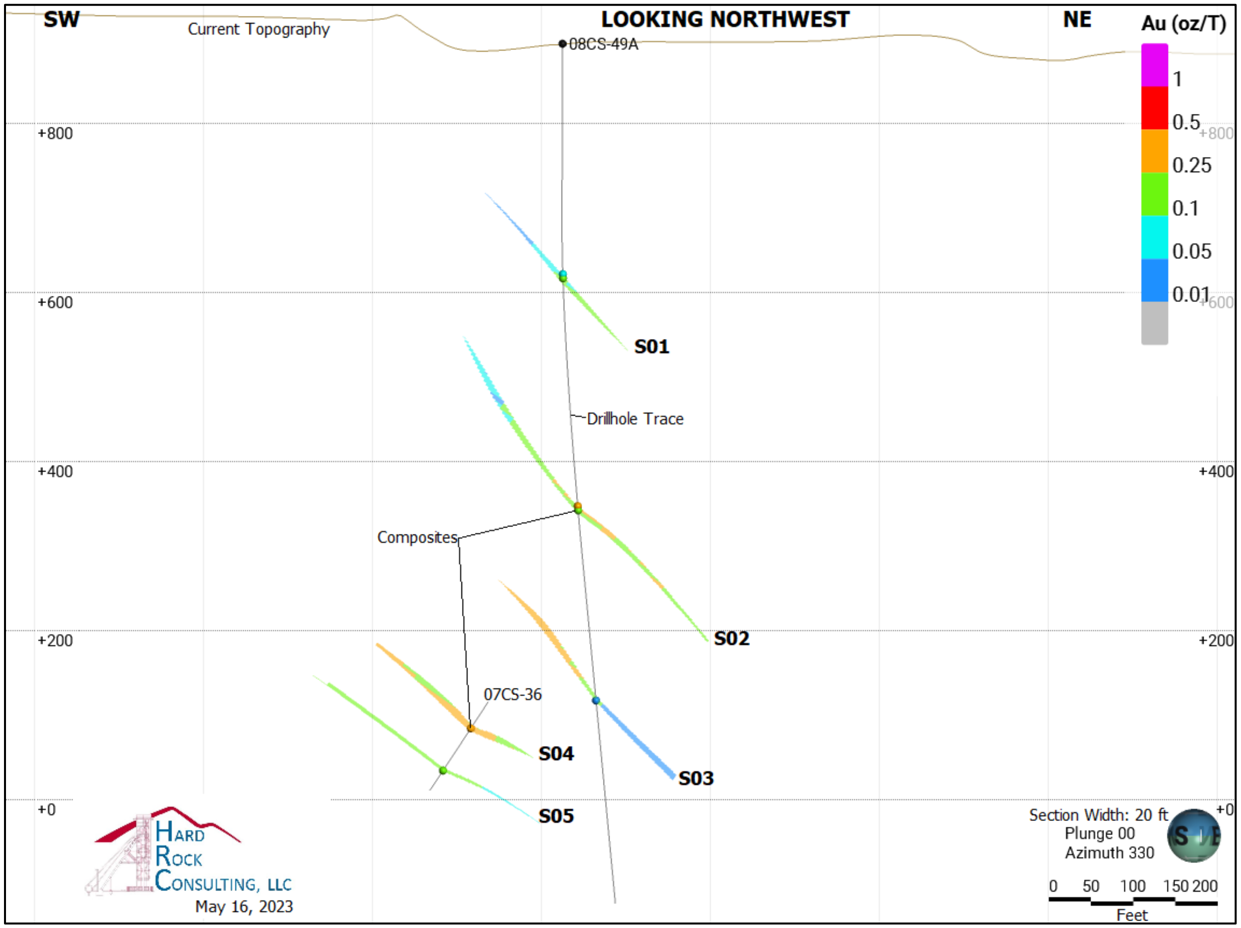

Figure 14-25 Cross Section showing the OK Gold Estimate Grades vs. Composite Gold Grades for the South Zone

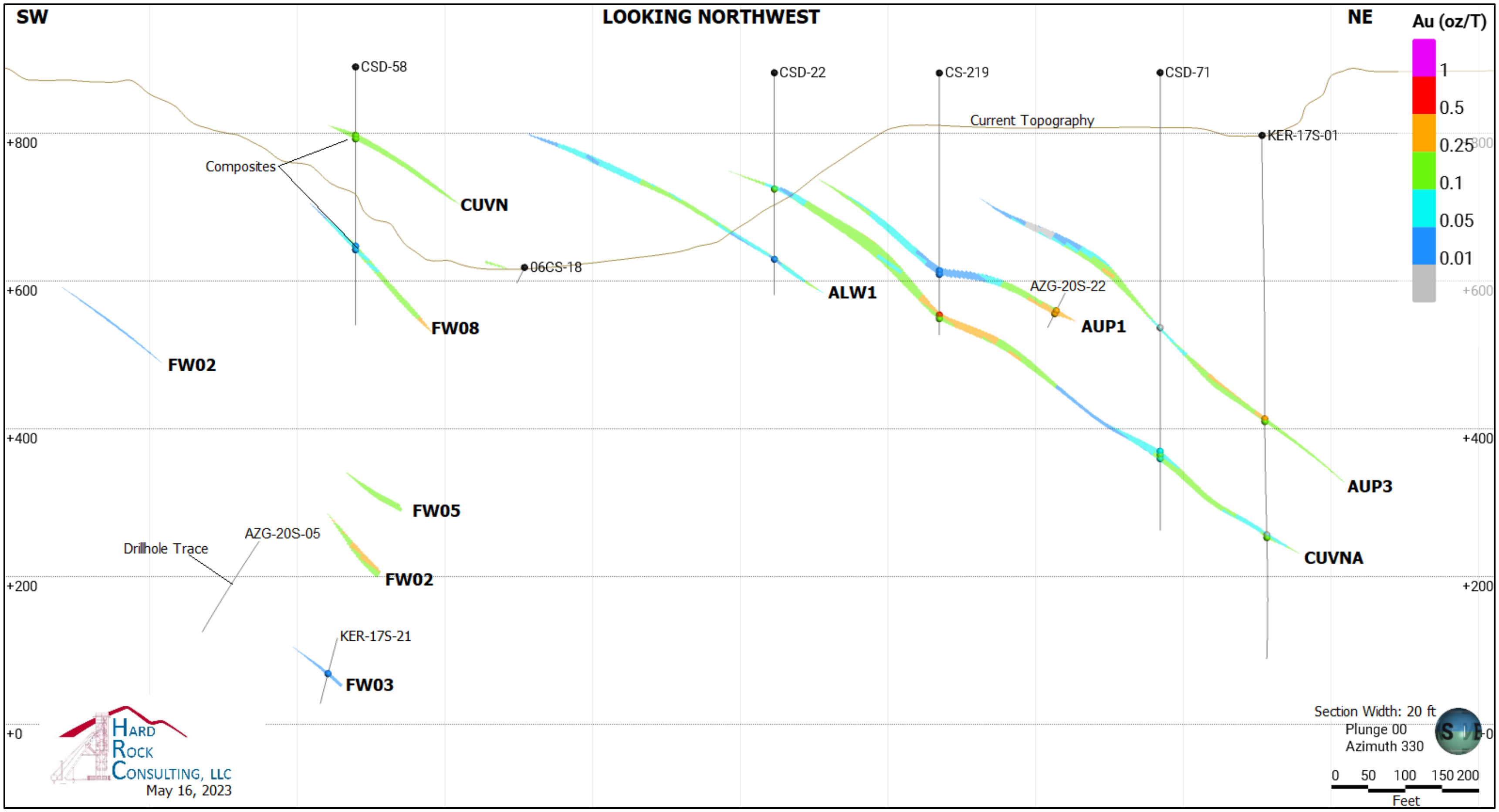

Figure 14-26 Cross Section showing the OK Gold Estimate Grades vs. Composite Gold Grades in the Footwall and A/B Zones

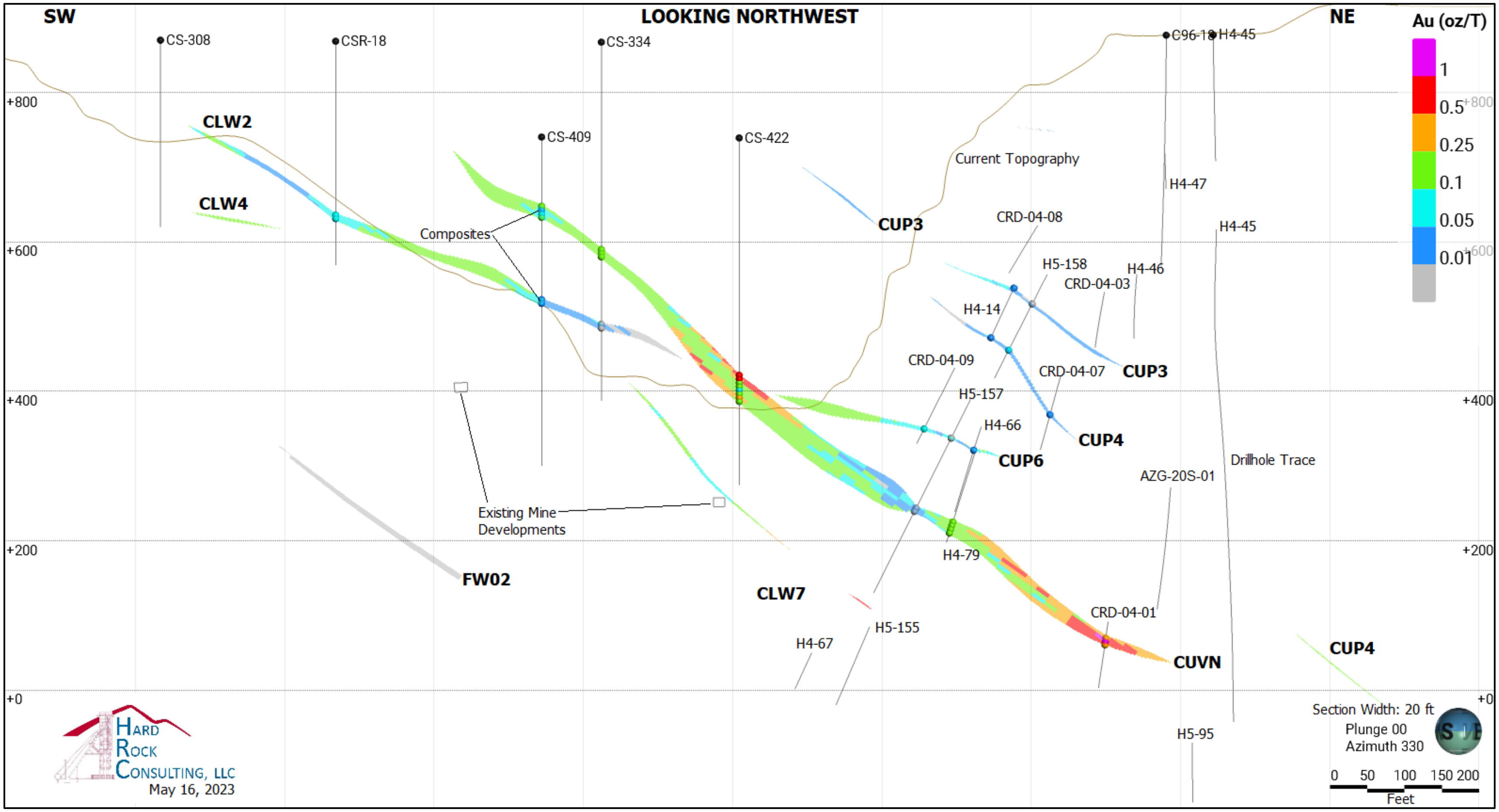

Figure 14-27 Cross Section showing the OK Gold Estimate Grades vs. Composite Gold Grades in the C Zone, Footwall Zone, and Upper Fracture Zone

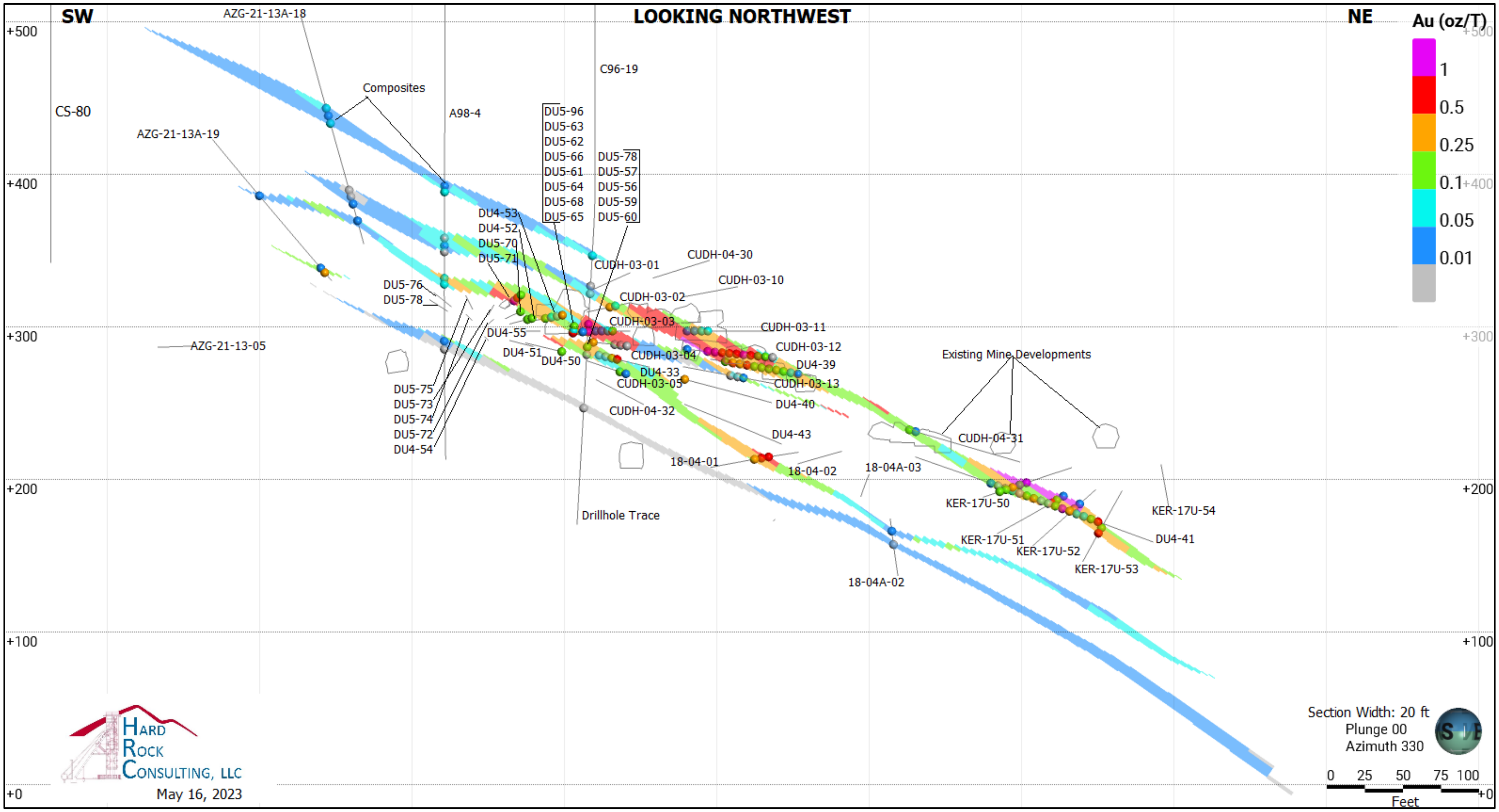

Figure 14-28 Cross Section showing the OK Gold Estimate Grades vs. Composite Gold Grades in the D Zone

14.4.8.2 Statistical Validation

Inverse distance 2.5 ("ID") and nearest neighbor ("NN") models were generated for comparison to the ordinary krige ("OK") estimate results. Descriptive statistics for all estimation results and drillhole composites for all estimated domains ("Global") are reported in Table 14-13. Descriptive statistics for all estimation results and drillhole composites by domain are reported in Tables 14-14 through 14-19. Statistics for the composites are weighted by length and use capped gold grades. Block model statistics are weighted by volume. Note that due to NN methodology, block counts between the NN, and OK and ID models may differ. Also, the OK model reports negative grades in the estimate. Negative grades in OK models are the result of two or more composites within close proximity having significantly different grades resulting in a negative weight assignment. Only 148 blocks (0.04%) were assigned a negative grade and do not significantly impact the estimate of gold grades. The overall reduction of the maximum and standard deviation within the OK and ID models represents an appropriate amount of smoothing to account for the point to block volume variance relationship while maintaining similar means.

Table 14-13 Global Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
Global	Comp	3,796	18,852	0.161	0.437	2.71	0.00009	0.07	13.54
Global	NN	401,302	91,515,375	0.131	0.254	1.94	0.00009	0.08	13.54
Global	ID	421,570	95,079,813	0.126	0.149	1.19	0.00009	0.09	8.42
Global	OK	421,570	95,079,813	0.128	0.138	1.07	-0.07620	0.10	5.49

Table 14-14 A/B Zone Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
CUVNA	Comp	167	833	0.181	0.146	0.81	0.0004	0.130	0.87
	NN	20,701	6,451,688	0.187	0.156	0.83	0.0004	0.130	0.87
	ID	20,834	6,468,625	0.182	0.107	0.59	0.0021	0.160	0.83
	OK	20,834	6,468,625	0.187	0.100	0.53	0.0051	0.168	0.68
CUVNB	Comp	67	335	0.161	0.184	1.14	0.0001	0.105	0.85
	NN	8,267	2,063,750	0.177	0.171	0.96	0.0001	0.142	0.85
	ID	8,503	2,094,563	0.162	0.110	0.68	0.0040	0.142	0.77
	OK	8,503	2,094,563	0.162	0.092	0.56	0.0089	0.149	0.67
ALW1	Comp	64	320	0.139	0.202	1.46	0.0001	0.059	1.06
	NN	13,558	2,946,625	0.140	0.191	1.36	0.0001	0.073	1.06
	ID	13,884	2,983,875	0.140	0.148	1.06	0.0011	0.098	1.00
	OK	13,884	2,983,875	0.142	0.127	0.89	0.0031	0.108	0.83
ALW2	Comp	8	40	0.153	0.059	0.39	0.1050	0.132	0.26
	NN	1,604	239,000	0.138	0.040	0.29	0.1050	0.132	0.26
	ID	1,604	239,000	0.145	0.031	0.21	0.1059	0.135	0.26
	OK	1,604	239,000	0.141	0.018	0.13	0.1145	0.140	0.23
ALW3	Comp	38	186	0.125	0.171	1.37	0.0001	0.080	0.84
	NN	4,052	1,126,188	0.097	0.135	1.39	0.0001	0.026	0.84
	ID	4,053	1,126,250	0.109	0.114	1.05	0.0002	0.087	0.84
	OK	4,053	1,126,250	0.104	0.089	0.86	0.0004	0.092	0.47
ALW4	Comp	35	172	0.097	0.120	1.24	0.0001	0.089	0.60
	NN	6,055	1,253,188	0.098	0.101	1.03	0.0001	0.103	0.60
	ID	6,133	1,264,750	0.099	0.075	0.75	0.0001	0.103	0.48
	OK	6,133	1,264,750	0.100	0.069	0.68	0.0001	0.098	0.33
AUP1	Comp	39	195	0.120	0.127	1.06	0.0100	0.071	0.44
	NN	5,794	1,214,063	0.105	0.120	1.15	0.0100	0.062	0.44
	ID	5,794	1,214,063	0.105	0.086	0.82	0.0102	0.073	0.42
	OK	5,794	1,214,063	0.107	0.072	0.67	0.0106	0.089	0.39
AUP2	Comp	42	208	0.141	0.087	0.62	0.0095	0.101	0.52
	NN	3,746	1,383,250	0.135	0.059	0.43	0.0095	0.126	0.52
	ID	3,866	1,400,938	0.143	0.050	0.35	0.0154	0.129	0.50
	OK	3,866	1,400,938	0.141	0.041	0.29	0.0449	0.129	0.27
AUP3	Comp	30	144	0.130	0.139	1.07	0.0001	0.115	0.39
	NN	4,387	880,250	0.150	0.146	0.97	0.0001	0.121	0.39
	ID	4,387	880,250	0.119	0.090	0.76	0.0046	0.104	0.39
	OK	4,387	880,250	0.125	0.082	0.65	0.0049	0.119	0.36
AUP4	Comp	27	138	0.190	0.257	1.35	0.0030	0.090	0.82
	NN	2,624	535,000	0.190	0.247	1.30	0.0030	0.102	0.82
	ID	2,652	537,563	0.186	0.167	0.90	0.0095	0.118	0.81
	OK	2,652	537,563	0.200	0.126	0.63	0.0100	0.172	0.57
AUP5	Comp	25	125	0.054	0.095	1.77	0.0001	0.006	0.39
	NN	2,509	398,000	0.046	0.087	1.89	0.0001	0.003	0.39
	ID	2,509	398,000	0.042	0.058	1.36	0.0003	0.018	0.28
	OK	2,509	398,000	0.038	0.047	1.23	0.0007	0.021	0.21

Table 14-15 C Zone Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
CUVN	Comp	785	3,877	0.188	0.239	1.27	0.0001	0.131	3.68
	NN	81,527	25,211,063	0.186	0.234	1.26	0.0001	0.128	3.68
	ID	81,602	25,223,750	0.186	0.153	0.82	0.0001	0.153	3.40
	OK	81,602	25,223,750	0.188	0.146	0.78	-0.0301	0.156	2.68
CLW1	Comp	142	711	0.161	0.314	1.95	0.0001	0.082	2.90
	NN	16,994	3,258,438	0.146	0.230	1.57	0.0001	0.076	2.90
	ID	16,994	3,258,438	0.142	0.135	0.95	0.0002	0.111	1.35
	OK	16,994	3,258,438	0.140	0.119	0.85	-0.0004	0.118	1.52
CLW2	Comp	112	565	0.079	0.095	1.21	0.0001	0.045	0.47
	NN	14,578	3,457,438	0.082	0.098	1.19	0.0001	0.050	0.47
	ID	14,891	3,497,250	0.071	0.059	0.83	0.0009	0.056	0.43
	OK	14,891	3,497,250	0.073	0.051	0.70	0.0021	0.062	0.29
CLW3	Comp	9	45	0.087	0.070	0.80	0.0030	0.109	0.20
	NN	1,516	204,938	0.087	0.064	0.73	0.0030	0.109	0.20
	ID	1,516	204,938	0.092	0.050	0.54	0.0039	0.110	0.20
	OK	1,516	204,938	0.085	0.034	0.40	0.0257	0.088	0.14
CLW4	Comp	13	60	0.187	0.134	0.72	0.0001	0.168	0.47
	NN	1,090	174,813	0.170	0.133	0.78	0.0001	0.153	0.47
	ID	1,090	174,813	0.160	0.083	0.52	0.0111	0.156	0.35
	OK	1,090	174,813	0.169	0.059	0.35	0.0570	0.170	0.31
CLW5	Comp	7	30	0.100	0.178	1.78	0.0030	0.056	0.51
	NN	1,298	149,563	0.137	0.195	1.42	0.0030	0.056	0.51
	ID	1,298	149,563	0.128	0.123	0.96	0.0050	0.074	0.50
	OK	1,298	149,563	0.108	0.071	0.66	0.0229	0.083	0.32
CLW6	Comp	9	45	0.181	0.068	0.37	0.0950	0.185	0.26
	NN	919	182,250	0.178	0.064	0.36	0.0950	0.185	0.26
	ID	919	182,250	0.179	0.047	0.27	0.1055	0.167	0.26
	OK	919	182,250	0.174	0.030	0.17	0.1254	0.166	0.26
CLW7	Comp	8	41	0.314	0.399	1.27	0.0070	0.232	1.09
	NN	2,108	230,750	0.219	0.266	1.21	0.0070	0.018	1.09
	ID	2,157	235,000	0.193	0.225	1.17	0.0083	0.103	1.08
	OK	2,157	235,000	0.194	0.208	1.07	0.0125	0.118	0.94
CLW8	Comp	10	51	0.079	0.096	1.22	0.0013	0.064	0.28
	NN	972	131,000	0.065	0.075	1.15	0.0013	0.069	0.28
	ID	972	131,000	0.069	0.060	0.87	0.0017	0.059	0.28
	OK	972	131,000	0.068	0.043	0.64	0.0038	0.056	0.26
CLW9	Comp	25	121	0.104	0.142	1.37	0.0001	0.040	0.50
	NN	1,080	127,688	0.088	0.135	1.53	0.0001	0.015	0.50
	ID	1,080	127,688	0.086	0.097	1.12	0.0019	0.051	0.44
	OK	1,080	127,688	0.090	0.078	0.86	0.0033	0.056	0.34
CUP6	Comp	19	94	0.112	0.096	0.86	0.0001	0.105	0.31
	NN	1,929	362,813	0.142	0.091	0.64	0.0001	0.120	0.31
	ID	1,929	362,813	0.123	0.056	0.46	0.0007	0.132	0.31
	OK	1,929	362,813	0.124	0.044	0.35	0.0041	0.130	0.24

Table 14-16 D Zone Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
CUVND	Comp	387	1,938	0.313	0.886	2.83	0.0004	0.099	13.54
	NN	7,378	1,543,875	0.293	0.996	3.40	0.0004	0.088	13.54
	ID	7,378	1,543,875	0.224	0.423	1.89	0.0028	0.141	6.16
	OK	7,378	1,543,875	0.244	0.392	1.61	-0.0281	0.155	5.49
CUVND2	Comp	351	1,747	0.176	0.734	4.16	0.0001	0.016	12.01
	NN	5,915	1,093,125	0.226	1.012	4.48	0.0001	0.019	12.01
	ID	5,915	1,093,125	0.142	0.425	3.00	0.0018	0.062	8.42
	OK	5,915	1,093,125	0.159	0.313	1.97	-0.0012	0.081	4.10
CUVND3	Comp	374	1,852	0.173	0.457	2.64	0.0001	0.024	4.85
	NN	7,675	1,384,188	0.158	0.427	2.71	0.0001	0.030	4.85
	ID	7,675	1,384,188	0.141	0.242	1.72	0.0007	0.067	2.89
	OK	7,675	1,384,188	0.158	0.222	1.40	-0.0762	0.091	2.27
CUVND4	Comp	208	1,033	0.080	0.395	4.91	0.0001	0.003	3.75
	NN	11,202	2,139,750	0.043	0.228	5.29	0.0001	0.003	3.75
	ID	11,685	2,245,125	0.045	0.116	2.61	0.0001	0.008	1.57
	OK	11,685	2,245,125	0.051	0.122	2.41	0.0001	0.011	1.72
CUVND5	Comp	31	139	0.164	0.569	3.47	0.0002	0.014	2.94
	NN	2,564	410,688	0.139	0.515	3.69	0.0002	0.015	2.94
	ID	2,567	411,000	0.097	0.304	3.14	0.0003	0.015	2.78
	OK	2,567	411,000	0.089	0.209	2.34	0.0013	0.015	1.47
CUVND6	Comp	21	102	0.034	0.072	2.13	0.0001	0.010	0.43
	NN	2,157	421,938	0.030	0.040	1.33	0.0002	0.024	0.37
	ID	2,151	421,563	0.032	0.068	2.09	0.0001	0.008	0.43
	OK	2,157	421,938	0.029	0.024	0.85	0.0021	0.026	0.25

Table 14-17 Footwall Zone Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
FW01	Comp	13	64	0.288	0.538	1.87	0.0020	0.009	1.50
	NN	141	12,750	0.261	0.434	1.66	0.0020	0.008	1.50
	ID	141	12,625	0.208	0.303	1.46	0.0020	0.062	1.38
	OK	141	12,625	0.199	0.281	1.41	0.0020	0.057	1.38
FW02	Comp	102	520	0.089	0.149	1.67	0.0001	0.011	0.59
	NN	36,840	7,510,688	0.062	0.116	1.86	0.0001	0.011	0.59
	ID	45,742	9,163,438	0.064	0.096	1.50	0.0001	0.021	0.59
	OK	45,742	9,163,438	0.068	0.094	1.37	0.0001	0.022	0.59
FW03	Comp	40	192	0.068	0.100	1.46	0.0001	0.045	0.52
	NN	13,891	3,140,750	0.074	0.085	1.15	0.0001	0.062	0.52
	ID	15,494	3,403,375	0.081	0.083	1.03	0.0002	0.070	0.52
	OK	15,494	3,403,375	0.083	0.083	1.00	0.0004	0.072	0.52
FW04	Comp	17	86	0.057	0.080	1.41	0.0001	0.018	0.22
	NN	11,838	2,296,813	0.050	0.073	1.46	0.0001	0.018	0.22
	ID	13,702	2,606,063	0.044	0.051	1.17	0.0001	0.027	0.22
	OK	13,702	2,606,063	0.045	0.049	1.09	0.0001	0.027	0.22
FW05	Comp	65	328	0.038	0.069	1.80	0.0001	0.005	0.32
	NN	16,504	3,472,813	0.044	0.062	1.41	0.0001	0.012	0.32
	ID	19,522	4,041,313	0.053	0.057	1.07	0.0001	0.029	0.32
	OK	19,522	4,041,313	0.055	0.053	0.96	0.0001	0.039	0.26
FW06	Comp	7	33	0.158	0.126	0.80	0.0070	0.148	0.40
	NN	7,782	1,182,625	0.130	0.098	0.75	0.0070	0.112	0.40
	ID	7,799	1,185,750	0.140	0.082	0.58	0.0069	0.125	0.38
	OK	7,799	1,185,750	0.141	0.080	0.57	0.0070	0.124	0.34
FW07	Comp	36	182	0.066	0.079	1.20	0.0001	0.042	0.38
	NN	7,239	1,468,563	0.055	0.086	1.55	0.0001	0.027	0.38
	ID	7,927	1,578,313	0.049	0.059	1.20	0.0001	0.035	0.35
	OK	7,927	1,578,313	0.050	0.051	1.03	0.0001	0.039	0.28
FW08	Comp	9	47	0.147	0.144	0.98	0.0001	0.202	0.37
	NN	2,656	414,438	0.099	0.138	1.40	0.0001	0.014	0.37
	ID	2,656	414,438	0.091	0.092	1.01	0.0001	0.051	0.36
	OK	2,656	414,438	0.105	0.075	0.71	0.0001	0.102	0.33

Table 14-18 South Zone Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
S01	Comp	58	289	0.099	0.099	1.00	0.0001	0.077	0.37
	NN	4,719	1,064,688	0.100	0.093	0.93	0.0001	0.081	0.37
	ID	4,750	1,067,438	0.101	0.066	0.66	0.0008	0.094	0.35
	OK	4,750	1,067,438	0.099	0.058	0.58	-0.0008	0.093	0.31
S02	Comp	24	117	0.132	0.121	0.91	0.0001	0.114	0.45
	NN	5,496	914,438	0.142	0.145	1.02	0.0001	0.123	0.45
	ID	5,956	968,313	0.142	0.088	0.62	0.0001	0.136	0.45
	OK	5,956	968,313	0.142	0.074	0.52	0.0001	0.136	0.45
S03	Comp	22	113	0.150	0.213	1.42	0.0010	0.067	0.76
	NN	8,929	2,521,188	0.132	0.181	1.37	0.0010	0.067	0.76
	ID	9,358	2,622,938	0.150	0.150	1.00	0.0010	0.067	0.76
	OK	9,358	2,622,938	0.155	0.149	0.96	0.0010	0.068	0.76
S04	Comp	24	121	0.117	0.131	1.12	0.0001	0.064	0.42
	NN	6,871	1,766,938	0.100	0.105	1.05	0.0001	0.064	0.42
	ID	7,551	1,871,938	0.114	0.102	0.89	0.0004	0.071	0.42
	OK	7,551	1,871,938	0.108	0.085	0.79	0.0018	0.088	0.42
S05	Comp	7	31	0.056	0.051	0.91	0.0001	0.054	0.15
	NN	6,227	892,813	0.055	0.047	0.86	0.0001	0.026	0.15
	ID	6,801	989,313	0.058	0.043	0.75	0.0001	0.043	0.15
	OK	6,801	989,313	0.057	0.042	0.73	0.0001	0.042	0.15
S06	Comp	6	29	0.071	0.097	1.37	0.0020	0.044	0.26
	NN	3,644	583,938	0.034	0.054	1.58	0.0020	0.007	0.26
	ID	3,686	590,125	0.039	0.041	1.06	0.0020	0.034	0.25
	OK	3,686	590,125	0.038	0.031	0.81	0.0020	0.035	0.20

Table 14-19 Upper Fracture Zone Descriptive Statistics Comparing Composite Gold Grades against Modeled Gold Grades

Domain	Name	Count	Length, Volume	Mean	Std. Dev.	CV	Minimum	Median	Maximum
CUP1	Comp	9	45	0.037	0.067	1.80	0.0030	0.003	0.16
	NN	394	41,938	0.035	0.062	1.78	0.0030	0.003	0.16
	ID	394	41,938	0.036	0.027	0.76	0.0031	0.032	0.14
	OK	394	41,938	0.033	0.020	0.60	0.0026	0.029	0.10
CUP2	Comp	19	96	0.137	0.236	1.72	0.0001	0.027	0.88
	NN	3,428	506,000	0.149	0.218	1.47	0.0001	0.030	0.88
	ID	3,428	505,875	0.134	0.139	1.04	0.0026	0.077	0.82
	OK	3,428	505,875	0.127	0.104	0.82	0.0030	0.098	0.52
CUP3	Comp	37	184	0.032	0.047	1.44	0.0001	0.011	0.18
	NN	4,542	673,813	0.037	0.049	1.33	0.0001	0.011	0.18
	ID	4,542	673,813	0.033	0.032	0.97	0.0008	0.023	0.18
	OK	4,542	673,813	0.033	0.025	0.75	0.0028	0.029	0.13
CUP4	Comp	90	448	0.038	0.071	1.87	0.0001	0.003	0.27
	NN	13,979	2,185,688	0.050	0.080	1.62	0.0001	0.006	0.27
	ID	14,087	2,198,563	0.048	0.064	1.33	0.0002	0.015	0.26
	OK	14,087	2,198,563	0.049	0.059	1.21	0.0015	0.022	0.24
CUP5	Comp	78	386	0.067	0.143	2.12	0.0001	0.009	0.62
	NN	10,351	1,646,000	0.064	0.126	1.96	0.0001	0.019	0.62
	ID	10,351	1,646,000	0.062	0.102	1.65	0.0018	0.022	0.62
	OK	10,351	1,646,000	0.062	0.091	1.47	0.0022	0.024	0.62
CUP7	Comp	21	98	0.062	0.113	1.82	0.0020	0.010	0.35
	NN	1,626	242,813	0.056	0.102	1.82	0.0020	0.009	0.35
	ID	1,626	242,813	0.060	0.066	1.11	0.0023	0.034	0.34
	OK	1,626	242,813	0.057	0.050	0.88	0.0037	0.045	0.27

Swath plots were generated to compare average estimated gold grade from the OK method to the two validation model methods (ID and NN). The results from the OK model, plus those for the validation ID model method are compared using the swath plot to the distribution derived from the NN model.

Three swath plots of gold grades were generated and reviewed for each domain. Swath plots for gold are presented for all estimated domains in the following figures: Figure 14-29 shows average gold grade from northwest to southeast along strike; Figure 14-30 shows average gold grade from southwest to northeast down dip, and Figure 14-31 shows average gold grade from top to bottom down thickness. All swath plots show gold grades in oz/ton for NN (Red line), ID (Blue line), and OK (Green line) models. Red and green bars show the block model volume for the NN and OK models respectively.

On a local scale, the nearest neighbor model does not provide a reliable estimate of grades. On a much larger scale, it represents an unbiased estimation of the grade distribution based on the total data set. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the distribution of grade from the nearest neighbor.

Correlation between the grade estimation methods appears reasonable. Variation between model estimates increases near model edges and is a result of lower drilling density.

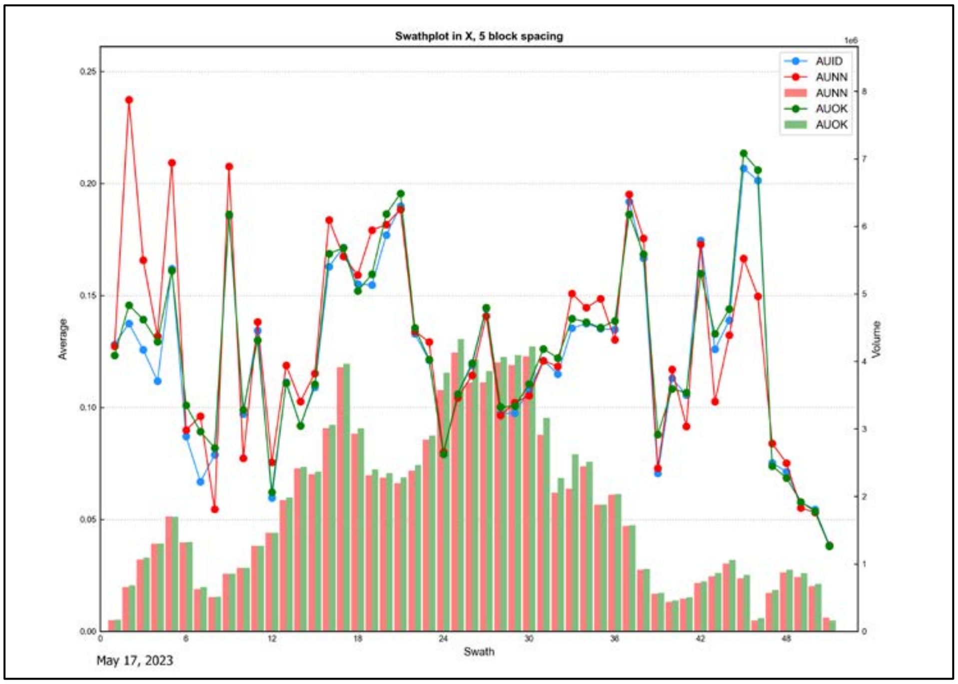

Figure 14-29 Swath Plot from Northwest to Southeast (Along Strike) for All Estimated Domains

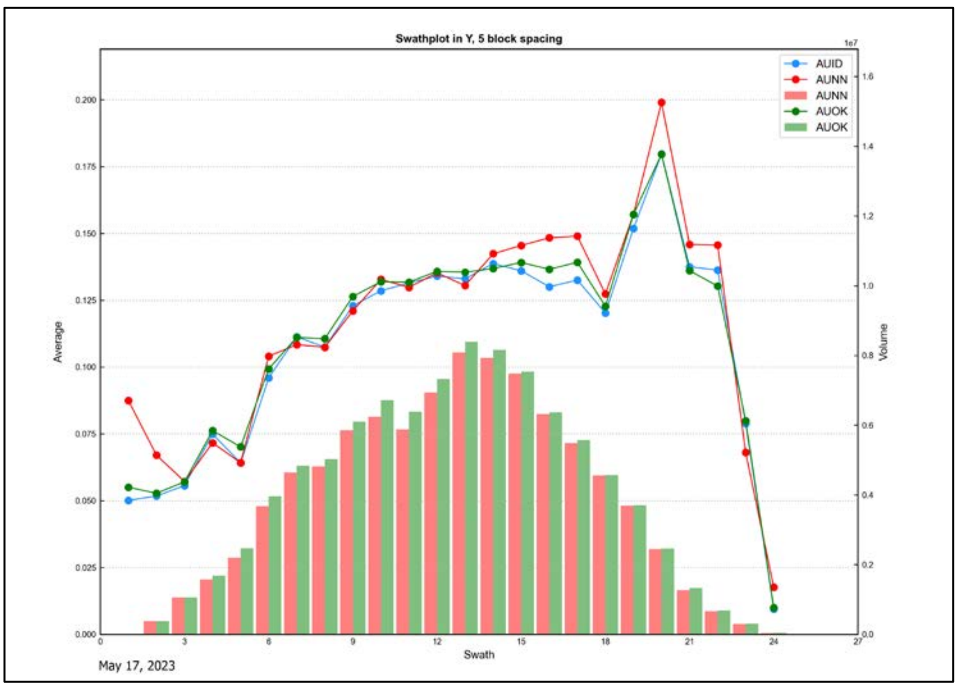

Figure 14-30 Swath Plot from Southwest to Northeast (Down Dip) for All Estimated Domains

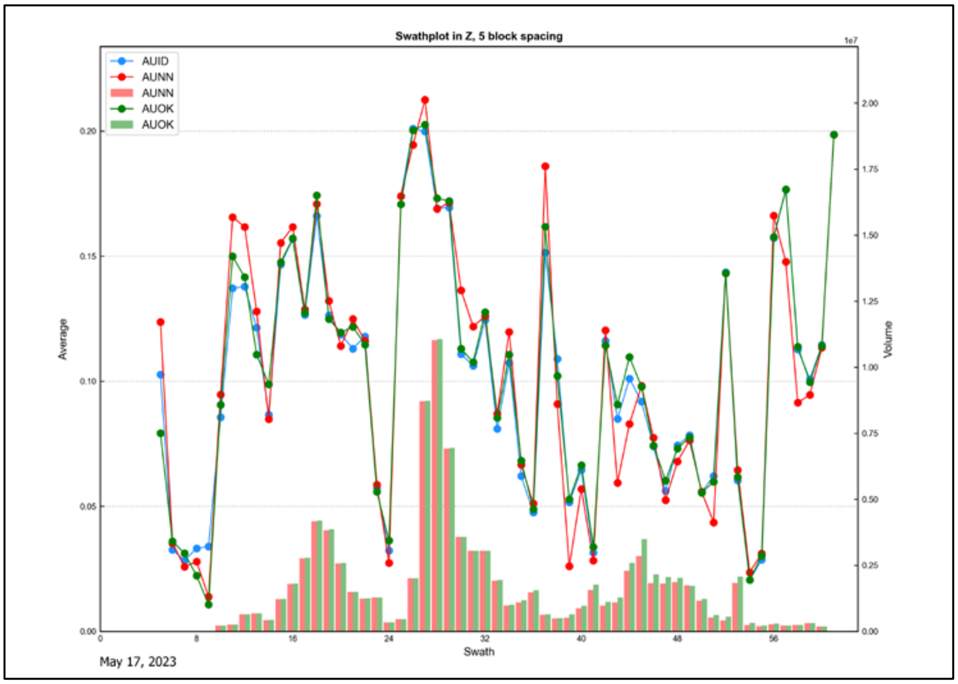

Figure 14-31 Swath Plot from Top to Bottom (Down Thickness) for All Estimated Domains

14.4.9 Density

Two hundred fifty-four density measurements from core drillholes along the strike length of the deposit were used to determine density for blocks inside, and outside estimation domains by stratigraphic unit. Length weighted means were used to assign densities to blocks by lithology inside estimation domains and outside estimation domains. Quaternary alluvium was assigned a density based on average density of packed sand (AMEC, 2006). Basalt was assigned the same density as quartz latite porphyry. Table 14-20 summarizes the densities applied to the block model.

Table 14-20 Summary of Densities Applied to Blocks Within the Copperstone Model

Lithology	Count	SG				tn (shrt)/ft³
		Min	Max	Mean	Length Weighted Mean	Density
Quaternary Alluvium						0.054
Basalt						0.084
Ironstone (Inside Estimation Domains)	42	2.31	4.67	3.39	3.35	0.105
Ironstone	14	2.02	4.47	2.89	2.91	0.091
Metasediments/Schist (Inside Estimation Domains)	16	2.32	4.08	2.88	2.84	0.089
Metasediments/Schist	39	2.36	3.67	2.65	2.65	0.083
Quartz Latite Porphyry (Inside Estimation Domains)	35	2.14	3.67	2.73	2.68	0.084
Quartz Latite Porphyry	108	0.94	4.35	2.68	2.68	0.084

14.4.10 Mineral Resource Classification

Mineral resources reported here are classified as Measured, Indicated and Inferred according to CIM standards adopted by CIM Counsel on May 10, 2014. The QP classified the mineral resources as Measured, Indicated, and Inferred using the minimum distance from the nearest composite, the number of composites used to estimate a block, and the spatial location and confidence of the estimated domains.

All estimated gold grades within domains in the Footwall, South, Upper Fracture zones, as well as nine domains in the A/B, C and D zones were assigned a classification of Inferred because of limited drilling across the strike length and down dip for the domains, a larger spatial separation from other domains, and/or limited, but bourgeoning understanding of the structural controls on the domains. The domain CLW9 was allowed Indicated and Inferred classifications for similar reasons as stated above. The remaining domains were allowed Measured, Indicated, and Inferred classifications. Table 14-21 summarizes the allowable classification by domain.

Table 14-21 Classification Allowance by Estimation Domain

Zone	Domain	Allowable Classification	Zone	Domain	Allowable Classification
A/B zone	ALW1	Measured, Indicated, & Inferred	Footwall zone	FW01	Inferred Only
	ALW2	Inferred Only		FW02	Inferred Only
	ALW3	Measured, Indicated, & Inferred		FW03	Inferred Only
	ALW4	Measured, Indicated, & Inferred		FW04	Inferred Only
	AUP1	Measured, Indicated, & Inferred		FW05	Inferred Only
	AUP2	Measured, Indicated, & Inferred		FW06	Inferred Only
	AUP3	Measured, Indicated, & Inferred		FW07	Inferred Only
	AUP4	Measured, Indicated, & Inferred		FW08	Inferred Only
	AUP5	Measured, Indicated, & Inferred	South zone	S01	Inferred Only
	CUVNA	Measured, Indicated, & Inferred		S02	Inferred Only
	CUVNB	Measured, Indicated, & Inferred		S03	Inferred Only
C zone	CLW1	Measured, Indicated, & Inferred		S04	Inferred Only
	CLW2	Measured, Indicated, & Inferred		S05	Inferred Only
	CLW3	Inferred Only		S06	Inferred Only
	CLW4	Inferred Only	Upper Fracture zone	CUP1	Inferred Only
	CLW5	Inferred Only		CUP2	Inferred Only
	CLW6	Inferred Only		CUP3	Inferred Only
	CLW7	Inferred Only		CUP4	Inferred Only
	CLW8	Inferred Only		CUP5	Inferred Only
	CLW9	Indicated & Inferred		CUP7	Inferred Only
	CUP6	Measured, Indicated, & Inferred			
	CUVN	Measured, Indicated, & Inferred			
D zone	CUVND	Measured, Indicated, & Inferred			
	CUVND2	Measured, Indicated, & Inferred			
	CUVND3	Measured, Indicated, & Inferred			
	CUVND4	Inferred Only			
	CUVND5	Measured, Indicated, & Inferred			
	CUVND6	Inferred Only			

Blocks within the remaining A/B and C zone domains were classified as Measured resources if the blocks were within 50 ft of a composite and estimated using at least 5 composites (three drillholes). Representing the distances at which continuity between drillholes is visually apparent. Additionally, the utmost confidence is reached by including the maximum number of composites based on the estimation parameters. Indicated resources are those blocks within 100 ft of a composite, or roughly between 2/3 and 1/2 the variogram range for the A/B and C zones respectively and estimated using at least three composites (two drillholes) to ensure confidence in the estimate. Inferred resources are the remaining estimated blocks and any blocks coded as Basalt.

Mineral resource classification is more restrictive in the D zone due to the orientation of underground drilling reducing confidence in the location and orientation of the modeled domains. Blocks within the D zone domains were classified as Measured resources if the blocks were within 25 ft of a composite and estimated with at least 10 composites (5 drillholes). Indicated Resources are those blocks within 50 ft of a composite and estimated with at least 5 composites (3 drillholes). Inferred resources are the remaining estimated blocks.

14.4.11 Removal of Mined Out Blocks

3D solids of the existing underground developments and stopes, including a decline development from Cyprus Mining were used to classify blocks as mined out if the block centroid was inside, or within 1 ft. of the development solid. These blocks could then be coded and removed from the mineral resource statement. The maximum extent of Cyprus Mines open pit operation is not completely known due to backfilling inside the pit. However, the QP utilized a surface of the maximum mined out pit constructed from plan view CAD drawings recently found at the Project geology office. Those blocks within that surface were coded as mined out and removed from the resource estimate.

14.4.12 Sensitivity

Prior to tabulation of mineral resources, blocks coded as quaternary alluvium were removed from the mineral resource. The block model tons and grade are shown in Figure 14-32 at variable economic cut-off grades as a sensitivity analysis.

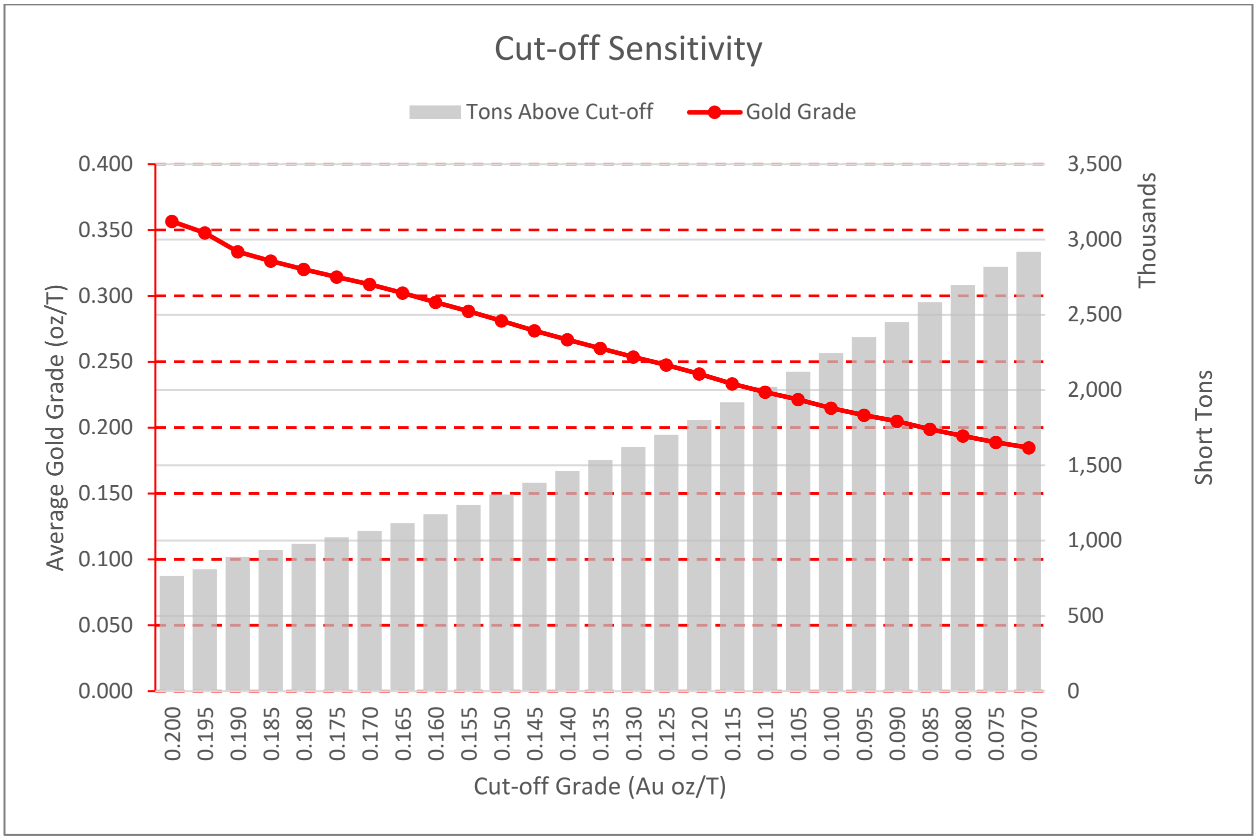

Figure 14-32 Cut-off Sensitivity Chart

14.4.13 Mineral Resources Statement

The undiluted Copperstone project mineral resource statement is presented in Tables 14-22 and 14-23 below. The results reported in the mineral resource have been rounded to reflect the approximation of grade and quantity which can be achieved at this level of resource estimation. Rounding may result in apparent differences when summing tons, grade and contained metal content. Tonnage and grade measurements are in U.S. Customary units and Metric units. 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. Inferred mineral resources are that part of a mineral resource for which the grade or quality are estimated on the basis of limited geological evidence and sampling. Inferred mineral resources do not have demonstrated economic viability and may not be converted to a mineral reserve. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.

The mineral resources are confined to material exceeding the gold cut-off grade of 0.092 oz/ton within coherent wireframe models. After the block grade estimations were complete, the estimated blocks at and above the cut-off grade for each domain were reviewed in long section by the QP. The majority of estimated blocks demonstrate grade continuity and meet the criteria of a minable shape. Small, and isolated blocks that did not meet the QP's opinion of a minable shape were excluded from the mineral resource statement. The application of a cut-off grade to estimated blocks which meet the criteria of a minable shape within coherent wireframes models meet the test of reasonable prospect for economic extraction. The cut-off is calculated based on the operating costs, royalties, recoveries and metal prices as presented in Table 14-22. A gold price of $1,800/oz was chosen which is the 36-month moving average price as of January 31, 2023. The effective date of the mineral resource estimate is February 15, 2023.

Table 14-22 Mineral Resource Cut-off Parameters

Mineral Resource Cut-off		
Mining	$/ore ton	$ 90.00
Processing	$/ore ton	$ 47.00
G&A	$/ore ton	$ 15.00
Recoveries	ton	95.0%
Royalties	%	3.0%
Refining & Shipping cost	per/oz	$ 12.00
Total cost	$/ore ton	$ 154.50
Gold Selling Price	oz	$ 1,800.00
Cut-off Grade		0.092

Table 14-23 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

1. The effective date of the mineral resource estimate is February 15, 2023. The QP for the estimate, Mr. Richard A. Schwering, P.G., SME-RM of HRC, is independent of Sabre Gold Mine Corp.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Inferred mineral resources are that part of a mineral resource for which the grade or quality are estimated on the basis of limited geological evidence and sampling. Inferred mineral resources do not have demonstrated economic viability and may not be converted to a mineral reserve. It is reasonably expected, though not guaranteed, that the majority of Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.
4. The mineral resource is reported at an underground mining cut-off of 0.092 oz/ton (3.15 g/t) Au beneath the historic open pit and within coherent wireframe models, and for estimated blocks which meet the criteria of a minable shape. The cut-off is based on the following assumptions: a gold price of $1,800/oz; assumed mining cost of $90/ton ($99.21/tonne), process costs of $47/ton ($51.81/tonne), general and administrative and property/severance tax costs of $15.00/ton ($16.53/tonne), refining and shipping costs of $12.00/oz, a metallurgical recovery for gold of 95%, and a 1.5% gross royalty.
5. Rounding may result in apparent differences when summing tons, grade and contained metal content. Tonnage and grade measurements are in U.S. Customary and Metric units. Grades are reported in troy ounces per short ton (oz/ton) and grams per tonne (g/t). Contained metal is reported as troy ounces.

15. MINERAL RESERVE ESTIMATE

This section is not applicable for a PEA-level report.

16. MINING METHODS

16.1 Summary

The PEA underground mine plan for the Copperstone Project includes approximately 1,222,300 tons of mineralized mill feed to be extracted by underground mining in 6 years. The underground mine designs and schedule utilize Inferred mineral resources as part of the analysis. Mineral resources that are not mineral reserves do not have demonstrated economic viability. This PEA is preliminary in nature in that it includes Inferred mineral resources that are considered too speculative to have economic considerations applied to them and should not be relied upon for that purpose. The mine production schedule calls for the production of 600 tpd for an annual production of 219,000 tons through the milling circuit. Mining recoveries of 95% were applied and overall dilution factors averaged 32%. 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 three eight-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 0.107 oz/ton gold for underground mining;
- For planning purposes, the stopes have been separated into six zones. A, B, C and D zones along with the Footwall zone, South and Upper Fracture zones;
- A production rate ramped up to 600 tpd;
- The underground design allowed for 28.9% planned dilution, 10% unplanned, and a mining recovery of 95%;
- Development drifting and raising of approximately 61,416 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 mining method proposed for the Copperstone Project is a mechanized cut and fill using CRF. Cut and fill was chosen for its flexibility in effectively mining low vein dip angles. The method also minimizes the amount of dilution during mining by careful geological and management control of the mining.

Underground mining methods were reviewed that will minimize dilution, capital, and operating costs, maximize recovery of the mineral resources while maintaining the design production throughput at the mill were reviewed. The Copperstone mineralization is relatively flat with an average dip of 38 degrees. Although there are some areas where the mineralization is steeper and will flow by gravity, above a 45-degree dip, the majority of the deposit is too flat to facilitate a long hole mining method.

The Copperstone mine had historic open pit production from 1987 through 1993 by Cyprus of approximately 514,000 oz of gold from 6,173,000 tons of mill feed grading 0.089 oz/ton Au. In 2012 American Bonanza Gold Corp started underground mining from two declines which were 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. Due to the historic underground mining that has taken place and an exploration drift that was put in during the summer of 2017 there is currently 12,800 ft of access development in place. This existing access includes two declines from the bottom of the pit and extends across 500 ft of strike. Therefore, a reduced amount of development is required to get the mine up to full production.

16.2 Geotechnical

There have been numerus geotechnical reports completed on the property with the most recent completed by Golder in 2006, Call and Nicolas in 2010, Langston and Associates in 2012, Tierra Group International in 2018 and Langston and Associates in 2021 and in 2022. Langston and Associates completed a Ground Control Management Plan (GCMP) in January 2022.

In 2018, Dr. Dermot Ross-Brown from Tierra Group International, Ltd completed a review of the past geotechnical studies and visited the current underground workings in order to provide an estimate of the required ground support and maximum opening sizes for the mine plan. Dr. Ross-Brown made three site visits to the mine to become familiar with the geology, the general conditions of the underground workings and to collect data on the quality of the rock mass in the region of potential stopes.

The main tools used in the 214ssessent of rock quality were the RMR-system (Rock Mass Rating, as described in Bienawski, 1974, Bienawski, 1989, and Gonzalez de Vallejo and Ferrer, 2011), and the Q-system (as described in Barton, 2002 and Norwegian Geotechnical Institute, 2015). To date, little weight has been given to these factors in assessing the stability of the underground excavations. The RMR and Q classification system are increasingly becoming more standard in the design of stopes, especially during the last 20 years. These methods have been briefly discussed in previous reports submitted to 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.

Mapping of core and exposures proceeded with the purpose of getting preliminary data on RMR and Q to help with the design of stopes in all four zones. On each borehole core, Copperstone's geologist marked off the mineralized zone, and Tierra Group's Engineer subdivided this zone, as well as the HW and FW sections, into geotechnical units (typically 3 or 4 units, with each unit having similar geological and geotechnical properties along its length). RMR and Q logging was undertaken on each unit. RMR data were then averaged over the applicable lengths to provide an estimate of RMR in all three sections (i.e. the mineralization, HW, and FW) for that borehole. A summary of the RMR data is displayed in Table 16-1.

Table 16-1 Productivity Rates

Code	Borehole No	Zone	Notes – Geology near mineralization	RMR_{76} HW	RMR_{76} Mineralized	RMR_{76} 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

For the eight **exposures mapped underground and in the pit**, the average RMR76 is 52. It is a similar average to that obtained from the boreholes, but the values obtained are for the 'rock mass' in each location and are not related to HW, gold mineralization, and FW as in the above table.

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.

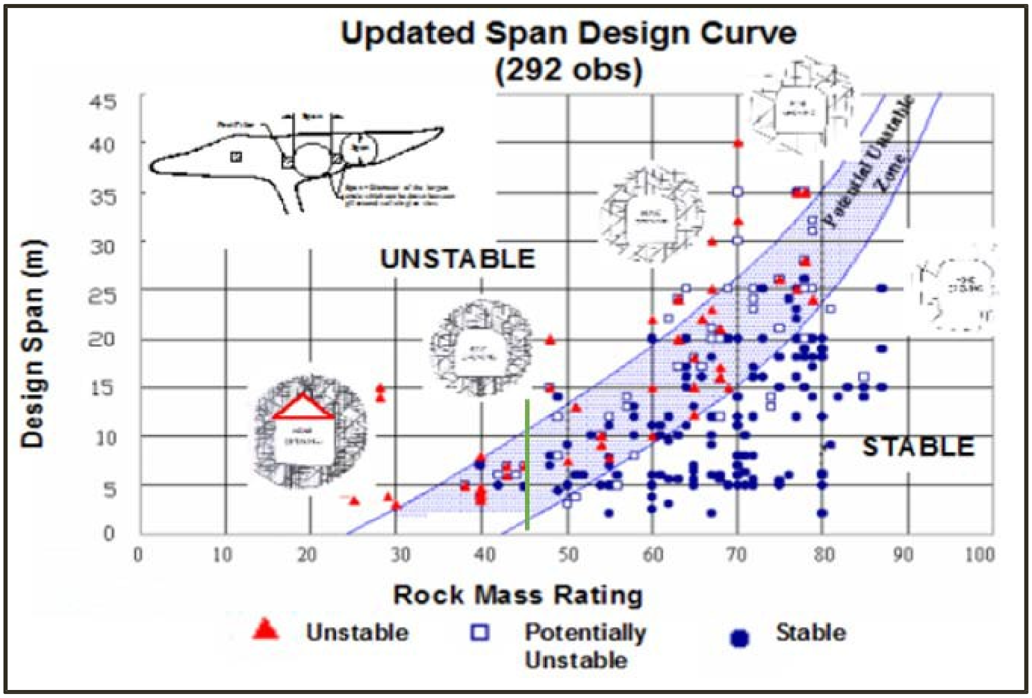

Figure 16-1 Critical Span Design Curve (Paknalis, 2002, updated 2014)

Note the definition of critical span (diameter of the largest circle drawn within boundaries of the excavation). The graph is based on short-term observations, 3 months to about 1 year and applies to no support or pattern bolting for local support (i.e. typical patterns of bolts 1.8 m long at 1.2 m x 1.2 m spacing). Note that sometimes it is necessary to make a correction for shallow joints; when jointing is near-parallel to the roof (<30°), in this case 10 should be subtracted from RMR_{76} when using 1.8-m bolts on a regular pattern.

With the limitations explained above, the estimated average Copperstone RMR value of 45 plots as a green vertical line on the graph; a span of 5 m (16 ft) occurs in the middle of the transition zone. Meaning that spans of 5 m should be stable in the very short-term (i.e. one or two weeks), as observed in the existing drifts at Copperstone. However, the risk of collapse increases as the excavation size is increased, and as the time to add effective support is delayed. At Copperstone, this means that excavations with a width and height of up to 5 m should be stable with local support applied quickly. The length of the stope is not as critical as the width and the height. The implications for the design of stopes, is that all excavations require at least 'normal support' in the roof and sometimes part way down the walls. Normal support is defined as being the equivalent of 1.8-m long bolts on a pattern of 1.2 m x 1.2 m; split sets may be used if the excavation has a life of less than 6 months. In addition, mesh and sometimes straps probably will be needed; in very bad ground, 5 cm of steel fiber-reinforced ("Sfr") shotcrete may be needed prior to bolting. The amount of support can be determined after each blast using the Q-system, which is described in detail in the NGI reference (NGI, 2015). For the PFS 10% of all development and stoping areas were assumed to require normal bolting (1.8 x 1.2 x 1.2 m), 70% of all areas were assumed to require bolting with mesh and occasional straps and 20% of all areas were assumed to require bolting with the prior addition of 5 cm Sfr shotcrete.

In 2021, Langston and Associates completed a site visit and a subsequent study on the stope span stability analysis. Span increments of 15 and 25 ft for the back and 15 and 30 ft along the hanging wall for production headings were analyzed to evaluate rock mass stability for each case. Analysis methods used were two-dimensional finite element modeling, kinematic wedge analysis, and rock mass quality evaluation. The analysis showed that back spans may be increased so that the amount of multiple pass mining may be limited to only the poorer rock mass quality areas and excessively wide mineralized intercepts. The analysis also demonstrates that the cemented rock backfill (CRF) becomes loaded after placement which means that is resisting deformation to some degree and generating support pressure to provide bearing support for the mined out areas. At a 15-ft span for either the back or hanging wall, very little no rock mass yield occurs in the back and none occurs in the hanging wall. When the span of either wall increases to 30 ft on the hanging wall and 25 ft in the back, rock mass yield remains small, displacements are also small – on the order of 2 to 3 mm. Assessment of potential wedge failure and rock mass quality indicated that these factors may be controlled by adapting the current development ground support standard to the production heading geometry.

Although the 2021 study by Langston and Associates concluded that wider stope spans are not problematic, than the 16-ft proposed by Tierra International in 2018, the PEA mine plan was developed using the more conservative 16-ft maximum span per pass. In good ground, wider spans are likely to be achievable which will allow for higher productivity in the 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 Schlumberger Water Services report.

16.3.1 Hydrology and Recharge

The Project is located in the Tyson Wash watershed. The Tyson Wash itself is located five miles to the southwest of the mine. There is no permanent or ephemeral local surface drainage. There are small local unnamed washes in the mine complex. The surface area of the existing mine pit catchment is approximately 654 acres. Catchment I is a surface water runoff area to the immediate north of the mine area, the area is in excess of 175 acres. Catchment II is a surface water runoff area to the immediate south of the mine area; this area is in excess of 2,854 acres.

The annual average precipitation is 4.7 inches of which an average of 0.84 inches falls in August. The nearest permanent surface waterbody is the Colorado River, nine miles to the west of the site. Based on studies elsewhere the net annual average recharge via infiltration of incident precipitation will be very low, likely 0.2 inches or less. The temperature ranges from 20°F to 121°F with an annual average of 71°F. The daily temperature from April to October is on average 89°F. The measured temperature of discharge water pumped from the decline was 95°F during a site visit in October 2009. The ambient air temperature was 71°F. This should be the temperature of the locally recharged ground water. The elevated ground water temperature would suggest that there is possibly a mixture of deeper, warmer water and locally sourced ground water being abstracted from the decline. The mine site area is a local topographic high. The mine site topography is flat to moderate, ranging from 860 ft to 902 ft. The surrounding topography, outside the mine property, is flat ranging from 900 ft to the east to 820 ft to the west and northwest.

16.3.1.1 Geological information

- **Sedimentary overburden:** There is between 0 and over 600 ft of unconsolidated to semi-consolidated sediments surrounding the existing pit. The original location of the pit itself was a series of outcropping low relief knolls. The sediments consist of: Sands, Clays and Conglomerates (of the Bouse Formation). The depth of unconsolidated sediments over the D zone is between 164 ft at the southern end to 284 ft at the northern end. In the vicinity of the open pit, the sedimentary overburden is believed to be unsaturated.
- **Bedrock:** The bedrock stratigraphy of the immediate property consists of: Miocene sedimentary breccia and basalt, Jurassic quartz latite porphyry (Brecciated where cut by the Copperstone Fault), Triassic metasediments which are the principal host rocks for the D zone and Triassic phyllites seen only in the footwall in the D and C zones.
- **Structure:** The brecciated and intensely fractured Copperstone fault detachment zone is between 45 ft and 185 ft wide, strikes to the northwest, and dips between 20° to 50° to the NE. The deposits themselves are brecciated. NW trending faults are high angle (>80°). Two such faults are found in the D zone which cross-cut the Copperstone fault. A massive cataclastic breccia zone was observed at the northern end of the pit in the D zone. Faulting alone does not account for the deformation and the presence of a breccia pipe is hypothesized. The overall implications of this structural and mineralization regime are that it is potentially conducive to the development of a high secondary permeability rock mass. The presence of sandy gouge material in faults could result in potentially permeable fault planes along strike, particularly if the material is washed out. Cataclastic zones could have the potential to be highly permeable. Mapping of the existing decline indicated heavily folded and faulted rock. The rock type changes rapidly over ten's of feet. Inspection of core both on site and from pictures taken at the time of drilling shows frequent fracturing, changes in rock types and extensive areas of brecciated materials especially in the underground core samples drilled into the D zone. The majority of fractures appear to be full of clay gouge, while others appear to be clean and potentially conductive especially those found within the brecciated zones. Inspection of photos from almost any given hole drilled in the underground drill bay into the D zone will confirm this. Another indication of the ability for water to move through the bedrock is that the pumped water from the decline sump, when deposited in the pit, infiltrates almost instantly with no apparent ponding, at a pumping rate of approximately 300 gallons per minute ("gpm").

16.3.1.2 Hydrostratigraphy

From a dewatering perspective, the bedrock is of primary interest. In the vicinity of mining, the sedimentary overburden is believed to be dry. There are no direct measurements of permeability for the bedrock units at the site. The bedrock geologic units bounding the underground mine and the shear zone are expected to have low hydraulic conductivity. The main control for groundwater inflow and dewatering flow rates will be the extent to which the shear zone and associated faulting contains fracturing that is open and contains minimal amounts of mineral or clay fill. Groundwater occurrence and movement will mostly occur in such zones. The available core data for the site indicates that most fractures will not be particularly conductive. However, some fractures appear clean, particularly those found within the brecciated zones. There are also extensive zones of fracturing within the porphyry that appear to have the potential for water movement.

Although there may be water transmission within the fault zone in close proximity to the mine, the system is in all probability bounded and limited in extent, which means that dewatering flow rates will tail off with time. There is evidence of this compartmentalization in piezometer data during periods when the decline and drill bay are pumped.

16.3.1.3 Local groundwater recharge estimates

The ground water and surface water catchments are assumed to be coincident.

Assuming a range of infiltration rates from 0.5 to 3% of mean annual precipitation, the total volume infiltrated on an annual basis within the pit footprint ranges from 417 thousand gallons to 2.5 million gallons. This works out to a continuous rate of 0.8 to 4.8 gallons per minute.

Assuming a range of infiltration rates from 0.5 to 3% of mean annual precipitation, the total volume infiltrated on an annual basis within the Catchment I ranges from 2.85 to 17.12 million gallons. This works out to a continuous rate of 5.4 to 32.6 gpm.

Ponding of storm water runoff within the pit is not anticipated to be a problem due to the rapid infiltration of water that was pumped from the decline and deposited in the pit. However, this implies the rapid recharge into the bedrock system which amplifies the need for the mine site to be dewatered.

16.3.2 Mine Dewatering

The new development will extend from 520 ft above mean sea level to -120 ft bmsl, which is approximately 190 ft deeper than the current underground workings. The maximum depth of the workings below the current water table will be 450 ft, which equates to a hydraulic pressure of approximately 210 psi. The Rock Quality Data (RQD) averages a classification of “Fair Rock” assuming a dry mine. If the rock mass is not dewatered, the values of RMR_{76} could be reduced by 8 (representing ‘wet’ conditions) or, in the worst case of flowing water, could be reduced by 15. Rock masses with RMR_{76} values below 40 (taking into account the water conditions) are difficult to mine, therefore, it will be beneficial to dewater the underground working to the fullest extent possible.

The main conclusions from SWS report are that the shear zone appears to be relatively broken and moderately permeable. The presence of warm groundwater implies deep circulating groundwater and discrete permeability within the system. The structural system appears to be bounded, therefore dewatering flow rates should stabilize or be reduced in the longer term and the annual recharge will be virtually insignificant. SWS dewatering flow rate requirement for the mine development was estimated to be on the order of 150 to 300 gpm, when accounting for recharge. However, a sensible contingency for pumping of sudden inflow during development needs to be factored, together with the possibility of ongoing recirculation due to operating practices. Therefore, the total inflow rate and pumping system capacity most likely needs to be on the order of 300 gpm.

The current mine dewatering system averages 278 gpm to maintain the water level in the mine. Currently the mine is using a Tsurumi 60hp pump in the 810 drift (160-ft elevation) to lift the water, thru a 6 inch pipe, to the 730 access finger cut sump (215-ft elevation). A 150 hp Tsurumi pump lifts the water, thru a 6 inch pipe, to a tank (590-ft elevation) on the surface and a 200 hp Cornell is used to pump through a 10 inch line to the tailings pond (894-ft elevation). A 30 hp Tsurumi pump is used to dewater individual finger cuts that do not drain to the main sump. Based on the current averages and the mine development only going 190 ft deeper than current workings, the prediction by SWS of a 300 gpm system appear to be accurate. Pumps will need to be repositioned as the mine develops but the overall pumping capacity is not anticipated to significantly increase from current levels. 500 ft of development is also included in the mine plan for creating a large water storage sump at the bottom of the C zone for management of storm water.

16.4 Mine Design

16.4.1 Stope Design

Underground mining methods were reviewed that will minimize dilution, capital, and operating costs, maximize recovery of the mineralized resources while maintaining the design production throughput at the mill. The Copperstone mineralization is relatively flat with an average dip of 38 degrees. Although there are some areas where the mineralized material will flow, above a 45-degree dip, the majority of the deposit is too flat to facilitate a long hole mining method. The mining method proposed for the Copperstone Project is a mechanized cut and fill using RF and CRF. Cut and fill was chosen for its flexibility in effectively mining low vein dip angles. The method also minimizes the amount of dilution during mining by careful geological and management control of the mining. Datamine's® Minable Stope Optimizer ("MSO") was used to generate the stopes utilizing a metal price of $1,800/oz for gold and a 0.107 oz/ton gold cut-off.

Cut and fill stoping involves accessing the mineralization from a main ramp. The initial stope access is driven down grade to the waste/mineralized contact and then extended through the mineralization to the hanging wall contact. Once the hanging wall has been located, longitudinal panels are mined perpendicular to the access drift, along strike, to the stope ends. The cut and fill design is based on 10-ft high stopes. The 10-ft height was chosen primarily to reduce dilution and to improve ground conditions but still allow for the proposed equipment to fit inside the stope. In narrow mineralized areas that are inclined at 38°, a significant amount of waste will be generated at the hanging wall and footwall contacts in order to recover all the mineralized material. This percentage of waste in narrow areas increases as a function of the stope height. As the mineralization width increases, the percentage dilution reduces significantly. A reduction in the amount of waste mined was achieved by implementing a shanty back of 60° on the stope hanging wall. Figure 16-3 shows multiple lift cut and fill mining with shanty backs and footwalls. Hanging wall instability can be a problem in shanty back stopes where bed separation can cause failures, this problem can be exacerbated with greater stope heights. Ground support will have to be installed quickly on the hanging wall in order to minimize stability issues with the stope back.

The mineralized body width varies considerably over the property and total stope widths range between 9 ft and 102 ft wide, the average total stope width is 30 ft. Where stope widths exceed 16 ft horizontally, it will be necessary to extract multiple side- by-side drifts (passes) on each cut in order to limit the mining span. After stopes are split into the required drift passes, the average actual mining width is 12 ft. Figure 16-2 shows the percentages of the stopes requiring multiple passes. Up to seven passes will be required but the majority of the stopes, approximately 89%, will be one, two or three pass stopes. Stope lengths average approximately 100 ft, with a few extending beyond 400 ft. If the stope widths are extended to a maximum 30-ft width as suggested by the Langston and Associates study then 63% of the stopes will be single pass, 30% will be 2 pass, 6% 3 pass and 1% will be 4 pass.

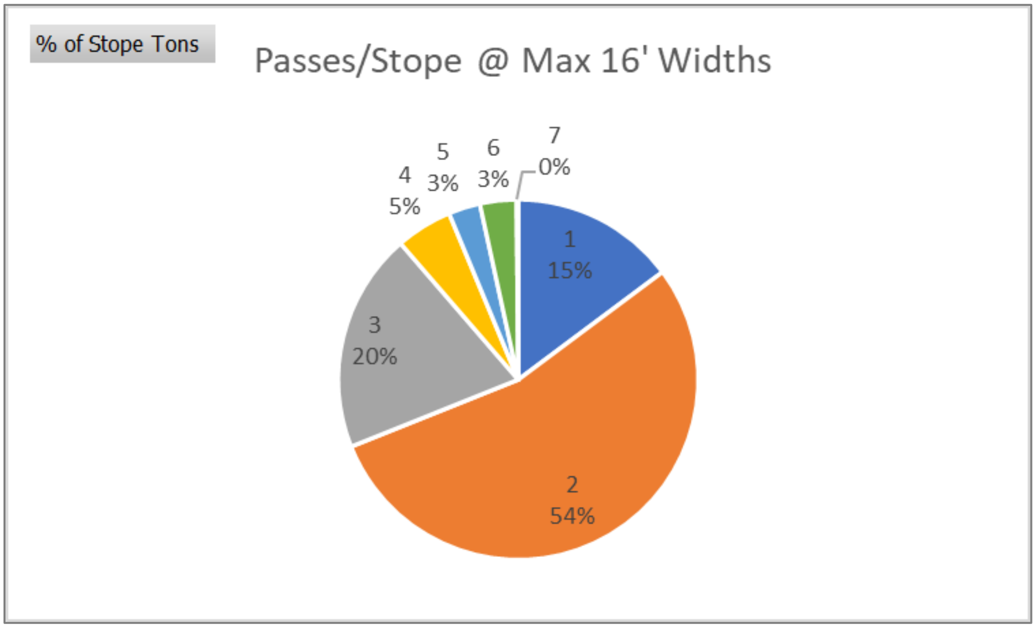

Figure 16-2 Passes per Stope

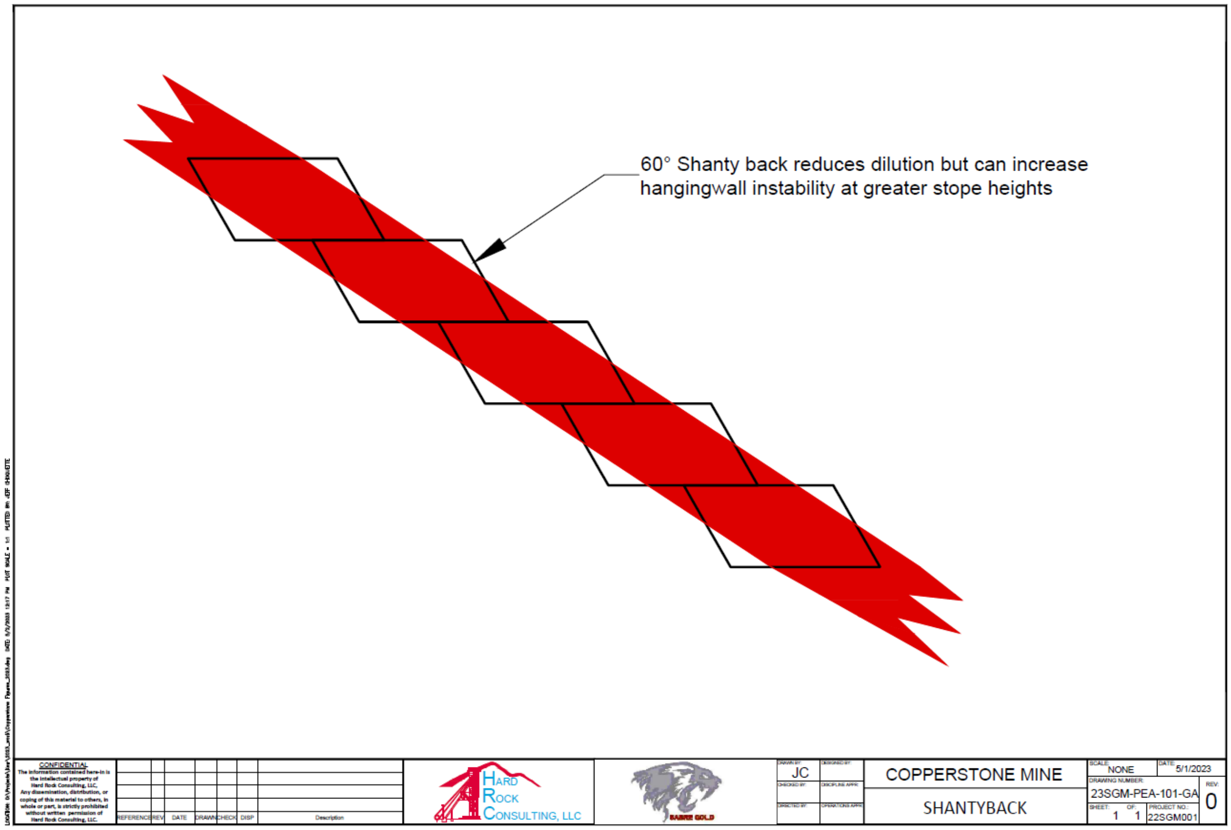

Figure 16-3 Stope Shanty Back

16.4.2 Cut-off

The mining breakeven cut-off grade was used to generate the stope designs in Datamine's MSO for defining the mine plan mineralized material. The cut-off is based on the following assumptions: a gold price of $1,800/oz; assumed mining cost of $115/ton, process costs of $46/ton, general and administrative and property/severance tax costs of $15/ton, refining costs of $12.00/oz and metallurgical recovery for gold of 95% and royalties of 3%. The resultant cut-off equates to 0.107 oz/ton for defining the mill feed tons within the mine stope designs.

16.4.3 Dilution and Mill Feed Loss

External dilution was applied in the amount of 10% at a grade of zero. In addition, the minimum stope width in MSO was set at six feet with a 1.5 foot dilution skin on the footwall and hanging wall for a total minimum stope width of 9 ft. Internal dilution is also applied based on any blocks that fall inside the stope shape but are below cut-off, the internal dilution is calculated at 28.9%. These factors resulted in an overall dilution factor of 32% for the mill feed material. A mining recovery of 95% is also applied to the reported stope design tonnage and metal content.

16.4.4 Development Design

The primary ramp development is planned in the footwall of the mineralization to access the cut and fill stopes. The main haulage drifts and ramps are planned to be developed at a 15-ft height x 15-ft width which is similar to the size of the existing development. The main ramp is designed to limit curves and turns to promote efficient truck haulage and reduce ventilation constraints. Muck bays 30 ft deep are planned near the stope access points or every 500 ft along the ramp to facilitate the development mucking process. As the development progresses these muck bays will be converted for use as sumps, transformer bays, storage areas and exploration drill bays.

The total length of the new main haulage ramps is 26,441 ft. One new portal in the pit bottom is planned, the portal will be installed towards the end of year one and will provide access to the A zone. There is also a historic decline that was put in by Cyprus towards the end of the operations of the open pit in the mid 1990's. The PEA plan includes costs to rehab 3,475 ft of this decline which will allow for an alternative haulage route to the mill for the Footwall zone mineralized material. The portal for this decline has been buried by the open pit waste material, to uncover the portal approximately 100,000 to 150,000 tons of material will also have to be moved on surface. Re-establishing this decline will also improve the ventilation circuit by providing an additional exhaust route for the air flow, should the decline be found to be deteriorated beyond re-pair a ventilation raise could be established in this area as an alternative option.

A vent raise from the 290-ft elevation down to the 80-ft elevation is also planned in the C zone to provide intake air down to the bottom of the C zone. Figure 16-4 shows the layout of the portals and other infrastructure in relation to the open pit and proposed underground workings.

The main haulage ramps are developed approximately 160 ft beyond the mineralization in the footwall. The stope access ramps are planned at 10-ft height x 10-ft width to allow sufficient access height for highly-productive mining equipment. A nominal level spacing of 60 ft was selected, providing access to six 10-ft high drift and fill cuts from a single access point. The first access ramp is driven at -15% to access the first of six lifts of the stope. The remaining five lifts are developed by backslashing and ramping up at +15% to access subsequent lifts. Figure 16-5 presents a typical stope access development layout. Each stope access point also includes a 30-ft muck bay. The total length of stope access ramps is planned at 11,407 ft and the total length of stope access backslash ramps is planned at 20,248 ft. The main haulage and stope access planned distance by year are presented below in Table 16-2.

Table 16-2 Development Distances

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	LOM Total
Main haulage ramp ft. (15'x15')	1,979	9,542	10,287	4,634	0	0	0	26,441
Muck Bays ft. (15'x15')	328	1,497	854	450	0	0	0	3,129
Borehole Raises ft. (8' dia.)	160	32	0	0	0	0	0	192
Stope access ramp ft. (10'x10')	671	1,436	1,480	1,964	2,228	2,700	927	11,407
Stope access backslash ft. (10'x10')	192	1,871	2,100	2,899	4,688	5,453	3,045	20,248
Total ft.	**3,330**	**14,378**	**14,721**	**9,947**	**6,916**	**8,153**	**3,972**	**61,416**

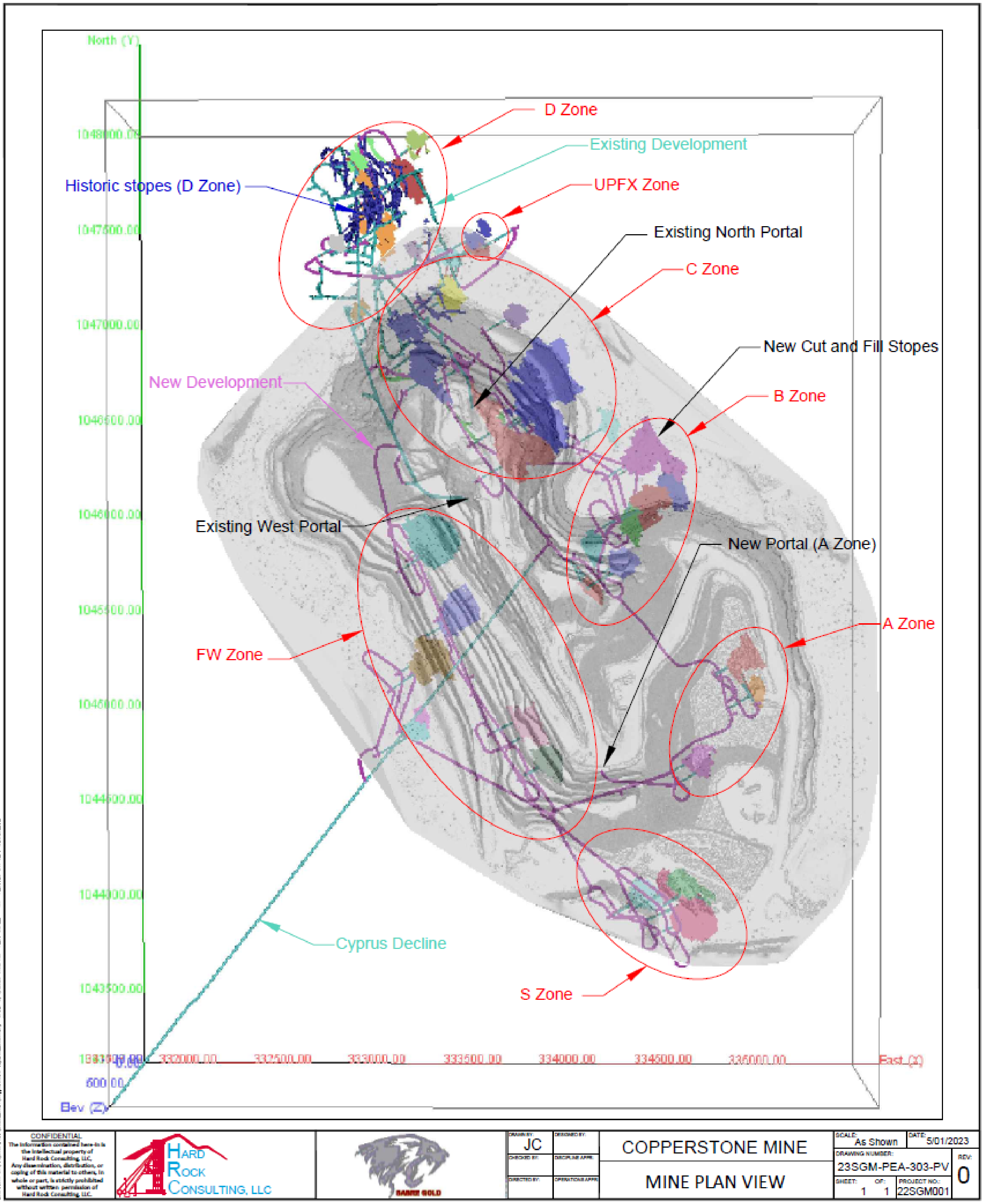

Figure 16-4 Mine Layout

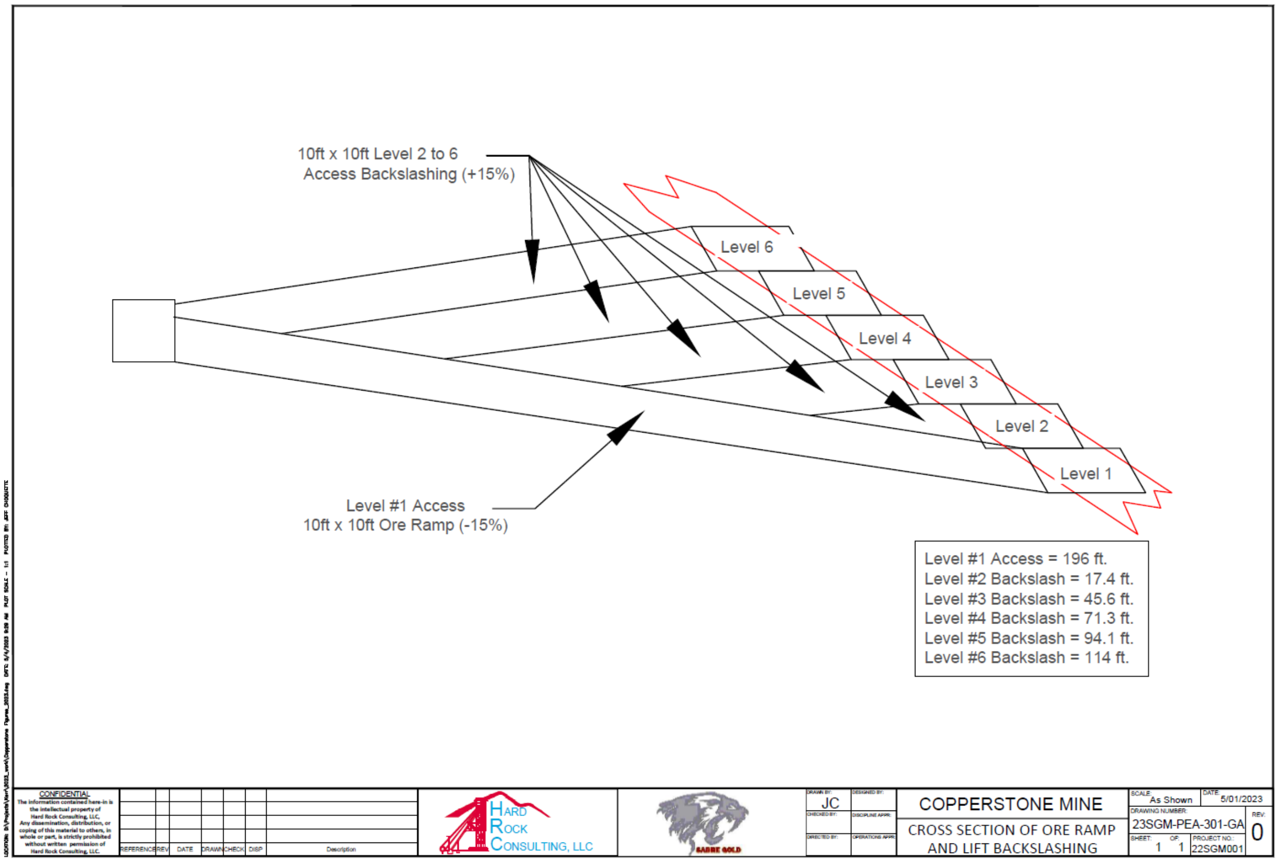

Figure 16-5 Stope Access Development

16.5 Production Schedule

16.5.1 Development Productivity

The development schedule is based on the development rate assumptions shown in Table 16-3.

Table 16-3 Productivity Rates

Activity Type	Dimensions feet (m)	Rate feet (m)
Main haulage ramp	15'x15' (4.6x4.6)	15.3 ft/d (4.7 m/d)
Stope access ramp	10'x10' (3.0x3.0)	12.5 ft/d (3.8 m/d)
Stope access backslash	10'x10' (3.0x3.0)	24.9 ft/d (7.6 m/d)
Vent raise	8' dia (2.4 dia)	7.3 ft/d (2.2 m/d)

The rate 15.3 ft/day for the larger headings is based on two 10hr shifts/day, cycling a 0.6 rounds/shift on average. The smaller stope access drifts are planned at a rate of 12.5 ft/day. Stope lift access backslashing will advance at a rate of 24.9 ft/day with these backslash areas mined by uppers out of the preceding stope access ramp, rather than as advancing faces. This process will increase productivity due to the fact that drilling, loading and ground support can now be continuous activities throughout the shift instead of cycling between each activity during a shift. For the backslash rounds, 31 ft of advance will be blasted at once instead of the typical 9.3 ft per round in the access heading. The vent raise in the C zone is planned to be put in by a contractor using a raise bore and driven at 7.3 ft/day. For the mine plan 10% of all development areas were assumed to require normal bolting (1.8 x 1.2 x 1.2 m), 70% of all areas were assumed to require bolting with mesh and occasional straps and 20% of all areas were assumed to require bolting with the prior addition of 5 cm Sfr shotcrete.

16.5.2 Stoping Productivity

During mining, each stope is calculated to produce at a rate of 176 tpd. In order to meet the required total mine production of 600 tpd, 3.5 stopes will have to be in production at all times. Based on the required backfill time, development and unanticipated delays a total of 6 active faces were scheduled at a time. The 176 tpd rate is based on two 10-hour shifts cycling a round/day.

The underground production requirement of 600 tpd on average is used to develop the production schedule. The nominal stope heading size will be 10 ft high x 14 ft wide. This heading size will allow the engineering design team and the grade control geology group the flexibility to maximize the gold grade from the underground mine. The mining cycle involves drilling, blasting, mucking and ground control cycle described below.

16.5.2.1 Drilling

The critical time path to complete a mining cycle in heading is the jumbo drilling time. Due to the need to keep the production heading size down to a minimum size of 10 ft by 9 ft in certain stopes to limit dilution a single boom jumbo is planned drilling 31 holes that are 45 mm diameter and 13 ft deep. The hydraulic drill has a penetration rate of 2.7 ft/min and will require 204 theoretical minutes to complete the required drilling. An allowance has been included to account for spotting the drill in the heading and maneuvering the drill booms between holes. The drill cycle time also includes allowances for mechanical problems, re-drills and grade control sampling so the actual calculated drilling time per round is 245 minutes.

16.5.2.2 Blasting

Once the jumbo drill has completed the drilling cycle the ANFO explosive is loaded into the holes with the respective nonel blasting cap and booster. The timing of the round with the nonel caps is extremely important and is critical to pulling the maximum amount of distance per round and minimizing over break. An estimated 105 kg of explosive is required for each round and using 150 mt ore/round the powder factor will be 0.704 kg/mt. The actual advance/round is 12 ft allowing for the difference between the drill depth and the actual pull depth.

16.5.2.3 Mucking and Hauling

A 3.9yd^3 diesel LHD is used to muck out the heading after blasting and clearing, the back and ribs are then scaled by hand. The LHD trams the mineralized material to the associated muckbay for each respective area located in the area of the access ramp. Once the heading is mucked out the bolting process can begin. While the bolting and drilling cycle has started on the next production cycle in the heading, the mineralized material previously placed in the muckbay is transferred by the 30 metric tonne haul trucks to the surface mill feed stockpile in the pit bottom. The mucking cycle including mechanical delays from the heading to the muckbay will be 215 minutes.

The productivity of the haul trucks is calculated based on the average distances and tons of development waste, stope mineralized material and backfill required throughout the mine schedule. An average of 2.5 and maximum of 3 trucks are required throughout the mine life in order to meet the production schedule for mining, backfill placement and development activities.

16.5.2.4 Ground Control

Ground control for the production area is slightly different from the development heading since it will be temporary in nature. Qualified individuals will inspect and rate the ground conditions in each heading and relay the information to the ground support crew. Ground will be classified as Good, Fair or Poor. Depending on the ground class the support plan will vary as per the Langston and Associates Ground Control Management Plan submitted in January 2022.

Development headings of 13 to 20 ft wide will utilize the following parameters.

Good Ground

- 6-ft long galvanized friction bolt with 6-inch square grade 2 plates on a 3-ft by 3-ft pattern and welded wire mesh panels as surface support installed in the back and ribs.
- 8-ft standard Swellex bolts on 6-ft by 6-ft pattern in the back.

Fair Ground

- 6-ft long galvanized friction bolt with 6-inch square grade 2 plates on a 3-ft by 3-ft pattern and welded wire mesh panels as surface support installed in the back and ribs.
- 8-ft standard Swellex bolts at 3 ft by 6 ft spacing in the back and alternate 8- and 12-ft Swellex bolt lengths.

Poor Ground

- 6-ft long galvanized friction bolt with 6-inch square grade 2 plates on a 3-ft by 3-ft pattern and welded wire mesh panels as surface support installed in the back and ribs.
- Alternate 8-ft and 12-ft long Swellex bolts in the back.
- 8-ft standard Swellex bolts added to support the ribs on 6-ft by 6-ft pattern.
- Shotcrete applied as needed with minimum thickness of 2 inches.

Production Headings

Good Ground

- 6-ft long black steel friction bolt with 6-inch square grade 2 plates on a 3-ft by 3-ft pattern and welded wire mesh panels as surface support installed in the back and ribs.
- 8-ft standard Swellex bolt installed every 6 ft along strike in the hanging wall.
- If heading/stope is greater than 15 ft wide additional 8-ft standard Swellex bolts added on 6-ft by 6-ft spacing in the back.

Fair Ground

- 6-ft long galvanized friction bolt with 6-inch square grade 2 plates on a 3-ft by 3-ft pattern and welded wire mesh panels as surface support installed in the back and ribs.
- Two 8-ft standard Swellex bolts installed every 6 feet along strike in the hanging wall.
- If heading/stope is wider than 15 ft, Swellex bolt pattern becomes 3 ft by 6 ft in the back alternating with 8-ft and 12-ft bolts.

Poor Ground

- 6-ft long galvanized friction bolt with 6-inch square grade 2 plates on a 3-ft by 3-ft pattern and welded wire mesh panels as surface support installed in the back and ribs.
- Three 8-ft standard Swellex bolts installed every 6 ft along strike in the hanging wall.
- If heading/stope is wider than 15 ft, Swellex bolt pattern becomes 3 ft by 6 ft in the back alternating with 8-ft and 12-ft bolts.
- Shotcrete applied as needed with minimum thickness of 2 inches.

If heading dimensions exceed those listed above or if ground conditions change the ground support will be modified to ensure a safe working environment for all personnel. Additional details on ground support is available in the 2022 GCMP.

This ground control process will be under constant review by management, engineering staff, safety department, MSHA and most importantly the miners to ensure a safe working environment. Once the ground control cycle is completed, the jumbo drill is brought back into the heading and the production cycle is started again.

16.5.3 Development and Production Schedule

The mine operations schedule is based on 365 days/year, 7 days/week, with two 10-hour shifts each working day. There are four crews scheduled working a four-on, four-off schedule. The production rate at full production is 600 tons per day with a 3-month ramp up period. Each stope is calculated to be able to produce 176 tpd, based on that assumption 3.5 active faces are required to meet production requirements. Due to inefficiencies in developing new stopes, backfill placement, and unplanned delays, a total of six active areas are scheduled in the mine plan.

Table 16-4 presents the annual mining schedule based on these assumptions. The stoping 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 development mill feed 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 16-4 Annual Mining Schedule

Production Schedule	Life-of-Mine	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
MINE PRODUCTION								
Tons Mill Feed Mined	1,222,317	18,148	199,678	240,468	232,153	200,989	222,630	108,251
Au, oz/ton	0.197	0.184	0.256	0.208	0.195	0.184	0.167	0.155
Development Feet	61,416	3,330	14,378	14,721	9,947	6,916	8,153	3,972
Development Waste	815,686	50,864	233,713	237,774	135,301	57,400	67,668	32,965
Total Tons Mined	**2,038,003**	**69,013**	**433,391**	**478,242**	**367,454**	**258,389**	**290,298**	**141,216**

Table 16-5 displays a summary of the resource classifications for the mill feed material.

Table 16-5 Resource Summary of Scheduled Material

Mineral Resource Class	Tons ('000's)	oz/ton	Contained Gold ('000 oz)	Dilution
Measured	562.5	0.230	117.8	30.6%
Indicated	220.0	0.219	43.7	32.0%
Measured + Indicated	**782.5**	**0.227**	**161.5**	**31.0%**
Inferred	439.9	0.198	79.0	33.7%

The cut and fill stopes were split up into seven different areas for the scheduling process. The LOM development and stopes by area are shown in Figure 16-8. As stated previously there will be two crews working on the main haulage ramps.

The scheduling process does not address the detail within each cut and fill stope level, instead the schedule assumes that as soon as the stope access ramps reach the mineralization, production will commence at a rate of 176 tpd until the complete level for that area is mined out. When the level mineralized material is mined out, backfill is placed at a rate of 263 yd^3/day until the stope is filled. When the stope is filled the stope access ramp backslashing for the next lift starts. Over the monthly timeframe used for the representation of the scheduling results in the economic model, this lack of in-stope scheduling detail is acceptable.

shows the stope mine schedule by area over the life of the mine. Figure 16-6 shows the mine schedule by stope area for the tons mined and subsequent grade and Figure 16-7 shows the tons mined and contained ounces for each stope area.

Table 16-6 Annual Mining Schedule

Zone	LOM	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
A Zone Tons	57,606	-	-	-	12,604	45,001	-	-
A Zone Au opt	0.161	-	-	-	0.157	0.162	-	-
A Zone Ounces	9,251	-	-	-	1,973	7,278	-	-
B Zone Tons	197,977	-	-	76,132	93,533	28,312	-	-
B Zone Au opt	0.159	-	-	0.148	0.173	0.139	-	-
B Zone Ounces	31,403	-	-	11,261	16,201	3,942	-	-
C Zone Tons	454,165	9,406	117,675	164,336	94,761	54,036	13,950	-
C Zone Au opt	0.221	0.136	0.250	0.235	0.203	0.182	0.148	-
C Zone Ounces	100,472	1,282	29,396	38,642	19,259	9,831	2,062	-
D Zone Tons	115,560	8,742	82,002	-	3,776	21,039	-	-
D Zone Au opt	0.253	0.236	0.264	-	0.195	0.228	-	-
D Zone Ounces	29,266	2,063	21,665	-	736	4,802	-	-
FW Zone Tons	213,026	-	-	-	-	-	116,323	96,703
FW Zone Au opt	0.163	-	-	-	-	-	0.172	0.153
FW Zone Ounces	34,758	-	-	-	-	-	19,986	14,772
S Zone Tons	162,709	-	-	-	27,478	52,601	77,537	5,093
S Zone Au opt	0.195	-	-	-	0.259	0.210	0.164	0.169
S Zone Ounces	31,709	-	-	-	7,117	11,037	12,694	861
UPFX Zone Tons	21,275	-	-	-	-	-	14,820	6,455
UPFX Zone Au opt	0.173	-	-	-	-	-	0.171	0.178
UPFX Zone Ounces	3,679	-	-	-	-	-	2,531	1,148
Total Tons	**1,222,317**	**18,148**	**199,678**	**240,468**	**232,153**	**200,989**	**222,630**	**108,251**
Total Au opt	**0.197**	**0.184**	**0.256**	**0.208**	**0.195**	**0.184**	**0.167**	**0.155**
Total Ounces	**240,538**	**3,345**	**51,061**	**49,903**	**45,286**	**36,890**	**37,272**	**16,781**

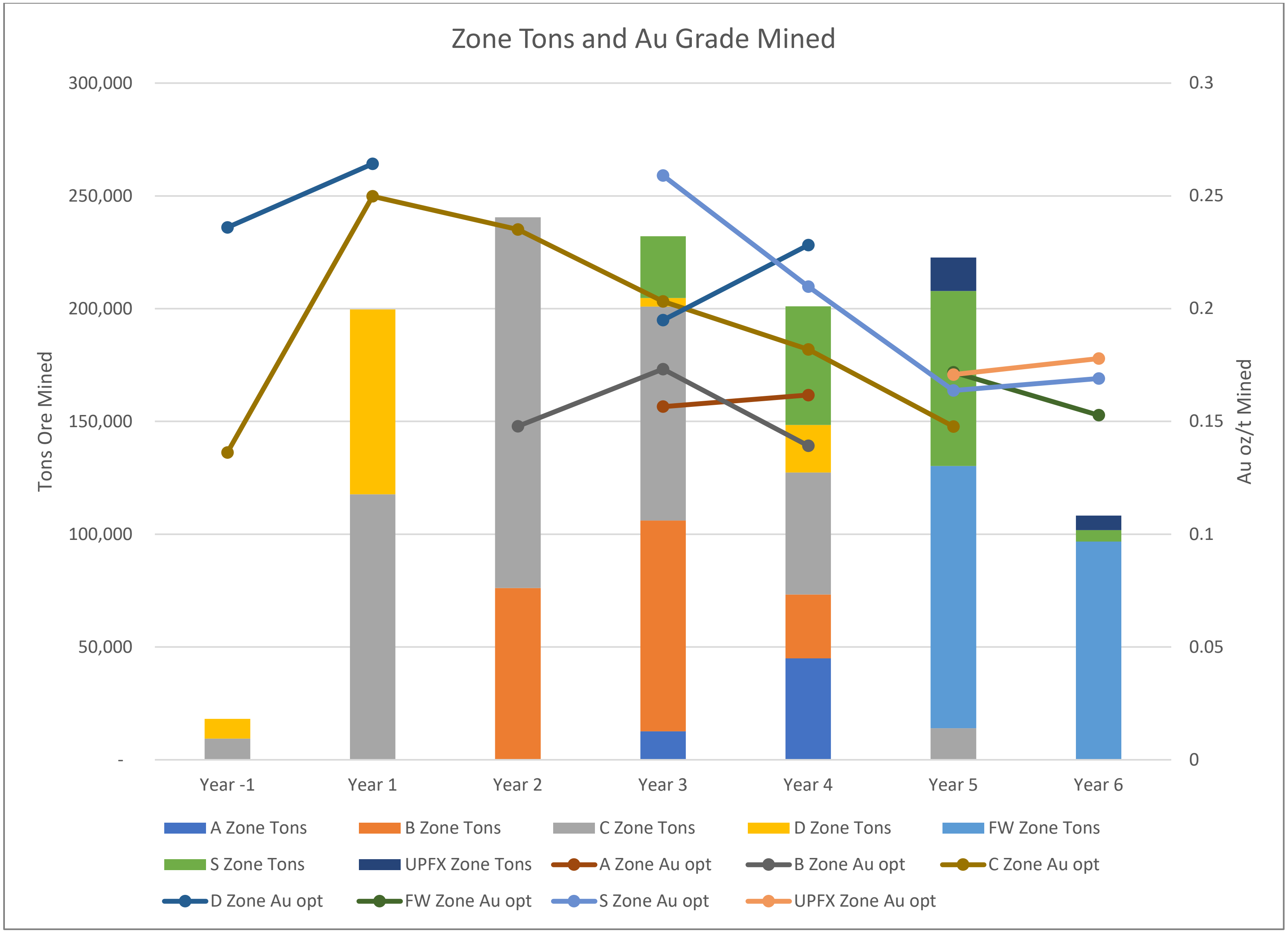

Figure 16-6 Stope Schedule by Area

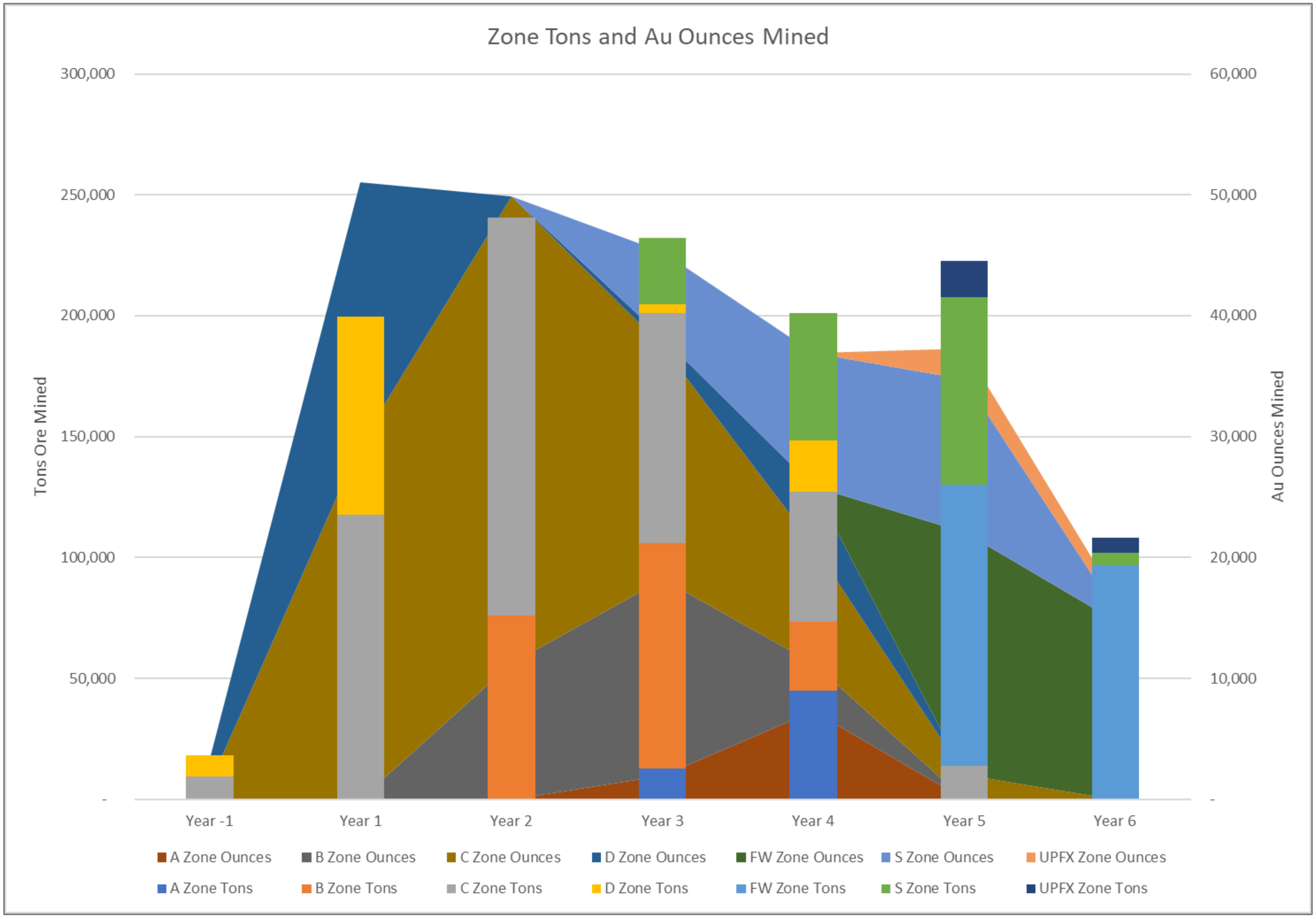

Figure 16-7 Stope Schedule by Area

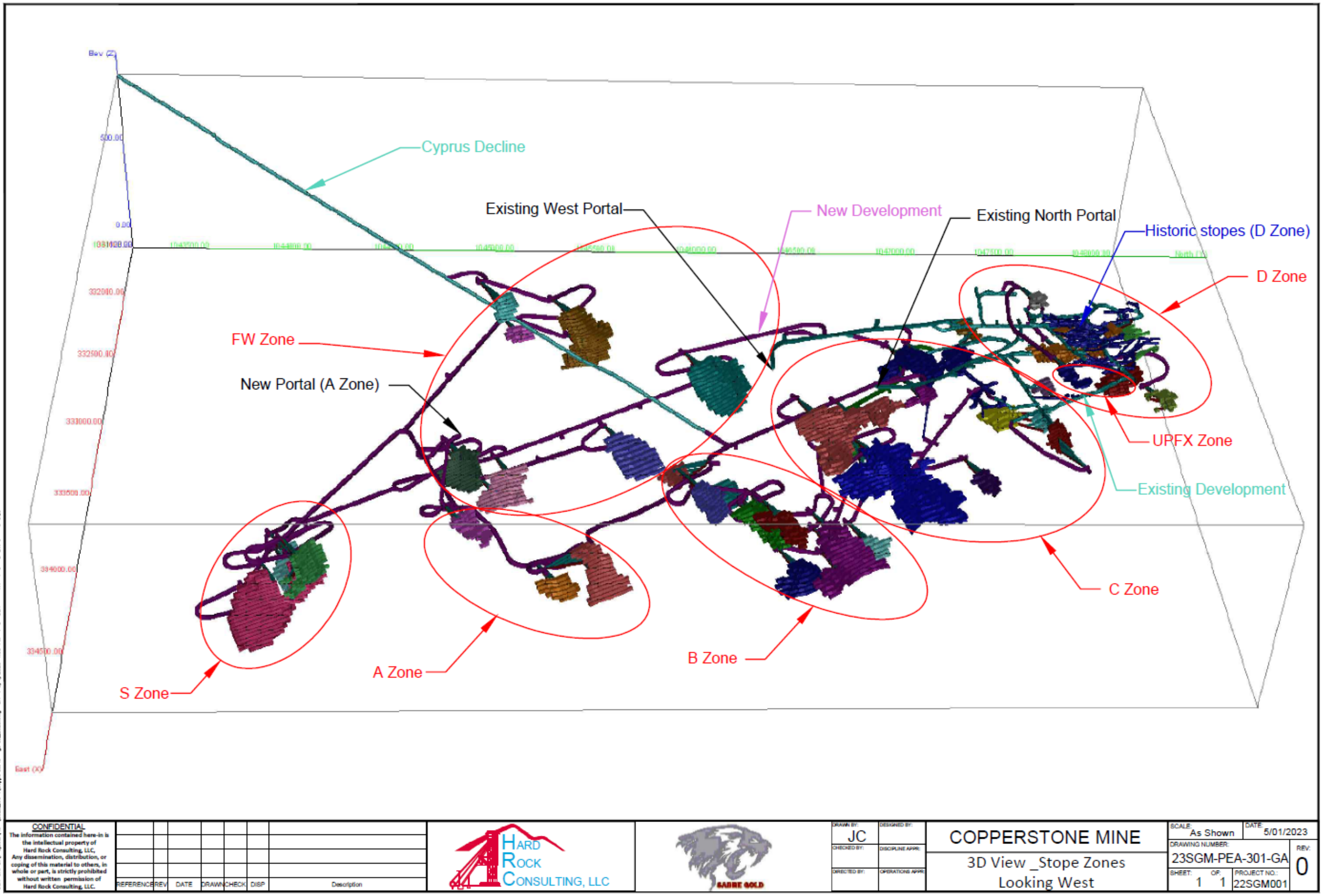

Figure 16-8 Stope Design by Area

16.5.4 Backfill

The primary consideration for backfilling is to achieve a consistent tight fill throughout the wider cut and fill areas. This is necessary where multiple drifts will be mined adjacent to each other on the same lift. If tight fill is not achieved, in these areas, the apparent span could get very large resulting in a potential for stope back stability issues. In order to achieve consistent tight fill the mine design will utilize a cemented rockfill ("CRF") and non-cemented rockfill ("RF") backfill methodology. The CRF is packed into the stope using a 3 m^3 LHD set up with a rammer thus ensuring a completely tight fill through the length of the stope. Backfill placement on a per stope basis is scheduled at an average rate of 263 yd^3/day. Taking into account that more than one stope will be in the backfill phase at one time the average backfill rate over the life of the mine is 236 yd^3/day.

In 2021, MineFill Services completed a study that included laboratory testing of aggregates to produce a suitable backfill mix design, and a review of backfill methodologies given the difficult ground conditions and proposed stope geometries. Based on the testing MineFill developed the following CRF general specifications:

- Aggregate intact rock density – 2.6 tonnes/m^3
- CRF density – 1.9 tonnes/m^3
- 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 to be placed in the pit so waste rock from either run of mine development waste or waste stockpiled in the pit from the open pit operations can be screened and mixed with slurry for the stoping areas that require CRF. All of the backfill will be placed by end-dumping and jamming tight to the back or hanging wall, hence segregation of the aggregate is not a risk. That being the case, MineFill still recommends avoiding waste fragments over 6-inches in size as it will impact the backfill density and hence the stiffness and strength. Even with the lack of an aggregate grading specification and a relatively coarse top size, it will be important to ensure the CRF contains a suitable proportion of fines (sizes less than 10 mm) or the resultant CRF will end up with significant void space and significantly reduced strengths. The selection of a suitable rock type for use as CRF aggregate will also be important to maintain CRF quality. In general, the phyllites should be avoided for batching CRF but the granites are expected to be suitable as was confirmed in the laboratory testing of CRF mixes.

16.5.5 Surface Mill Feed Haulage

Mill feed haulage from the bottom of the open pit to the mill feed stockpiles at the mill is scheduled using a 4m^3 front end loader and two 40 tonne articulated dump trucks. The loader is also used for loading the underground trucks when backfill is required from the open pit waste dump stockpile. The schedule for the surface mill feed haul assumes that there would be two crews working a four on four off schedule 10 hours per day with no night shift. Based on these parameters, two trucks are required to meet the mill schedule.

16.6 Mine Ventilation

16.6.1 Existing System

The existing ventilation circuit consists of a Jet-air 150hp fan with vent ducting and a Spendrup 100hp fan blowing air down the West portal. The air travels down the west decline to an intersection of the D zone declines, West decline and North decline, where the air travels up the North decline to exit out the North portal. At the intersection, the air flow is directed to the D zone through a 50hp fan that carries the air through ducting to the 730 access drift intersection. A 25hp fan carries the air to the dewatering station in the 730 HW decline. At the C zone decline the air travels through a Spendrup 100hp fan via ducting to the exploration drift developed during the summer of 2017. The current system has been measured to have air flows up to 110,000 cfm.

16.6.2 Ventilation System Upgrade

The total ventilation requirements for the mine at a production rate of 600 tpd are estimated to be 226,600 cfm as shown below in Table 16-7. The planned ventilation upgrade includes a full fan drift and airlock system to provide flow through ventilation. The plan includes 160 feet of drift to be driven out of the west decline, a bulkhead and a 150 hp fan will be installed in the fan drift. This fan will provide approximately 180,000 cfm at 2 inch of water gauge and will need to be upgraded as the mine expands in production, initial Ventsim models have been developed but will need to be updated as a more detailed mine plan is developed for the Project. An airlock consisting of two Hoffman bi-fold door systems with suitable PLC, electronic eyes, and hydraulic controls will be placed in the main decline to provide a positive seal and allows equipment to pass through without interruption to the ventilation circuit as shown if Figure 16-9. Once installed, the fans, ducting, and roll-up door at the west portal can be removed which will allow the west portal to be used as a main haulage. The North portal will continue to function as an exhaust and the A zone and Cyprus decline will function as exhaust portals for mining of the A, B , S and Footwall zones. As discussed previously a vent raise is planned to provide air from the West decline intake down to the bottom of the C zone which will then travel out the current exploration drift and eventually exhaust out the North decline.

Additional fans will be required in the underground areas of the mine to direct the air flow as required. The two current 100 hp Spendrup, fans and two current 25hp and 50hp current jet-air fans will be used as booster fans. Smaller auxiliary stope fans and flexible vent bag are planned to direct the ventilation air to the working faces. The 30hp auxiliary ventilation stope fans are used to direct fresh air from the ramp into each working stope. There will be up to six auxiliary stope fans running at any one time. These improvements to the ventilation circuit have been accounted for in the capital schedule for the underground mine development and operation.

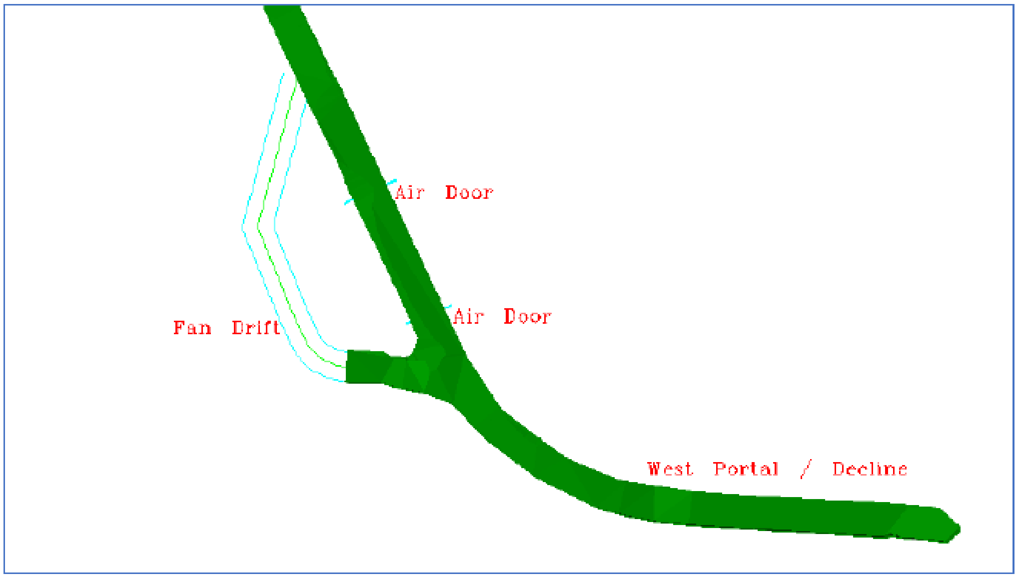

Figure 16-9 West Decline Fan Drift

Table 16-7 Ventilation Airflow Requirements

ITEM/DESCRIPTION	OPERATING QUANITITY (EA)	MOTOR SIZE (hp)	OPERATING FACTOR	VENTILATION REQ. (cfm)
DEVELOPMENT/PRODUCTION EQUIPMENT				
JUMBO Single boom	2	75	10%	1,500
JUMBO Double Boom	1	150	10%	1,500
LHD - 3 m3	6	215	65%	83,900
LHD - 1.3 m3	2	67	65%	8,700
TRUCK 30 Tonne	3	415	68%	84,700
SUPPORT EQUIPMENT				
Scissor Lift - Bolter	4	147	25%	14,700
Lube/Fuel Truck	1	147	35%	5,100
Boom Truck	1	147	20%	2,900
Man carrier	1	147	5%	700
Road Grader	1	147	40%	5,900
Tractor	2	58	15%	1,700
Buggies	4	25	10%	1,000
Telehandler	1	147	10%	1,500
MISC. EQUIPMENT			25%	8,375
People	22	200		4,400
TOTAL				**226,600**
VENTILATION REQUIREMENT	100	CFM/BHP		

16.7 Electrical Services and Compressed Air

A review of the current electrical system and the required upgrades for the mine to operate at 600 tpd were provided by Bruno Engineering. Based on the power requirements for the new mine plan the main substation is planned to be upgraded to a 15/20 MVA oil-filled power transformer with a new capacitor bank and circuit breaker. A new motor control center, switchgear, overhead power line and infrastructure for new underground feeders are also planned.

For the ventilation circuit a new PLC automation control system is planned along with a new fan motor starter and feeder. Two new power centers along with switch houses and motor start assemblies for each center are planned. New underground 15kV, Type MP-GC cable is planned for the power centers. Power feeder cable is also planned for the jumbo's, inline fans, stope fans, dewatering pumps and face lighting.

Compressed air will be required in the development and production headings to operate the jumbos, handheld jackleg and stoper drills required for drilling holes for ground control bolts, and utility requirements throughout the mine. The current compressor equipment, located on the surface, will be used to supply the compressed air requirements for the underground mine. The compressor (Atlas Copco 381hp model GA315VSD) is rated at 1,394 ft^3/min at a pressure of 150 psi.

16.8 Mining Equipment

The mine plan is based on contract mining for the initial pre-production development and then for the first seven months of production during Year 1. 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 16-8. A two boom Jumbo drill is planned for the main ramp development and two single boom jumbos are planned for the stope access development headings and stope headings. The electric/hydraulic drills will be operated by one operator, with help from the mining crew during setup and tear down for each heading.

Six 3.9 yd^3 LHD's are planned for the development headings and mill feed production headings, in addition two smaller 1.7 yd^3 LHD's are planned for any narrow mill feed headings in order to minimize dilution. The LHD's will muck the development and/or mill feed production headings to the respective muckbay for that area. Once the heading is mucked out the bolting process can begin. The mill feed/waste in the muck bay is subsequently loaded onto the haul trucks for transport to the surface mill feed stockpile or designated backfill area underground. Development waste will be stored in muckbays near the production areas where the waste will subsequently be used for CRF backfill placement to allow an optimal turnaround time for the heading.

A total of three 33 ton trucks will be required to haul the mill feed material to the surface and waste material to the backfill area as required. The trucks are estimated to hold 30 ton/trip and will be loaded by the 3.9 yd^3 LHD's.

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 yd^3 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 16-8 Mining Equipment

Type	Quantity
UNDERGROUND EQUIPMENT	
Truck 33 t (30 tonne)	3
LHD – 3.9 yd^3 (3 m^3)	6
LHD – 1.7 yd^3 (1.3 m^3)	2
Jumbo 2 boom	1
Jumbo 1 boom	2
Bolter	2
Scissor Lift	2
Grader	1
Lube/Fuel Truck	1
Personnel Buggies	4
Shotcrete Sprayer	1
Boom Truck	1
Explosives Truck	1
Pallet Handler	1
SURFACE EQUIPMENT	
Front Loader 4 m^3 - Used	1
Articulated Truck 40t - Used	2
5,000 gallon water truck - Used	1
Surface dozer - D6 Used	1

17. RECOVERY METHODS

17.1 Flowsheet Development

Metallurgical testing for the Copperstone deposit has evaluated three process options:

- Grinding and flotation to produce a gold concentrate;
- Grinding, flotation, and cyanide leaching of the gold flotation concentrate; and
- Grinding and whole ore cyanide leaching.

The final two options produce doré bars which provide a marketing advantage over the production of flotation concentrates. Testing was conducted by RDI for screening these flowsheet options. The test work results indicated a 97% recovery of gold, while the other processes were 88% to 90% recovery. Based upon economic and technical analysis, whole ore leaching was chosen as the best processing option for the Copperstone deposit. Although the operating costs for the whole ore leaching of Copperstone mineralized material are the highest operating costs of the three options, the increase in recoveries and elimination of smelter charges make this option economically superior.

The flowsheet developed for the Copperstone deposit is shown in Figure 17-1. It takes advantage of the existing primary crusher, secondary crusher, screen, ore bin, conveyors and ball mill. This equipment will be refurbished prior to commissioning the process plant. The target grind size is 104µm; the existing ball mill is larger than required for the target 600 short tons per day throughput and will operate with reduced steel load and draw lower power.

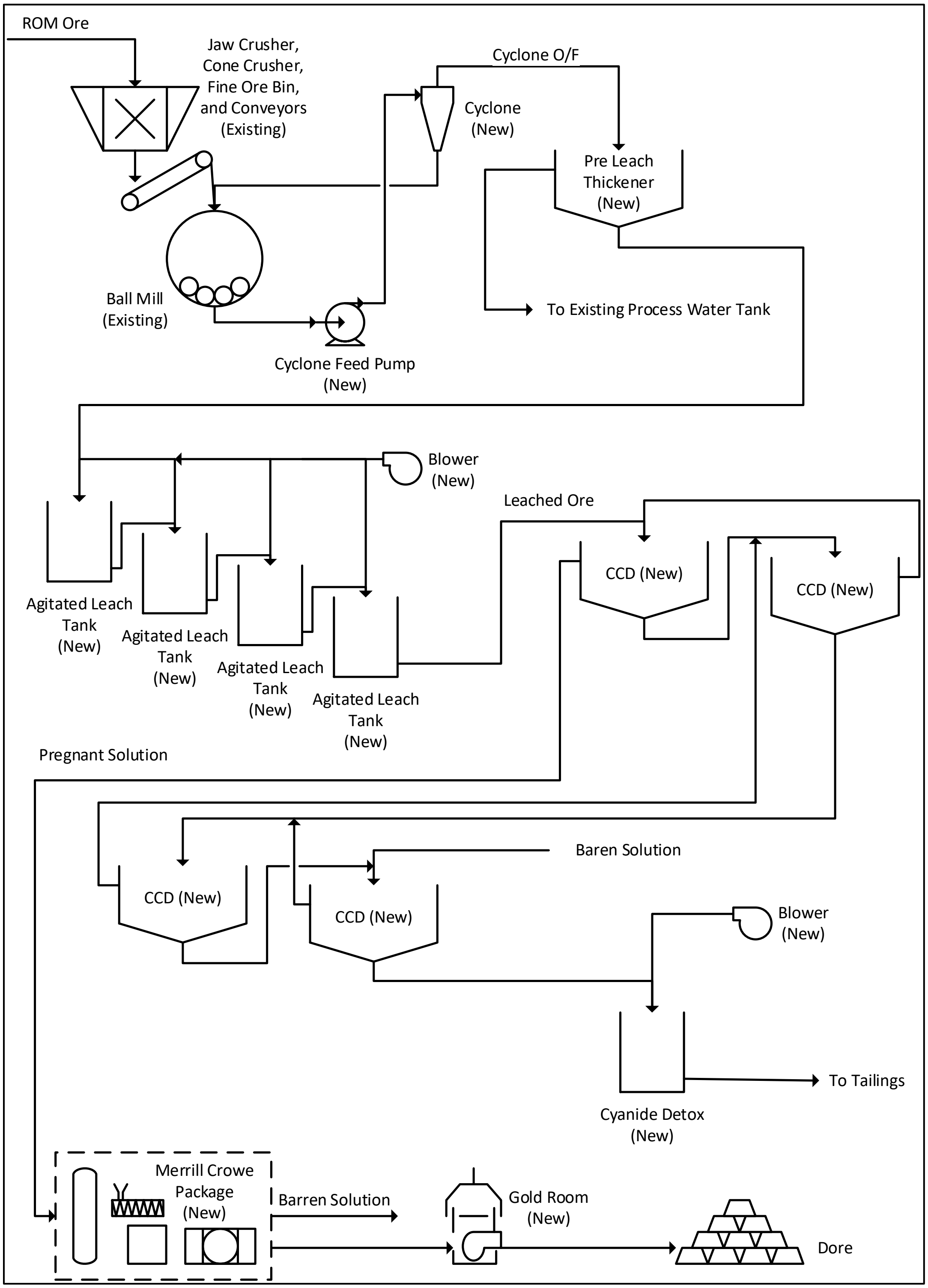

Figure 17-1 Whole Ore Leach Process Flow Diagram

17.2 Design Criteria

The Copperstone process plant design is based on the mine delivering 198,560 tonnes of mineralized material per year (219,000 short tons per year). The process plant will operate seven days per week with a plant availability of 92% establishing a design capacity of 592 tonnes per day (652 short tons per day). Table 17-1 presents the general design parameters for the process plant.

Table 17-1 General Design

General		
Plant Tonnage	Tonnes Per Day	544
	Short Tons Per Day	600
Plant Availability	Percent	92
Operation	Hours Per Day	24
	Days per Week	7
Design Capacity	Tonnes Per Day	592
	Short Tons Per Day	652
ROM F80	mm	102
ROM Specific Gravity	-	3.18

The crushing and grinding circuit is currently in place and will require only minor modification. Hence, the design criteria for the grinding circuit is presented in Table 17-2. The design criteria for the selected whole ore leach processing scheme discussed above is given in Table 17-3.

Table 17-2 Crushing and Grinding Design

Crushing and Grinding		
Primary Crusher	Type	Existing Jaw
Secondary Crusher	Type	Existing Cone
Milling Feed F80	µm	12500
Fine Ore Bin	-	Existing
Bond Ball Mill Work Index	Average, kwh/t	13.94
	Maximum, kwh/t	18.03
	Minimum, kwh/t	10.04
Ball Mill	Type	Existing Hardinge
	Motor Power, hp	700
	Percent of Critical	75
	Ball Diameter, Inches	3
	Ball Charge, Percent	25-35
Cyclone	Type	New 20 inch
	Number	1
Circulating load, Design	Percent	250
Cyclone Overflow P_{80}	µm	104

The new whole ore leaching equipment and the solution recovery process will be installed as per Tables 17-3 and 17-4. The design implements a pre-leach thickener followed by 4 leaching tanks providing 27 hours of total residence time. Pregnant solution is recovered through four CCD thickeners. Pregnant solution is processed in a Merrill Crowe circuit to precipitate gold, silver, and copper. The product will be batch leached with acid in the gold room to extract the residual copper, which can be recovered using cementation process, prior to smelting and pouring doré bars.

Table 17-3 Whole Ore Leaching Design

Leach Circuit		
Pre-Leach Thickener	Type	New High Rate
	Diameter, m	6.1
	Diameter, ft	20
	Settling Rate, ft^2/tons per day	295
	Underflow Density, Wt%	45
Leach Tanks	Feed Specific Gravity	1.45
	Design Leach Time, hr	18
	Quantity	4
	Height, ft	30
	Diameter, ft	20
	Operating Volume, m^3 each	258
	Operating Volume, gal each	68150
	3 Tanks Residence Time, hr	20.4
	4 Tanks Residence Time, hr	27.2
	Free Board, %	10
Sodium Cyanide Concentration	ppm	1750
Sodium Cyanide Consumption	Design lbs per short ton	4.3
	Design kg per tonne	2.15

Table 17-4 Solution Recovery and Merrill Crowe Design

Solution Recovery and Merrill Crowe		
CCD Thickeners	Quantity	4
	Wash Ratio (Barren/Feed Solids)	2.3
	Wash Ratio (Barren/Feed Aqueous)	2.7
	Settling Rate, m^2/tonnes per day	0.04625
	Settling Rate, ft^2/tons per day	0.452
	Thickener Diameter, ft	20
	Thickener Diameter, m	6
	Thickener UF Density, wt%	50
Pregnant Solution Tank	Residence Time, hr	0.67
	Volume, m^3	58
Barren Solution Tank	Residence Time, hr	2
	Volume, m^3	174
Detox Tank	Feed Percent Solids, wt%	50
	Residence Time, hr	4
	SMBS Consumption, lb/ton	0.248
	SMBS Consumption, kg/tonne	0.497
	Detox Tank Volume, m^3	128
Merrill Crowe	Feed, gpm	383
	Feed, m^3/hr	87
	Feed Design, gpm	459
	Feed Design, m^3/hr	104
	Pregnant Solution, Au g/m^3	1.82
	Pregnant Solution, Ag g/m^3	0.12
	Pregnant Solution, Cu g/m^3	262
	Pregnant Solution, CN g/m^3	383
	Zinc Consumption, kg/hr	36

17.3 Plant Equipment

Dr. Deepak Malhotra visited the Copperstone property in February 2018 and reviewed the plant flowsheet and equipment in the plant. The present test work indicated that the deposit does not have free coarse gold. In addition, samples representing softer material and harder material were tested for Bond's Ball Mill Work Index at a closed size of 100 mesh. The two composite samples yielded BWi values of 10.74 kWh/ton and 18.04 kWh/ton. The samples were also submitted for Bond's Abrasion Index determination. The Ai values were 0.0259 and 0.359. Based on the gravity testing and Work Index testing the Knelson concentrator and rod mill, which have been removed will not be required to be replaced.

The list of major equipment with specifications and projected throughput is given in Table 17-5. The list of equipment that needs to be removed from the plant along with the additional equipment required for the whole ore leach process is given in Table 17-6. The site previously had a rod mill which has been removed already. Pumps, reagent tanks and reagent pumps have been lumped together into miscellaneous equipment and have not been classified as major equipment.

Table 17-5 Existing Equipment

Equipment	Make	Size	HP	No.	Projected Throughput TPH
Jaw Crusher	Nordberg	32 inch X 40inch	150	1	185
Conveyor Belt	Nordberg	36 inch wide	10	1	
Triple Deck Screen	Trio	6 ft X 16 ft	40	1	
Cone Crusher	Symons	4 ft	150	1	110-130
Ball Mill	Hardinge	11 ft 4 inch X 6 ft 10 inch	700	1	26-43
Ball Mill Sump Pump	Vacseal	4 inch X 3 inch VD	30	1	
Rougher Flotation Cells	Denver	50 ft^3	64	16	19-28
Cleaner Flotation Cells	Denver	25 ft^3	8	2	
Cleaner Flotation Sump Pump	Sala	2 inch	10	1	
Cleaner 2 Flotation Cells	Denver	12 ft^3	8	2	
Tailings Pump	Warman Style	3 inch X 2 inch	75	1	

Table 17-6 Additional Major Equipment Required

Whole Ore Leach Additional Equipment
Ball Mill Cyclone
20-ft Diameter Leach Feed Thickener
Four Leach Tanks
Four 20-ft diameter CCD Thickeners
Merrill Crowe Plant and Gold Room
Reagent Systems

17.4 Process Plant Operation

The complete plant has been designed for 92% plant availability. The new process will use the existing crushing circuit, conveyed to the ball mill, where it is ground to P_{80} of 104 µm, in a closed-circuit arrangement. The cyclone overflow will then be pumped to the new leach feed thickener. The thickener underflow is pumped at 50% solids for cyanide leaching followed by counter current decantation for pregnant solution recovery. Pregnant solution is processed by Merrill Crowe to recover the gold. The precipitated Merrill Crowe product will be high in copper which would be batch leached in the gold room to leach the copper prior to smelting and pouring doré bars. The tailings will be detoxified by SMBS and aeration prior to being sent to the tailing's storage facility. The process plant consumables summary is shown in Table 17-7.

Table 17-7 Process Plant Consumables Summary

Process Consumables		
Jaw Crusher Liners	Set/Year	1
Cone Crusher Liners	Set/Year	1
Ball Mill Liners	lbs/short ton milled	0.9
Grinding Media	lbs/short ton milled	2.3
Quicklime, 90% CaO w/w	lbs/short ton milled	5.2
Cyanide, NaCN 20%	lbs/short ton milled	4.3
Zinc Powder	lbs/short ton milled	3.2
Hydrated Lime, 90% $Ca(OH)_2$ w/w	lbs/short ton milled	0.39
SMBS, Na_2SO_5	lbs/short ton milled	0.97
Smelter Flux, Silica	lbs/short ton milled	0.0068
Smelter Flux, Borax	lbs/short ton milled	0.0101
Smelter Flux, Sodium Nitrate	lbs/short ton milled	0.0101
Smelter Flux, Fluorspar	lbs/short ton milled	0.0033
Flocculant	lbs/short ton milled	0.16

The new whole ore leaching plant will be constructed to the north of the former flotation plant location. The planned layout is shown in Figure 17-2 and isometric views are shown in Figure 17-3 and 17-4.

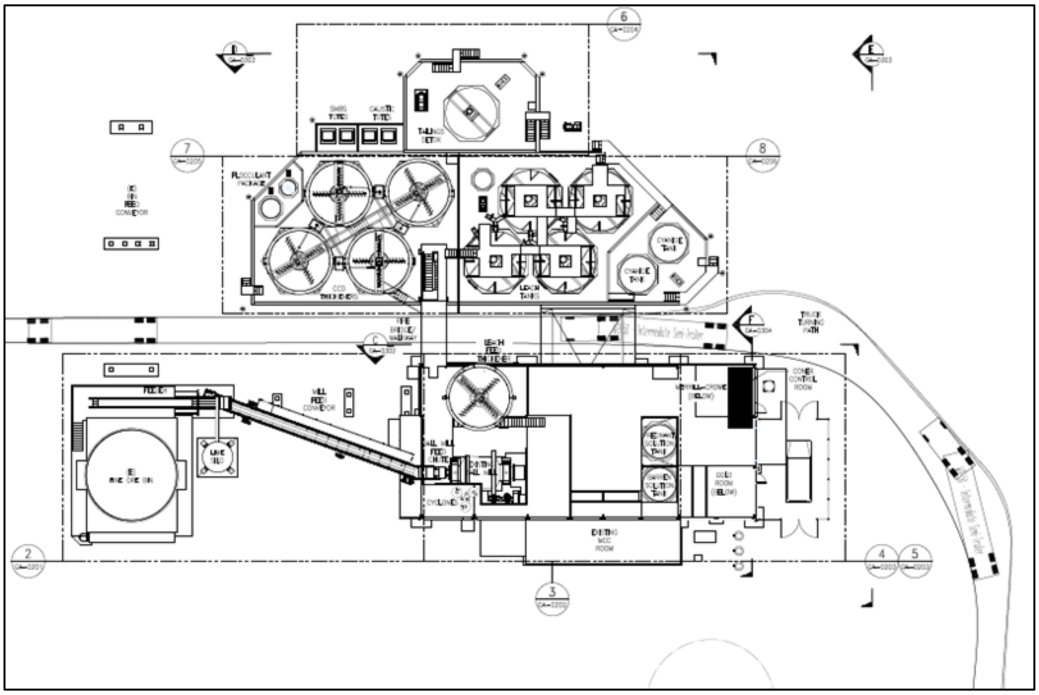

Figure 17-2 Plant Layout

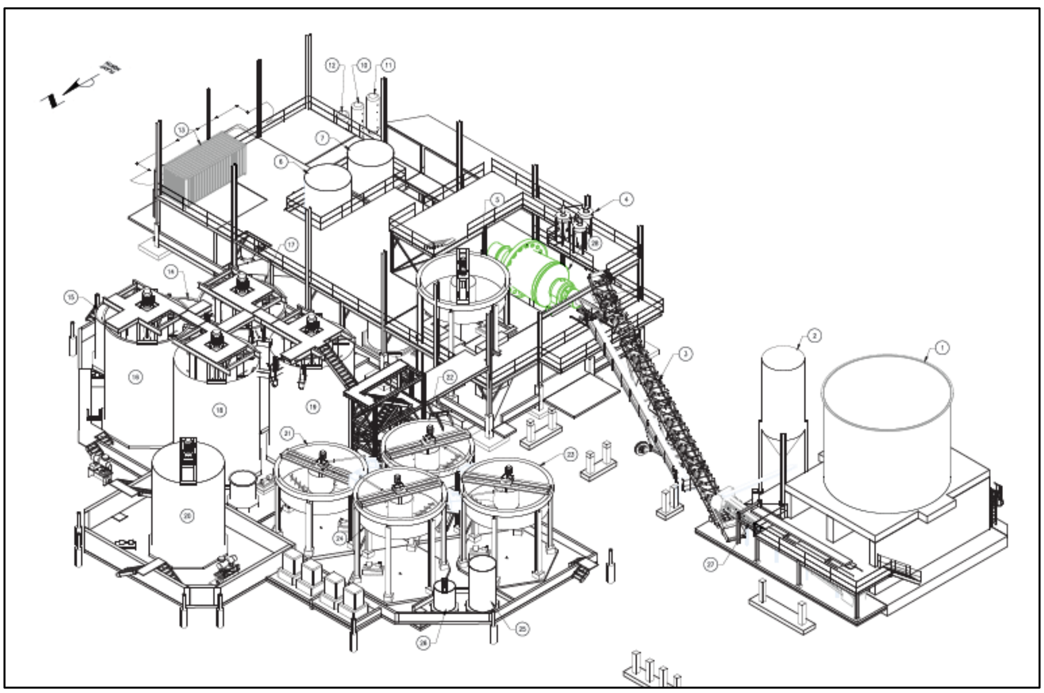

Figure 17-3 Process Plant Isometric View

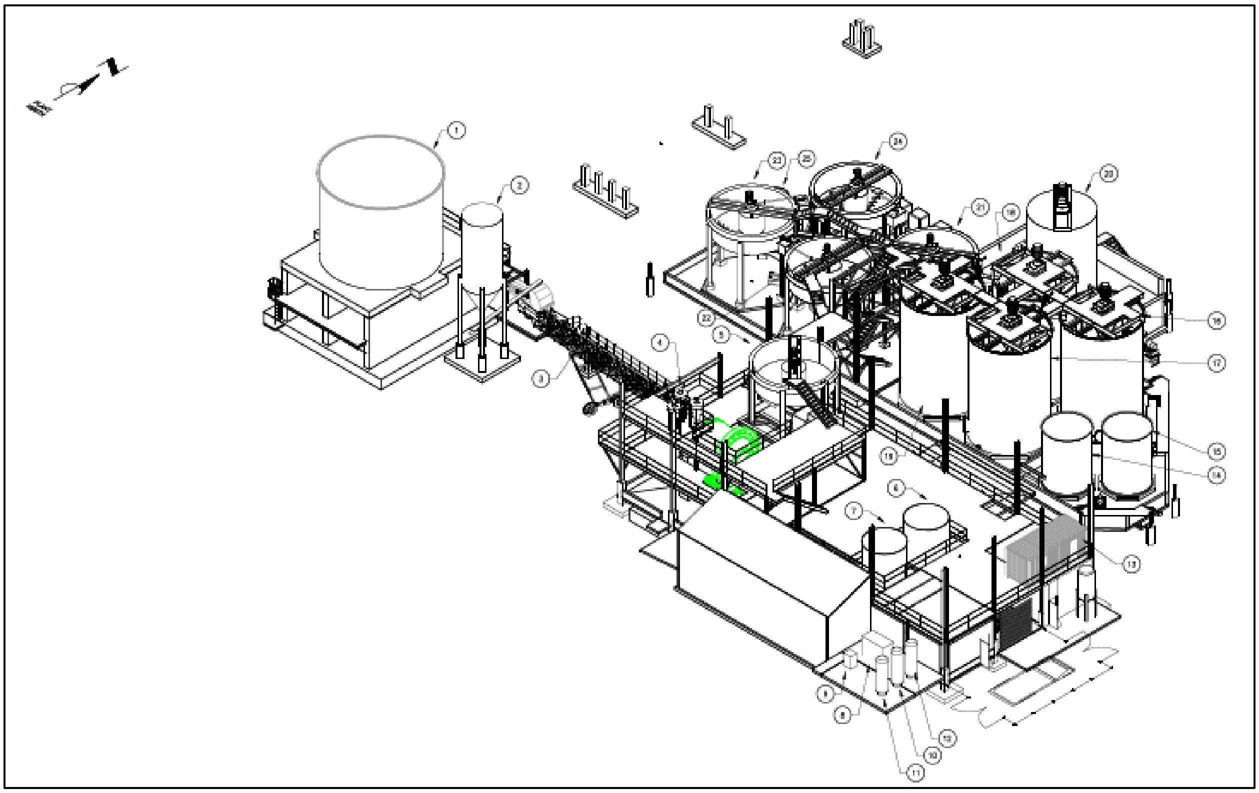

Figure 17-4 Alternative Isometric View

18. PROJECT INFRASTRUCTURE

The Copperstone deposit is favorably located across flat, sandy desert. A 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 SGLD'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 complex is located to the northwest and south of the process plant. See Figure 18-1 for the current infrastructure layout.

This office facility accommodates the accounting, purchasing, human resources, health and safety, and environmental personnel, as well as the general management and engineering staff. Senior mine and maintenance personnel will have offices in the mine shop. An office is included in the laboratory for the lab manager.

The general and administrative offices have both internal and external communication service.

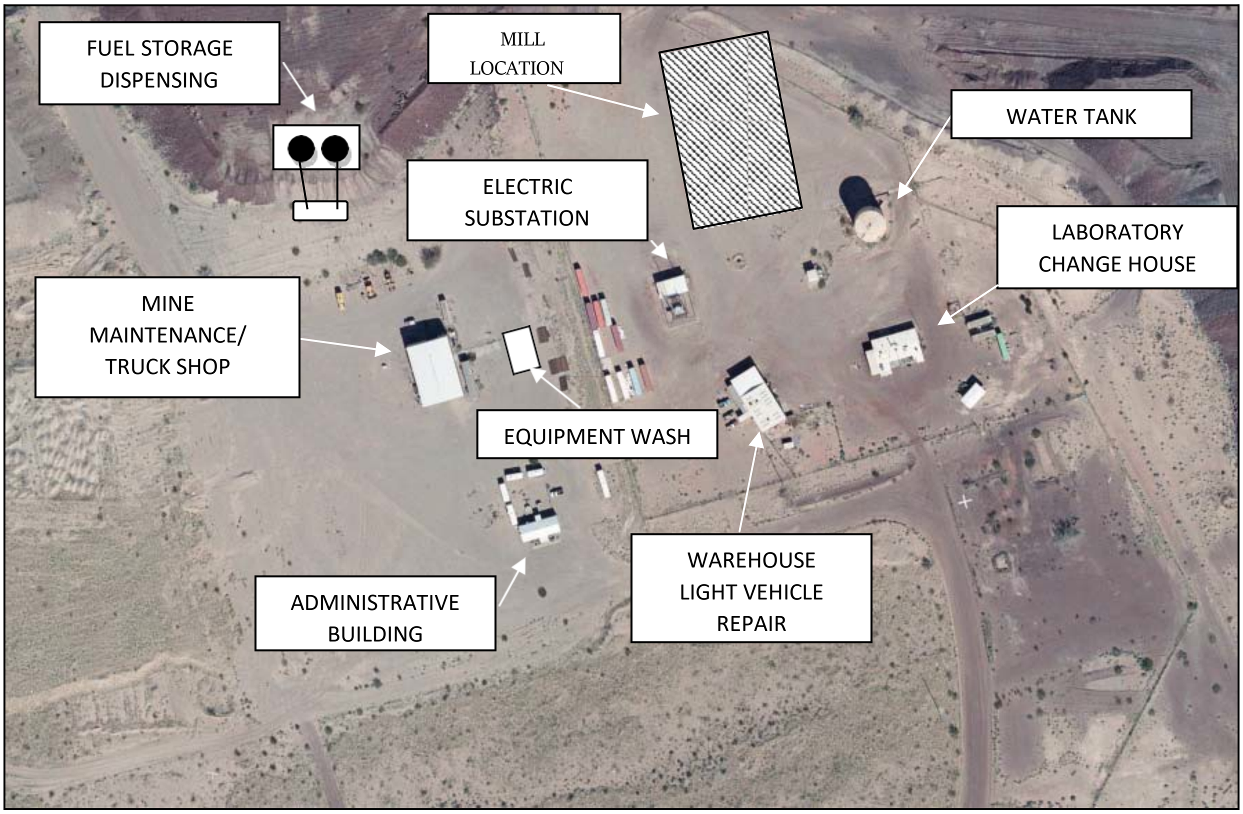

Figure 18-1 Ancillary Facilities

18.1.1 Office

The currently constructed administration building is approximately 1,950 ft^2 and will house all the administrative and management personnel.

The administration 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 Shop

The mine shop is a prefabricated steel construction butler type building with tin siding and is approximately 5,400 sf. Two-inch rolled fiberglass insulation covers the roof and walls with plywood around the base of the walls. The facility is located on the northwest side of the facilities area. Two truck bays, compressed air piping, welding equipment outlets, offices and a tool store are included. The mine shop is a "drive- through" facility with rail placed in the floor to protect the concrete from tracked equipment. Maintenance on all heavy vehicles will be conducted in this shop.

18.1.3 Warehouse

The warehouse is located on the south end of the facilities area and is approximately 2,600 sf. The warehouse is a prefabricated steel construction building with tin siding. All project supplies, with the exception of process plant reagents, are received, stored and dispersed from this facility. The facility includes inside storage for parts and supplies, an office and a tool crib/small parts area. A fenced storage yard south of the warehouse is used to store large items or bulk materials which can withstand exposure to the elements. The outside storage areas consist of a compacted, graveled area.

Most reagents and chemicals delivered to the project are for use at the process plant or at the laboratory. These materials are checked in at the main warehouse, then delivered to these facilities and stored there. An attached area to the existing plant facility provides storage of bulk reagents.

The lubricants and oil storage is located outside on the northeast side of the mine shop. These are stored in their individual containers and all are within a single steel containment unit.

Light vehicles and small mobile equipment maintenance is performed in the smaller shop attached to the warehouse. The area has reinforced concrete and an overhead crane.

18.1.4 Laboratory/Change House Area

The analytical laboratory is a single story pre-engineered building with corrugated roofing and siding located with the change house just east of the entrance to the plant. The laboratory is approximately 1,750 ft² and consists of a sample preparation area, wet laboratory, fire assay metallurgical laboratory, environmental laboratory, offices, lunchroom and restrooms. A 10-ft double door provides access at the north end of the building to receive materials into the sample preparation area.

The sample preparation area is isolated from the analytical laboratory by a wall. It will contain sample dryers, crushers, pulverizers, sample splitters, and a dust collection system to capture and contain any dust generated from this operation.

The analytical laboratory contains the wet laboratory, reagent storage area, balance rooms, and analytical equipment. Also included is a facility to collect and manage waste chemicals in the laboratory. Disposal of the chemical or laboratory wastes follows appropriate regulatory requirements dependent upon the waste generated. Laboratory offices are also included in this facility.

The employee change house is a single story pre-engineered steel building with corrugated metal roofing and siding located with the laboratory building just to the east of the plant area, inside the fence line. Employees park in the designated parking area inside the fence and walk about 50 ft to the change house. The Change House facility is approximately 1,750 ft² with separate changing rooms, with showers and bathrooms, provided for men and women.

A full septic system capable of 1,500 gallons per day is in place.

Waste products from the laboratory are generally compatible with the milling operation and are returned to the milling circuit.

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 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 on the 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 contained concrete pad is equipped for washing heavy and light vehicles. The wash station is located to the west of the mine shop and designed as a drive-through facility. Pressure washers and associated hoses/tanks exist. This pad is sloped to the central collection area to prevent any contaminated solution from being discharged. A concrete sump designed to be emptied with a loader is used to remove mud and material from the facility.

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 Cyprus 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.

One elevated ammonium nitrate silo, with 75-ton capacity, will be located to the south of the mine entrance in a location that allows for easy loading and turnaround. This area is also convenient for the mine ANFO trucks to fill up with ammonium nitrate and diesel before going to the mine. The ammonium nitrate and diesel are not mixed until ready to place in blast holes.

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 an 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**
Merril Crowe	67	50	0.82	**41**
Gold Room	13	10	0.80	**8**
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
During Operations Water usage		
Mine	107	gpm
Plant	52	gpm
Road etc.	57	gpm
Total	**217**	**gpm**
Water produced from mine	**278**	**gpm**
Excess water	61	gpm

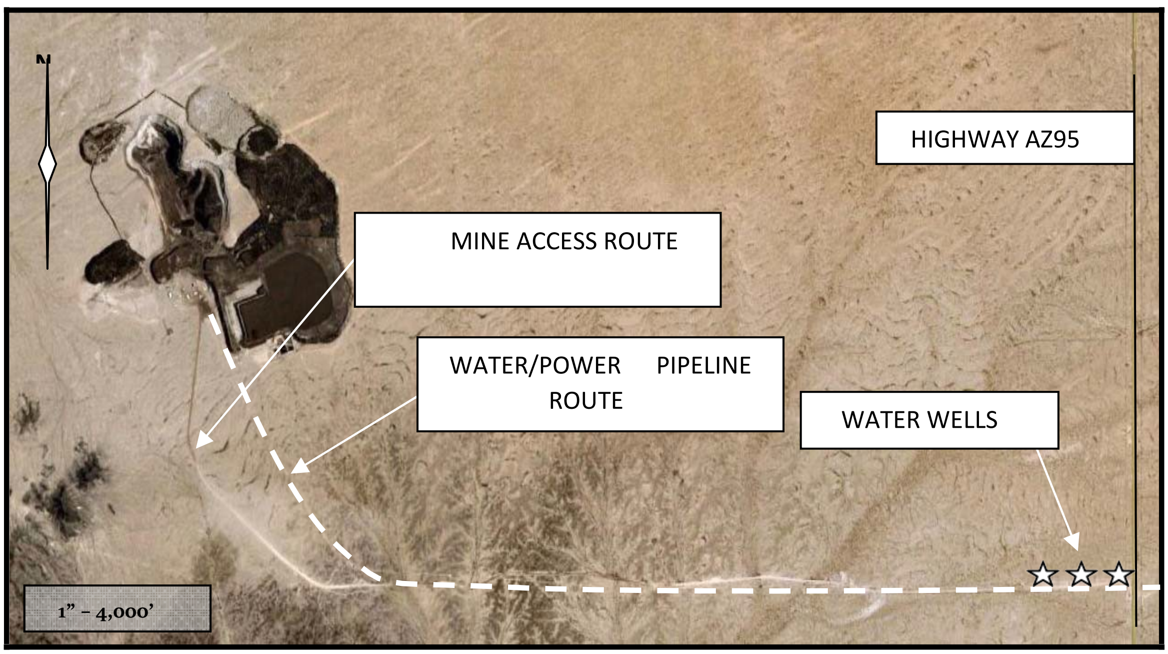

Figure 18-2 Water Well Location

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 by 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 TSF was designed for 1.24 million tons storage capacity. That design was completed by Schlumberger and approved by ADEQ. The tailings facility built was 17 ft high and included phase 1A - a portion of the overall footprint, allowing for the first 366,000 tons of material. During operation in 2012 and 2013, an estimated 101,500 tons of tailings were deposited into the tailings facility. At the anticipated throughput of the plant and matching up with the mining plan, the remaining capacity of the tailings facility as-built will allow for 1+ years of production. Therefore, the expansion and construction of phase 1B will need to occur during the first year of operation. Expansion by downstream raise of the impoundment another 4 ft will be needed by year 3 of operation.

Figures 18-3 and 18-4 show the development of the TSF from the existing facility to the ultimate size.

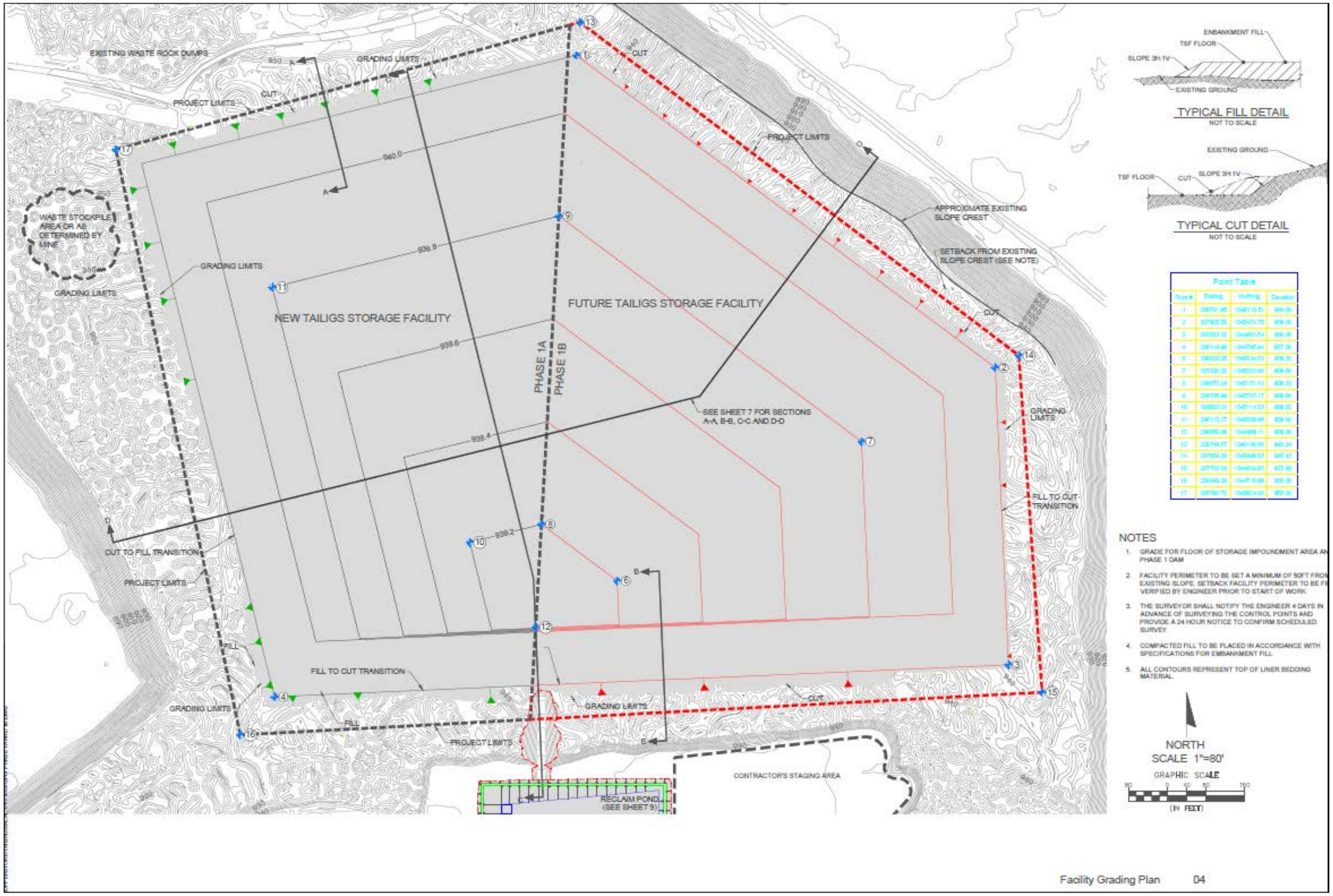

Figure 18-3 Existing TSF Phase 1A (left) and 1B (right)

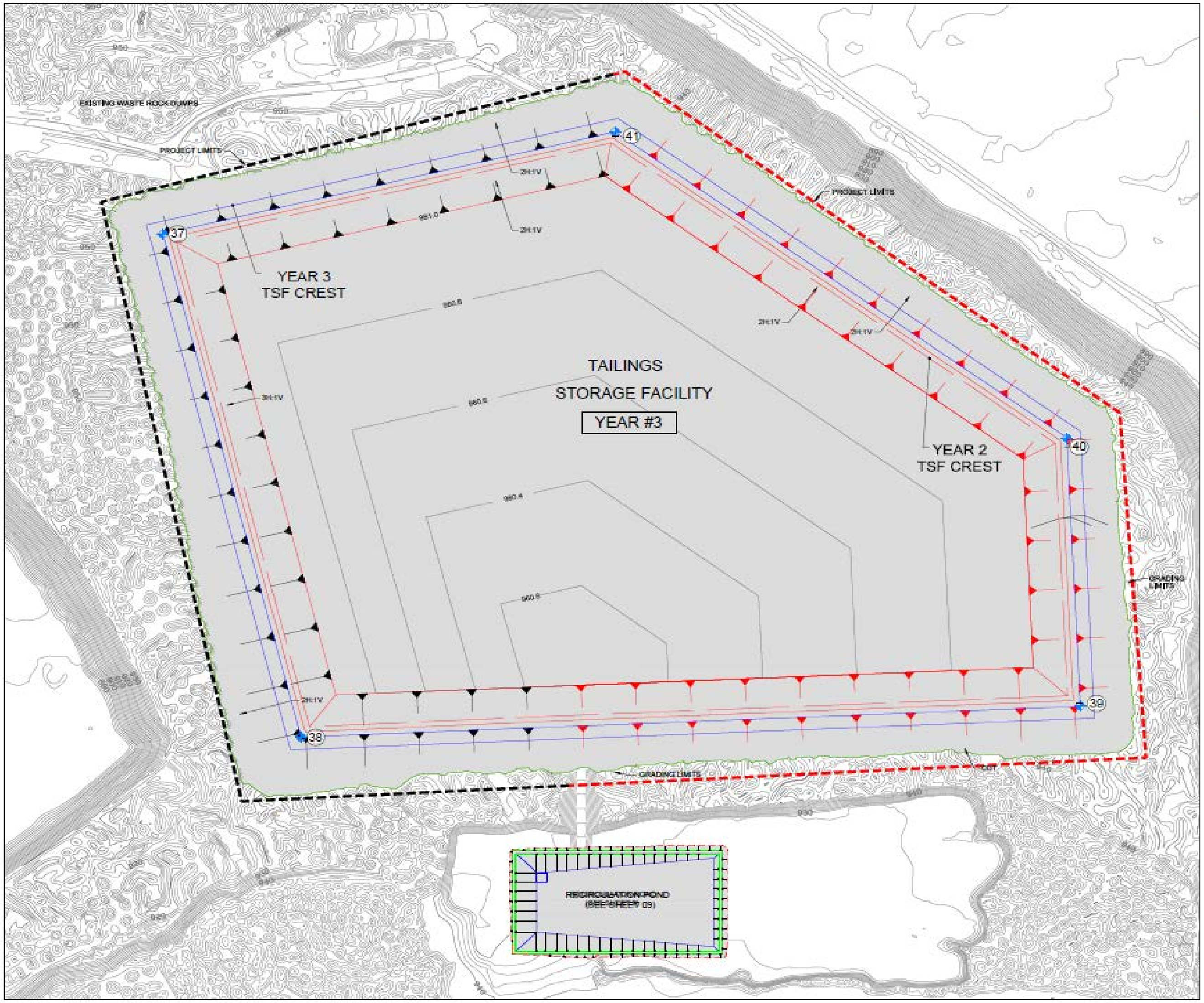

Figure 18-4 Ultimate Size TSF

19. MARKET STUDIES AND CONTRACTS

Gold markets are stable, transparent, global markets serviced by well-known smelters and refiners located throughout the world. Gold doré will be refined to 0.9999 or 0.99999 purity in the refinery, and, as such, would be a fungible commodity bought and sold universally. Therefore, no contracts have been negotiated at this stage of the Project. The Project does not plan to have forward sales of gold, nor plan to have any hedging programs at this time.

SGLD has not conducted any market studies, as gold is widely traded in the world markets. The Copperstone mine would produce doré from whole ore leaching in the processing plant which is projected to provide strong recovery while eliminating concentrate treatment and transportation costs.

The gold doré produced on site at the Project gold recovery plant can be transported to a number of reputable refiners that can improve the metal product into LME acceptable fineness and sizes for final sale. A long-established, dynamic, worldwide market exists for the buying and selling of gold and silver. It is reasonable to assume that the product from the Project will be salable. A selling price of $1,800/oz. for gold has been used to develop this PEA, which is the same as the 36-monht trailing average at the end of January 2023, as tabulated from public data from the website www.kitco.com. The month-end closing spot price at that time was $1,924/oz. for gold. The historical gold prices are presented in Figure 19-1.

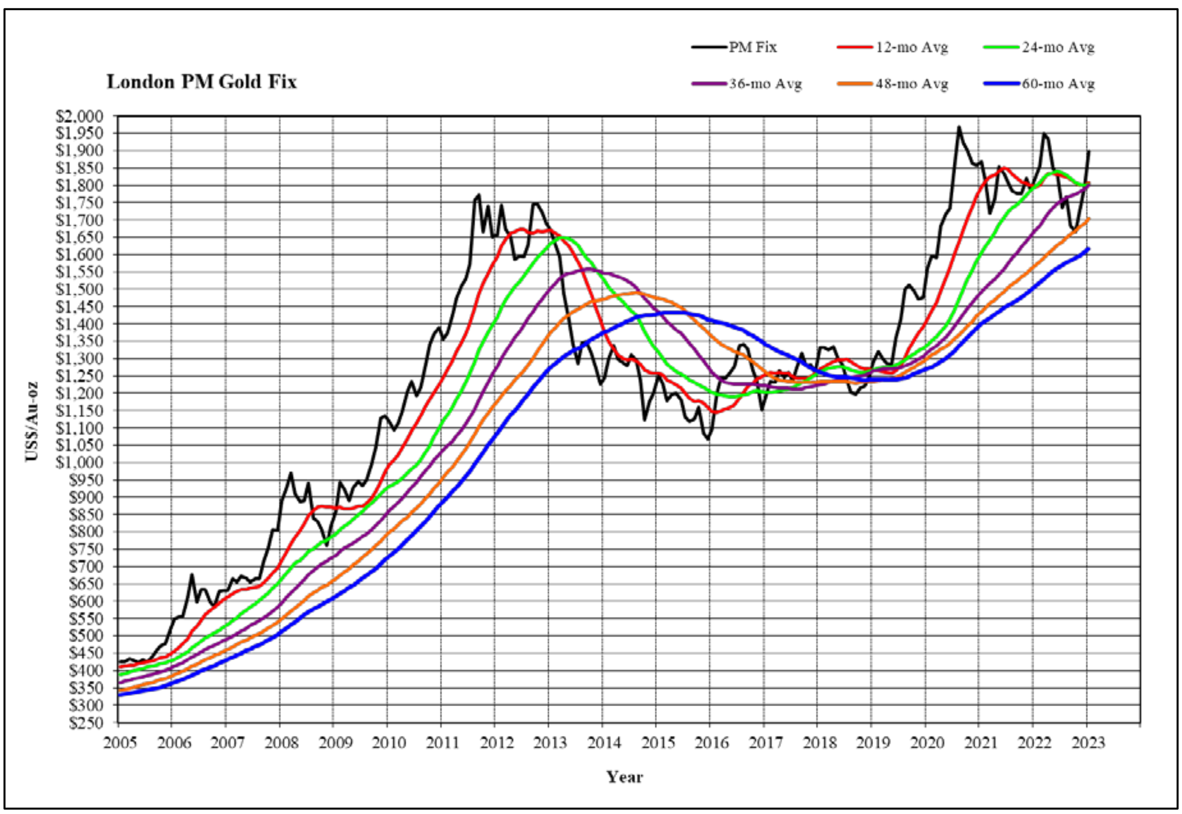

Figure 19-1 Historical Gold Prices

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

To support the development and approval of the 2008 MPO and Updated Interim Management Plan approved August 12, 2019, and State environmental permitting requirements, the following environmental studies/surveys were conducted:

- Water Resource Assessment (surface and groundwater)
- Threatened and Endangered Species and Wildlife Assessment
- Air Quality Assessment
- Biologic Survey
- Cultural Resources Assessment

20.1.1 Water Quality

Water quality samples were collected during the hydrogeological investigation, as well as from point of compliance ("POC") wells designated by Aquifer Protection Permit No. P106172 and they serve as a benchmark for water quality at the project site. A summary of typical POC tests are shown 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				
Specfic Condutivity (lab)	1580	uS/cm			
TOC	1	mg/L			
TDS	898	mg/L			
Alkalinty	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
Berylium	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				
Specfic Condutivity (lab)	1460	uS/cm			
TOC	1	mg/L			
TDS	838	mg/L			
Alkalinty	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
Berylium	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			

The water quality testing shows the water to be slightly alkaline with a pH of approximately 8. With the exception of fluoride, which is known to be naturally elevated in the area, there have been no quality parameters detected above Arizona Aquifer Water Quality Standards (AWQS), and in summary the water appears to be very suitable for use as process water.

20.1.2 Air Quality

The major sources of degradation to air quality will be dust from the crusher/screening system. The crusher/screening dust will be mitigated with water spray bars at the jaw crusher in order to pre-wet the mill feed. Dust from the mining operations will be less significant and will be confined to the underground mine operations, mill feed stockpiles, tailings impoundment, and the haul roads. Dust in the temporary coarse mill feed stockpile is expected to be minimal as the mineralized material coming from underground will be moist. As the mineralized material dries in storage, any dust issues will be controlled by using a water spray and spraying the mill feed pile. Dust in the tailings area will be controlled using proper tailings deposition and keeping the pond area moist. The top of the tailings will be wetted during operation and will ultimately be covered with 6 inches of crushed waste rock to eliminate long term dust issues. Dust generated on the haul roads will be mitigated with frequent watering of the road surface and possibly through the application of a dust suppressant, if economic. The milling and leaching for gold recovery are wet processes with no potential for fugitive dust. A dilute sodium cyanide solution is used to extract gold from the milled mineralized material as it passes through the leach tanks. The amounts of emissions are well below any regulatory thresholds.

20.1.3 Noise

The major sources of noise at the Project will be the crushing operations. Noise emissions from the crusher are mitigated by the remote location of the mill from any fixed base receptors such as residences and popular recreation sites. The noise from mining operations will primarily be confined to below ground and are not expected to be noticeable beyond the historic pit margins. The vibrations from blasting operations should not be noticeable beyond the project site given the relatively small charges that will be used.

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, adequate containment and good housekeeping facilities. Drainages downgradient from the mine site are ephemeral in nature and the mine intends to maintain zero discharge of process waters at the facility. The surface water flows in the Copperstone mine are ephemeral, with no outstanding waters, no impaired waters, and no outfalls. During periods of intense rainfall there may be sheet runoff from the travel ways and parking sites around the mineral processing facility. There are no areas of concentrated flow and the site does not accept run-on from adjacent areas.

20.1.5 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 contaminates in the groundwater.
- Annual and quarterly facility inspections, visual assessment and/or comprehensive facility inspection of storm water control systems, including routine inspections and audits of cyanide transport, handling and storage as required by the International Cyanide Management Code.
- Routine air quality control equipment performance testing as required by 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.

20.1.6 Hazard Operations Plans

The Copperstone mine will prepare a number of hazard operations plans or procedures addressing fluid management, worker health and safety, training, emergency response, and monitoring and reporting, as well as various operating practices. These plans may take several forms, including but not limited to formalized manuals, standard operating procedures, checklists, signs, work orders and training materials. These plans may be limited solely to issues involving cyanide management, but not necessarily. The intent of the Code is that management systems and procedures demonstrate that the operation understands the practices necessary to manage cyanide in a manner that prevents and controls releases to the environment and exposures to workers and the community.

The mine will obtain conditional certification of its operation per International Cyanide Management Code ("ICMC") protocol for operations that is not yet active but are sufficiently advanced in its planning and design phases so that its site plans and proposed operating procedures can be audited for conformance with the ICMC principles and standards of practice. Verification of compliance may be done as appropriate by a third-party auditor in assessing whether the pre-operational mining operation can be conditionally certified based on the expectation that it will meet the standards of the ICMC.

20.2 Disposal, Monitoring, and Management Plans

20.2.1 Disposal Facilities and Operations

Most of the existing infrastructure for the processing facilities, mill feed handling conveyors, tailings conveyance and disposal impoundments, and miscellaneous support buildings will be operated as indicated in the current MPO without modification. These facilities are described in Section 18 of this report. Several operational modifications for the planned future operations that are not included in the current MPO will require the following new infrastructure and disposal facilities.

- Cyanide vat leaching pad and associated equipment.
- Doré electro winning and associated equipment.
- Gold room furnace, propane fuel tank and associated equipment.

20.2.2 Closure and Post-Closure

Throughout production closure and post closure, process water, surface water, and groundwater within the area of impact will be closely managed. Within 90 days submitting a notification of closure to ADEQ, SGLD will prepare a Closure Plan that meets the requirements of A.R.S 49-252 and A.A.C R-18-9-A209(B)(3) as required by APP P106172. At closure, mill and process plant fluids will be managed as part of the decommissioning process. Dewatering will be significantly reduced or eliminated during closure, but will be recycled to the process plant as required for stockpiled mill feed processing, or evaporated/infiltrated in the infiltration gallery facility until dewatering is suspended. There is not anticipated to be residual surface or groundwater impacts at the mine that will prevent achieving a clean closure determination from ADEQ, therefore a Post Closure Plan to address ongoing discharges from the facility is not anticipated.

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 around feature perimeters;
- Placing silt fences and straw bales around the perimeters of disturbed areas;
- Placing erosion control fabric on slopes during revegetation establishment;
- Site grading to route stormwater to constructed channels (i.e., diversion channels, terrace channels and down chute 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.

20.3.2 Compliance History

The permits and approvals that were granted for the historical operations at the Copperstone mine specified certain requirements that needed to be met. With respect to the APP, this included an ongoing obligation to monitor and report groundwater quality in down gradient POC wells, and submittal of quarterly and annual reports that were contained in a compliance schedule. In 2021 the site failed in collecting the required samples and analysis. In April 2022 the site completed all required samples and analysis on time to meet permit requirements. Site management is in contact with ADEQ to maintain a good working relationship.

The site started to experience higher levels of fluoride in the POC wells starting in June of 2018. The Arizona state fluoride limit is 4.0 mg/L. Over the past years the site has worked with a consulting firm WSP to conduct a Parker Basin Fluoride Study showing that fluoride levels are elevated throughout the Parker Basin. The study showed that the fluoride levels in the parker basin are elevated in eight of the nine wells that were tested. Testing the POC wells on site and seven of the closest wells to the mine; fluoride levels ranged from 3.7 mg/L to 7.3 mg/L. The POC wells at site have a Fluoride level of 6.6 mg/L (POC 1) and 4.1 mg/L (POC 2) in the most recent analysis. The Parker basin study has been submitted to the ADEQ for review and a request to raise the fluoride exceedance levels is being prepared for submittal.

The air permit requires monthly method 9 opacity test be completed and reported to ADEQ when the processing plant is running. The Company has an excellent history of permit compliance and has fulfilled all the obligations for data collection, monitoring, and reporting. The current air permit will expire on December 17, 2023, SGLD is in the process of renewing the permit. Once the permit is renewed it will be valid through 2028. A minor modification will be required when the final design of the processing plant is completed.

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 mining operations will require preparation of an MPO amendment that will be reviewed by the Federal (BLM) State (ADEQ) and regulatory authorities to evaluate if the changes do not meet design requirements in the existing permits and therefore will require permit modification applications and additional public notice and review. Such changes are routine and regularly submitted during final design.

In order to proceed with the redevelopment of the Copperstone mine, it is anticipated that SGLD will have to obtain the following permit modifications and approvals:

- Aquifer Protection Permit Amendment
    - Fluoride Limit increase
- Air Quality Permit Minor Modification
    - Minor Modification to process plant with new tank emission calculations
- MPO Addendum
    - Minor Modification – to reflect Whole Ore Leach (WOL)
    - Removal of the floatation plant
    - Removal of the gravity circuit
    - Removal of the Rod Mill
    - Removal of the SART process
    - Adjustments to the underground access (portals) and mine ventilation (vent raises to surface)

- Arizona Storm Water Multi Sector General Permit (SWPPP)
    - A complete addressing to the storm water discharge from the new facilities prior to initiation of full-scale mining operations will be required.

An APP amendment will be required for a new Fluoride limit for the Copperstone mine; in which SGLD is working to obtain an increase to the Arizona state limits. An update to the current SWPPP will be completed to address the control of storm water discharges from the new facilities prior to initiation of full-scale mining operations.

SGLD will have to operate in conformance within the parameters of the current air permit until a modification issued to address the new facilities. Approval of the air permit modification is required before Sabre can start initial construction of any permit related facilities, such as the cyanide vat leaching pad and associated equipment.

The proposed Copperstone mine operation will not require an amendment to the Reclamation Plan approval through the Arizona State Mine Inspector's Office, because there will be no significant changes to the area of disturbance outlined in the current MPO. However, as required by Aggregate Mine Land Reclamation Act (A.R.S. Title 27, Chapter 6), SGLD will obtain the 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.

20.3.4 Permit Submittals and Approvals

An application will need to be submitted to the ADEQ for a minor amendment to the existing Air Quality Permit to address the new milling and refining facilities. The permit application will include a detailed Process flow diagram, emissions calculations for predicted emissions from the proposed operations, an assessment of best available control technology ("BADCT"), analysis of new source performance standards ("NSPS"), review of applicability of National Emission Standards for Hazardous Air Pollutants ("NESHAPS"), and federal acid rain regulations. Preparation of the air permit modification will commence prior to the completion of the MPO amendment and is expected to require 60 to 90 days for review, revision and approval.

Preparation of the permit amendment application package for the APP will also commence prior to the completion of the MPO amendment, but the application package will not be complete until preliminary BLM review of the MPO amendment is completed. According to the regulations for the APP program, a permit must be issued within 180 days of the submittal of a complete application. That 180 days only includes the time that the application is under review and does not include the amount of time necessary for an applicant to respond to deficiencies identified during the review. ADEQ has completed significant improvements to its review and approval procedure in order to reduce the time to permit issuance with a great deal of success.

Preparation of the MPO amendment will occur concurrent with the preparation of the APP amendment and air permit applications. The amendment will be submitted to the Yuma BLM Field Office. Typical processing for a MPO is approximately 250 days, including NEPA review and public comment.

The Copperstone mine is covered under the Mining 2010 Multi-Sector General Permit ("MSGP") (AZMSG 2010- 003) which authorizes storm water discharges associated with Sector G: Metal Mining (Ore mining and Dressing). The mine site is configured to be a zero-discharge facility and therefore does not have any engineered storm water outfalls to maintain and monitor. Permit AZMSG2010-003 is valid through January 2016 and is extended by rule until a new general permit is issued. An update to the facility SWPPP is required every five years and/or as the new facilities are added. There is no bonding requirement or approval process for operation under this general permit, however a Notice of Intent ("NOI") may be submitted when the new MSGP is issued by the State of Arizona.

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 being designed to not affect any undisturbed land and only use previously disturbed land from previous mining. As a component of the overall environmental stewardship policy of SGLD, a reclamation plan has been designed to promote the final closure of the Copperstone mine when all mining is completed. This approach provides for the ultimate closure of the facility without long-term mitigation efforts or environmental problems. Design criteria for the overall approach to mining, processing, and the sequencing of tailings placement within the final landform for optimum reclamation and closure conditions are addressed. The Plan contains provisions for protection of the environment during the operations phase using best management practices. These practices are primarily guided by the protection of surface water and groundwater resources. The proposed reclamation/closure mitigation elements for the mining operation include employment of concurrent reclamation of the facilities where practical. Therefore, reclamation obligations will be incrementally reduced as the operation progresses.

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 127 acres which is entirely within the original disturbed footprint. SGLD'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. SGLD 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 back fill of the pit.

The reclamation will be completed concurrent with mining operations as practicable during the life of the mine. 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 with no new disturbance;
- 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 establish a long-term productive biotic community compatible with proposed post-mining land uses; and
- Consideration of public safety through stabilization, removal, berming and/or limited fencing of structures or land forms that could constitute a public hazard; minimize the outward regrading or reshaping of slopes to reduce further impacts to undisturbed wildlife habitat; and consideration of the long-term visual character of the reclaimed area.

Because Copperstone is being built on previously disturbed land, the reclamation will include improving the currently disturbed areas by partially backfilling the pit bottoms, smoothing off areas as necessary and general clean-up and final closure of the buildings and ancillary facilities. The reclamation requirements that are stipulated in the MPO are summarized below.

20.4.1 Contouring and Shaping

Final grading will create land forms 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, and ripping and/or disking or other mechanical manipulation to leave the surfaces in a rough condition, and
- Use of tillage implements, as needed, for all areas to be reclaimed that can safely be worked by surface equipment to create a friable surface with favorable bulk density.

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 the 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

In the tailings areas, the final design TSF is not expected to extend above the elevation of the current facility more than 15 ft. As the tailings are deposited behind the buttress, another lift of buttress material will be placed until the full height of the dam is developed. The material used to make the buttress will be similar to the capping material used by the previous operator.

When TSF use is completed, the structure will be leveled to a 3H:1V slope, capped using an impervious cover material such as a plastic sheet or clay and covered with 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 mineralized material milling has stopped and the tailings pond decant has stopped flowing from the TSF into the reclaim pond, the pond solution would be allowed to evaporate in place. The sludge in the reclaim pond would be sampled. Should sampling demonstrate that the sludge is considered toxic according to federal and state regulations, then the sludge would be removed from the site and disposed of in accordance with state and federal regulations. 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 from the 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 a concrete containment pads and associated ball mill, tanks, and gold doré circuit. Closure of the plant area will consist of a number of activities. Milling and mineralized material processing equipment will be sold and removed. All structural steel will either 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 APS 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 effected by SGLD'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, SGLD plans to only reclaim the areas outlined in the MPO. The cost to reclaim other areas of the site that were undisturbed by SGLD are not included in this 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. Since 2016, AZG and subsequently, SGLD has engaged in constructive conversation with the La Paz County Administration as well as the Tribal Council of the Colorado River Indian Tribes, with positive feedback in all cases. SGLD currently plans to hold town hall meetings in the communities of Quartzsite and Parker in mid-2024, but does not anticipate any significant impediments to future plans or operations from a social or community perspective. The QP knows of no other social or community related requirements or plans for the Project.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The capital costs for this PEA are estimated from HRC's in-house database of projects and studies, budgetary quotes from a mining contractor, budgetary quotes for mine equipment, Hanlon Engineering and Associates ("HEA") for the process plant and price quotes obtained by SGLD for infrastructure upgrades. 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 Development	$9.2	$24.4	$33.6
Tailings Management Facility	-	$4.4	$4.4
Mineral Processing Plant	$11.6	-	$11.6
On-site Infrastructure	$8.9	$17.7	$26.6
Total Direct Costs	**$29.7**	**$46.5**	**$76.2**
Owner Costs and Reclamation	$0.6	$1.2	$1.8
Project Indirect Costs	$1.7	-	$1.7
Contingency	$4.3	$4.4	$8.7
Total Indirect Costs	**$6.6**	**$5.6**	**$12.2**
TOTAL	**$36.3**	**$52.1**	**$88.4**

21.1.1 Initial Capital

Initial capital for this study includes items purchased in Year -1 before production, production is estimated to start in month 1 of year 1. Initial capital costs were developed for the plant upgrade, infrastructure needs, 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 $36.3 million.

21.1.1.1 Initial Mine Capital Cost

Initial mine equipment and mine infrastructure capital costs are presented in

by unit cost, units required and total cost. As discussed previously the mine plan is based on using a mining contractor for the initial pre-production development and continued on for the first seven 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. Other miscellaneous infrastructure items as outlined in Table 21-2 are also included in the initial capital costs.

Table 21-2 Initial Mine Department Capital Costs

Area	Description	Cost/Unit $US_1,000's	# Initial Units	Initial Capital Cost $US_1,000's
Mine	Front Loader 4m3 - Used	$390	1	$390
Mine	Articulated Truck 40t - Used	$190	2	$380
Mine	5,000 gallon water truck - Used	$110	1	$110
Mine	Surface dozer - D6 Used	$270	1	$270
Mine	Dewatering pumps	$52	0	$5
Mine	Surface Substation	$916	1	$916
Mine	Electrical Cable from main area to pit to ug	$473	1	$473
Mine	Surface facility electrical upgrades	$236	1	$236
Mine	Expanded Dry	$382	1	$382
Mine	Bobcat	$66	1	$66
Mine	Survey Equipment	$200	1	$200
Mine	Software-Mine Planning	$60	5	$300
Mine	Software-AutoCAD	$2	3	$6
Mine	Software-Vent Sim	$5	1	$5
Mine	Computer Hardware	$3	10	$30
Mine	Plotter and printers	$10	1	$10
Mine	Open pit Re-config & civil works.	$750	1	$750
Mine	Leaky Feeder	$363	1	$363
Mine	UG Mine Contractor Mobilization	$210	1	$210
Mine	Rehab of existing workings to prep for production	$250	1	$250
Mine	Degob various access points	$153	1	$153
Mine	C Zone Failure Area Clean up	$147	1	$147
Mine	Main fan, W-5150-C, 150 hp, 78" x 30" x 1200 rpm	$50	1	$50
Mine	Main fan bulkhead and install	$50	1	$50
Mine	Air Doors (Hoffman)	$40	2	$80
Mine	Fan Drift & Slabbing for airlock	$281	1	$282
Mine	Air door & install North/West decline intersect	$85	1	$85
Mine	Underground MPCs	$345	2	$690
Mine	North Portal Construction/North	$760	1	$760
Mine	CRF pit and liner	$100	1	$100
Mine Development	Initial Mine Development			$9,166
Total Mine Initial Capital				**$16,913**

21.1.1.2 *Initial Process Plant Capital Cost*

The processing method selected is whole ore leaching followed by Merril Crowe for gold recovery. New equipment for the whole ore leach method will be installed except for the crushing plant and ball mill which will be refurbished.

The estimate includes all costs associated with project management, process engineering, design engineering, drafting, procurement, and commissioning services required to construct and commission the processing facility and its associated support infrastructure. In addition, the estimate includes costs associated with the spare parts and the provision of first fills and consumables required for the commencement of operations of the WOL plant expansion.

The estimate has been based upon preliminary engineering designs, material quantity estimates taken from these designs, budget price quotations for major process equipment, and budget rates for the supply of bulk commodities. Unit rates for site installation works were based on market enquiries and benchmarked against those recently achieved on similar resource projects.

Pricing for the estimate was obtained predominantly during the third quarter of 2022. Pricing is provided in US dollars ($US). Where pricing was received in a foreign currency, it was converted to $US using the foreign exchange rates set in the third quarter of 2022. The estimate accuracy is ±20% and based on the following:

- Material quantities developed from preliminary engineering design calculations and design drawings.
- Budgetary quotations obtained for major equipment items and site-based contract works.

The capital cost estimate was compiled using a conventional work breakdown structure (WBS) based on plant areas (e.g., crushing, milling, leach, CIP, thickening, metal recovery, and refining) and sub-categories of discipline groupings (e.g., civil, concrete, steel, platework, mechanical, piping, electrical, and I&C). Table 21 3 lists the initial capital required for the processing plant.

Table 21-3 Initial Plant Capital Costs

Area	Description	Cost/Unit $US_1,000's	# Initial Units	Initial Capital Cost $US_1,000's
Plant	Front Loader 4m3- Plant	$390	1	$390
Plant	Crushing Plant (refurbish)	$207	1	$207
Plant	Ball Mill (refurbish)	$651	1	$651
Plant	General Site	$440	1	$440
Plant	Crushing & Conveying	$305	1	$305
Plant	Grinding	$38	1	$38
Plant	Whole Ore Leaching	$6,467	1	$6,467
Plant	Merrill Crowe	$1,122	1	$1,122
Plant	Gold Room	$286	1	$286
Plant	Tailings	$60	1	$60
Plant	Indirect Field Construction Costs	$814	1	$814
Plant	Freight	$342	1	$342
Plant	Assay Lab Refurbishment	$500	1	$500
Total Process Initial Capital				**$11,624**

21.1.1.3 Initial G&A, Indirects and Contingency Capital Cost

Table 21-4 lists the initial capital for the G&A infrastructure along with the estimated indirects and contingency. Indirect costs for the establishment and operation of temporary construction facilities were based on recent experience with similar sized projects with allowances made for the project's location. Provision has been made for all the Contractors onsite office accommodation, stores, workshops, communications, and toilet and crib facilities. The EPCM estimate was based on the hours required for a multidisciplined project engineering team, consisting of suitably qualified and experienced personnel, able to undertake the activities required to successfully complete the Project within the scheduled timeframe. The involvement of each design discipline was estimated based on the complexity of the individual tasks and benchmarked against actual EP costs incurred on recent similar projects. Owner's costs have been compiled inclusive of the following; owner's project management team, owner's consultants, operational readiness items such as pre-operations costs, business systems, insurance and approvals. The contingency for mining portion is included at 20% and the plant and G&A is estimated at an average of 19.3%.

Table 21-4 Initial G&A, Indirects and Contingency Capital Cost

Area	Description	Cost/Unit $US_1,000's	# Initial Units	Initial Capital Cost $US_1,000's
G&A	Pickups	$60	5	$300
G&A	Septic system Expansion	$30	1	$30
G&A	Technical Services Office Building	$147	1	$147
G&A	Site IT upgrades	$100	1	$100
G&A	Site Access Road - Resurface and Dust Abatement	$210	1	$210
G&A	Warehouse Tool Cage/Room	$18	1	$18
G&A	Refurbish Vehicle Wash Bay	$13	1	$13
G&A	Line Fuel station/storage	$2	1	$2
G&A	Repair freshwater tank	$29	1	$29
G&A	Repair Existing Water Supply System	$62	1	$62
G&A	Upgrade Potable water system	$20	1	$20
G&A	Re-Furbish Guard shack	$4	1	$4
G&A	Install Auto Gates and re-configure Fencing	$45	1	$45
G&A	First Aid Room	$10	1	$10
G&A	First Fill PPE & Safety Gear	$20	1	$20
G&A	Used Ambulance	$15	1	$15
G&A	First Fill Warehouse Tools Allowance	$85	1	$85
G&A	Water Stand at Highway Wells	$23	1	$23
Indirects	Operations Staff During Construction	$283		$283
Indirects	Sales tax	$253		$253
Indirects	Indirect EPCM, final Eng, first fills	$1,721		$1,721
Contingency	Contingency 19.3%	$4,344		$4,344
Total G&A. Indirects and Contingency Initial Capital				**$7,733**

21.1.2 Sustaining Capital

Sustaining capital costs are included for major mine equipment component rebuilds and additional mine equipment requirements. In month eight of Year one, a fleet of underground mining equipment is planned to be purchased so the mine can transition from contract mining 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 16-8. The mine equipment is included in the economic model as a capital lease which includes a 3-year term at 10% interest and a 25% down payment.

Sustaining capital costs are also included for the continued mine primary development, the A zone portal, equipment rebuilds, and two tailings dam raises. The sustaining capital requirements by year and area are presented in Table 21-5. Reclamation closure costs are estimated at an additional $1.2 million, currently there is a reclamation bond posted in the amount of $1.6 million for the Project.

Table 21-5 Sustaining Capital Costs

Sustaining Capital $US_1,000's	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Mine Equipment and infrastructure	$14,270	$995	$900	$591	$610	$375	$17,741
Mine Primary Development	$13,904	$7,138	$3,387	$0	$0	$0	$24,429
Plant Tailings lifts	$2,500	$0	$1,900	$0	$0	$0	$4,400
Indirects & Contingency	$3,323	$192	$540	$114	$118	$72	$4,358
Reclamation Closure Costs	$0	$0	$0	$0	$0	$1,200	$1,200
Total Sustaining capital	**$33,997**	**$8,325**	**$6,727**	**$705**	**$727**	**$1,647**	**$52,128**

21.1.3 Mine Development Costs

The primary development costs are included in the initial and sustaining capital cost estimates presented above. As production begins, the secondary development in the mineralized zones are included as an operating cost. The total LOM development and overall costs, including labor, are presented in Table 21-6.

Table 21-6 Capital Mine Development Requirements and Costs

	Year -1	Year 1	Year 2	Year 3
feet	3,330	11,071	11,141	5,084
$/ft	$2,752	$1,256	$641	$666
$/m	$9,028	$4,120	$2,101	$2,185

21.2 Operating Cost Estimates

The operating costs include the ongoing cost of operations related to mining, processing, tailings disposal and general administration activities. The operating costs were determined based on HRC's in-house database of projects and studies, budgetary quotes from a mining contractor and Hanlon Engineering and Associates for the process costs. Following the AISC guidelines, LOM average Cash Operating Cost is projected to be $864/oz of gold sold. The AISC LOM average base case Total Operating Cost (including royalties and production taxes) is expected to be $1,012/oz. The total AISC summary per ton of mill feed and per ounce of gold is expected to be $1,286/oz as shown in Table 21-7.

Table 21-7 Total Operating Cost Summary

Operating Costs	$/oz Au	$/ton mill feed
Mining	$512.88	$95.79
Processing	$253.69	$47.38
Site G&A	$85.05	$15.89
Transportation & Refining	$12.84	$2.40
CASH OPERATING COSTS	**$864.46**	**$161.46**
Royalties and Stream	$138.89	$25.95
Production Taxes	$8.76	$1.64
TOTAL CASH COSTS	**$1,012.11**	**$189.05**
Reclamation	$5.26	$0.98
Sustaining Capital	$268.65	$50.17
ALL-IN SUSTAINING COSTS	**$1,286.02**	**$240.20**

21.2.1 Mine Operating Costs

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 QP also used data from operating mines of similar size in developing the operating costs for the mine. The costs details by department over the life of the mine are shown in 8 and Figure 21-1 shows the distribution of these costs by department.

Table 21-8 Mine Operating Costs

Department	Average Yearly Costs	$/ton Mill Feed	$/oz Au Sold
Mine G&A	$2,643,288	$12.41	$66.45
Secondary Development	$2,067,011	$9.70	$51.96
UG Stoping	$5,742,591	$26.96	$144.36
Backfill	$2,934,615	$13.78	$73.77
UG Mill Feed Transport	$1,275,955	$5.99	$32.07
Surface Haulage	$561,118	$2.63	$14.11
Mine Services & Maint.	$2,799,141	$13.14	$70.36
Engineering	$652,771	$3.06	$16.41
Geology	$1,228,593	$5.77	$30.88
Contingency	$497,627	$2.34	$12.51
Grand Total	**$20,402,712**	**$95.79**	**$512.88**

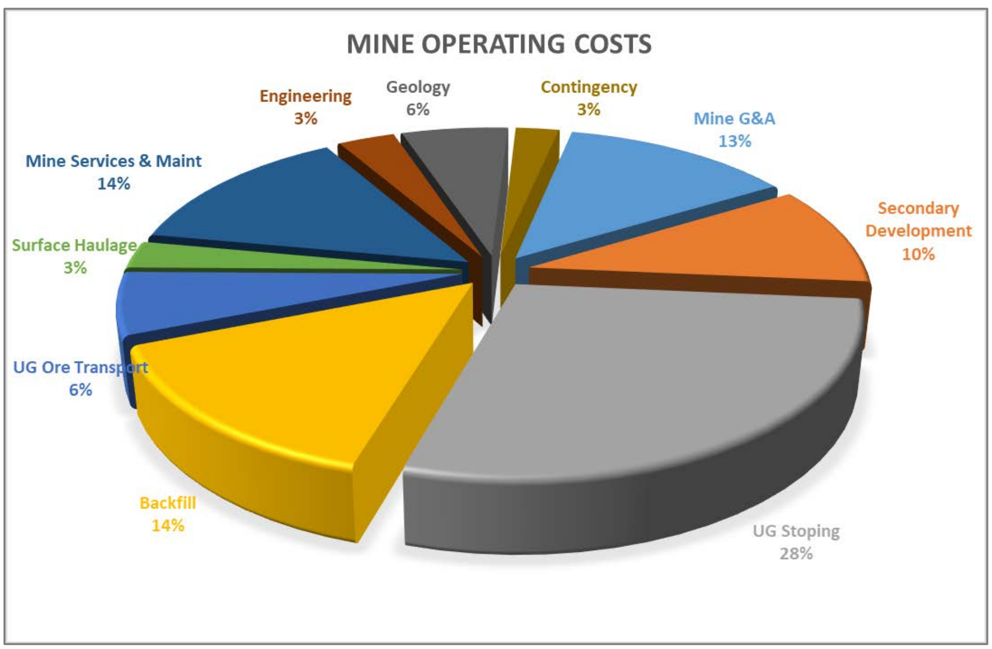

Figure 21-1 Mine Operating Cost Distribution

Table 21-9 shows the distribution of the estimated mine operating costs by cost category for a typical year. On average the mine operating costs are estimated $95.79 per ton of mineralized material mined or $513 per ounce of gold sold.

Table 21-9 Mine Operating Costs by Category

Category	Average Yearly Costs	$/ton	$/oz Au Sold
Labor & Benefits	$8,290,374	$38.92	$208.40
Labor Alloc	-$139,109	-$0.65	-$3.50
Energy	$885,748	$4.16	$22.27
Fuel & Lubes	$2,603,606	$12.22	$65.45
Wear Parts	$409,318	$1.92	$10.29
Maint. Parts	$748,751	$3.52	$18.82
Materials/Supplies	$3,810,407	$17.89	$95.79
Blasting Supplies	$346,274	$1.63	$8.70
Service	$2,929,589	$13.75	$73.64
Lease	$20,127	$0.09	$0.51
Contingency	$497,627	$2.34	$12.51
Grand Total	**$20,402,712**	**$95.79**	**$512.88**

21.2.2 Plant Operating Costs

Processing costs were estimated by Hanlon based on equipment requirements built up in detail utilizing hourly costs for equipment components, supplies, consumables 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 21-10 and Figure 21-2 shows the distribution of these costs by department.

Table 21-10 Process Plant Operating Costs

Department	Average Yearly Costs	$/ton	$/oz Au Sold
General Site	$1,524,627	$6.96	$37.27
Crushing	$726,375	$3.32	$17.78
Grinding	$1,612,864	$7.36	$39.41
Whole Ore Leaching	$3,028,569	$13.83	$74.05
Merrill Crowe	$1,766,482	$8.07	$43.21
Gold Room	$412,033	$1.88	$10.07
Laboratory	$644,435	$2.94	$15.74
Tailings	$660,453	$3.02	$16.17
Grand Total	$10,375,838	$47.38	$253.70

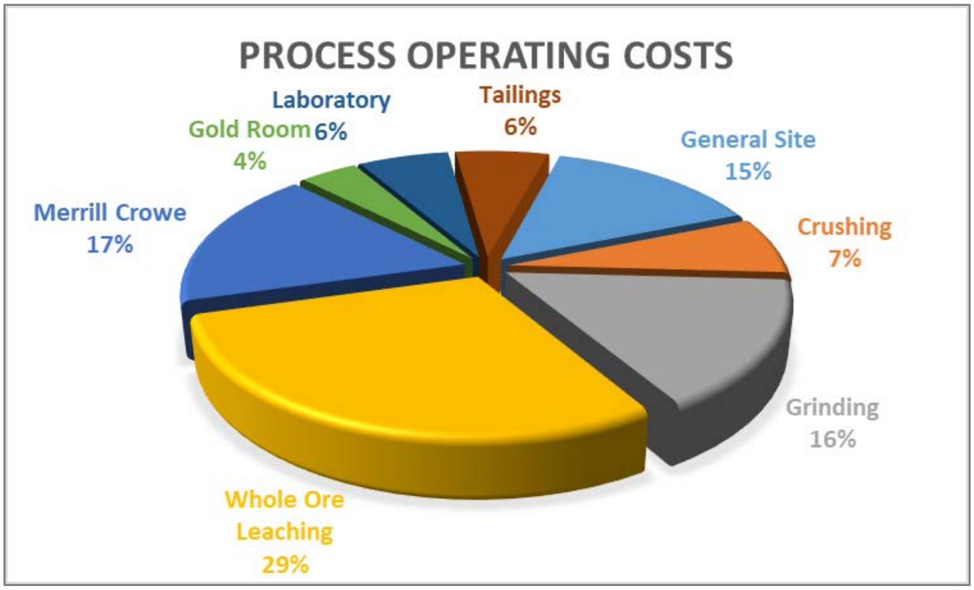

Figure 21-2 Process Operating Costs Distribution

Table 21-11 shows the distribution of the estimated plant operating costs by cost category for a typical year. On average the plant operating costs are estimated $47.38 per ton of mineralized material milled or $253 per ounce of gold sold.

Table 21-11 Process Plant Operating Costs by Category

Category	Average Yearly Costs	$/ton	$/oz Au Sold
Labor	$3,939,600	$17.99	$96.33
Operating Consumables	$4,460,117	$20.37	$109.05
Power	$1,285,200	$5.87	$31.42
Plant Maintenance	$372,753	$1.70	$9.11
Plant Mobile Equipment	$70,483	$0.32	$1.72
Laboratory	$247,685	$1.13	$6.06
Grand Total	$10,375,838	$47.38	$253.70

21.2.3 G&A Operating Cost

The G&A costs have been developed from the QP'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. The costs details by department over the life of the mine are shown in Table 21-12 and Figure 21-3 shows the distribution of these costs by department. On average the G&A operating costs are estimated to be $15.41 per ton of mineralized material milled or $83 per ounce of gold sold.

Table 21-12 G&A Operating Costs

Department	Average Yearly Costs	$/ton	$/oz Au Sold
Admin	$1,512,531	$7.10	$38.02
Human Relations	$169,756	$0.80	$4.27
Accounting	$253,003	$1.19	$6.36
Environmental	$336,753	$1.58	$8.47
Purchasing	$273,945	$1.29	$6.89
Security & Safety	$655,911	$3.08	$16.49
Contingency	$80,047	$0.38	$2.01
Grand Total	**$3,281,946**	**$15.41**	**$82.50**

Figure 21-3 G&A Operating Costs Distribution

21.2.4 Labor

Operating labor rates and burdens percentages are based on mines of similar size and compared to Cost Mine data for labor rates in mines in Arizona. Staffing levels and rates for the life of the reserve are shown in Table 21-13 through Table 21-16. Overtime was estimated at 2.5% and payroll burdens were estimated at 38% with a 50% bonus for the underground miners.

Table 21-13 Manpower Requirements for Mine Operations

Manpower Summary		Rate	Year-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Average
Mining G&A										
Mine Superintendent	Mine G&A	$155,000	1	1	1	1	1	1	1	1
Mine Foreman	Mine G&A	$120,000	0	2	4	4	4	4	4	3
Mine G&A			**1**	**3**	**5**	**5**	**5**	**5**	**5**	**4**
Mining G&A										
Mine Clerk	Mine G&A	$21.00	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Capital Development										
Miner 1	Development	$34.00	4	4	4	2	0	0	0	2
Miner 2	Development	$30.00	8	8	8	5	0	0	0	4
Helper	Development	$28.00	4	4	4	2	0	0	0	2
Development			**16**	**16**	**16**	**9**	**0**	**0**	**0**	**8**
Secondary Development										
Miner 1	Ore Access	$32.00	2	2	2	2	2	2	2	2
Miner 2	Ore Access	$30.00	3	4	4	4	4	4	4	4
Development			**5**	**6**	**6**	**6**	**6**	**6**	**6**	**6**
U/G Production										
Miner 1	UG Stoping	$34.00	6	8	8	8	8	8	8	8
Miner 2	UG Stoping	$30.00	3	4	4	4	4	4	4	4
Helper	UG Stoping	$28.00	2	2	2	2	2	2	2	2
Backfill	Backfill	$32.00	1	2	2	2	2	2	2	2
Blaster	UG Stoping	$34.00	1	2	2	2	2	2	2	2
Stoping			**12**	**18**	**18**	**18**	**18**	**18**	**18**	**17**
Haulage										
Miner 1	Surface Ore	$30.00	3	5	6	6	5	6	5	5
Miner 3	Ore/Waste	$30.00	3	6	7	6	4	4	4	5
Miner 4	Ore/Waste	$27.00	3	6	7	6	4	4	4	5
Haulage			**9**	**18**	**19**	**17**	**13**	**14**	**12**	**14**
Mine Services										
U/G Helper	Mine Services	$27.00	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Mine Maintenance										
Lead Mechanic	UG Stoping	$37.00	4	4	4	4	4	4	4	4
Mechanic	UG Stoping	$32.00	8	8	8	8	8	8	8	8
Mechanic/Welder	Ore Access	$27.00	6	8	8	8	8	8	8	8
Electrician	Development	$32.50	1	1	1	1	1	1	1	1
Total Mine Maintenance			**19**	**21**	**21**	**21**	**21**	**21**	**21**	**21**
Total Mine Operations			**63**	**85**	**88**	**80**	**66**	**67**	**64**	**73**

Table 21-14 Manpower Requirements for Engineering and Geology

Manpower Summary		Rate	Year-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Average
Engineering										
Sr Mining Engineer	Engineering	$125,000	1	1	1	1	1	1	1	1
Jr Mining Engineer	Engineering	$105,000	1	1	1	1	1	1	1	1
Chief Surveyor	Engineering	$110,000	1	1	1	1	1	1	1	1
Surveyor Tech	Engineering	$24.00	2	2	2	2	2	2	2	2
Engineering			**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
Geology & Grade Control										
Sr Geologist	Geology	$110,000	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Grade Control										
Geologist	Geology	$100,000	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Sampler	Geology	$24.00	**3**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
Geology			**5**	**7**	**7**	**7**	**7**	**7**	**6**	**7**
Total Engineering and Geology			**10**	**12**	**12**	**12**	**11**	**12**	**11**	**11**

Table 21-15 Manpower Requirements for Process Plant Operations

Manpower Summary			Year-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Average
Processing Plant										
Salary Personnel										
Plant Superintendent	Plant G&A	$110,000	0	1	1	1	1	1	1	1
Merrill Crowe Supt.	Plant G&A	$90,000	0	1	1	1	1	1	1	1
Plant Metallurgist	Plant G&A	$110,000	0	1	1	1	1	1	1	1
Maintenance Supervisor	Plant Mntnc	$90,000	0	1	1	1	1	1	1	1
Process Salary			**0**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
Hourly Personnel										
Crushing Operators	Primary Crushing	$28.85	0	4	4	4	4	4	4	4
Milling Operators	Grinding	$28.85	0	4	4	4	4	4	4	4
Leaching/CCD Operators	Leaching	$38.46	0	4	4	4	4	4	4	4
DT/Tailings	Tails	$28.85	0	8	8	8	8	8	8	8
Merrill Crowe Oper.	Merrill Crowe	$33.65	0	4	4	4	4	4	4	4
Lab Analyst	Assay Lab	$38.46	0	1	1	1	1	1	1	1
Lab Technicians	Assay Lab	$24.04	0	2	2	2	2	2	2	2
Maintenance Leadmen	Plant Mntnc	$33.65	0	4	4	4	4	4	4	4
Welders/Mechanics	Plant Mntnc	$33.65	0	2	2	2	2	2	2	2
Electricians	Plant Mntnc	$33.65	0	4	4	4	4	4	4	4
Trades Assistants/Helpers	Plant Mntnc	$24.04	0	2	2	2	2	2	2	2
Process Hourly			**0**	**39**	**39**	**39**	**39**	**39**	**39**	**39**
Total Process Operations			**0**	**43**	**43**	**43**	**43**	**43**	**43**	**43**

Table 21-16 Manpower Requirements for G&A and Totals for Project

Manpower Summary			Year-1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Average
General and Administrative										
Mine Manager	Admin	$200,000	1	1	1	1	1	1	1	1
Environmental Manager	Environmental	$125,000	0	1	1	1	1	1	1	1
HR Manager	Human Relations	$105,000	0	1	1	1	1	1	1	1
Safety Superintendent	Security & Safety	$100,000	1	1	1	1	1	1	1	1
Controller	Accounting	$85,000	1	1	1	1	1	1	1	1
Purchasing Manager	Purchasing	$100,000	1	1	1	1	1	1	1	1
Environmental Tech	Environmental	$80,000	0	1	1	1	1	1	1	1
IT Tech	Admin	$75,000	0	1	1	1	1	1	1	1
Payroll	Accounting	$22.00	1	1	1	1	1	1	1	1
Accounts Payable	Accounting	$20.00	1	1	1	1	1	1	1	1
Admin Assistant	Admin	$17.00	1	1	1	1	1	1	1	1
Safety/Security	Security & Safety	$22.00	6	8	8	8	7	8	7	7
Warehousemen	Purchasing	$22.00	2	2	2	2	2	2	2	2
Total G&A			**14**	**21**	**21**	**21**	**20**	**21**	**19**	**20**
Totals for all Departments										
Total Salary			**10**	**21**	**23**	**23**	**22**	**23**	**22**	**21**
Total Hourly			**78**	**140**	**141**	**133**	**117**	**120**	**115**	**126**
Total Property			**88**	**161**	**164**	**156**	**140**	**143**	**137**	**147**

22. ECONOMIC ANALYSIS

The Project has been evaluated using a constant US dollar, after-tax discounted cashflow methodology. This valuation method requires projecting material balances estimated from operations and calculating the economic analysis. Cashflows are calculated from sales of metal plus net equipment salvage value, less cash outflows such as operating costs, capital costs, working capital changes, any applicable taxes and reclamation costs. Resulting annual cash flows are used to calculate the NPV and IRR of the Project. 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. Information contained and certain statements made herein are considered forward-looking within the meaning of applicable Canadian securities laws. These statements address future events and conditions and so involve inherent risks and uncertainties. Actual results could differ from those currently projected. This PEA is preliminary in nature, and includes Inferred mineral resources that are considered too speculative geologically to have economic consideration applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

22.1 Summary

The Project is planned to be an underground mining operation with milling and WOL followed by a Merril Crowe plant for gold recovery. The life of mine PEA plan includes 1,222,000 tons of mill feed with an average grade of 0.197 oz/ton gold. The process operations are planned to run at a rate of 600 tons per day with a metallurgical gold recovery of 95%.

The economic analysis of the base case scenario for the Project uses a price of US$1,800/oz for gold, which is the 3 year trailing average price as of end of January, 2023. The economic model shows an After-Tax Net Present Value @ 5% ("NPV-5") of $61.78 million using a 0.107 oz/ton gold mining cut-off grade, as well as an After-Tax Internal Rate of Return ("IRR") of 50.5%. Table 22-1 summarizes the projected Cashflow; Net Present Value at varying rates; Internal Rate of Return; years of positive cash flows to repay the negative cash flow ("payback period"); multiple of positive cash flows compared to the maximum negative cash flow ("payback multiple") for the Project on both After-Tax and Before-Tax bases.

Table 22-1 Summary of Copperstone Economic Results

Project Valuation Overview	After Tax	Before Tax
Net Cashflow (millions)	$86.77	$89.78
NPV @ 5.0%; (millions)	$61.78	$63.97
NPV @ 7.5%; (millions)	$52.21	$54.09
NPV @ 10.0%; (millions)	$44.15	$45.76
Internal Rate of Return	50.5%	51.2%
Payback Period, Years	1.81	1.81
Payback Multiple	2.90	3.53
Total Initial Capital (millions)	-$36.27	-$36.27
Max Neg. Cashflow (millions)	-$45.66	-$35.54

22.1.1 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.

22.1.2 Royalties

A 3.0% production royalty along with the Star Streaming Agreement have been calculated on the gross proceeds less transportation and refining costs as required by underlying agreements provided by SGLD.

22.1.3 Corporate Income Taxes

United States and State corporate taxes have been considered in the economic analysis.

22.1.4 Cashflows

The projected total lifespan of the Project is 6.9 years: 1.2 years of pre-production and construction, and 5.7 years of operations. Approximately 240,500 oz of gold is projected to be mined, with 228,200 oz recovered and produced for sale. An initial capital investment of $36.2 million, including contingency/working capital is projected. Following the AISC guidelines, life-of-mine average base case Cash Operating Cost is projected to be $864/oz of gold sold. The AISC life-of-mine average base case Total Operating Cost (including royalties and production taxes), is expected to be $1,012/oz. The AISC is projected to be $1,286/oz.

Table 22-2 Copperstone Project Total Operating Cost/ounce Gold & per ton Mill Feed

Operating Costs	$/oz Au	$/ton mill feed
Mining	$512.88	$95.79
Processing	$253.69	$47.38
Site G&A	$85.05	$15.89
Transportation & Refining	$12.84	$2.40
CASH OPERATING COSTS	**$864.46**	**$161.46**
Royalties and Stream	$138.89	$25.95
Production Taxes	$8.76	$1.64
TOTAL CASH COSTS	**$1,012.11**	**$189.05**
Reclamation	$5.26	$0.98
Sustaining Capital	$268.65	$50.17
ALL-IN SUSTAINING COSTS	**$1,286.02**	**$240.20**

As previously mentioned, the gold price used in the economic analysis (US $1,800/oz Au) is the 36-month trailing average price as of the end of January 2023. The economic analysis was done on an annual cashflow basis; the cashflow output is shown in Table 22-3.

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Table 22-3 Copperstone Project Schedule and Cash Flow

Note: All Dollars are in US	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Life-of-Mine
MINE PRODUCTION									
Tons Ore Mined		18,148	199,678	240,468	232,153	200,989	222,630	108,251	1,222,317
Au, oz/tn		0.184	0.256	0.208	0.195	0.184	0.167	0.155	0.197
Development Feet		3,330	14,378	14,721	9,947	6,916	8,153	3,972	61,416
Development Waste		50,864	233,713	237,774	135,301	57,400	67,668	32,965	815,686
Total Tons Mined		**69,013**	**433,391**	**478,242**	**367,454**	**258,389**	**290,298**	**141,216**	**2,038,003**
PROCESS PRODUCTION									
Tons Ore Processed			214,350	219,000	219,000	219,600	219,000	131,367	**1,222,317**
Au, oz/t			0.250	0.208	0.196	0.185	0.170	0.156	0.197
Income Statement									
Contained Oz Au to Mill			53,542	45,583	42,879	40,732	37,298	20,504	240,538
Saleable Oz Au, post 99.9% Refinery credit			50,814	43,260	40,694	38,657	35,398	19,460	228,283
Gross Revenue			$91,465,920	$77,868,360	$73,249,020	$69,582,780	$63,715,680	$35,027,460	$410,909,220
Transportation and Refinery Charges			($637,788)	($569,732)	($546,611)	($528,262)	($498,897)	($149,190)	($2,930,480)
Net Refined Revenue			$90,828,132	$77,298,628	$72,702,409	$69,054,518	$63,216,783	$34,878,270	$407,978,740
Royalties			($2,743,978)	($2,336,051)	($2,197,471)	($2,087,483)	($1,911,470)	($1,050,824)	($12,327,277)
Star Royalties- Stream			($4,527,563)	($3,854,484)	($3,625,826)	($3,444,348)	($3,153,926)	($773,855)	($19,380,002)
Net Revenue			**$83,556,591**	**$71,108,093**	**$66,879,112**	**$63,522,687**	**$58,151,386**	**$33,053,592**	**$376,271,462**
OPERATING EXPENSES									
Mining									
Mine G&A	$0	$0	($2,636,553)	($2,684,458)	($2,684,458)	($2,691,499)	($2,684,458)	($1,787,292)	($15,168,717)
Secondary Development	$0	$0	($1,927,015)	($1,508,703)	($1,871,963)	($2,425,681)	($2,653,421)	($1,474,925)	($11,861,707)
UG Stoping	$0	$0	($9,712,079)	($5,255,661)	($5,152,429)	($4,825,470)	($5,090,650)	($2,918,020)	($32,954,310)
UG Ore Transport	$0	$0	($1,960,658)	($1,239,735)	($1,202,893)	($1,049,963)	($1,228,339)	($640,581)	($7,322,169)
Surface Haulage	$0	$0	($604,650)	($693,073)	($690,050)	($606,600)	($625,650)	$0	($3,220,023)
Mine Services & Maint	$0	$0	($1,329,917)	($2,878,952)	($3,092,874)	($3,291,186)	($3,231,925)	($2,238,239)	($16,063,093)
Backfill	$0	$0	($3,629,659)	($3,044,161)	($3,099,855)	($2,676,614)	($2,822,484)	($1,567,744)	($16,840,516)
Engineering	$0	$0	($674,546)	($674,546)	($674,546)	($637,756)	($674,546)	($410,040)	($3,745,980)
Geology	$0	$0	($1,231,458)	($1,324,358)	($1,305,420)	($1,189,448)	($1,283,732)	($715,962)	($7,050,377)
Contingency, @ 2.5%	$0	$0	($592,663)	($482,591)	($494,362)	($484,855)	($507,380)	($293,820)	($2,855,671)
Total Mining		**$0**	**($24,299,197)**	**($19,786,236)**	**($20,268,849)**	**($19,879,071)**	**($20,802,585)**	**($12,046,623)**	**($117,082,561)**
Processing & ROM handling									
Plant G&A	$0	$0	($1,491,876)	($1,524,240)	($1,524,240)	($1,528,416)	($1,524,240)	($914,316)	($8,507,328)
Crushing	$0	$0	($711,642)	($727,080)	($727,080)	($729,072)	($727,080)	($436,139)	($4,058,093)
Grinding	$0	$0	($1,577,616)	($1,611,840)	($1,611,840)	($1,616,256)	($1,611,840)	($966,863)	($8,996,255)
Whole Ore Leach	$0	$0	($2,964,461)	($3,028,770)	($3,028,770)	($3,037,068)	($3,028,770)	($1,816,808)	($16,904,647)
Merril Crowe	$0	$0	($1,729,805)	($1,767,330)	($1,767,330)	($1,772,172)	($1,767,330)	($1,060,133)	($9,864,100)
Gold Room	$0	$0	($402,978)	($411,720)	($411,720)	($412,848)	($411,720)	($246,970)	($2,297,956)
Laborayory	$0	$0	($630,189)	($643,860)	($643,860)	($645,624)	($643,860)	($386,220)	($3,593,613)
Tailings	$0	$0	($647,337)	($661,380)	($661,380)	($663,192)	($661,380)	($396,729)	($3,691,398)
Total Processing	**$0**	**$0**	**($10,155,903)**	**($10,376,220)**	**($10,376,220)**	**($10,404,648)**	**($10,376,220)**	**($6,224,178)**	**($57,913,389)**
Site General & Administration									
Admin	$0	$0	($1,535,715)	($1,535,715)	($1,535,715)	($1,527,318)	($1,535,715)	($1,009,597)	($8,679,776)
Human Relations	$0	$0	($177,120)	($177,120)	($177,120)	($162,805)	($177,120)	($102,875)	($974,160)
Security & Safety	$0	$0	($680,135)	($680,135)	($680,135)	($636,697)	($680,135)	($406,758)	($3,763,995)
Accounting	$0	$0	($263,978)	($263,978)	($263,978)	($242,642)	($263,978)	($153,324)	($1,451,877)
Purchasing	$0	$0	($281,814)	($281,814)	($281,814)	($269,986)	($281,814)	($174,813)	($1,572,054)
Environmental	$0	$0	($346,044)	($346,044)	($346,044)	($332,578)	($346,044)	($215,729)	($1,932,483)
Contingency, @ 2.5%	$0	$0	($82,120)	($82,120)	($82,120)	($79,301)	($82,120)	($51,577)	($459,358)
Total G&A		**$0**	**($3,366,926)**	**($3,366,926)**	**($3,366,926)**	**($3,251,327)**	**($3,366,926)**	**($2,114,672)**	**($18,833,703)**
Property Tax	$0	($64,528)	($106,918)	($101,808)	($101,583)	($73,605)	($69,433)	($65,235)	($583,110)
Mine Severance Tax	$0	$0	($559,439)	($387,519)	($359,576)	($344,163)	($263,471)	($86,181)	($2,000,349)
Total Operating Costs	**$0**	**($64,528)**	**($38,488,383)**	**($34,018,709)**	**($34,473,153)**	**($33,952,815)**	**($34,878,634)**	**($20,536,890)**	**($196,413,112)**

Copperstone Project Schedule and Cash Flow (cont.)

Continued	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Life-of-Mine
Operating Margin (EBITDA)	**$0**	**($64,528)**	**$45,068,208**	**$37,089,385**	**$32,405,959**	**$29,569,872**	**$23,272,752**	**$12,516,702**	**$179,858,349**
Development Deduction	$0	($6,416,052)	($9,733,061)	($4,996,587)	($2,370,698)	$0	$0	$0	($23,516,398)
Amortization	$0	($274,974)	($692,105)	($906,244)	($1,007,846)	($1,007,846)	($1,007,846)	($5,181,596)	($10,078,457)
Depreciation	$0	$0	($8,144,092)	($12,665,940)	($9,822,740)	($7,912,836)	($7,523,916)	($7,533,057)	($53,602,581)
Reclamation Deduction	$0	$0	$0	$0	$0	$0	$0	($1,200,000)	($1,200,000)
Interest Expense	$0	($28,916)	($526,937)	($709,976)	($364,111)	($50,789)	$0	$0	($1,680,730)
Income - before NOL & Perc Depletion	**$0**	**($6,784,470)**	**$25,972,014**	**$17,810,638**	**$18,840,563**	**$20,598,401**	**$14,740,990**	**($1,397,951)**	**$89,780,184**
Net Operating Loss Adjustment	$0	$6,784,470	($25,972,014)	($17,810,638)	($18,840,563)	($11,861,255)	$0	$1,397,951	($66,302,049)
Depletion	$0	$0	$0	$0	$0	($4,215,673)	($7,112,527)	$674,512	($10,653,688)
State Income Tax	$0	$0	$0	$0	$0	($305,800)	($515,935)	$48,928	($772,807)
Federal Income Tax	$0	$0	$0	$0	$0	($885,291)	($1,493,631)	$141,647	($2,237,275)
Taxable Income, less Tax	**$0**	**$0**	**$0**	**$0**	**$0**	**$3,330,382**	**$5,618,897**	**$865,087**	**$9,814,366**
Cash Flow Calculation									
Adjustments for Non Cash Items									
Development Deduction	$0	$6,416,052	$9,733,061	$4,996,587	$2,370,698	$0	$0	$0	$23,516,398
Amortization	$0	$274,974	$692,105	$906,244	$1,007,846	$1,007,846	$1,007,846	$5,181,596	$10,078,457
Depreciation/Reclamation/Salvage	$0	$0	$8,144,092	$12,665,940	$9,822,740	$7,912,836	$7,523,916	$8,733,057	$54,802,581
Net Operating Loss Adjustment	$0	($6,784,470)	$25,972,014	$17,810,638	$18,840,563	$11,861,255	$0	($1,397,951)	$66,302,049
Depletion	$0	$0	$0	$0	$0	$4,215,673	$7,112,527	($674,512)	$10,653,688
Total Adjustments for Non Cash Items	**$0**	**($93,444)**	**$44,541,271**	**$36,379,409**	**$32,041,847**	**$24,997,610**	**$15,644,289**	**$11,842,190**	**$165,353,172**
Capital									
Investment - Mine	$0	($7,747,290)							($7,747,290)
Investment - Primary Development	$0	($9,165,788)							($9,165,788)
Investment - Plant	($1,338,840)	($10,284,860)	$0						($11,623,700)
Investment - G&A	$0	($1,132,200)	$0						($1,132,200)
Capital Indirects & Contingency	($600,913)	($5,999,687)	$0						($6,600,600)
Total Capital	**($1,939,753)**	**($34,329,825)**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**($36,269,578)**
Sustaining Capital - Mine			($14,269,608)	($994,878)	($900,320)	($591,245)	($609,843)	($374,759)	($17,740,654)
Sustaining - Primary Development			($13,904,373)	($7,137,982)	($3,386,712)	$0	$0	$0	($24,429,067)
Sustaining Capital - Plant			($2,500,000)	$0	($1,900,000)	$0	$0	$0	($4,400,000)
Sustaining Capital - Indirects& Contingency			($3,322,935)	($191,750)	($539,727)	($113,955)	($117,540)	($72,230)	($4,358,137)
Reclamation Closure Costs	$0	$0	$0	$0	$0	$0	$0	($1,200,000)	($1,200,000)
Total Capital & Sustaining	**($1,939,753)**	**($34,329,825)**	**($33,996,915)**	**($8,324,611)**	**($6,726,759)**	**($705,200)**	**($727,383)**	**($1,646,989)**	**($88,397,436)**
Working capital		$0	($2,835,000)	$0	$0	$0	$0	$2,835,000	$0
Equipment Financing	$0	$911,850	$9,487,500	$0	$0	$0	$0	$0	$10,399,350
Principal Payments	$0	($87,183)	($1,719,576)	($3,316,717)	($3,546,483)	($1,729,391)	$0	$0	($10,399,350)
Total Capital & Working Capital	**($1,939,753)**	**($33,505,158)**	**($29,063,991)**	**($11,641,328)**	**($10,273,242)**	**($2,434,592)**	**($727,383)**	**$1,188,011**	**($88,397,436)**
Beginning Cash	**$0**	**($1,939,753)**	**($35,538,355)**	**($20,061,075)**	**$4,677,006**	**$26,445,611**	**$52,339,012**	**$72,874,814**	
Period Net Cash Flow	**($1,939,753)**	**($33,598,602)**	**$15,477,280**	**$24,738,080**	**$21,768,605**	**$25,893,400**	**$20,535,803**	**$13,895,287**	**$86,770,102**
Ending Cash	**($1,939,753)**	**($35,538,355)**	**($20,061,075)**	**$4,677,006**	**$26,445,611**	**$52,339,012**	**$72,874,814**	**$86,770,102**	**$86,770,102**

22.2 Sensitivity Analysis

22.2.1 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$100/oz Au would change the NPV-5 by 24%, or approximately $14.6 million (Figure 22-1). Table 22-4 also shows the economic sensitivities, due to the change in gold price, in the Net Cash Flow, the Net Present Value at 5%, the Internal Rate of Return, the Payback Period, and the Payback Multiple.

Table 22-4 Gold Price Sensitivity Economic Results

Au Price	Net Cash Flow, millions	NPV - 5.0%, millions	IRR	Payback Period, Years	Payback Multiple
$1,500	$27.75	$15.47	16.8%	3.6	1.5
$1,600	$48.42	$31.63	28.6%	2.9	2.0
$1,700	$69.10	$47.78	40.0%	2.2	2.4
$1,800	$86.77	$61.78	50.5%	1.8	2.9
$1,900	$104.19	$75.54	60.8%	1.5	3.4
$2,000	$121.74	$89.34	71.1%	1.3	4.0
$2,100	$139.22	$103.07	81.3%	1.2	4.6

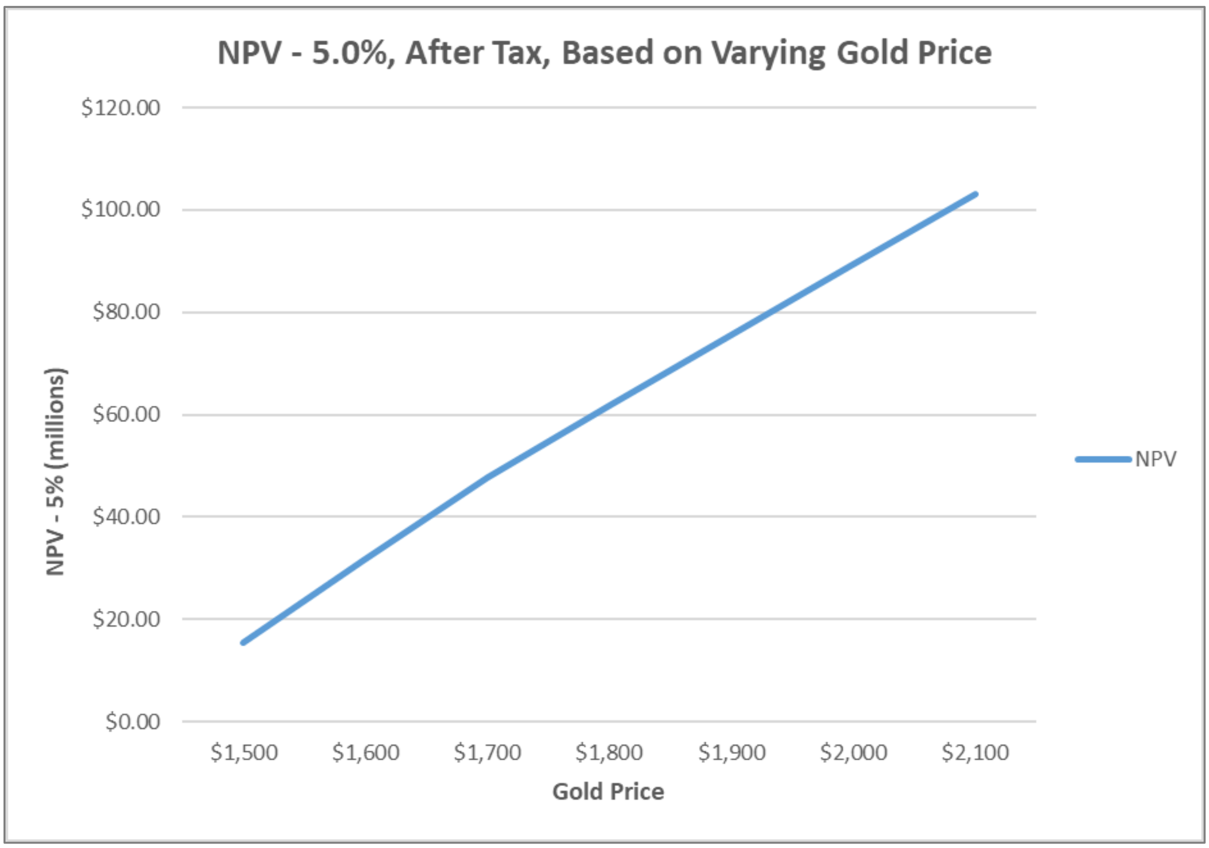

Figure 22-1 Project Gold Price Sensitivity Analysis

22.2.2 Operating and Capital Costs Sensitivity

The Project is very sensitive to the cost of operations, incurring an approximately 20% 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 11% in decline in the NPV-5 for each increase of 10% in the capital costs, as shown in Figure 22-2.

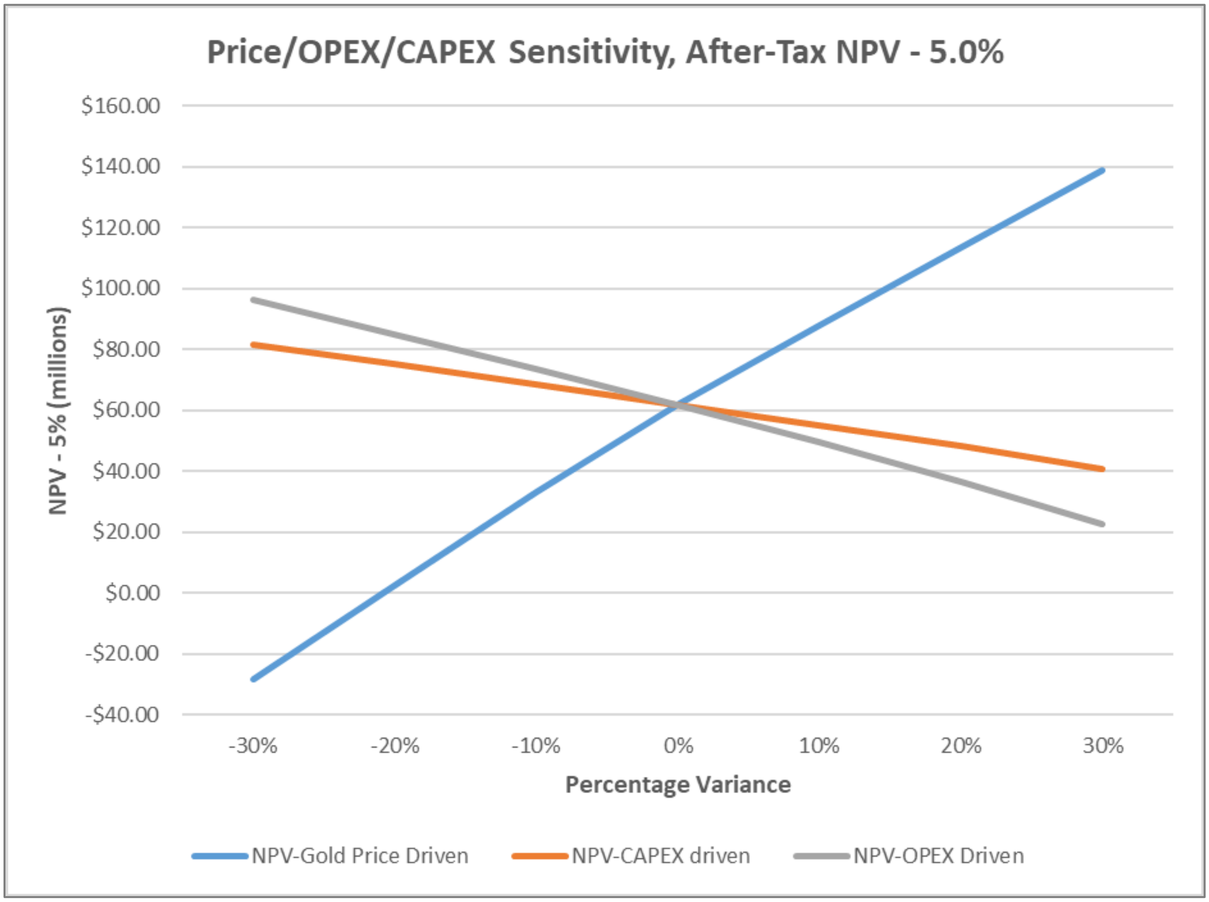

Figure 22-2 Project Operating Cost & Capital Cost Sensitivity Analysis

22.3 Conclusion

The Project would be economically viable based on the parameters considered in this study. The base case scenario produces approximately 228,200 salable ounces of gold over a 5.7-year period. The Project is most sensitive to the gold price and to operating costs, but sensitive to a lesser extent to capital costs.

The base case economic analysis of the Project at a gold price of US$1,800/oz shows positive economics with a $61.78 million post-tax NPV (5%) and 50.5% post-tax IRR. The PEA 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

The QP knows of no other relevant technical or other data or information that might materially impact the interpretations and conclusions presented herein, nor of any additional information necessary to make the report more understandable or not misleading.

25. INTERPRETATION AND CONCLUSIONS

25.1 Introduction

Results of the PEA indicate that the Project shows positive economics, benefitting from significant in-place infrastructure, underground development and permitting. There are numerous targets to increase the mineral resources and economic grade mill feed appears to be readily available for mining and processing. The base case scenario produces approximately 228,200 salable ounces of gold over a 5.7-year period. The Project is most sensitive to the gold price and to operating costs, but sensitive to a lesser extent to capital costs. The base case economic analysis of the Project at a gold price of US$1,800/oz shows positive economics with a $61.78 million post-tax NPV (5%) and 50.5% post-tax IRR. The PEA supports the decision to carry out additional detailed studies.

25.2 Geology and Mineral Resource Estimate

The Copperstone deposit is a mid-Tertiary, detachment 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 the 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 drillhole database was vetted to identify missing values, duplicate records, interval overlap errors, from-to data exceeding maximum collar depth, and special (i.e. non-numeric or less than zero) values. Errors identified by the mechanical audit were reviewed with SGLD staff and resolved prior to modelling and calculation of the mineral resource estimate. A random manual check of 4.4% of the historic assay database against original certificates was conducted by the QP. The error rate within the database is considered to be less than 1% based on the number of samples spot checked. Identified errors were corrected prior to modelling and calculation of the mineral resource estimate. All assays from 2017 to present were verified against their certificate values by the QP.

The QP concludes that the sample preparation, security and analytical procedures are appropriate and adequate to ensure the integrity of the sample data. The sample methods and density are appropriate, and the samples are of sufficient quality to comprise a representative, unbiased database.

Mineral resource estimation was performed only for gold. Copper may be a by-product of mining economic gold ores and extracting the copper during the processing of the economic gold material at the Copperstone mine is at the early stages of being evaluated. The economics of monetizing copper as a by-product are potentially attractive as the cash costs of production are shared with the cash cost of producing the primary product – gold. Further exploration drilling, assaying and modelling work of copper bearing gold material is required. Metallurgical testing for the economic extraction of copper is ongoing, but currently incomplete, and further testing is required. Copper may be incorporated into a mineral resource dependent upon favorable metallurgical results, and a complete review of copper data for adequacy.

Factors that may affect the mineral resources include changes to geological or grade interpretations, including grade shell considerations; changes to the modelling method or approach; changes to metallurgical recovery assumptions; and changes to any of the social, political, economic, permitting, and environmental assumptions considered when evaluating reasonable prospects for eventual economic extraction.

25.3 Metallurgical Testing

The mineralized material in the deposit is variable in gold grade and silica content. A careful blending program would be required for consistent processing operation. Bond's Ball Mill Work Index testing at a closed size of 100 mesh indicate that the samples are of medium hardness. BWi values ranged from 11.34 kwh/ton in zone D to 14.29 kwh/ton in zone C.

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 P_{80} 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.

The amount of cyanide soluble copper should be incorporated into the mine plan and economic analysis. Currently, cyanide soluble copper assays make up 0.05% of the total gold assay database. When compared to gold assays greater than 0.05 oz/ton, cyanide soluble copper assays make up 13% of the database. There is reasonable spatial distribution of cyanide copper soluble assays along strike and down dip of available mineral resources. This information will be useful in determining the amount of cyanide that will be consumed and the amount of copper that can be recovered in the conceptual process flowsheet developed for the Project.

A trade-off study should be undertaken to determine the selection of filters or CCD circuit for the process. Additionally, geo-metallurgical testing is recommended for continuing the viability of the process flowsheet in handling varying feed material.

25.4 Mining Methods

Underground mining methods were reviewed that will minimize dilution, capital, and operating costs, maximize recovery of the mineral resources while maintaining the design production throughput at the mill. The Copperstone mineralization is relatively flat with an average dip of 38 degrees. Although there are some areas where the mineralized material will flow, above a 45-degree dip, the majority of the deposit is too flat to facilitate a long hole mining method. The mining method proposed for the Copperstone Project is a mechanized cut and fill using Rock Fill (RF) and Cemented Rock Fill (CRF). Cut and fill was chosen for its flexibility in effectively mining low vein dip angles. The method also minimizes the amount of dilution during mining by careful geological and management control of the mining. Datamine's® Minable Stope Optimizer ("MSO") was used to generate the stopes utilizing a metal price of $1,800/oz for gold and a 0.107 oz/ton gold cut-off.

Cut and fill stoping involves accessing the mineralization from a main ramp. The initial stope access is driven down grade to the waste/mineralized contact and then extended through the mineralization to the hanging wall contact. Once the hanging wall has been located, longitudinal panels are mined perpendicular to the access drift, along strike, to the stope ends. The cut and fill design is based on 10-ft high stopes. The 10-ft height was chosen primarily to reduce dilution and to improve ground conditions but still allow for the proposed equipment to fit inside the stope. In narrow mineralized areas that are inclined at 38°, a significant amount of waste will be generated at the hanging wall and footwall contacts in order to recover all the mineralized material. This percentage of waste in narrow areas increases as a function of the stope height. As the mineralization width increases, the percentage dilution reduces significantly. A reduction in the amount of waste mined was achieved by implementing a shanty back of 60° on the stope hanging wall. Hanging wall instability can be a problem in shanty back stopes where bed separation can cause failures, this problem can be exacerbated with greater stope heights. Ground support will have to be installed quickly on the hanging wall in order to minimize stability issues with the stope back.

Mineralized width varies considerably over the property and total stope widths range between 9 ft and 102 ft wide, the average total stope width is 30 ft. Where stope widths exceed 16 ft horizontally, it will be necessary to extract multiple side- by-side drifts (passes) on each cut in order to limit the mining span. After stopes are split into the required drift passes the average actual mining width is 12 ft. Up to seven passes will be required but the majority of the stopes, approximately 89%, will be one, two or three pass stopes. Stope lengths average approximately 100 ft, with a few extending beyond 400 ft. If the stope widths are extended to a maximum 30-ft width, as suggested by the Langston and Associates study, then 63% of the stopes will be single pass, 30% will be 2 pass, 6% 3 pass and 1% will be 4 pass.

25.4.1 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 assumptions as to permitting, environmental, and social license to operate.

Other than the above, the QP is unaware of any legal, political, environmental or other risks that could materially affect the mine plan.

25.4.2 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 to amount of gold recovered from the resource;
- Conduct a more detailed analysis to see if there is a high enough percentage of stopes that may be able to be mined by long hole methods to justify bringing in a long hole drill for mining these areas which will reduce development and 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.

25.5 Mineral Processing

Metallurgical testing for the Copperstone deposit has evaluated three process options:

- Grinding and flotation to produce a gold concentrate;
- Grinding, flotation, and cyanide leaching of the gold flotation concentrate; and
- Grinding and whole ore cyanide leaching.

The final two options produce doré bars which provide a marketing advantage over the production of flotation concentrates. Testing was conducted by RDI for screening these flowsheet options. The test work results indicated a 97% recovery of gold, while the other processes were 88% to 90% recovery. Based upon economic and technical analysis, whole ore leaching was chosen as the best processing option for the Copperstone deposit. Although the operating costs for the whole ore leaching of Copperstone mineralized material are the highest operating costs of the three options, the increase in recoveries and elimination of smelter charges make this option economically superior.

25.6 Infrastructure and Permitting

The Copperstone mine benefits from extensive existing infrastructure development including 10,000 ft of underground development with two portals to access the underground mine from the bottom of the historic open pit mine. Supporting the underground development are electrical equipment, compressors and ventilation. Also present are a crushing grinding circuit, a tailings storage facility expandable, within existing permits, to contain the life of mine mineralized material tailings as defined in the 2023 PEA; and other surface infrastructure including line power, office buildings, maintenance shop, fuel bay, wash rack, assay lab, warehouse and a mine dry. The entire mine-site layout is compact with the underground operations proximal to the process plant, tailings facility and site buildings.

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 cyanide for recovery of gold from mineralized material using captive steel tanks located in the gold mineralized material processing facility, whole ore leaching.
- 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.

25.7 Economic Analysis

Information contained and certain statements made herein are considered forward-looking within the meaning of applicable Canadian securities laws. These statements address future events and conditions and so involve inherent risks and uncertainties. Actual results could differ from those currently projected. This PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have economic consideration applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Results of the PEA indicate that the Project shows positive economics, benefitting from significant in-place infrastructure, underground development and permitting. The base case scenario produces approximately 228,200 salable ounces of gold over a 5.7-year period. The Project is most sensitive to the gold price and to operating costs, but sensitive to a lesser extent to capital costs.

The base case economic analysis of the Project at a gold price of US$1,800/oz shows positive economics with a $61.78 million post-tax NPV (5%) and 50.5% post-tax IRR. The PEA supports a decision to carry out additional detailed studies.

26. RECOMMENDATIONS

The following tasks are recommended to be completed to advance the Project and to prepare for Project development and operation.

26.1 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, and it is recommended that these efforts are continued. All drillhole information should be migrated into a formal database software such as AcQuire or Access.

26.2 Structural Understanding

Structural understanding in the Footwall zone and South zone is reasonably assumed to be parallel to the Copperstone shear zone. Support for this structural interpretation of the Footwall and South zones can be accomplished by systematically reviewing intercepts from core drilling and attempting to measure the orientation of mineralization relative to the core axis. Future drilling could incorporate the use of down-hole Televiewer imagery to produce accurate in-situ structural measurements which can be linked to the geological and assay logs for modeling purposes. If a Televiewer is not used or available core drilling with oriented across the strike and dip of the Footwall and South zones could also be considered. Confirmation of the structural understanding of the Footwall and South zones could upgrade portions of the mineral resource estimate currently classified as Inferred to Indicated.

26.3 Additional Drilling

26.3.1 <u>Step Out Drilling</u>

SGLD intends to complete a surface core drillhole program totaling approximately 14,000 ft (2,270 m) at an expected cost of approximately 1.5 million $US. The drilling plan will utilize wedges to cover more area from a single initial collar location. The drillhole spacing is planned to be between 100 and 150 ft and targets down dip and down plunge mineralization in the C and D zones, down dip extension of the South zone, as well as an exploration drillhole testing for the furthest down dip expression of the Footwall zone.

Anomalous gold grades have been intersected by drilling northeast of the C zone. Drillhole C96-15 has two intercepts greater 0.10 oz/ton gold (810-815 ft grading 0.185 oz/ton Au, 855-860 ft grading 0.112 oz/ton Au). Additionally, hole C96-16 encountered two intervals of greater than 0.10 oz/ton Au (1135-1140 ft grading 0.182 oz/ton Au; 1205-1210 ft grading 0.192 oz/ton Au). C96-16 is 283 ft northwest from C96-15. Drillhole C96-14 is 287 ft southeast of C96-15, but returned no significant gold values. No drilling has been completed to the northeast or southwest of C96-15. The intercepts could represent either the down dip extension of the C zone, or the expression of a new mineralization zone. Two drillholes are recommended to test between C96-15 & C96-14, and C96-15 & C96-16; 2 drillholes testing up dip between C96-16 & H4-63 on approximately 200-ft centers; and a step out drillhole 100 ft northeast of C96-15 to test down dip. Drilling these targets will require drilling through previously mined dump material as well as a significant amount of overburden.

Drillhole H5-108 encountered significant gold grade intercepts at depth (1084-1088 ft grading 0.3 oz/ton Au; 1121-1124 ft grading 0.64 oz/ton Au; 1124-1129 ft grading 0.35 oz/ton Au). The hole ended in low grade gold. These intercepts could be the expression of the Footwall zone beneath the A/B zone. Follow up drilling is recommended surrounding this intercept to test the extent and continuity of mineralization, as well as its relation to the South zone.

26.3.2 Infill Drilling

Infill drilling in the current Footwall zone and South zone to convert Inferred mineral resources to Indicated or Measured, expand mineral resources, and improve the understanding of geometry and orientation of mineralized structures could be continued in future drilling programs. Figure 26-1 shows the block model for domains classified as Measured, Indicated, and Inferred resources. Areas colored dark blue and white space are areas for infill drilling in A/B, C, and D zones.

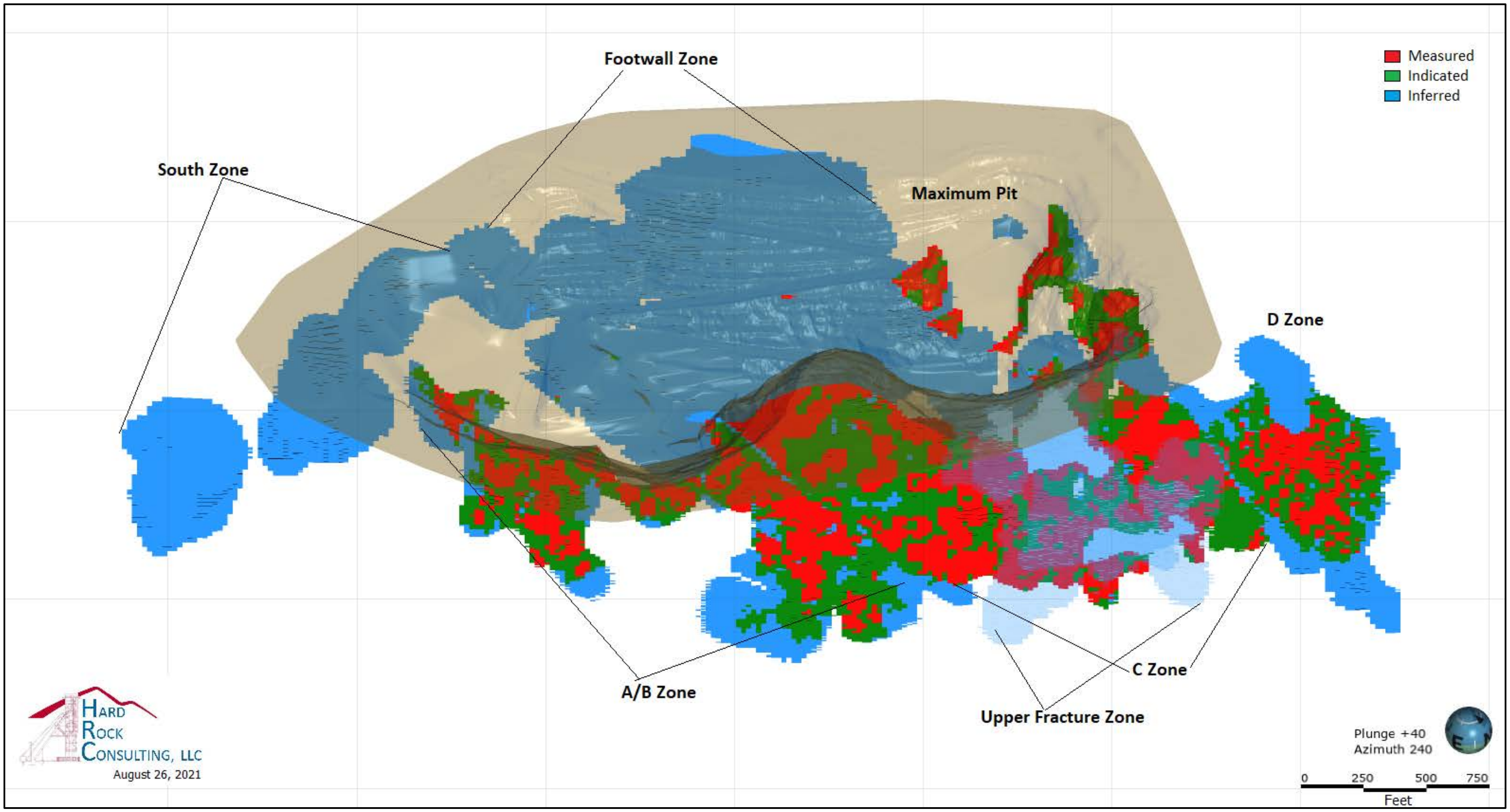

Figure 26-1 Block Model showing Domains with Classification

26.3.3 Exploration Drilling

The following targets are recommended for exploration drilling:

- Test for the expression of Footwall zone mineralization at depth below D zone;
- Historic drill hole 06CS-20 intercepted 5 ft of gold mineralization grading 0.6 oz/ton from 1035-1040 ft approximately 3,000 ft southwest of the Copperstone pit and has not been followed up on. No drilling is within 500 ft of the drillhole;
- Testing around CS-266 which had a 10-ft intercept of 0.1 oz/ton Au from 780-790 ft. The intercept is approximately 650 ft southwest from the Copperstone pit, and is not currently incorporated into any known zone of mineralization; and
- Expansion of the Southwest target located approximately 2,500 ft. southwest of the Copperstone mine, to determine mineralization extent.

26.4 Mineral Processing and Recovery Methods

The Copperstone Project has undergone significant value engineering during the course of the significant engineering work completed to date. The opportunities identified by SGLD, Hanlon Engineering and Associates (HEA), and Forte Dynamics are:

- Purchase of a new tire driven ball mill;
- Purchase of a new cone crusher;
- Rental of portable crushers;
- Sourcing used equipment in lieu of new equipment where feasible, and
- Process re-design to utilize filtration technologies as an alternative to CCD’s.

Utilizing a new tire-driven ball mill and a new cone crusher offer operation and maintenance opportunities with comparative capital cost. Rental of portable crushers will shift expenditures from sustaining capital to operating costs. Purchasing used equipment will be reviewed in more detail in the next phase to evaluate potential capital cost and schedule benefits. Finally, evaluating the use of filtration for pregnant solution recovery will be studied at a preliminary level under a separate trade-off proposal.

26.5 Mining

Based on the favorable results of the PEA, it is recommended that the mine design and mine plan be advanced to a pre-feasibility level prior to a production decision. The following areas are recommended for further study during the next phase of work:

- 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 in order to increase to amount of gold recovered from the mineral resource;
- Develop grade control procedures based on the recent stope infill drilling programs that have been completed;

- Further investigate contract mining versus owner mining;
- Hire key underground technical and management staff on a priority basis to facilitate the pre-feasibility design phase; and
- Optimize the ventilation, water management, and electrical power systems.

SGLD should continue to build the geotechnical database concurrently with the geological database, so that the core is logged geotechnically, particularly in the vicinity of potential mineralized zones. This includes core photos in the boxes before the core is split with estimates of core recovery and RQD (both as percentages), and notes in the geological log about shear zones, faults, and altered zones (already being noted), and any other geotechnical observations that could help with mine design. Old core should be re-evaluated by SGLD to expand their geological models. New geotechnical holes may need to be drilled, but this should not be done until the relevant geotechnical data has been assessed from the existing array of boreholes. Core samples from the main rock types should be selected for strength testing, particularly Unconfined Compressive Strength testing (at least four tests to start with for each of the main rock types in the vicinity of the mineralized zones). Other laboratory strength tests also may be required. In addition to mapping rock type and structure, new exposures in tunnels should be mapped using the Q and RMR methods and the data used to specify support in the tunnels. This data can be used later to help with the detailed design of the stopes and the standoff distance of access drifts. Rock falls and collapses in tunnels should be logged in a geotechnical log book, and any persistent underground movements should be monitored and measured.

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
Drillhole Database	**$10,000**
Structural Understanding	**$30,000**
Step Out Drilling	**$1,500,000**
Mineral Processing & Recovery Methods Trade Off Study	**$10,000**
Mining	
Optimize short term mine design	$25,000
Review use of lower cut-off grades	$5,000
Grade Control Program	$5,000
Contract Mining vs Owner Mining analysis	$10,000
Key underground mining staff	$100,000
Optimize Ventilation, Water, and Power systems	$25,000
Total Mining	**$170,000**
Update PEA or Pre-Feasibility Study	**$150,000**
Total Budget	**$1,870,000**

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APPENDIX A

Project List of Patented Claims and State Claims

Bonanza

AMERICAN BONANZA GOLD CORP.
COPPERSTONE PROPERTY, LA PAZ COUNTY, ARIZONA
PLAN OF OPERATIONS

Attached are the following unpatented lode mining claims situated in the Plomosa Mining District, La Paz County, State of Arizona, located in Sections 18-22, Township 6 North, Range 19 West; Sections 1, 2, 11-15, 22-27, Township 6 North, Range 20 West, GSRM, described as follows:

NAME OF CLAIM	ORIGINAL RECORDATION BOOK	PAGE	PRESENT RECORDATION INSTRUMENT	BLM SERIAL NUMBER (AMC)
Iron Reef # 1-10	1168	69-88		105953-105962
Copperstone # 1-14			95-05841	335231-335244
Formerly: 1128	65-80		86-2300 – 2313	91283-91296
Copperstone # 15-17			95-05841	335245-335247
Formerly: 1129	627-632		86-2314 – 2316	88612-88614
Copperstone # 18-29			95-05841	335248-335259
Formerly: 1131	294-309		86-2317 – 2328	95246-95257
Copperstone # 30-40	1151	145-157	86-2329 – 2339	98423-98433
Copperstone # 41-53	1152	181-205	86-2340 – 2352	98957-98969
Copperstone # 54-57	1152	763-770	86-2353 – 2356	98970-98973
Copperstone # 58-62	1152	771-779	86-2357 – 2361	98974-98978
Copperstone # 63	1152	781	86-2362	98979
Copperstone # 64-65	1173	716-719	86-2363 – 2364	108058-108059
Copperstone # 101-115	1254	76-105	86-2365 – 2379	144884-144898
Copperstone # 116A	1254	107	86-2380	144899
Copperstone # 117-120	1254	109-115	86-2381 – 2384	144900-144903
Copperstone # 122-127	1254	119-129	86-2385 – 2390	144905-144910
Copperstone # 129-131	1254	133-138	86-2391 – 2393	144912-144914
Copperstone # 132-133	1254	139-140	86-2394 – 2395	144915-144916
Copperstone # 134	1254	142	86-2396	144917
Copperstone # 136-139	1254	147-154	86-2397 – 2400	144919-144922
Copperstone # 140-150	1254	155-175	86-2401 – 2411	144923-144933
Copperstone # 151-161	1254	176-197	86-2412 – 2422	144934-144944
Copperstone # 162-171	1276	349-371	86-2423 – 2432	164418-164427
Copperstone # 172A	1276	373	86-2433	164428
Copperstone # 183A	1276	395	86-2434	164439
Copperstone # 184-191	1276	397-410	86-2435 – 2442	164440-164447
Copperstone # 192A	1276	412	86-2443	164448
Copperstone # 210-315	1276	448-658	86-2444 – 2549	164466-164571
Copperstone # 316-328	84-2460-2472		86-2550 – 2562	220648-220660
Copperstone # 329-339	86-4548-4558			260459-260469

Claim Name	BLM/AMC#	La Paz Co. Recordation #
CSA 1 – CSA 51	362237 – 362287	2004-03993 to 2004-04043

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APPENDIX B

Drillhole Collar Locations by Year

Appendix B: Drillholes

All coordinates are in Arizona State Place West NAD 27 feet. Total depths are in feet. Negative dips point down.

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CSD-2	333780.0	1044874.0	884.9	294	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-20	333683.1	1044973.0	885.6	250	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-21	333877.9	1044975.0	883.6	350	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-23	333478.9	1045366.0	881.3	220	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-25	333286.0	1045368.0	891.3	200	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-26	333372.9	1045684.0	879.9	250	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-27	332878.6	1045972.0	871.0	180	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-28	333097.2	1045975.0	876.0	200	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-3	333371.4	1045303.0	885.8	241	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-31	333974.6	1045080.0	881.8	400	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-33	333381.2	1045475.0	881.8	220	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-34	333581.0	1045479.0	881.6	300	-90.0	0.0	1984	DDH	Cyprus	Included
CSD-59	333980.7	1044676.0	883.2	301	-90.0	0.0	1984	DDH	Cyprus	Included
CR-380	333528.7	1045925.0	740.2	360	-90.0	0.0	1985	RC	Cyprus	Included
CR-381	333594.5	1045817.0	739.0	410	-90.0	0.0	1985	RC	Cyprus	Included
CR-382	333565.3	1046076.0	740.8	385	-90.0	0.0	1985	RC	Cyprus	Included
CS-103	332123.0	1046968.0	868.3	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-106	332695.0	1047778.0	867.0	640	-90.0	0.0	1985	RC	Cyprus	Included
CS-107	335085.0	1045767.0	876.4	680	-90.0	0.0	1985	RC	Cyprus	Included
CS-108	335060.0	1045368.0	882.2	680	-90.0	0.0	1985	RC	Cyprus	Included
CS-109	332295.0	1047780.0	868.0	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-111	332273.0	1047385.0	862.0	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-112	331540.3	1044815.9	851.0	450	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-113	331131.3	1042853.8	852.2	400	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-115	333176.0	1047385.0	868.0	790	-90.0	0.0	1985	RC	Cyprus	Included
CS-116	331528.8	1045984.9	851.1	500	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-117	331489.7	1043967.5	849.7	500	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-119A	336770.9	1043692.8	871.6	500	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-120	328302.5	1038613.8	855.5	285	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-121	327765.3	1038947.7	903.0	300	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-122	328824.9	1037564.0	960.0	440	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-123	328628.4	1038222.1	918.0	280	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-124	331085.8	1041855.0	860.0	300	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-125	328691.2	1037247.7	960.0	160	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-126	330618.1	1038527.5	957.0	325	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-127	330079.2	1038730.4	925.0	290	-90.0	0.0	1985	RC	Cyprus	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-128	328662.9	1037557.9	990.0	200	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-129	330795.0	1037946.5	902.0	250	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-130	331786.9	1042251.3	860.0	250	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-131	327596.6	1038253.6	920.0	200	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-132	328115.5	1038050.8	920.0	270	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-133A	326097.0	1034769.6	950.0	800	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-134	325308.0	1032341.7	890.0	340	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-135A	326405.0	1032329.9	885.0	600	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-136	326202.9	1035130.0	920.0	400	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-137	332020.9	1042439.1	860.0	300	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-137A	332009.8	1042970.1	876.0	820	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-138	328489.0	1040590.9	875.0	500	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-139	328882.0	1038730.0	905.0	480	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-140	332467.0	1038768.0	870.0	300	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-141	329521.7	1033388.2	880.0	670	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-151	333184.9	1045960.0	891.6	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-152	333097.6	1045877.0	891.0	160	-90.0	0.0	1985	RC	Cyprus	Included
CS-153	332885.4	1046071.0	869.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-154	332789.4	1045981.0	869.1	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-155	332886.8	1045883.0	870.8	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-156	333013.0	1046172.0	870.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-157	332988.4	1045973.0	872.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-158	333087.7	1046068.0	878.3	240	-90.0	0.0	1985	RC	Cyprus	Included
CS-159	333180.7	1045777.0	890.5	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-160	333282.7	1045873.0	892.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-161	333385.4	1045970.0	878.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-162	332781.7	1046179.0	866.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-163	333288.6	1046084.0	873.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-164	333385.6	1045875.0	884.0	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-165	333136.7	1045823.0	892.1	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-166	333574.3	1044793.0	894.5	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-167	333682.8	1044877.0	885.7	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-168	333780.5	1044972.0	884.1	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-169	333878.0	1044872.0	883.9	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-170	333778.2	1044771.0	887.6	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-171	333678.6	1044676.0	892.6	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-172	334386.1	1044679.0	879.3	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-173	334483.9	1044777.0	879.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-174	333983.2	1044771.0	883.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-175	334479.8	1045165.0	882.9	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-176	333636.0	1044726.0	892.5	200	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-177	333526.0	1044630.0	891.2	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-178	333286.8	1045677.0	882.7	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-179	333234.7	1045921.0	893.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-180	333191.0	1045593.0	880.1	100	-90.0	0.0	1985	RC	Cyprus	Included
CS-181	333033.5	1045724.0	879.2	100	-90.0	0.0	1985	RC	Cyprus	Included
CS-182	333384.8	1045577.0	878.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-183	333284.2	1045475.0	884.3	100	-90.0	0.0	1985	RC	Cyprus	Included
CS-184	333282.8	1044984.0	892.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-185	333079.3	1045180.0	888.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-186	333487.8	1045474.0	879.3	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-187	333384.5	1045368.0	882.7	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-188	333282.0	1045269.0	884.1	100	-90.0	0.0	1985	RC	Cyprus	Included
CS-189	334564.5	1044874.0	880.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-190	334367.3	1044963.0	880.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-191	334170.6	1045075.0	882.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-192	334072.8	1044975.0	883.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-193	333579.9	1045377.0	881.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-194	333481.7	1045277.0	882.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-195	333354.2	1045168.0	890.0	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-196	333681.6	1045275.0	884.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-197	333585.0	1045177.0	888.9	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-198	333485.4	1045073.0	888.9	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-199	333682.8	1045071.0	885.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-200	334164.7	1045268.0	880.0	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-201	332887.1	1045585.0	876.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-202	334109.6	1045417.0	884.9	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-203	333883.4	1045084.0	880.9	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-204	334272.8	1045077.0	882.6	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-205	334078.0	1044270.0	879.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-206	334181.7	1044372.0	879.8	305	-90.0	0.0	1985	RC	Cyprus	Included
CS-207	334078.5	1044473.0	878.4	220	-90.0	0.0	1985	RC	Cyprus	Included
CS-208	332618.7	1046105.0	866.0	225	-90.0	0.0	1985	RC	Cyprus	Included
CS-209	332789.3	1046278.0	868.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-210	333581.9	1044974.0	886.6	225	-90.0	0.0	1985	RC	Cyprus	Included
CS-211	333778.3	1045175.0	885.7	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-212	334062.2	1045267.0	880.1	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-213	334063.9	1045169.0	879.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-214	334262.6	1045268.0	879.5	340	-90.0	0.0	1985	RC	Cyprus	Included
CS-215	334264.7	1045168.0	880.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-216	334368.1	1045167.0	880.3	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-217	334164.0	1044765.0	880.8	300	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-218	334280.6	1044867.0	881.3	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-219	334467.8	1045074.0	881.6	355	-90.0	0.0	1985	RC	Cyprus	Included
CS-220	334468.2	1044870.0	884.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-221	334564.2	1044973.0	880.1	340	-90.0	0.0	1985	RC	Cyprus	Included
CS-222	334281.6	1044672.0	879.7	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-223	334383.2	1044774.0	879.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-224	334569.1	1044773.0	879.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-225	334669.9	1044877.0	879.6	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-226	334472.2	1044667.0	878.8	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-227	334180.5	1044973.0	883.0	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-228	334081.2	1044871.0	882.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-229	333971.0	1044974.0	882.7	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-230	334088.1	1045067.0	881.1	305	-90.0	0.0	1985	RC	Cyprus	Included
CS-231	334164.9	1045169.0	880.2	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-232	334365.5	1045272.0	880.2	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-233	334281.1	1044575.0	879.7	345	-90.0	0.0	1985	RC	Cyprus	Included
CS-234	333881.8	1044675.0	885.1	275	-90.0	0.0	1985	RC	Cyprus	Included
CS-235	333030.7	1045930.0	877.4	225	-90.0	0.0	1985	RC	Cyprus	Included
CS-236A	332887.0	1046270.0	866.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-237	334164.5	1045367.0	882.5	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-238	333892.0	1045295.0	883.8	385	-90.0	0.0	1985	RC	Cyprus	Included
CS-238A	333892.0	1045295.0	883.8	275	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-239	333791.7	1045384.0	883.3	365	-90.0	0.0	1985	RC	Cyprus	Included
CS-240	333684.7	1045476.0	881.9	375	-90.0	0.0	1985	RC	Cyprus	Included
CS-241	333585.0	1045575.0	878.8	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-242	333488.7	1045684.0	877.9	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-243	333381.1	1045773.0	882.4	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-244	332686.8	1046281.0	865.1	260	-90.0	0.0	1985	RC	Cyprus	Included
CS-245	333189.4	1046180.0	871.6	305	-90.0	0.0	1985	RC	Cyprus	Included
CS-246	333349.4	1046179.0	870.5	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-247	333011.2	1045811.0	880.2	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-248	333120.0	1045705.0	892.3	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-249	333658.2	1045548.0	878.3	365	-90.0	0.0	1985	RC	Cyprus	Included
CS-250	333882.8	1045179.0	883.8	345	-90.0	0.0	1985	RC	Cyprus	Included
CS-251	334183.6	1044571.0	882.1	245	-90.0	0.0	1985	RC	Cyprus	Included
CS-252	334278.3	1044465.0	881.0	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-253	334375.0	1044466.0	881.9	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-254	334373.6	1044568.0	881.9	275	-90.0	0.0	1985	RC	Cyprus	Included
CS-255	333217.7	1045436.0	891.9	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-256	334346.4	1046043.0	878.4	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-257	332647.0	1047330.0	871.0	605	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-258	334349.9	1046244.0	877.3	805	-90.0	0.0	1985	RC	Cyprus	Included
CS-259	334673.0	1045167.0	882.2	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-260	332669.6	1045586.0	873.7	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-261	333196.0	1045086.0	893.4	705	-90.0	0.0	1985	RC	Cyprus	Included
CS-262	332894.3	1044969.0	917.7	540	-90.0	0.0	1985	RC	Cyprus	Included
CS-263	340429.1	1047619.0	884.4	600	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-264	332595.0	1047485.0	866.5	880	-90.0	0.0	1985	RC	Cyprus	Included
CS-265	332083.0	1044776.8	867.3	570	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-266	332596.3	1044536.0	879.4	905	-90.0	0.0	1985	RC	Cyprus	Included
CS-267	333985.0	1046892.0	875.0	1160	-90.0	0.0	1985	RC	Cyprus	Included
CS-268	332530.0	1048180.0	872.0	905	-90.0	0.0	1985	RC	Cyprus	Included
CS-269	334563.4	1045080.0	883.0	425	-90.0	0.0	1985	RC	Cyprus	Included
CS-270	334665.2	1045076.0	882.5	425	-90.0	0.0	1985	RC	Cyprus	Included
CS-271	334568.0	1045170.0	883.9	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-273	334950.0	1045990.0	880.0	1140	-90.0	0.0	1985	RC	Cyprus	Included
CS-274	333785.2	1045576.0	881.9	485	-90.0	0.0	1985	RC	Cyprus	Included
CS-275	333882.8	1045491.0	882.5	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-276	333985.0	1044173.0	882.7	305	-90.0	0.0	1985	RC	Cyprus	Included
CS-277	334075.7	1044073.0	882.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-278	333984.6	1044280.0	883.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-279	332775.0	1046269.0	870.5	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-280	332984.0	1046379.0	870.1	450	-90.0	0.0	1985	RC	Cyprus	Included
CS-281	334383.0	1045373.0	880.9	425	-90.0	0.0	1985	RC	Cyprus	Included
CS-282	334283.0	1045473.0	882.7	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-283	333680.0	1045673.0	879.2	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-284	334480.0	1045273.0	881.4	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-285	333434.5	1046301.0	870.2	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-286	333257.0	1046298.0	869.3	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-287	334681.0	1045273.0	881.7	605	-90.0	0.0	1985	RC	Cyprus	Included
CS-288	333480.0	1046073.0	875.6	375	-90.0	0.0	1985	RC	Cyprus	Included
CS-289	333955.0	1044373.0	881.3	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-290	334030.0	1044333.0	883.0	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-291	333480.0	1045873.0	875.6	265	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-292	334130.0	1044333.0	881.3	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-293	333930.0	1044223.0	884.5	175	-90.0	0.0	1985	RC	Cyprus	Included
CS-294	334030.0	1044223.0	882.6	175	-90.0	0.0	1985	RC	Cyprus	Included
CS-295	334775.0	1045173.0	880.9	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-296	333985.0	1045273.0	880.6	425	-90.0	0.0	1985	RC	Cyprus	Included
CS-297	334280.0	1044273.0	882.0	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-298	334145.0	1044223.0	880.0	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-299	334180.0	1044173.0	882.2	350	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-300	333985.0	1045173.0	879.9	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-301	334035.0	1044123.0	882.2	275	-90.0	0.0	1985	RC	Cyprus	Included
CS-302	333583.0	1045768.0	878.1	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-303	333883.0	1045573.0	881.2	530	-90.0	0.0	1985	RC	Cyprus	Included
CS-304	334085.0	1044673.0	859.6	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-305	332935.0	1046323.0	868.0	375	-90.0	0.0	1985	RC	Cyprus	Included
CS-306	333480.0	1045873.0	875.6	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-307	332835.0	1046223.0	868.5	275	-90.0	0.0	1985	RC	Cyprus	Included
CS-308	332685.0	1046073.0	869.9	250	-90.0	0.0	1985	RC	Cyprus	Included
CS-309	334783.0	1044973.0	879.9	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-310	332585.0	1046173.0	865.6	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-311	334380.0	1045478.0	882.1	550	-90.0	0.0	1985	RC	Cyprus	Included
CS-312	332735.0	1046223.0	867.8	305	-90.0	0.0	1985	RC	Cyprus	Included
CS-313	334480.0	1045378.0	880.7	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-314	333085.0	1046473.0	867.1	345	-90.0	0.0	1985	RC	Cyprus	Included
CS-315	334080.0	1045578.0	881.4	610	-90.0	0.0	1985	RC	Cyprus	Included
CS-316	333080.0	1046273.0	868.5	395	-90.0	0.0	1985	RC	Cyprus	Included
CS-317	334183.0	1045476.0	881.8	565	-90.0	0.0	1985	RC	Cyprus	Included
CS-318	334285.0	1045573.0	882.2	555	-90.0	0.0	1985	RC	Cyprus	Included
CS-319	333584.3	1045679.0	881.1	450	-90.0	0.0	1985	RC	Cyprus	Included
CS-320	333773.9	1045462.0	879.3	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-321	333883.9	1045382.0	881.1	550	-90.0	0.0	1985	RC	Cyprus	Included
CS-322	334185.8	1045675.0	881.5	545	-90.0	0.0	1985	RC	Cyprus	Included
CS-323	334388.9	1045668.0	880.8	656	-90.0	0.0	1985	RC	Cyprus	Included
CS-324	334079.3	1045381.0	881.3	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-325	333781.3	1045690.0	880.0	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-326	333679.6	1045786.0	876.4	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-327	333579.2	1045886.0	875.2	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-328	333480.7	1045985.0	872.5	400	-90.0	0.0	1985	RC	Cyprus	Included
CS-329	334387.2	1045280.0	880.5	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-330	334579.2	1045278.0	879.9	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-331	334779.7	1045274.0	882.6	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-332	334776.8	1045069.0	877.2	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-333	334581.1	1045474.0	881.9	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-334	333198.4	1046365.0	867.4	480	-90.0	0.0	1985	RC	Cyprus	Included
CS-335	334187.7	1045870.0	877.0	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-336	334187.5	1046075.0	875.3	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-337	334481.1	1045576.0	881.4	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-338	333976.7	1045684.0	878.0	565	-90.0	0.0	1985	RC	Cyprus	Included
CS-339	334878.0	1045174.0	879.8	620	-90.0	0.0	1985	RC	Cyprus	Included
CS-340	334289.6	1045968.0	877.8	685	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-341	334393.2	1045859.0	879.6	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-342	334093.0	1045959.0	877.3	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-343	334094.1	1046162.0	877.1	615	-90.0	0.0	1985	RC	Cyprus	Included
CS-344	333964.2	1046094.0	877.4	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-345	334777.6	1045486.0	882.0	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-346	334878.7	1045386.0	881.1	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-347	333978.4	1046275.0	877.7	1000	-90.0	0.0	1985	RC	Cyprus	Included
CS-348	333989.3	1045868.0	879.8	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-349	334082.7	1045781.0	860.6	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-350	333176.1	1046559.0	868.8	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-351	333080.0	1046675.0	868.0	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-352	333886.2	1045771.0	860.6	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-353	334078.0	1046159.0	877.4	625	-90.0	0.0	1985	RC	Cyprus	Included
CS-354	333780.0	1045876.0	860.0	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-355	334584.7	1045378.0	880.9	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-356	334478.6	1045474.0	881.1	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-357	334381.2	1045574.0	881.8	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-358	334180.3	1045575.0	881.2	500	-90.0	0.0	1985	RC	Cyprus	Included
CS-359	333879.2	1045671.0	860.0	550	-90.0	0.0	1985	RC	Cyprus	Included
CS-360	334080.5	1045674.0	882.1	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-361	334880.8	1044875.0	879.6	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-362	334631.2	1044724.0	860.1	450	-90.0	0.0	1985	RC	Cyprus	Included
CS-363	334980.0	1044575.0	875.0	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-364	335030.4	1045224.8	882.3	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-365	334182.6	1045771.0	879.9	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-366	334081.3	1045875.0	878.1	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-367	334279.1	1045874.0	879.8	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-368	334483.3	1045770.0	879.7	680	-90.0	0.0	1985	RC	Cyprus	Included
CS-369	334435.8	1046307.0	876.4	880	-90.0	0.0	1985	RC	Cyprus	Included
CS-370	334193.3	1046277.0	875.5	795	-90.0	0.0	1985	RC	Cyprus	Included
CS-371	334431.7	1046115.0	877.4	690	-90.0	0.0	1985	RC	Cyprus	Included
CS-372	333291.1	1046670.0	869.3	550	-90.0	0.0	1985	RC	Cyprus	Included
CS-373	333190.0	1046780.0	862.0	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-374	333182.0	1046875.0	869.5	690	-90.0	0.0	1985	RC	Cyprus	Included
CS-375	332988.0	1046882.0	869.5	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-376	333080.0	1046977.0	869.0	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-377	333981.4	1045775.0	861.2	640	-90.0	0.0	1985	RC	Cyprus	Included
CS-378	332990.0	1046680.0	870.0	550	-90.0	0.0	1985	RC	Cyprus	Included
CS-379	334185.0	1046472.0	880.0	835	-90.0	0.0	1985	RC	Cyprus	Included
CS-383	333377.1	1046274.0	740.3	410	-90.0	0.0	1985	RC	Cyprus	Included
CS-384	333478.0	1046275.0	740.5	450	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-385	333478.5	1046174.0	739.9	385	-90.0	0.0	1985	RC	Cyprus	Included
CS-386	333378.8	1046372.0	739.4	440	-90.0	0.0	1985	RC	Cyprus	Included
CS-387	333228.4	1046424.0	739.7	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-388	333129.1	1046522.0	740.9	440	-90.0	0.0	1985	RC	Cyprus	Included
CS-389	333297.7	1046492.0	739.4	480	-90.0	0.0	1985	RC	Cyprus	Included
CS-390	332977.3	1046476.0	741.3	385	-90.0	0.0	1985	RC	Cyprus	Included
CS-391	332827.1	1046324.0	740.6	310	-90.0	0.0	1985	RC	Cyprus	Included
CS-392	333427.6	1046326.0	740.7	470	-90.0	0.0	1985	RC	Cyprus	Included
CS-393	333280.5	1046373.0	740.3	405	-90.0	0.0	1985	RC	Cyprus	Included
CS-394	333556.7	1046188.0	740.0	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-395	332626.0	1046223.0	739.8	285	-90.0	0.0	1985	RC	Cyprus	Included
CS-396	332726.0	1046324.0	740.2	325	-90.0	0.0	1985	RC	Cyprus	Included
CS-397	332622.7	1046314.0	740.4	310	-90.0	0.0	1985	RC	Cyprus	Included
CS-398	332575.2	1046270.0	740.1	310	-90.0	0.0	1985	RC	Cyprus	Included
CS-399	332538.8	1046216.0	740.2	445	-90.0	0.0	1985	RC	Cyprus	Included
CS-400	333129.5	1046424.0	740.5	385	-90.0	0.0	1985	RC	Cyprus	Included
CS-401	333226.2	1046323.0	740.1	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-402	333331.1	1046324.0	740.8	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-403	333327.0	1046223.0	740.0	345	-90.0	0.0	1985	RC	Cyprus	Included
CS-404	333428.3	1046225.0	740.5	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-405	333429.4	1046123.0	740.3	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-406	333527.5	1046026.0	740.6	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-407	333329.5	1046423.0	737.8	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-408	333179.2	1046473.0	740.0	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-409	333128.5	1046326.0	740.3	440	-90.0	0.0	1985	RC	Cyprus	Included
CS-410	333232.0	1046525.0	739.5	465	-90.0	0.0	1985	RC	Cyprus	Included
CS-411	333028.0	1046523.0	740.8	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-412	333572.8	1045977.0	739.2	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-413	333526.6	1046124.0	740.6	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-414	333554.7	1046252.0	741.3	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-415	333416.8	1046413.0	740.1	420	-90.0	0.0	1985	RC	Cyprus	Included
CS-416	333462.9	1046361.0	741.0	485	-90.0	0.0	1985	RC	Cyprus	Included
CS-417	333512.0	1046303.0	740.3	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-418	333097.8	1046557.0	739.6	410	-90.0	0.0	1985	RC	Cyprus	Included
CS-419	332827.1	1046416.0	739.8	340	-90.0	0.0	1985	RC	Cyprus	Included
CS-420	332938.8	1046506.0	742.5	460	-90.0	0.0	1985	RC	Cyprus	Included
CS-421	333019.9	1046438.0	740.5	385	-90.0	0.0	1985	RC	Cyprus	Included
CS-422	333361.4	1046452.0	739.1	465	-90.0	0.0	1985	RC	Cyprus	Included
CS-423	333517.2	1046213.0	739.1	440	-90.0	0.0	1985	RC	Cyprus	Included
CS-424	333591.4	1046487.0	872.9	670	-90.0	0.0	1985	RC	Cyprus	Included
CS-425	333435.6	1046624.0	870.4	700	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-426	333628.0	1045925.0	760.5	430	-90.0	0.0	1985	RC	Cyprus	Included
CS-427	333688.2	1045686.0	758.2	405	-90.0	0.0	1985	RC	Cyprus	Included
CS-428	333728.1	1045626.0	755.2	440	-90.0	0.0	1985	RC	Cyprus	Included
CS-429	333541.8	1046535.0	874.1	750	-90.0	0.0	1985	RC	Cyprus	Included
CS-430	333728.9	1046224.0	882.0	785	-90.0	0.0	1985	RC	Cyprus	Included
CS-431	333703.2	1045893.0	802.4	520	-90.0	0.0	1985	RC	Cyprus	Included
CS-432	333828.7	1045727.0	800.4	485	-90.0	0.0	1985	RC	Cyprus	Included
CS-433	333777.1	1045775.0	799.7	560	-90.0	0.0	1985	RC	Cyprus	Included
CS-434	333588.7	1045759.0	721.4	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-434-A	333577.8	1045775.0	721.7	125	-90.0	0.0	1985	RC	Cyprus	Included
CS-435	333624.0	1045723.0	720.0	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-436	333846.9	1045638.0	799.7	525	-90.0	0.0	1985	RC	Cyprus	Included
CS-437	333738.0	1045739.0	800.5	515	-90.0	0.0	1985	RC	Cyprus	Included
CS-438	333981.3	1045580.0	800.2	545	-90.0	0.0	1985	RC	Cyprus	Included
CS-439	333822.2	1045817.0	800.8	525	-90.0	0.0	1985	RC	Cyprus	Included
CS-440	333930.6	1045826.0	801.4	545	-90.0	0.0	1985	RC	Cyprus	Included
CS-441	333927.7	1045725.0	800.0	555	-90.0	0.0	1985	RC	Cyprus	Included
CS-442	333932.2	1045626.0	800.9	525	-90.0	0.0	1985	RC	Cyprus	Included
CS-443	333934.7	1045527.0	800.5	515	-90.0	0.0	1985	RC	Cyprus	Included
CS-444	334028.7	1045633.0	800.0	560	-90.0	0.0	1985	RC	Cyprus	Included
CS-445	333489.0	1046580.0	870.7	665	-90.0	0.0	1985	RC	Cyprus	Included
CS-446	333380.2	1046674.0	869.4	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-447	333302.9	1046708.0	870.6	725	-90.0	0.0	1985	RC	Cyprus	Included
CS-448	333231.2	1046734.0	871.6	650	-90.0	0.0	1985	RC	Cyprus	Included
CS-449	333428.7	1046632.0	870.1	625	-79.0	225.0	1985	RC	Cyprus	Included
CS-45	332265.0	1046365.0	859.9	320	-90.0	0.0	1985	RC	Cyprus	Included
CS-450	333641.3	1046436.0	872.2	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-451	333534.9	1046539.0	871.3	620	-80.0	225.0	1985	RC	Cyprus	Included
CS-452	333373.3	1046667.0	868.9	140	-75.5	225.0	1985	RC	Cyprus	Included
CS-453	333628.6	1046426.0	873.3	640	-82.0	225.0	1985	RC	Cyprus	Included
CS-454	333727.4	1046228.0	884.3	645	-81.0	225.0	1985	RC	Cyprus	Included
CS-455	333727.2	1046125.0	884.7	600	-80.0	225.0	1985	RC	Cyprus	Included
CS-456	331168.9	1043270.0	857.4	765	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-457	332123.4	1044049.5	871.9	665	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-458	331723.5	1044077.0	859.6	922	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-459	331101.8	1045402.8	849.4	600	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-46	332275.0	1046160.0	864.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CS-460	331287.2	1044273.0	854.8	745	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-461	342087.9	1050225.9	889.0	360	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-462	344189.5	1048618.9	882.8	600	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-463	352150.0	1060290.0	882.0	870	*Assumed Vertical*		1985	RC	Cyprus	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-464	342419.8	1059917.1	899.2	720	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-465	348420.0	1054500.0	885.0	660	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-466	342026.4	1048365.6	886.3	625	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-467	337490.7	1040424.8	858.7	600	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-468	333984.8	1043800.4	886.4	520	-90.0	0.0	1985	RC	Cyprus	Included
CS-469	334578.2	1044676.2	659.0	130	-90.0	0.0	1985	RC	Cyprus	Included
CS-470	334578.4	1044775.0	660.0	200	-90.0	0.0	1985	RC	Cyprus	Included
CS-471	334479.5	1044777.3	659.7	160	-90.0	0.0	1985	RC	Cyprus	Included
CS-472	334377.7	1044875.5	660.0	180	-90.0	0.0	1985	RC	Cyprus	Included
CS-473	334377.5	1044975.5	660.8	180	-90.0	0.0	1985	RC	Cyprus	Included
CS-474	334377.3	1045076.3	661.1	210	-90.0	0.0	1985	RC	Cyprus	Included
CS-475	334278.1	1045174.8	659.7	215	-90.0	0.0	1985	RC	Cyprus	Included
CS-476	341224.4	1048479.8	886.7	710	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-477	341260.1	1049449.1	887.6	805	-90.0	0.0	1985	RC	Cyprus	Excluded
CS-478	327440.6	1034614.2	907.7	385	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-479	326712.7	1035055.3	931.3	350	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-480	326405.8	1034292.1	929.2	500	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-481	332824.9	1046925.3	871.2	580	-80.0	135.0	1985	RC	Cyprus	Included
CS-482	332775.6	1046874.8	874.2	620	-90.0	0.0	1985	RC	Cyprus	Included
CS-483	332770.5	1046776.6	867.3	520	-90.0	0.0	1985	RC	Cyprus	Included
CS-484	332748.4	1046674.4	866.1	600	-90.0	0.0	1985	RC	Cyprus	Included
CS-485	333027.5	1046725.2	560.0	320	-65.0	315.0	1985	RC	Cyprus	Included
CS-486	333381.5	1045972.5	559.4	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-487	333341.9	1046032.3	558.8	170	-90.0	0.0	1985	RC	Cyprus	Included
CS-488	333127.8	1046227.5	559.4	160	-90.0	0.0	1985	RC	Cyprus	Included
CS-489	333230.4	1046125.7	600.0	180	-90.0	0.0	1985	RC	Cyprus	Included
CS-490	333175.8	1046176.3	600.3	150	-90.0	0.0	1985	RC	Cyprus	Included
CS-491	334226.5	1045873.3	725.1	480	*Assumed Vertical*		1985	RC	Cyprus	Excluded
CS-492	334175.2	1046031.6	875.5	700	-90.0	0.0	1985	RC	Cyprus	Included
CS-493	334257.4	1045890.9	725.7	545	-82.0	315.0	1985	RC	Cyprus	Included
CS-494	332833.3	1046922.7	870.1	450	-68.0	135.0	1985	RC	Cyprus	Included
CS-495	332784.0	1046867.6	869.7	440	-61.0	135.0	1985	RC	Cyprus	Included
CS-496	332740.3	1046814.1	869.2	400	-63.0	135.0	1985	RC	Cyprus	Included
CS-55	332462.0	1046576.0	862.0	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-56	332260.0	1046580.0	862.5	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-57	332066.0	1046570.0	862.9	350	-90.0	0.0	1985	RC	Cyprus	Included
CS-58	332105.0	1046780.0	868.3	360	-90.0	0.0	1985	RC	Cyprus	Included
CS-59	333674.0	1046975.0	870.0	820	-90.0	0.0	1985	RC	Cyprus	Included
CS-62	333080.0	1046782.0	863.8	550	-90.0	0.0	1985	RC	Cyprus	Included
CS-64	332881.0	1046783.0	862.6	725	-90.0	0.0	1985	RC	Cyprus	Included
CS-72	332685.0	1046785.0	861.7	550	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CS-73	332881.0	1046981.0	867.0	675	-90.0	0.0	1985	RC	Cyprus	Included
CS-74	332678.0	1047384.0	867.0	540	-90.0	0.0	1985	RC	Cyprus	Included
CS-80	332480.0	1047385.0	867.0	525	-90.0	0.0	1985	RC	Cyprus	Included
CS-99	332479.0	1046978.0	869.8	725	-90.0	0.0	1985	RC	Cyprus	Included
CSD-1	333301.9	1045971.0	879.8	364	-90.0	0.0	1985	RC	Cyprus	Included
CSD-10	331492.6	1047008.1	854.4	852	-90.0	0.0	1985	RC	Cyprus	Excluded
CSD-11	334651.8	1044978.0	876.6	546	-90.0	0.0	1985	RC/DDH	Cyprus	Included
CSD-12	334301.1	1044237.0	875.0	546	-55.0	225.0	1985	DDH	Cyprus	Included
CSD-13	333879.0	1044789.0	884.3	222.5	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-15	333278.0	1045590.0	882.2	130	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-16	333675.3	1044572.0	893.3	269	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-17	333662.3	1044773.0	891.4	130	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-18	333881.2	1044388.0	881.8	304	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-19	334078.3	1044171.0	879.9	249	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-22	334272.2	1044966.0	882.0	301.5	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-24	333685.6	1045376.0	881.6	392	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-29	333258.2	1046177.0	872.5	320	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-30	333096.7	1046384.0	870.7	400	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-32	333782.2	1045077.0	882.0	270	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-35	333188.7	1045878.0	895.3	400	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-36	333584.0	1045276.0	884.7	250	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-37	333570.0	1045070.0	891.8	250	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-38	333787.3	1045273.0	886.0	370	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-39	332779.0	1045865.0	870.2	150	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-4	333379.0	1045673.0	879.8	347	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-40	332675.7	1045959.0	868.6	170	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-41	332775.3	1046063.0	868.9	220	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-42	332991.2	1046071.0	871.1	221	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-43	332988.4	1046271.0	868.7	246	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-43A	332994.5	1046269.0	867.2	351	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-44	333192.0	1046271.0	870.7	341	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-45	333283.9	1046381.0	869.6	301	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-46	333975.2	1044871.0	883.2	400	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-47	334173.0	1044870.0	883.1	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-48	334270.9	1044767.0	880.8	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-49	334365.7	1044873.0	881.4	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-5	334179.7	1044282.0	879.5	400.5	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-50	332983.1	1045874.0	878.1	189	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-51	333586.5	1044872.0	889.6	150	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-52	333577.3	1044467.0	884.5	150	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-53	333685.5	1044373.0	883.0	200	-90.0	0.0	1985	DDH	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CSD-54	333389.0	1045880.0	885.3	200	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-55	333775.7	1044278.0	881.3	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-56	333187.6	1045676.0	887.9	150	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-57	333576.9	1044677.0	895.4	200	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-58	333783.0	1044679.0	889.9	350	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-6	332695.7	1046174.0	867.1	282	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-60	333188.4	1046080.0	877.7	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-61	333977.7	1044482.0	881.5	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-62	334175.0	1044674.0	881.0	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-63	334376.8	1045078.0	884.4	301	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-64	334178.7	1044484.0	880.5	300	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-65	333780.7	1044469.0	880.9	199	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-66	333781.1	1044897.0	885.7	285.5	-55.0	225.0	1985	DDH	Cyprus	Included
CSD-67	333085.5	1044977.0	897.9	432	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-68	333208.7	1045489.0	885.9	300	-45.0	270.0	1985	DDH	Cyprus	Included
CSD-69	334127.5	1044525.0	880.1	721	-45.0	270.0	1985	DDH	Cyprus	Included
CSD-7	333275.0	1046975.0	872.0	721	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-70	332843.0	1044341.0	883.3	210	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-73	334180.5	1045971.0	877.7	722	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-74	333629.4	1045723.0	857.8	585.4	-90.0	0.0	1985	DDH	Cyprus	Included
CSD-75	333177.5	1045131.4	868.5	510	-90.0	0.0	1985	DDH	Cyprus	Included
CSR-1	333479.6	1044785.0	898.7	125	-90.0	0.0	1985	RC	Cyprus	Included
CSR-10	334075.1	1044384.0	880.0	840	-90.0	0.0	1985	RC	Cyprus	Included
CSR-100	334240.0	1046142.0	871.8	840	-90.0	0.0	1985	RC	Cyprus	Included
CSR-100A	334245.0	1046121.0	871.0	825	-90.0	0.0	1985	RC	Cyprus	Included
CSR-101	335087.0	1046167.0	874.4	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-102	334687.0	1046169.0	872.7	750	-90.0	0.0	1985	RC	Cyprus	Included
CSR-104	334689.0	1046569.0	878.7	700	-90.0	0.0	1985	RC	Cyprus	Included
CSR-105	334289.0	1046571.0	870.0	750	-90.0	0.0	1985	RC	Cyprus	Included
CSR-11	333487.2	1044976.0	889.9	130	-90.0	0.0	1985	RC	Cyprus	Included
CSR-110	334869.4	1044976.0	876.1	600	-90.0	0.0	1985	RC	Cyprus	Included
CSR-114	332878.0	1047383.0	872.8	690	-90.0	0.0	1985	RC	Cyprus	Included
CSR-118	333464.3	1044785.0	898.7	635	-90.0	0.0	1985	RC	Cyprus	Included
CSR-11A	333486.6	1044963.0	890.0	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-12	333091.3	1045977.0	873.0	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-13	333086.5	1045585.0	879.9	265	-90.0	0.0	1985	RC	Cyprus	Included
CSR-14	333491.9	1045189.0	885.9	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-142	333666.2	1044781.0	889.3	280	-90.0	0.0	1985	RC	Cyprus	Included
CSR-143	333065.2	1044944.0	899.3	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-144	333092.7	1045775.0	891.4	280	-90.0	0.0	1985	RC	Cyprus	Included
CSR-145	333196.9	1044870.0	895.0	500	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CSR-146	332894.8	1044761.0	903.1	325	-90.0	0.0	1985	RC	Cyprus	Included
CSR-147	332980.5	1045076.0	908.6	430	-90.0	0.0	1985	RC	Cyprus	Included
CSR-148	332783.9	1044868.0	898.4	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-15	333688.4	1044978.0	884.4	225	-90.0	0.0	1985	RC	Cyprus	Included
CSR-16	333272.1	1045197.0	884.4	200	-90.0	0.0	1985	RC	Cyprus	Included
CSR-17	333484.6	1045374.0	880.2	220	-90.0	0.0	1985	RC	Cyprus	Included
CSR-18	332889.6	1046188.0	868.6	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-19	333872.6	1044200.0	879.9	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-2	333678.8	1044584.0	893.3	200	-90.0	0.0	1985	RC	Cyprus	Included
CSR-20	334074.5	1044187.0	878.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-21	334073.2	1044591.0	881.4	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-22	333874.0	1044784.0	884.3	270	-90.0	0.0	1985	RC	Cyprus	Included
CSR-23	333889.2	1044987.0	883.0	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-24	333689.3	1045191.0	887.1	145	-90.0	0.0	1985	RC	Cyprus	Included
CSR-24A	333675.7	1045191.0	887.1	130	-90.0	0.0	1985	RC	Cyprus	Included
CSR-25	333689.8	1045386.0	881.1	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-26	333490.2	1045585.0	878.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-27	333488.9	1045785.0	879.0	220	-90.0	0.0	1985	RC	Cyprus	Included
CSR-28	332887.9	1045978.0	871.1	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-29	332689.1	1046164.0	867.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-3	333282.8	1045376.0	891.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-30	332688.7	1046379.0	863.6	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-31	332894.3	1046378.0	866.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-32	333288.7	1045962.0	880.7	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-33B	334302.7	1044199.0	877.7	305	-90.0	0.0	1985	RC	Cyprus	Included
CSR-34	334696.0	1043986.9	878.8	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-35	334480.5	1043980.6	882.4	400	-90.0	0.0	1985	RC	Cyprus	Included
CSR-36	334687.4	1043787.1	877.3	400	-90.0	0.0	1985	RC	Cyprus	Included
CSR-37	334891.7	1043594.7	874.9	380	-90.0	0.0	1985	RC	Cyprus	Included
CSR-38	334081.3	1044790.0	883.1	430	-90.0	0.0	1985	RC	Cyprus	Included
CSR-39	333719.4	1045185.0	888.1	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-4	333088.4	1045770.0	891.1	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-40	332979.2	1045674.0	889.5	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-41	333068.5	1046176.0	871.6	350	-90.0	0.0	1985	RC	Cyprus	Included
CSR-42	333273.4	1046179.0	872.0	450	-90.0	0.0	1985	RC	Cyprus	Included
CSR-43	332480.8	1046170.4	866.9	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-44	332469.2	1046373.0	860.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-47	334287.0	1043983.0	882.0	360	-90.0	0.0	1985	RC	Cyprus	Included
CSR-48	334275.5	1043773.4	879.2	340	-90.0	0.0	1985	RC	Cyprus	Included
CSR-49	334481.5	1043780.2	877.8	305	-90.0	0.0	1985	RC	Cyprus	Included
CSR-5	333282.7	1045796.0	893.7	46	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CSR-50	334687.5	1043575.1	876.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-51	333488.3	1045780.0	879.7	310	-90.0	0.0	1985	RC	Cyprus	Included
CSR-52	333488.3	1045798.0	878.9	470	-90.0	0.0	1985	RC	Cyprus	Included
CSR-53	333683.7	1046581.0	873.0	110	-90.0	0.0	1985	RC	Cyprus	Included
CSR-54	333670.4	1046565.0	873.2	750	-90.0	0.0	1985	RC	Cyprus	Included
CSR-5A	333266.6	1045801.0	894.0	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-6	333879.2	1044595.0	883.4	160	-90.0	0.0	1985	RC	Cyprus	Included
CSR-60	333086.1	1046385.0	866.7	420	-90.0	0.0	1985	RC	Cyprus	Included
CSR-61	332883.8	1046579.0	860.7	530	-90.0	0.0	1985	RC	Cyprus	Included
CSR-62	333081.8	1046781.7	863.8	550	-90.0	0.0	1985	RC	Cyprus	Included
CSR-63	333391.7	1046077.0	873.3	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-65	332681.2	1045373.0	874.9	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-66	334277.7	1044978.0	883.9	320	-90.0	0.0	1985	RC	Cyprus	Included
CSR-67	333480.0	1044572.0	883.4	400	-90.0	0.0	1985	RC	Cyprus	Included
CSR-68	332468.8	1044381.0	892.4	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-69	332575.6	1043881.8	872.2	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-7	333278.3	1045577.0	882.2	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-70	333893.3	1045985.0	877.9	700	-90.0	0.0	1985	RC	Cyprus	Included
CSR-71	333485.8	1046376.0	873.4	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-75A	331463.8	1047007.9	853.9	470	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-76	331275.6	1047144.3	855.3	500	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-77	334466.9	1044972.0	877.6	400	-90.0	0.0	1985	RC	Cyprus	Included
CSR-78A	334674.4	1044975.0	876.5	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-79	334670.0	1044775.0	876.7	430	-90.0	0.0	1985	RC	Cyprus	Included
CSR-8	333684.7	1044774.0	888.6	190	-90.0	0.0	1985	RC	Cyprus	Included
CSR-81	335074.2	1044777.2	876.2	700	-90.0	0.0	1985	RC	Cyprus	Included
CSR-82A	335093.9	1044372.0	877.4	870	-90.0	0.0	1985	RC	Cyprus	Included
CSR-83	333470.0	1044171.0	877.8	595	-90.0	0.0	1985	RC	Cyprus	Included
CSR-84	333072.3	1044167.0	879.7	700	-90.0	0.0	1985	RC	Cyprus	Included
CSR-85	334673.7	1044371.0	877.0	700	-90.0	0.0	1985	RC	Cyprus	Included
CSR-86	334672.9	1045373.0	876.6	550	-90.0	0.0	1985	RC	Cyprus	Included
CSR-87A	334288.0	1044188.0	874.7	650	-90.0	0.0	1985	RC	Cyprus	Included
CSR-88	334475.1	1044568.0	877.1	560	-90.0	0.0	1985	RC	Cyprus	Included
CSR-89	333956.9	1045464.0	876.7	550	-90.0	0.0	1985	RC	Cyprus	Included
CSR-9	333895.8	1044376.0	882.8	300	-90.0	0.0	1985	RC	Cyprus	Included
CSR-90	334265.4	1045368.0	877.1	500	-90.0	0.0	1985	RC	Cyprus	Included
CSR-90A	334285.6	1045370.0	876.5	200	-90.0	0.0	1985	RC	Cyprus	Included
CSR-91	331639.0	1047279.0	856.6	500	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-92	334685.0	1045769.0	880.7	800	-90.0	0.0	1985	RC	Cyprus	Included
CSR-93	330909.8	1047441.9	859.1	350	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-94	335069.5	1044968.9	876.2	750	-90.0	0.0	1985	RC	Cyprus	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CSR-95	330770.5	1047061.6	855.1	400	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-96A	336322.5	1042518.2	864.1	815	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-97	333422.2	1040894.9	870.5	600	-90.0	0.0	1985	RC	Cyprus	Excluded
CSR-98	334285.0	1045771.0	877.1	775	-90.0	0.0	1985	RC	Cyprus	Included
CSD-71	334731.6	1045215.0	882.2	620	-90.0	0.0	1986	DDH	Cyprus	Included
CSD-72	334283.0	1045673.0	879.5	569	-90.0	0.0	1986	DDH	Cyprus	Included
CSD-76	332384.2	1044149.0	875.3	274.3	-90.0	0.0	1988	DDH	Cyprus	Included
CSD-8	333884.9	1046383.0	874.2	950	-90.0	0.0	1988	DDH	Cyprus	Included
CSD-9	332678.0	1047410.0	866.4	676	-90.0	0.0	1988	DDH	Cyprus	Included
DCU-1	333265.0	1044640.0	890.0	700	-65.0	242.0	1993	RC	Santa Fe	Excluded
DCU-10	331390.0	1043940.0	855.0	700	-65.0	238.0	1993	RC	Santa Fe	Excluded
DCU-11	331730.0	1044120.0	859.0	700	-65.0	241.0	1993	RC	Santa Fe	Excluded
DCU-12	337380.0	1045930.0	883.0	700	-90.0	0.0	1993	RC	Santa Fe	Excluded
DCU-13	331490.0	1044760.0	856.0	700	-65.0	239.0	1993	RC	Santa Fe	Excluded
DCU-14	331130.0	1044550.0	852.0	700	-65.0	239.0	1993	RC	Santa Fe	Excluded
DCU-15	336600.0	1046550.0	884.0	500	-90.0	0.0	1993	RC	Santa Fe	Excluded
DCU-16	336700.0	1046740.0	883.0	800	-90.0	0.0	1993	RC	Santa Fe	Excluded
DCU-17	340500.0	1048500.0	885.0	700	-90.0	0.0	1993	RC	Santa Fe	Excluded
DCU-2	332940.0	1045380.0	881.0	700	-65.0	240.0	1993	RC	Santa Fe	Included
DCU-3	334160.0	1046240.0	878.0	1000	-90.0	0.0	1993	RC	Santa Fe	Included
DCU-4	333943.0	1046565.0	875.0	800	-90.0	0.0	1993	RC	Santa Fe	Included
DCU-5	332800.0	1044140.0	881.0	700	-65.0	240.0	1993	RC	Santa Fe	Included
DCU-6	332310.0	1044770.0	872.0	700	-65.0	242.0	1993	RC	Santa Fe	Excluded
DCU-7	332500.0	1043550.0	870.0	700	-65.0	245.0	1993	RC	Santa Fe	Excluded
DCU-8	333760.0	1045350.0	540.0	1000	-90.0	0.0	1993	RC	Santa Fe	Included
DCU-9	332120.0	1043330.0	867.0	700	-65.0	240.0	1993	RC	Santa Fe	Excluded
C95-01	334989.7	1045574.2	882.4	744.5	-68.0	225.0	1995	DDH	Royal Oak	Included
C95-02	333062.8	1044953.3	901.8	500	-55.0	226.0	1995	DDH	Royal Oak	Included
C95-03	333677.1	1046868.9	875.8	850	-90.0	0.0	1995	RC	Royal Oak	Included
C95-04	333744.0	1046843.8	876.6	820	-74.0	225.0	1995	RC	Royal Oak	Included
C95-05	333624.5	1046923.4	875.8	850	-75.0	225.0	1995	RC	Royal Oak	Included
C95-06	333431.4	1047036.0	873.3	800	-70.0	225.0	1995	RC	Royal Oak	Included
C95-07	334327.9	1046428.5	877.2	820	-90.0	0.0	1995	RC	Royal Oak	Included
C95-08	333218.8	1047114.7	872.5	785	-90.0	0.0	1995	RC	Royal Oak	Included
C95-09	334154.7	1046363.0	875.5	796	-90.0	0.0	1995	RC	Royal Oak	Included
C95-10	332770.8	1047066.3	871.4	624	-90.0	0.0	1995	RC	Royal Oak	Included
C95-11	333369.4	1047384.9	875.9	795	-90.0	0.0	1995	RC	Royal Oak	Included
C95-12	334081.9	1046695.1	878.7	867	-90.0	0.0	1995	RC	Royal Oak	Included
C95-13	333998.6	1046396.7	874.7	750	-65.0	224.0	1995	RC	Royal Oak	Included
C96-14	334470.0	1047170.0	920.8	1227	-90.0	0.0	1996	RC/DDH	Royal Oak	Included
C96-15	334225.0	1047320.0	921.7	1313	-90.0	0.0	1996	DDH	Royal Oak	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
C96-16	334030.0	1047520.0	875.0	1245	-90.0	0.0	1996	DDH	Royal Oak	Included
C96-17	334513.8	1046528.3	895.9	1007	-90.0	0.0	1996	DDH	Royal Oak	Included
C96-18	333853.4	1046744.0	876.6	958	-90.0	0.0	1996	RC	Royal Oak	Included
C96-19	332792.4	1047558.1	874.3	704	-90.0	0.0	1996	DDH	Royal Oak	Included
C97-20	335084.8	1045660.4	883.2	950	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-21	333588.6	1047269.6	876.3	710	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-22	334392.4	1046766.4	905.3	1123	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-23	332928.0	1047124.0	873.9	628	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-24	332828.0	1047720.0	874.9	700	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-25	332726.0	1048018.0	875.3	801	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-26	332630.0	1047620.0	872.7	650	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-27	332928.0	1047818.0	875.5	600	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-28	333175.0	1047568.0	881.8	834	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-29	333520.0	1047220.0	876.2	906	-90.0	0.0	1997	DDH	Royal Oak	Included
C97-30	333880.0	1046873.0	879.3	931	-90.0	0.0	1997	DDH/RC	Royal Oak	Included
C97-31	332933.0	1048037.3	875.4	870	-90.0	0.0	1997	RC	Royal Oak	Included
C97-32	334528.0	1046225.0	881.4	950	-90.0	0.0	1997	RC	Royal Oak	Included
C97-33	332780.0	1047270.0	880.0	571	-90.0	0.0	1997	RC/DDH	Royal Oak	Included
C97-34	332978.0	1047572.0	875.3	734	-90.0	0.0	1997	DDH	Royal Oak	Included
A98-1	332744.2	1047643.3	873.8	665.2	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-10	332771.8	1047171.3	873.9	651.5	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-11	332883.6	1047823.8	874.3	734.3	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-12	332969.6	1048170.3	875.8	943.2	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-13	332872.0	1047573.6	874.5	693.9	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-14	332911.7	1047911.8	874.6	745	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-15	332957.5	1047758.1	875.1	747.1	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-2	332878.0	1047677.5	874.7	731	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-3	332825.8	1047824.4	874.0	728	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-4	332707.9	1047507.4	873.1	660	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-5	332775.6	1047475.5	873.2	690	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-6	333424.3	1047323.9	876.6	900	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-7	333433.7	1047132.8	875.3	630	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-8	333300.9	1047300.3	875.2	915	-90.0	0.0	1998	DDH	Asia Minerals	Included
A98-9	332698.4	1046997.7	870.5	545	-90.0	0.0	1998	DDH	Asia Minerals	Included
A00-1	332819.3	1048319.6	885.9	534	-90.0	0.0	2000	DDH	Asia Minerals	Included
A00-10	333711.8	1045430.8	539.4	649	-89.7	159.0	2000	DDH	Asia Minerals	Included
A00-11	332799.7	1046994.7	871.7	700	-90.0	0.0	2000	RC	Asia Minerals	Included
A00-2	332835.1	1047934.3	875.2	743.5	-87.8	350.0	2000	DDH	Asia Minerals	Included
A00-3	332925.0	1048416.1	878.1	595	-90.0	0.0	2000	RC	Asia Minerals	Included
A00-4	332762.2	1047365.6	873.8	645.5	-87.8	174.0	2000	DDH	Asia Minerals	Included
A00-5	332729.5	1047128.5	872.4	637.5	-87.4	223.0	2000	DDH	Asia Minerals	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
A00-6	333024.7	1048512.8	878.0	890	-90.0	0.0	2000	RC	Asia Minerals	Included
A00-7	333028.9	1048715.8	876.1	1035	-90.0	0.0	2000	RC	Asia Minerals	Included
A00-8	332824.7	1048519.8	876.0	880	-90.0	0.0	2000	RC	Asia Minerals	Included
A00-9	333223.5	1048714.6	877.5	1300	-90.0	0.0	2000	RC	Asia Minerals	Included
CDH-1	332746.2	1047073.8	281.9	77	-10.0	62.0	2001	DDH	Asia Minerals	Included
CDH-10	332741.3	1047077.2	285.1	92	12.0	35.0	2001	DDH	Asia Minerals	Included
CDH-2	332746.2	1047073.8	284.6	68	12.0	62.0	2001	DDH	Asia Minerals	Included
CDH-3	332744.8	1047074.7	285.4	80	12.0	51.0	2001	DDH	Asia Minerals	Included
CDH-4	332744.8	1047074.7	282.5	75	-11.0	51.0	2001	DDH	Asia Minerals	Included
CDH-5	332747.5	1047071.8	285.9	118	12.0	76.0	2001	DDH	Asia Minerals	Included
CDH-5A	332747.5	1047071.8	282.0	8	-10.0	76.0	2001	DDH	Asia Minerals	Included
CDH-6	332745.9	1047067.8	284.2	75	10.5	103.0	2001	DDH	Asia Minerals	Included
CDH-7	332744.3	1047065.3	284.4	101	10.0	125.0	2001	DDH	Asia Minerals	Included
CDH-8	332747.5	1047071.8	283.0	98	5.0	76.0	2001	DDH	Asia Minerals	Included
CDH-9	332747.5	1047071.8	288.8	101	20.0	76.0	2001	DDH	Asia Minerals	Included
CRD-03-01	333708.9	1045438.4	538.7	600	-90.0	0.0	2003	RC	Bonanza	Included
CRD-03-02	333705.9	1045444.5	538.6	600	-90.0	0.0	2003	DDH	Bonanza	Included
CRD-03-03	333702.4	1045453.3	537.4	520	-85.0	330.0	2003	DDH	Bonanza	Included
CRD-03-04	333759.0	1045360.0	548.9	595	-90.0	0.0	2003	RC	Bonanza	Included
CRD-03-05	333758.7	1045363.0	548.9	600	-90.0	0.0	2003	DDH	Bonanza	Included
CRD-03-06	333738.0	1045397.0	548.0	650	-90.0	0.0	2003	DDH	Bonanza	Included
CRD-03-07	333731.0	1045394.0	548.0	600	-85.0	240.0	2003	DDH	Bonanza	Included
CRD-03-08	333757.0	1045355.6	549.3	600	-85.0	240.0	2003	DDH	Bonanza	Included
CRD-03-09	333761.8	1045355.6	548.7	600	-85.0	150.0	2003	DDH	Bonanza	Included
CRD-03-10	333760.2	1045398.4	546.0	600	-85.0	282.0	2003	DDH	Bonanza	Included
CRD-03-11	333801.8	1045376.3	547.5	600	-90.0	0.0	2003	DDH	Bonanza	Included
CRD-03-12	333810.3	1045421.6	547.2	600	-85.0	330.0	2003	DDH	Bonanza	Included
CRD-03-13	333777.3	1045414.9	547.0	600	-90.0	0.0	2003	DDH	Bonanza	Included
CUDH-03-01	332718.2	1047576.3	300.5	138	20.0	96.0	2003	DDH	Bonanza	Included
CUDH-03-02	332718.2	1047576.3	299.0	196.5	10.0	96.0	2003	DDH	Bonanza	Included
CUDH-03-03	332718.2	1047576.3	297.5	172	0.0	96.0	2003	DDH	Bonanza	Included
CUDH-03-04	332718.2	1047576.3	296.0	199	-5.0	96.0	2003	DDH	Bonanza	Included
CUDH-03-05	332718.2	1047576.3	294.5	340.1	-15.0	96.0	2003	DDH	Bonanza	Included
CUDH-03-06	332717.5	1047580.0	300.5	124	20.0	66.0	2003	DDH	Bonanza	Included
CUDH-03-07	332717.5	1047580.0	299.0	128	10.0	66.0	2003	DDH	Bonanza	Included
CUDH-03-08	332717.5	1047580.0	297.5	167	0.0	66.0	2003	DDH	Bonanza	Included
CUDH-03-09	332717.5	1047580.0	296.0	244	-5.0	66.0	2003	DDH	Bonanza	Included
CUDH-03-10	332718.1	1047578.8	299.0	159	10.0	79.0	2003	DDH	Bonanza	Included
CUDH-03-11	332718.1	1047578.8	297.5	222.5	0.0	79.0	2003	DDH	Bonanza	Included
CUDH-03-12	332718.1	1047578.8	296.0	226.2	-5.0	79.0	2003	DDH	Bonanza	Included
CUDH-03-13	332718.1	1047578.8	295.0	349	-10.0	79.0	2003	DDH	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
CUDH-03-14	332714.2	1047583.4	296.0	299	-5.0	45.0	2003	DDH	Bonanza	Included
CUDH-03-15	332714.2	1047583.4	297.5	274	0.0	45.0	2003	DDH	Bonanza	Included
CRD-04-01	333797.3	1046833.2	877.0	885	-82.0	180.0	2004	RC/DDH	Bonanza	Included
CRD-04-02	333819.1	1046821.6	877.0	920	-90.0	0.0	2004	RC/DDH	Bonanza	Included
CRD-04-03	333790.7	1046831.5	877.0	920	-76.0	189.0	2004	RC/DDH	Bonanza	Included
CRD-04-04	333927.2	1046728.9	876.0	988	-90.0	0.0	2004	RC/DDH	Bonanza	Included
CRD-04-05	333880.1	1046613.5	874.0	896	-90.0	0.0	2004	RC/DDH	Bonanza	Included
CRD-04-06	333886.8	1046622.6	875.0	901	-80.0	229.0	2004	RC/DDH	Bonanza	Included
CRD-04-07	333786.3	1046828.4	877.0	869	-71.0	203.0	2004	RC/DDH	Bonanza	Included
CRD-04-08	333786.3	1046819.3	877.0	850	-64.0	214.0	2004	RC/DDH	Bonanza	Included
CRD-04-09	333786.2	1046819.3	877.0	951	-59.0	219.0	2004	RC/DDH	Bonanza	Included
CRD-04-10	333714.8	1046874.9	876.0	850	-66.0	219.0	2004	RC/DDH	Bonanza	Included
CRD-04-11	333896.8	1046639.6	876.0	847	-73.0	239.0	2004	RC/DDH	Bonanza	Included
CRD-04-12	333988.9	1046771.6	879.0	987	-90.0	0.0	2004	RC/DDH	Bonanza	Included
CRD-04-13	333715.6	1046889.2	877.0	977	-90.0	0.0	2004	RC/DDH	Bonanza	Included
CUDH-04-16	332714.2	1047583.4	295.0	299	-10.0	45.0	2004	DDH	Bonanza	Included
CUDH-04-17	332716.0	1047582.5	297.5	267.5	0.0	48.0	2004	DDH	Bonanza	Included
CUDH-04-18	332716.0	1047582.5	296.0	255	-5.0	48.0	2004	DDH	Bonanza	Included
CUDH-04-19	332716.0	1047582.5	295.0	284	-10.0	48.0	2004	DDH	Bonanza	Included
CUDH-04-20	332712.9	1047584.0	294.5	353	-12.0	28.0	2004	DDH	Bonanza	Included
CUDH-04-21	332712.9	1047584.0	295.5	279	-8.0	28.0	2004	DDH	Bonanza	Included
CUDH-04-22	332712.9	1047584.0	296.3	250	-4.0	28.0	2004	DDH	Bonanza	Included
CUDH-04-23	332712.9	1047584.0	295.5	509	-16.0	28.0	2004	DDH	Bonanza	Included
CUDH-04-24	332715.8	1047583.4	295.0	344	-10.0	45.0	2004	DDH	Bonanza	Included
CUDH-04-25	332715.8	1047583.4	294.0	508.6	-15.0	45.0	2004	DDH	Bonanza	Included
CUDH-04-26	332715.8	1047583.4	294.0	749	-18.0	45.0	2004	DDH	Bonanza	Included
CUDH-04-27	332712.9	1047584.0	294.0	46	-19.0	28.0	2004	DDH	Bonanza	Included
CUDH-04-28	332712.0	1047580.0	296.5	219	5.0	45.0	2004	DDH	Bonanza	Included
CUDH-04-29	332713.0	1047579.0	294.0	345	-15.0	48.0	2004	DDH	Bonanza	Included
CUDH-04-30	332715.0	1047576.5	300.0	150	18.0	79.0	2004	DDH	Bonanza	Included
CUDH-04-31	332717.5	1047578.0	295.0	348	-10.0	66.0	2004	DDH	Bonanza	Included
CUDH-04-32	332718.2	1047576.3	294.0	764	-22.0	96.0	2004	DDH	Bonanza	Included
DU4-33	332718.2	1047576.3	295.0	499	-10.0	96.0	2004	DDH	Bonanza	Included
DU4-34	332713.5	1047583.5	298.0	266	7.0	37.0	2004	DDH	Bonanza	Included
DU4-35	332713.5	1047583.5	296.0	322	0.0	37.0	2004	DDH	Bonanza	Included
DU4-36	332713.5	1047583.5	295.5	352	-8.0	37.0	2004	DDH	Bonanza	Included
DU4-37	332713.5	1047583.5	294.0	489	-12.0	37.0	2004	DDH	Bonanza	Included
DU4-38	332713.5	1047583.5	294.0	867	-18.0	37.0	2004	DDH	Bonanza	Included
DU4-39	332718.1	1047579.6	296.0	264	-5.0	76.0	2004	DDH	Bonanza	Included
DU4-40	332718.1	1047578.8	294.0	349	-12.0	79.0	2004	DDH	Bonanza	Included
DU4-41	332717.5	1047580.0	294.0	537	-14.0	66.0	2004	DDH	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
DU4-42	332715.8	1047583.4	292.0	914	-21.0	48.0	2004	DDH	Bonanza	Included
DU4-43	332715.0	1047576.5	293.0	374	-17.0	79.0	2004	DDH	Bonanza	Included
DU4-44	332714.2	1047583.4	293.0	494	-20.0	45.0	2004	DDH	Bonanza	Included
DU4-45	332704.9	1047584.2	298.0	424	4.0	0.0	2004	DDH	Bonanza	Included
DU4-46	332708.6	1047584.8	297.5	349	0.0	14.0	2004	DDH	Bonanza	Included
DU4-47	332701.0	1047582.9	298.5	411	5.0	345.0	2004	DDH	Bonanza	Included
DU4-48	332701.0	1047582.9	296.5	279	12.0	345.0	2004	DDH	Bonanza	Included
DU4-49	332701.0	1047582.9	295.5	186.5	-2.0	345.0	2004	DDH	Bonanza	Included
DU4-50	332711.9	1047555.6	295.5	188	-9.0	100.0	2004	DDH	Bonanza	Included
DU4-51	332711.3	1047554.2	295.5	184	-9.0	112.0	2004	DDH	Bonanza	Included
DU4-52	332711.3	1047554.2	298.0	183.5	9.0	112.0	2004	DDH	Bonanza	Included
DU4-53	332711.9	1047555.6	298.0	179	9.0	100.0	2004	DDH	Bonanza	Included
DU4-54	332708.3	1047552.8	296.0	237	-5.0	140.0	2004	DDH	Bonanza	Included
DU4-55	332711.9	1047555.6	297.5	46	0.0	100.0	2004	DDH	Bonanza	Included
F4-1	333756.6	1045403.6	545.9	624	-83.0	290.0	2004	DDH	Bonanza	Included
F4-2	333757.9	1045399.1	546.6	789	-72.0	277.5	2004	DDH	Bonanza	Included
F4-3	333758.3	1045393.8	546.5	600	-75.0	256.0	2004	DDH	Bonanza	Included
F4-4	333764.6	1045359.1	549.3	660	-79.0	247.0	2004	DDH	Bonanza	Included
F4-5	333767.9	1045353.6	549.0	880	-84.0	218.0	2004	DDH	Bonanza	Included
F4-6	333767.5	1045345.4	549.3	776	-83.0	180.0	2004	DDH	Bonanza	Included
F4-7	333717.6	1045449.5	540.8	760	-80.0	286.0	2004	DDH	Bonanza	Included
F4-8	333784.4	1045294.8	561.5	750	-83.0	180.0	2004	DDH	Bonanza	Included
F4-9	333774.0	1045348.8	549.2	655	-79.0	345.0	2004	DDH	Bonanza	Included
H4-14	333782.6	1046827.2	877.0	850	-59.0	214.0	2004	DDH	Bonanza	Included
H4-15	333720.8	1046884.6	879.0	900	-73.0	219.0	2004	DDH	Bonanza	Included
H4-16	333724.6	1046884.9	879.0	1151	-78.0	219.0	2004	DDH	Bonanza	Included
H4-17	333627.9	1046919.4	877.0	1053	-90.0	0.0	2004	DDH	Bonanza	Included
H4-18	333779.3	1046912.7	878.0	872	-71.0	210.0	2004	DDH	Bonanza	Included
H4-19	333773.8	1046918.6	878.0	922	-70.8	209.4	2004	DDH	Bonanza	Included
H4-20	333630.7	1046941.3	877.0	940	-86.5	352.5	2004	DDH	Bonanza	Included
H4-21	333619.2	1046942.7	876.0	883	-59.0	219.0	2004	DDH	Bonanza	Included
H4-22	333725.5	1046889.5	879.0	972	-84.0	219.0	2004	DDH	Bonanza	Included
H4-23	333550.2	1046996.3	877.0	752	-59.0	219.0	2004	DDH	Bonanza	Included
H4-24	333553.6	1047000.0	877.0	819	-66.0	219.0	2004	DDH	Bonanza	Included
H4-25	333546.1	1047001.3	875.0	848	-72.0	219.0	2004	DDH	Bonanza	Included
H4-26	333445.9	1047045.3	875.0	782	-59.0	219.0	2004	DDH	Bonanza	Included
H4-27	333448.7	1047048.8	875.0	801	-66.0	219.0	2004	DDH	Bonanza	Included
H4-28	333435.0	1047050.8	874.0	863	-73.0	219.0	2004	DDH	Bonanza	Included
H4-29	333800.9	1046969.1	878.0	1051	-90.0	0.0	2004	DDH	Bonanza	Included
H4-30	333609.4	1047066.1	878.0	1013	-90.0	0.0	2004	DDH	Bonanza	Included
H4-31	333510.7	1047121.6	875.0	972	-90.0	0.0	2004	DDH	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
H4-32	333709.5	1047016.5	878.0	1063	-90.0	0.0	2004	DDH	Bonanza	Included
H4-33	333623.1	1046947.5	877.0	812	-66.0	219.0	2004	DDH	Bonanza	Included
H4-34	333625.9	1046950.5	877.0	850	-73.0	219.0	2004	DDH	Bonanza	Included
H4-35	333629.4	1046955.4	877.0	1012	-78.0	219.0	2004	DDH	Bonanza	Included
H4-36	333443.2	1047058.3	873.0	1000	-78.0	219.0	2004	DDH	Bonanza	Included
H4-37	333446.4	1047071.9	870.0	975	-84.0	219.0	2004	DDH	Bonanza	Included
H4-38	333450.5	1047072.6	874.0	975	-90.0	0.0	2004	DDH	Bonanza	Included
H4-39	333562.5	1047016.4	876.0	915	-78.0	219.0	2004	DDH	Bonanza	Included
H4-40	333554.6	1047005.8	876.0	970	-84.0	219.0	2004	DDH	Bonanza	Included
H4-41	333908.6	1046622.2	877.0	950	-90.0	0.0	2004	DDH	Bonanza	Included
H4-42	333911.7	1046696.6	876.0	952	-90.0	0.0	2004	DDH	Bonanza	Included
H4-43	333887.6	1046633.9	874.0	968	-71.0	229.0	2004	DDH	Bonanza	Included
H4-44	333548.8	1047011.3	876.0	996.5	-90.0	0.0	2004	DDH	Bonanza	Included
H4-45	333903.4	1046783.4	877.0	954	-90.0	0.0	2004	DDH	Bonanza	Included
H4-46	333794.0	1046809.2	878.0	999	-65.0	215.0	2004	DDH	Bonanza	Included
H4-47	333803.1	1046811.8	878.0	971	-76.0	200.0	2004	DDH	Bonanza	Included
H4-48	333757.0	1046886.5	877.0	920	-82.0	180.0	2004	DDH	Bonanza	Included
H4-49	334088.1	1046163.5	878.0	817	-90.0	0.0	2004	DDH	Bonanza	Included
H4-50	334295.6	1046191.8	878.0	1072	-90.0	0.0	2004	DDH	Bonanza	Included
H4-51	333295.0	1047296.1	876.0	877	-90.0	0.0	2004	DDH	Bonanza	Included
H4-52	333281.8	1047280.6	878.0	822	-75.0	219.0	2004	DDH	Bonanza	Included
H4-53	333581.1	1047184.3	874.0	989	-90.0	0.0	2004	DDH	Bonanza	Included
H4-54	333650.8	1047257.4	877.0	1054	-90.0	0.0	2004	DDH	Bonanza	Included
H4-55	334367.7	1046562.6	894.0	1200	-85.0	163.0	2004	DDH	Bonanza	Included
H4-56	334370.1	1046575.7	894.0	1000	-83.1	178.2	2004	DDH	Bonanza	Included
H4-57	334289.0	1046572.5	890.0	1024	-90.0	0.0	2004	DDH	Bonanza	Included
H4-58	333498.6	1047178.1	872.0	1000	-90.0	0.0	2004	DDH	Bonanza	Included
H4-59	333353.5	1047197.2	878.0	834.5	-76.1	208.3	2004	DDH	Bonanza	Included
H4-60	333354.8	1047197.5	878.0	983	-90.0	0.0	2004	DDH	Bonanza	Included
H4-61	333432.9	1047138.5	875.0	843	-75.0	220.2	2004	DDH	Bonanza	Included
H4-62	333437.4	1047139.8	877.0	1008	-90.0	0.0	2004	DDH	Bonanza	Included
H4-63	333843.7	1046928.4	879.0	1047	-90.0	0.0	2004	DDH	Bonanza	Included
H4-64	334013.0	1046519.5	875.0	891	-75.2	220.6	2004	DDH	Bonanza	Included
H4-65	334015.1	1046521.9	875.0	888	-90.0	0.0	2004	DDH	Bonanza	Included
H4-66	333777.1	1046826.1	878.0	900	-64.5	215.6	2004	DDH	Bonanza	Included
H4-67	333774.9	1046828.6	877.0	1002	-58.1	230.5	2004	DDH	Bonanza	Included
H4-68	333936.0	1046567.3	878.0	932	-58.9	229.7	2004	DDH	Bonanza	Included
H4-69	333942.8	1046567.6	875.0	970	-68.3	228.8	2004	DDH	Bonanza	Included
H4-70	333946.4	1046569.9	875.0	934	-76.6	226.7	2004	DDH	Bonanza	Included
H4-71	332999.1	1047260.9	877.0	784	-75.7	223.1	2004	DDH	Bonanza	Included
H4-72	333005.2	1047263.1	878.0	800	-90.0	0.0	2004	DDH	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
H4-73	333230.3	1047330.3	875.0	813	-75.5	223.5	2004	DDH	Bonanza	Included
H4-74	333229.9	1047337.6	891.0	875	-90.0	0.0	2004	DDH	Bonanza	Included
H4-75	333174.9	1047382.9	868.0	804	-75.7	214.5	2004	DDH	Bonanza	Included
H4-76	333287.2	1047390.6	876.0	852	-90.0	0.0	2004	DDH	Bonanza	Included
H4-77	333590.7	1047270.3	877.0	1022	-90.0	0.0	2004	DDH	Bonanza	Included
DU5-56	332778.8	1047550.2	298.0	198	-7.0	85.0	2005	DDH	Bonanza	Included
DU5-57	332778.8	1047549.3	298.0	181	-7.0	105.0	2005	DDH	Bonanza	Included
DU5-58	332778.8	1047549.3	300.0	161	10.0	105.0	2005	DDH	Bonanza	Included
DU5-59	332778.8	1047549.3	297.5	231	-12.0	105.0	2005	DDH	Bonanza	Included
DU5-60	332778.8	1047549.3	297.0	317.6	-18.0	105.0	2005	DDH	Bonanza	Included
DU5-61	332778.5	1047547.5	298.0	178.2	-5.0	115.0	2005	DDH	Bonanza	Included
DU5-62	332778.5	1047547.5	299.0	135	5.0	115.0	2005	DDH	Bonanza	Included
DU5-63	332778.5	1047547.5	301.0	114	15.0	115.0	2005	DDH	Bonanza	Included
DU5-64	332778.5	1047547.5	297.0	295	-15.0	115.0	2005	DDH	Bonanza	Included
DU5-65	332778.5	1047547.5	296.0	344	-20.0	115.0	2005	DDH	Bonanza	Included
DU5-66	332778.5	1047547.0	298.5	148.5	0.0	125.0	2005	DDH	Bonanza	Included
DU5-67	332778.5	1047547.0	301.0	119	15.0	125.0	2005	DDH	Bonanza	Included
DU5-68	332778.5	1047547.0	297.0	350.7	-10.0	125.0	2005	DDH	Bonanza	Included
DU5-69	332778.5	1047547.0	302.5	109	27.0	125.0	2005	DDH	Bonanza	Included
DU5-70	332711.9	1047555.6	300.0	178.4	12.0	120.0	2005	DDH	Bonanza	Included
DU5-71	332711.9	1047555.6	302.0	159	25.0	120.0	2005	DDH	Bonanza	Included
DU5-72	332710.6	1047553.2	298.0	230	10.0	140.0	2005	DDH	Bonanza	Included
DU5-73	332710.6	1047553.2	301.0	189	20.0	140.0	2005	DDH	Bonanza	Included
DU5-74	332707.6	1047552.8	300.0	349	12.0	160.0	2005	DDH	Bonanza	Included
DU5-75	332707.6	1047552.8	302.0	274	25.0	160.0	2005	DDH	Bonanza	Included
DU5-76	332703.8	1047553.1	302.0	373.4	23.0	190.0	2005	DDH	Bonanza	Included
DU5-78	332701.6	1047553.0	301.0	274.8	18.0	190.0	2005	DDH	Bonanza	Included
H4-78	332825.7	1047776.0	875.0	744	-90.0	0.0	2005	DDH	Bonanza	Included
H4-79	333777.1	1046826.1	878.0	1200	-65.6	221.9	2005	DDH	Bonanza	Included
H4-80	332775.4	1047826.1	875.0	769	-90.0	0.0	2005	DDH	Bonanza	Included
H4-81	332829.0	1047863.0	875.0	904	-90.0	0.0	2005	DDH	Bonanza	Included
H4-82	333445.0	1047384.8	877.0	924	-90.0	0.0	2005	DDH	Bonanza	Included
H4-83	333232.6	1047130.9	873.0	1060	-77.1	218.2	2005	DDH	Bonanza	Included
H4-84	333235.8	1047135.7	873.0	876	-90.0	0.0	2005	DDH	Bonanza	Included
H4-85	333371.0	1047447.4	878.0	914	-90.0	0.0	2005	DDH	Bonanza	Included
H4-86	333294.7	1047459.7	877.0	900	-90.0	0.0	2005	DDH	Bonanza	Included
H5-100	334306.8	1046135.1	878.0	1027	-90.0	0.0	2005	DDH	Bonanza	Included
H5-101	333364.0	1047320.6	876.0	937	-90.0	0.0	2005	DDH	Bonanza	Included
H5-102	334227.4	1046192.6	905.0	944	-90.0	0.0	2005	DDH	Bonanza	Included
H5-103	333827.5	1046896.7	878.0	975	-90.0	0.0	2005	DDH	Bonanza	Included
H5-104	334146.0	1046312.1	878.0	440	-90.0	0.0	2005	RC	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
H5-105	333620.5	1047026.9	878.0	200	-90.0	0.0	2005	RC	Bonanza	Included
H5-106	334136.9	1046314.2	878.0	1026	-90.0	0.0	2005	DDH	Bonanza	Included
H5-107	333620.5	1047026.9	878.0	1040	-90.0	0.0	2005	RC	Bonanza	Included
H5-108	334175.1	1046125.9	878.0	1137	-90.0	0.0	2005	RC	Bonanza	Included
H5-109	334410.0	1046253.7	879.1	1228	-90.0	0.0	2005	RC	Bonanza	Included
H5-110	332764.0	1047883.1	874.9	778	-90.0	0.0	2005	RC	Bonanza	Included
H5-111	332716.5	1047842.8	874.3	814	-90.0	0.0	2005	RC	Bonanza	Included
H5-112	332765.5	1047739.5	874.5	711	-90.0	0.0	2005	RC	Bonanza	Included
H5-113	334173.4	1046127.0	878.6	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-114	333328.2	1047345.1	876.6	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-115	333336.2	1047343.9	876.2	920	-90.0	0.0	2005	RC	Bonanza	Included
H5-116	334183.4	1046128.3	877.7	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-117	334163.6	1046125.4	878.9	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-118	334299.3	1046132.6	878.6	1035	-80.6	209.2	2005	RC	Bonanza	Included
H5-119	334169.9	1046137.2	878.6	834.8	-90.0	0.0	2005	RC	Bonanza	Included
H5-120	333524.8	1047337.7	878.3	360	-90.0	0.0	2005	RC	Bonanza	Included
H5-121	333391.7	1047171.9	875.7	924	-90.0	0.0	2005	RC	Bonanza	Included
H5-122	333605.8	1047366.6	879.2	540	-90.0	0.0	2005	RC	Bonanza	Included
H5-123	334200.7	1046229.1	876.5	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-124	334204.8	1046230.3	876.9	929	-90.0	0.0	2005	RC/DDH	Bonanza	Included
H5-125	333202.3	1047601.0	877.7	300	-90.0	0.0	2005	RC	Bonanza	Included
H5-126	333661.0	1047346.8	878.6	1070	-90.0	0.0	2005	RC	Bonanza	Included
H5-127	333200.8	1047609.3	878.2	260	-90.0	0.0	2005	RC	Bonanza	Included
H5-128	333245.6	1047555.3	881.8	420	-90.0	0.0	2005	RC	Bonanza	Included
H5-129	333238.2	1047465.9	878.8	640	-90.0	0.0	2005	RC	Bonanza	Included
H5-130	333800.1	1048002.5	881.0	1210	-90.0	0.0	2005	RC	Bonanza	Excluded
H5-131	333631.7	1047237.2	878.5	640	-90.0	0.0	2005	RC	Bonanza	Included
H5-132	333941.3	1046845.7	881.7	630	-90.0	0.0	2005	RC	Bonanza	Included
H5-133	333171.8	1047480.0	878.1	640	-90.0	0.0	2005	RC	Bonanza	Included
H5-134	333926.8	1046926.0	881.2	200	-90.0	0.0	2005	RC	Bonanza	Included
H5-135	333906.0	1046597.3	877.0	300	-79.0	219.0	2005	RC	Bonanza	Included
H5-136	333078.6	1048277.6	877.6	1455	-90.0	0.0	2005	RC	Bonanza	Included
H5-137	332289.6	1048304.7	874.6	620	-90.0	0.0	2005	RC	Bonanza	Included
H5-138	333187.7	1048820.3	877.3	1600	-90.0	0.0	2005	RC	Bonanza	Included
H5-139	332353.3	1048869.4	874.3	1202	-90.0	0.0	2005	RC	Bonanza	Excluded
H5-140	332399.8	1049502.3	869.8	1200	-90.0	0.0	2005	RC	Bonanza	Excluded
H5-141	333058.2	1044889.9	903.4	600	-90.0	0.0	2005	RC	Bonanza	Included
H5-142	332253.0	1047698.9	871.6	1202	-90.0	0.0	2005	RC	Bonanza	Included
H5-143	332553.2	1047561.8	872.2	790	-90.0	0.0	2005	RC	Bonanza	Included
H5-144	333898.8	1046601.2	874.9	300	-90.0	0.0	2005	RC	Bonanza	Included
H5-145	333904.6	1046608.6	875.9	320	-90.0	0.0	2005	RC	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
H5-146	333910.3	1046615.7	875.7	300	-90.0	0.0	2005	RC	Bonanza	Included
H5-147	333036.2	1044972.6	904.7	500	-90.0	0.0	2005	RC	Bonanza	Included
H5-148	333138.7	1044883.0	899.0	60	-90.0	0.0	2005	RC	Bonanza	Included
H5-149	334269.4	1047769.2	881.5	730	-90.0	0.0	2005	RC	Bonanza	Included
H5-150	332268.5	1047206.0	870.5	663	-89.8	250.6	2005	RC	Bonanza	Included
H5-151	334848.5	1048094.2	882.5	600	-90.0	0.0	2005	RC	Bonanza	Excluded
H5-152	333933.9	1047363.6	879.5	800	-81.0	109.6	2005	RC	Bonanza	Included
H5-153	334002.7	1047461.5	880.8	600	-85.0	119.0	2005	RC	Bonanza	Included
H5-154	332611.8	1047423.1	872.2	400	-89.6	40.3	2005	RC	Bonanza	Included
H5-155	333744.3	1046871.4	877.1	1029	-62.0	218.8	2005	RC	Bonanza	Included
H5-156	333751.9	1046880.4	877.8	1000	-66.4	217.8	2005	RC	Bonanza	Included
H5-157	333827.3	1046788.4	876.3	1000	-60.2	229.3	2005	RC	Bonanza	Included
H5-158	333832.6	1046793.3	876.5	1040	-62.0	226.0	2005	RC	Bonanza	Included
H5-159	333024.8	1048224.9	876.8	600	-90.0	0.0	2005	RC	Bonanza	Included
H5-160	333076.8	1048170.1	878.1	1200	-89.4	120.4	2005	RC	Bonanza	Included
H5-161	333131.7	1048222.0	878.4	400	-89.4	306.9	2005	RC	Bonanza	Included
H5-162	331482.1	1047023.0	862.1	1064	-89.7	15.8	2005	RC	Bonanza	Excluded
H5-163	333031.0	1045077.6	901.9	150	-90.0	0.0	2005	RC	Bonanza	Included
H5-87	331657.9	1047005.3	864.8	885	-90.0	0.0	2005	RC	Bonanza	Excluded
H5-88	333366.9	1047309.1	877.0	640	-90.0	0.0	2005	RC	Bonanza	Included
H5-89	333479.5	1047291.7	877.0	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-90	333543.2	1047285.3	878.0	260	-90.0	0.0	2005	RC	Bonanza	Included
H5-91	333535.7	1047277.8	877.0	990	-90.0	0.0	2005	RC	Bonanza	Included
H5-92	333471.0	1047296.3	877.0	965	-90.0	0.0	2005	RC	Bonanza	Included
H5-93	333437.9	1047450.1	877.0	942	-90.0	0.0	2005	RC	Bonanza	Included
H5-94	334208.8	1046232.7	888.0	240	-90.0	0.0	2005	RC	Bonanza	Included
H5-95	333863.6	1046768.9	877.0	1028	-90.0	0.0	2005	RC	Bonanza	Included
H5-96	333508.2	1047060.7	876.0	928	-90.0	0.0	2005	RC	Bonanza	Included
H5-97	333445.7	1047202.7	876.0	260	-90.0	0.0	2005	RC	Bonanza	Included
H5-98	333438.9	1047210.7	876.0	360	-90.0	0.0	2005	RC	Bonanza	Included
H5-99	333438.9	1047210.7	876.0	985	-90.0	0.0	2005	RC	Bonanza	Included
06CS-01	330696.3	1047751.4	871.5	1100.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-02	330963.6	1048015.0	873.8	1040.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-03	331412.6	1047898.5	878.3	1080.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-04	330893.0	1047374.9	867.5	1100.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-05	335481.2	1041373.0	878.4	980.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-06	335949.1	1041529.4	875.0	900.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-07	337657.8	1041874.8	870.5	950.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-08	337991.2	1042183.4	868.0	900.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-09	337477.6	1045911.3	890.4	865.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-10	332624.1	1042577.6	881.6	1160.0	-90.0	0.0	2006	DDH	Bonanza	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
06CS-11	335058.4	1045148.7	887.7	910.0	-90.0	0.0	2006	DDH	Bonanza	Included
06CS-12	334858.5	1043670.9	941.1	1160.0	-90.0	0.0	2006	DDH	Bonanza	Included
06CS-13	339525.0	1036682.0	876.0	620.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-14	335918.5	1044120.1	950.7	940.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-15	333799.8	1045400.1	549.7	600.0	-60.0	210.0	2006	DDH	Bonanza	Included
06CS-16	332692.1	1046157.3	752.2	760.0	-59.6	164.6	2006	DDH	Bonanza	Included
06CS-17	333011.7	1045941.5	730.0	600.0	-60.0	210.0	2006	DDH	Bonanza	Included
06CS-18	333985.7	1044785.6	618.4	540.0	-60.0	210.0	2006	DDH	Bonanza	Included
06CS-19	331019.8	1043618.8	859.7	965.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-20	331609.2	1041750.6	881.0	1105.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-21	331456.2	1042441.8	870.8	880.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-22	330830.6	1043400.3	859.0	1025.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-23	330790.1	1044422.4	856.0	1000.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-24	334754.4	1043646.1	940.7	1170.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-25	334822.7	1043639.3	940.8	1140.0	*Assumed Vertical*		2006	DDH	Bonanza	Excluded
06CS-26	337454.1	1045425.8	940.0	1020.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
06CS-27	333208.7	1050145.1	869.3	1000.0	-90.0	0.0	2006	DDH	Bonanza	Excluded
07CS-28	332884.1	1049718.3	882.0	780.0	*Assumed Vertical*		2007	DDH	Bonanza	Excluded
07CS-29	337031.7	1046272.5	889.2	1203.0	*Assumed Vertical*		2007	DDH	Bonanza	Excluded
07CS-30	330740.4	1043407.3	859.0	1203.0	-88.8	16.3	2007	DDH	Bonanza	Excluded
07CS-31	330823.7	1043290.4	859.0	1129.0	-88.2	210.5	2007	DDH	Bonanza	Excluded
07CS-32	334744.5	1043712.1	939.5	1184.0	-89.3	100.5	2007	DDH	Bonanza	Included
07CS-33	334873.3	1043580.4	942.1	1200.0	-89.1	338.5	2007	DDH	Bonanza	Included
07CS-34	334753.7	1044191.6	891.1	1017.0	-60.0	225.0	2007	DDH	Bonanza	Included
07CS-35	337378.7	1041842.8	973.5	800.0	*Assumed Vertical*		2007	DDH	Bonanza	Excluded
07CS-36	334679.3	1044544.5	876.1	1102.0	-54.0	225.0	2007	DDH	Bonanza	Included
07CS-37	334741.6	1045110.3	808.5	1095.0	-89.0	131.4	2007	DDH	Bonanza	Excluded
07CS-38	332854.3	1050543.3	865.1	1130.0	-89.5	31.4	2007	DDH	Bonanza	Excluded
07CS-39	332570.7	1049315.1	873.2	1200.0	-89.3	173.6	2007	DDH	Bonanza	Excluded
07CS-40	336131.3	1047462.5	889.0	1200	-89.1	76.3	2007	DDH	Bonanza	Excluded
07CS-41	336572.3	1047917.3	891.6	1190	-89.4	4.9	2007	DDH	Bonanza	Excluded
07CS-42	333487.3	1042314.4	884.5	1200	*Assumed Vertical*		2007	DDH	Bonanza	Excluded
07CS-43	330766.7	1042224.2	864.8	930	*Assumed Vertical*		2007	DDH	Bonanza	Excluded
07CS-44	337159.1	1041384.0	870.0	420	*Assumed Vertical*		2007	DDH	Bonanza	Excluded
08CS-45	330801.9	1043358.4	854.5	1130	-90.0	0.0	2008	DDH	Bonanza	Excluded
08CS-46	330770.4	1043276.1	854.8	1000	-90.0	0.0	2008	DDH	Bonanza	Excluded
08CS-47	330887.0	1043297.7	856.2	1282.5	-89.5	125.3	2008	DDH	Bonanza	Excluded
08CS-48	330837.6	1043226.6	856.3	1130	-89.7	187.9	2008	DDH	Bonanza	Excluded
08CS-49A	334360.6	1044119.8	893.7	1020	-90.0	0.0	2008	DDH	Bonanza	Included
08CS-50	334424.0	1044012.9	897.7	900	-89.5	177.3	2008	DDH	Bonanza	Included
08CS-51	334264.6	1044307.6	894.4	869	-90.0	0.0	2008	DDH	Bonanza	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
08CS-52	334258.4	1044144.3	897.0	910	-90.0	0.0	2008	DDH	Bonanza	Included
08CS-53	332826.8	1047465.4	874.8	660	-90.0	0.0	2008	DDH	Bonanza	Included
08CS-54	332875.0	1047425.8	876.8	690	-90.0	0.0	2008	DDH	Bonanza	Included
08CS-55	332937.9	1047465.3	876.6	715	-90.0	0.0	2008	DDH	Bonanza	Included
08CS-56	330791.8	1043461.3	852.6	1091	-90.0	0.0	2008	DDH	Bonanza	Excluded
08CS-57	330752.3	1043379.4	852.7	1000	-90.0	0.0	2008	DDH	Bonanza	Excluded
08CS-58	334323.2	1043995.4	890.9	900	-90.0	0.0	2008	DDH	Bonanza	Included
08CS-59	334205.9	1044199.8	895.0	850	-90.0	0.0	2008	DDH	Bonanza	Included
DZ12-1	332664.9	1047794.0	230.4	100	55.0	80.0	2012	Missing	DZ Holes	Excluded
DZ12-2	332664.9	1047794.0	230.4	100	55.0	80.0	2012	Missing	DZ Holes	Excluded
DZ12-3	332664.9	1047794.0	230.4	100	55.0	80.0	2012	Missing	DZ Holes	Excluded
DZ12-4	332664.9	1047794.0	230.4	100	55.0	80.0	2012	Missing	DZ Holes	Excluded
DZ12-5	332664.9	1047794.0	230.4	100	55.0	80.0	2012	Missing	DZ Holes	Excluded
DZ12-6	332664.9	1047794.0	230.4	100	55.0	80.0	2012	Missing	DZ Holes	Excluded
DZ-3	332689.4	1047345.9	304.0	100	45.0	120.0	2012	Missing	DZ Holes	Excluded
DZ-4	332690.4	1047354.2	305.9	100	45.0	67.0	2012	Missing	DZ Holes	Excluded
520-1	332625.0	1047500.0	360.0	125	60.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-10	332625.0	1047400.0	360.0	185	35.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-11	332625.0	1047400.0	360.0	225	15.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-12	332625.0	1047350.0	360.0	175	75.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-13	332625.0	1047350.0	360.0	165	55.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-14	332625.0	1047350.0	360.0	185	24.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-15	332625.0	1047350.0	360.0	225	0.0	0.0	2013	Percussion	Bonanza Underground	Excluded
520-16	332625.0	1047300.0	360.0	175	50.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-17	332625.0	1047300.0	360.0	185	20.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-2	332625.0	1047500.0	360.0	170	35.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-3	332625.0	1047500.0	360.0	185	10.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-4	332625.0	1047450.0	360.0	125	70.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-5	332625.0	1047450.0	360.0	150	50.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-6	332625.0	1047450.0	360.0	180	25.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-7	332625.0	1047450.0	360.0	200	5.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-8	332625.0	1047400.0	360.0	150	85.0	90.0	2013	Percussion	Bonanza Underground	Excluded
520-9	332625.0	1047400.0	360.0	150	65.0	90.0	2013	Percussion	Bonanza Underground	Excluded
650W-1	332764.0	1047583.4	318.4	64	16.0	264.0	2013	Percussion	Bonanza Underground	Excluded
650W-10	332777.1	1047496.6	318.7	64	64.0	277.0	2013	Percussion	Bonanza Underground	Excluded
650W-12	332772.7	1047468.6	308.3	56	15.0	278.0	2013	Percussion	Bonanza Underground	Excluded
650W-13	332775.1	1047467.9	313.9	52	59.0	267.0	2013	Percussion	Bonanza Underground	Excluded
650W-14	332786.8	1047469.8	314.1	100	53.0	104.0	2013	Percussion	Bonanza Underground	Excluded
650W-15	332772.3	1047443.4	303.0	100	8.0	274.0	2013	Percussion	Bonanza Underground	Excluded
650W-16	332774.4	1047443.5	308.1	100	63.0	261.0	2013	Percussion	Bonanza Underground	Excluded
650W-17	332783.8	1047442.0	306.9	100	57.0	95.0	2013	Percussion	Bonanza Underground	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
650W-2	332764.5	1047583.5	317.7	60	60.0	266.0	2013	Percussion	Bonanza Underground	Excluded
650W-3	332779.6	1047584.9	314.0	52	38.0	81.0	2013	Percussion	Bonanza Underground	Excluded
650W-4	332770.8	1047531.8	315.4	60	12.0	259.0	2013	Percussion	Bonanza Underground	Excluded
650W-5	332771.8	1047531.9	314.9	60	32.0	266.0	2013	Percussion	Bonanza Underground	Excluded
650W-6	332773.2	1047532.3	320.4	60	67.0	260.0	2013	Percussion	Bonanza Underground	Excluded
650W-8	332774.7	1047497.0	314.1	52	33.0	277.0	2013	Percussion	Bonanza Underground	Excluded
650W-9	332776.8	1047496.7	319.4	56	55.0	279.0	2013	Percussion	Bonanza Underground	Excluded
654-1	332815.8	1047424.0	297.4	100	20.0	145.0	2013	Percussion	Bonanza Underground	Excluded
654-2	332818.9	1047426.9	298.4	100	20.0	123.0	2013	Percussion	Bonanza Underground	Excluded
654-50	332817.5	1047681.6	303.0	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
654-51	332822.2	1047684.7	303.0	52	12.0	32.0	2013	Percussion	Bonanza Underground	Excluded
654-52	332834.3	1047656.1	302.0	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
654-53	332839.3	1047659.8	302.0	100	12.0	35.0	2013	Percussion	Bonanza Underground	Excluded
654-54	332854.7	1047616.7	302.0	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
654-55	332863.1	1047619.3	302.0	52	12.0	23.0	2013	Percussion	Bonanza Underground	Excluded
654-56	332835.3	1047554.3	302.0	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
654-57	332842.5	1047553.3	302.0	100	12.0	97.0	2013	Percussion	Bonanza Underground	Excluded
654-58	332836.7	1047522.9	302.0	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
654-59	332844.6	1047523.0	302.0	52	12.0	71.0	2013	Percussion	Bonanza Underground	Excluded
654-60	332836.8	1047493.9	302.0	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
654-61	332844.1	1047493.3	302.0	52	12.0	100.0	2013	Percussion	Bonanza Underground	Excluded
654cc-1	332788.6	1047481.4	314.1	60	15.0	95.0	2013	Percussion	Bonanza Underground	Excluded
654cc-2	332790.3	1047524.8	314.3	60	15.0	78.0	2013	Percussion	Bonanza Underground	Excluded
654cc-3	332789.8	1047530.8	315.9	100	15.0	51.0	2013	Percussion	Bonanza Underground	Excluded
690-1	332774.7	1047858.9	241.0	52	10.0	75.0	2013	Percussion	Bonanza Underground	Excluded
690-10	332760.8	1047772.4	250.5	52	10.0	33.0	2013	Percussion	Bonanza Underground	Excluded
690-11	332758.6	1047770.7	244.4	52	45.0	77.0	2013	Percussion	Bonanza Underground	Excluded
690-12	332759.9	1047770.1	246.7	52	80.0	79.0	2013	Percussion	Bonanza Underground	Excluded
690-13	332757.2	1047741.7	252.7	52	10.0	48.0	2013	Percussion	Bonanza Underground	Excluded
690-14	332756.0	1047741.5	248.5	52	45.0	63.0	2013	Percussion	Bonanza Underground	Excluded
690-15	332755.5	1047740.9	246.0	52	80.0	75.0	2013	Percussion	Bonanza Underground	Excluded
690-19	332719.4	1047882.0	237.3	84	11.0	29.0	2013	Percussion	Bonanza Underground	Excluded
690-2	332774.7	1047858.6	240.9	52	45.0	75.0	2013	Percussion	Bonanza Underground	Excluded
690-20	332717.1	1047861.9	240.3	88	11.0	115.0	2013	Percussion	Bonanza Underground	Excluded
690-21	332717.0	1047861.6	236.6	76	45.0	117.0	2013	Percussion	Bonanza Underground	Excluded
690-22	332717.3	1047874.8	241.8	88	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
690-23	332724.4	1047874.9	239.9	88	45.0	55.0	2013	Percussion	Bonanza Underground	Excluded
690-24	332725.6	1047874.9	236.4	64	10.0	60.0	2013	Percussion	Bonanza Underground	Excluded
690-25	332721.8	1047831.9	238.4	88	10.0	89.0	2013	Percussion	Bonanza Underground	Excluded
690-26	332721.7	1047830.9	238.2	88	45.0	80.0	2013	Percussion	Bonanza Underground	Excluded
690-28	332754.2	1047821.8	237.2	66	10.0	41.0	2013	Percussion	Bonanza Underground	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
690-29	332755.7	1047821.5	239.9	60	45.0	36.0	2013	Percussion	Bonanza Underground	Excluded
690-3	332774.7	1047859.7	240.2	44	80.0	75.0	2013	Percussion	Bonanza Underground	Excluded
690-31	332774.9	1047819.5	231.4	28	-90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
690-32	332767.5	1047759.6	231.4	28	-90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
690-33	332721.0	1047650.0	260.0	100	55.0	103.0	2013	Percussion	Bonanza Underground	Excluded
690-34	332721.0	1047650.0	260.0	100	86.0	170.0	2013	Percussion	Bonanza Underground	Excluded
690-35	332721.0	1047650.0	260.0	85	55.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-36	332721.0	1047650.0	260.0	70	85.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-38	332630.0	1047650.0	260.0	125	60.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-39	332726.0	1047700.0	260.0	76	50.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-40	332726.0	1047700.0	260.0	72	90.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-41	332714.0	1047700.0	260.0	90	90.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-42	332714.0	1047700.0	260.0	125	45.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-43	332757.0	1047750.0	260.0	60	70.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-44	332757.0	1047750.0	260.0	100	35.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-45	332557.0	1047750.0	255.0	120	37.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-46	332557.0	1047750.0	255.0	100	60.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-47	332763.0	1047800.0	250.0	80	65.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-48	332654.0	1047800.0	250.0	110	65.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-49	332654.0	1047800.0	250.0	120	90.0	90.0	2013	Percussion	Bonanza Underground	Excluded
690-50	332724.0	1047850.0	245.0	125	40.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-51	332724.0	1047850.0	245.0	100	60.0	270.0	2013	Percussion	Bonanza Underground	Excluded
690-7	332765.3	1047799.9	243.6	52	10.0	73.0	2013	Percussion	Bonanza Underground	Excluded
690-8	332765.1	1047799.6	240.4	52	45.0	80.0	2013	Percussion	Bonanza Underground	Excluded
726-1	332819.4	1047934.0	227.0	52	12.0	86.0	2013	Percussion	Bonanza Underground	Excluded
726-10	332850.4	1047850.0	233.3	100	12.0	47.0	2013	Percussion	Bonanza Underground	Excluded
726-11	332845.2	1047844.3	235.9	100	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
726-13	332876.7	1047825.4	233.2	52	12.0	44.0	2013	Percussion	Bonanza Underground	Excluded
726-14	332870.9	1047820.0	233.6	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
726-2	332813.0	1047933.8	236.4	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
726-3	332805.0	1047933.3	229.2	100	70.0	272.0	2013	Percussion	Bonanza Underground	Excluded
726-4	332817.3	1047899.5	233.2	52	12.0	90.0	2013	Percussion	Bonanza Underground	Excluded
726-5	332810.1	1047899.5	236.1	52	90.0	0.0	2013	Percussion	Bonanza Underground	Excluded
726-6	332800.5	1047900.1	233.8	52	80.0	279.0	2013	Percussion	Bonanza Underground	Excluded
726-7	332824.3	1047872.4	234.0	52	12.0	45.0	2013	Percussion	Bonanza Underground	Excluded
730-11	332952.5	1047637.8	243.0	52	86.0	180.0	2013	Percussion	Bonanza Underground	Excluded
730-13	332947.7	1047674.1	230.8	120	47.0	93.0	2013	Percussion	Bonanza Underground	Excluded
730-14	332933.8	1047674.3	231.0	132	64.0	259.0	2013	Percussion	Bonanza Underground	Excluded
730-15	332842.6	1047833.7	246.3	60	79.0	101.0	2013	Percussion	Bonanza Underground	Excluded
730-16	332862.6	1047813.0	236.6	20	88.0	137.0	2013	Percussion	Bonanza Underground	Excluded
730-17	332896.7	1047767.5	231.8	80	81.0	35.0	2013	Percussion	Bonanza Underground	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
730-18	332903.5	1047761.8	230.9	100	54.0	116.0	2013	Percussion	Bonanza Underground	Excluded
730-19	332842.7	1047833.7	245.0	28	7.0	90.0	2013	Percussion	Bonanza Underground	Excluded
730-2	332936.0	1047579.0	248.9	100	50.0	137.0	2013	Percussion	Bonanza Underground	Excluded
730-20	332896.9	1047762.0	222.6	56	-89.0	180.0	2013	Percussion	Bonanza Underground	Excluded
730-21	332920.4	1047723.9	230.8	120	88.0	302.0	2013	Percussion	Bonanza Underground	Excluded
730-22	332927.9	1047723.5	227.7	152	32.0	94.0	2013	Percussion	Bonanza Underground	Excluded
730-23	332908.7	1047732.7	221.6	76	-75.0	163.0	2013	Percussion	Bonanza Underground	Excluded
730-3	332934.0	1047580.9	239.5	104	-25.0	151.0	2013	Percussion	Bonanza Underground	Excluded
730-4	332932.0	1047587.5	237.9	64	-82.0	45.0	2013	Percussion	Bonanza Underground	Excluded
730-5	332938.5	1047594.0	251.1	100	87.0	3.0	2013	Percussion	Bonanza Underground	Excluded
730-7	332949.8	1047613.1	245.9	52	89.0	220.0	2013	Percussion	Bonanza Underground	Excluded
730-8	332947.0	1047613.5	244.8	132	66.0	275.0	2013	Percussion	Bonanza Underground	Excluded
730-9	332944.1	1047617.6	240.9	132	12.0	264.0	2013	Percussion	Bonanza Underground	Excluded
750-1	333070.1	1047144.8	194.0	52	12.0	237.0	2013	Percussion	Bonanza Underground	Excluded
750-10	333236.8	1047015.9	194.0	96	12.0	206.0	2013	Percussion	Bonanza Underground	Excluded
750-11	333239.9	1047025.2	202.0	60	80.0	159.0	2013	Percussion	Bonanza Underground	Excluded
750-12	333244.1	1047033.4	194.0	52	12.0	23.0	2013	Percussion	Bonanza Underground	Excluded
750-13	333138.3	1047074.2	194.0	52	12.0	43.0	2013	Percussion	Bonanza Underground	Excluded
750-14	333173.6	1047044.2	194.0	52	43.0	12.0	2013	Percussion	Bonanza Underground	Excluded
750-17	333248.1	1047003.6	194.0	100	12.0	165.0	2013	Percussion	Bonanza Underground	Excluded
750-18	333253.8	1047015.9	194.0	100	12.0	96.0	2013	Percussion	Bonanza Underground	Excluded
750-2	333076.7	1047147.7	202.0	48	80.0	344.0	2013	Percussion	Bonanza Underground	Excluded
750-20	333214.5	1047026.6	194.0	40	12.0	203.0	2013	Percussion	Bonanza Underground	Excluded
750-3	333083.4	1047150.3	194.0	52	12.0	57.0	2013	Percussion	Bonanza Underground	Excluded
750-4	333145.0	1047044.0	194.0	100	12.0	223.0	2013	Percussion	Bonanza Underground	Excluded
750-5	333151.2	1047049.5	202.0	56	80.0	136.0	2013	Percussion	Bonanza Underground	Excluded
750-6	333155.2	1047054.9	194.0	52	12.0	43.0	2013	Percussion	Bonanza Underground	Excluded
750-7	333186.9	1047022.3	194.0	100	12.0	215.0	2013	Percussion	Bonanza Underground	Excluded
750-8	333194.1	1047034.1	202.0	60	80.0	82.0	2013	Percussion	Bonanza Underground	Excluded
750-9	333201.1	1047044.7	194.0	52	12.0	30.0	2013	Percussion	Bonanza Underground	Excluded
810-10	333107.6	1047310.7	156.2	100	51.0	294.0	2013	Percussion	Bonanza Underground	Excluded
810-12	333107.9	1047309.8	155.9	104	19.0	343.0	2013	Percussion	Bonanza Underground	Excluded
810-13	333030.2	1047281.3	162.9	104	18.0	323.0	2013	Percussion	Bonanza Underground	Excluded
810-14	333030.3	1047281.2	167.0	100	47.0	328.0	2013	Percussion	Bonanza Underground	Excluded
810-15	332995.0	1047251.3	173.7	100	30.0	163.0	2013	Percussion	Bonanza Underground	Excluded
810-16	332995.0	1047251.3	173.7	100	60.0	163.0	2013	Percussion	Bonanza Underground	Excluded
810-16a	333127.9	1047410.6	160.2	100	59.0	259.0	2013	Percussion	Bonanza Underground	Excluded
810-17	333140.5	1047411.8	156.1	108	9.0	108.0	2013	Percussion	Bonanza Underground	Excluded
810-18	333127.9	1047364.0	159.5	100	58.0	290.0	2013	Percussion	Bonanza Underground	Excluded
810-19	333140.1	1047360.9	154.6	100	9.0	70.0	2013	Percussion	Bonanza Underground	Excluded
810-2	333079.5	1047344.3	160.0	100	10.0	52.0	2013	Percussion	Bonanza Underground	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
810-20	333136.7	1047400.0	150.6	56	-90.0	360.0	2013	Percussion	Bonanza Underground	Excluded
810-21	333136.7	1047374.7	150.5	56	-90.0	360.0	2013	Percussion	Bonanza Underground	Excluded
810-3	333077.5	1047337.9	162.0	100	45.0	52.0	2013	Percussion	Bonanza Underground	Excluded
810-4	333075.7	1047337.0	162.0	100	45.0	80.0	2013	Percussion	Bonanza Underground	Excluded
810-8	333064.0	1047343.0	162.0	100	45.0	313.0	2013	Percussion	Bonanza Underground	Excluded
810-9	333065.0	1047347.0	160.0	100	10.0	313.0	2013	Percussion	Bonanza Underground	Excluded
KER-15-01	333738.0	1045390.0	544.0	752	-90.0	0.0	2015	DDH	Kerr	Included
KER-15-02	333738.0	1045390.0	544.0	691	-65.0	215.0	2015	DDH	Kerr	Included
KER-15-03	333738.0	1045390.0	544.0	1021.5	-45.0	220.0	2015	DDH	Kerr	Included
KER-15-04	333539.2	1045699.0	505.0	581	-90.0	0.0	2015	DDH	Kerr	Included
KER-17S-01	334849.0	1045287.6	797.1	709	-90.0	0.0	2017	DDH	Kerr	Included
KER-17S-02	333750.0	1045254.5	557.8	665	-65.0	240.0	2017	DDH	Kerr	Included
KER-17S-03	333747.0	1045253.0	558.4	665	-45.0	240.0	2017	DDH	Kerr	Included
KER-17S-04	333852.2	1045044.9	583.4	775	-65.0	240.0	2017	DDH	Kerr	Included
KER-17S-06	333513.2	1045341.5	652.3	860	-89.6	243.9	2017	RC	Kerr	Included
KER-17S-07	333535.2	1045333.2	650.5	831	-62.0	240.0	2017	DDH	Kerr	Included
KER-17S-08	333462.1	1045427.1	663.5	240	-65.0	240.0	2017	DDH	Kerr	Included
KER-17S-08B	333461.7	1045426.3	663.6	262	-55.0	240.0	2017	DDH	Kerr	Included
KER-17S-09	333427.0	1045480.2	669.9	970	-79.4	236.4	2017	RC	Kerr	Included
KER-17S-10	333365.9	1045567.6	681.9	855	-86.0	240.0	2017	DDH	Kerr	Included
KER-17S-11	333363.3	1045566.7	682.0	949	-60.0	240.0	2017	DDH	Kerr	Included
KER-17S-12	333218.7	1045731.0	702.4	1040	-89.7	236.0	2017	DDH	Kerr	Included
KER-17S-13	333112.7	1045888.0	720.3	980	-89.9	193.2	2017	RC	Kerr	Included
KER-17S-14	334835.9	1045316.4	794.0	660	-89.7	265.5	2017	RC	Kerr	Included
KER-17S-15	334836.3	1045318.8	794.0	800	-67.2	11.2	2017	RC	Kerr	Included
KER-17S-16	333740.7	1045241.9	558.1	760	-89.4	346.1	2017	RC	Kerr	Included
KER-17S-17	333229.8	1045726.2	702.0	842	-53.2	239.1	2017	DDH	Kerr	Included
KER-17S-18	333112.5	1045892.0	720.9	428	-60.0	240.0	2017	DDH	Kerr	Included
KER-17S-19	333816.7	1045055.5	580.7	820	-89.2	114.1	2017	RC	Kerr	Included
KER-17S-20	333440.1	1045459.2	667.1	330	-88.8	331.0	2017	RC	Kerr	Included
KER-17S-21	333894.6	1044783.3	612.7	620	-64.3	235.7	2017	RC	Kerr	Included
KER-17S-22	334376.4	1045442.8	693.0	280	-89.4	328.3	2017	RC	Kerr	Included
KER-17S-23	334198.3	1044201.7	887.8	280	-51.4	258.0	2017	RC	Kerr	Included
KER-17U-01	333091.2	1047769.2	220.1	130	-35.0	240.0	2017	DDH	Kerr	Included
KER-17U-02	333093.5	1047771.3	220.2	175	-80.0	240.0	2017	DDH	Kerr	Included
KER-17U-03	333105.5	1047776.1	220.4	176	-65.0	60.0	2017	DDH	Kerr	Included
KER-17U-04	333096.0	1047780.5	220.8	200	-50.0	358.0	2017	DDH	Kerr	Included
KER-17U-05	333093.1	1047780.9	221.4	218	-35.0	330.0	2017	DDH	Kerr	Included
KER-17U-06	333079.7	1047776.2	221.2	220	-20.0	277.0	2017	DDH	Kerr	Included
KER-17U-07	332746.0	1047184.2	293.5	206	-80.0	240.0	2017	DDH	Kerr	Included
KER-17U-08	332751.9	1047185.1	305.5	151	88.0	240.0	2017	DDH	Kerr	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
KER-17U-09	332752.2	1047184.8	305.7	151	45.0	330.0	2017	DDH	Kerr	Included
KER-17U-10	332789.8	1047368.7	300.0	238	-11.0	41.0	2017	DDH	Kerr	Included
KER-17U-11	332789.8	1047363.5	296.0	220.5	-15.0	65.0	2017	DDH	Kerr	Included
KER-17U-12	332788.0	1047366.0	300.0	284	-11.0	44.0	2017	DDH	Kerr	Included
KER-17U-13	332788.0	1047366.0	300.0	170	-15.0	72.0	2017	DDH	Kerr	Included
KER-17U-14	332788.0	1047366.0	300.0	243	-12.0	48.0	2017	DDH	Kerr	Included
KER-17U-15	332772.0	1047362.7	292.7	154	-42.0	264.0	2017	DDH	Kerr	Included
KER-17U-16	332773.3	1047364.6	302.6	178	35.0	284.0	2017	DDH	Kerr	Included
KER-17U-17	332790.2	1047367.2	295.3	220	-17.0	45.0	2017	DDH	Kerr	Included
KER-17U-18	332605.6	1047237.5	343.9	16.5	-90.0	0.0	2017	DDH	Kerr	Included
KER-17U-18B	332604.1	1047243.9	342.0	193.5	25.0	105.0	2017	DDH	Kerr	Included
KER-17U-19	332589.4	1047232.9	356.0	150	60.0	62.0	2017	DDH	Kerr	Included
KER-17U-20	332589.9	1047232.9	341.9	302	-40.0	36.0	2017	DDH	Kerr	Included
KER-17U-21B	332590.2	1047238.7	341.9	134	-70.0	291.0	2017	DDH	Kerr	Included
KER-17U-22B	332588.7	1047239.2	357.8	175	35.0	288.0	2017	DDH	Kerr	Included
KER-17U-23	332588.7	1047239.2	357.8	200	60.0	320.0	2017	DDH	Kerr	Excluded
KER-17U-24	332590.2	1047238.7	341.9	19	-70.0	320.0	2017	DDH	Kerr	Included
KER-17U-25	332625.9	1047636.1	265.1	227	61.0	177.0	2017	DDH	Kerr	Included
KER-17U-26	332627.3	1047641.2	267.9	152	78.0	113.0	2017	DDH	Kerr	Included
KER-17U-27	332627.3	1047641.2	267.9	202	50.0	107.0	2017	DDH	Kerr	Included
KER-17U-28	332627.3	1047641.2	267.9	152	62.0	93.0	2017	DDH	Kerr	Included
KER-17U-29	332812.0	1046826.3	318.2	202	40.8	58.6	2017	DDH	Kerr	Included
KER-17U-30	332811.3	1046826.7	315.4	252	25.3	57.2	2017	DDH	Kerr	Excluded
KER-17U-34	332812.0	1046826.3	318.2	204	74.1	57.7	2017	DDH	Kerr	Included
KER-17U-50	333079.5	1047756.0	219.8	105	-19.0	222.5	2017	DDH	Kerr	Included
KER-17U-51	333084.7	1047762.2	219.6	110	-32.7	222.5	2017	DDH	Kerr	Included
KER-17U-52	333087.5	1047763.0	220.0	100	-40.7	215.5	2017	DDH	Kerr	Included
KER-17U-53	333090.7	1047764.3	219.8	98	-52.1	200.0	2017	DDH	Kerr	Included
KER-17U-54	333100.1	1047764.1	220.9	280.5	-52.7	136.0	2017	DDH	Kerr	Included
KER-17U-55	333087.9	1047779.2	225.8	270	-3.8	311.0	2017	DDH	Kerr	Included
KER-17U-56	333086.9	1047779.2	224.9	151	-9.2	305.0	2017	DDH	Kerr	Included
KER-17U-57	333081.5	1047776.8	222.7	160	-17.3	288.0	2017	DDH	Kerr	Included
KER-17U-58	333082.8	1047774.4	220.2	251	-35.3	274.0	2017	DDH	Kerr	Included
KER-17U-59	333130.9	1047621.9	207.0	182.5	-11.7	223.0	2017	DDH	Kerr	Included
KER-17U-60	333132.6	1047619.3	200.5	220	-53.2	202.0	2017	DDH	Kerr	Included
KER-17U-61	333136.8	1047620.0	199.3	187	-53.0	202.0	2017	DDH	Kerr	Included
KER-17U-62	333147.6	1047628.6	202.8	190	-56.2	88.0	2017	DDH	Kerr	Included
KER-17U-63	333141.8	1047637.8	203.1	178	-54.7	25.0	2017	DDH	Kerr	Included
KER-17U-64	333139.1	1047637.4	202.4	171	-65.4	330.0	2017	DDH	Kerr	Included
KER-17U-65	333128.3	1047630.0	203.4	150	-50.2	282.0	2017	DDH	Kerr	Included
KER-17U-66	333128.7	1047627.7	206.9	180	-15.3	260.0	2017	DDH	Kerr	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
KER-17U-67	333157.9	1047528.2	194.9	201	-9.4	225.0	2017	DDH	Kerr	Included
KER-17U-68	333158.1	1047528.3	190.5	169	-55.0	205.0	2017	DDH	Kerr	Included
KER-17U-69	333176.3	1047523.0	194.7	279	-5.1	130.0	2017	DDH	Kerr	Excluded
KER-17U-70	333176.6	1047521.9	188.5	180	-45.6	134.0	2017	DDH	Kerr	Included
KER-17U-71	333167.8	1047524.0	188.5	151	-70.0	147.0	2017	DDH	Kerr	Included
KER-17U-72	333173.5	1047529.7	190.3	242	-60.4	88.0	2017	DDH	Kerr	Included
KER-17U-73	333171.4	1047535.2	191.7	224	-56.3	61.7	2017	DDH	Kerr	Included
KER-17U-74	333166.3	1047528.1	189.5	162	-89.1	0.0	2017	DDH	Kerr	Included
KER-17U-75	333164.5	1047520.1	187.9	172	-61.0	241.7	2017	DDH	Kerr	Included
KER-17U-76	333158.6	1047525.2	194.5	200	-10.5	240.0	2017	DDH	Kerr	Included
KER-17U-77	333164.3	1047523.2	188.7	240	-53.0	168.0	2017	DDH	Kerr	Included
18-01-01	332797.0	1046858.0	316.0	280	64.8	62.0	2019	RC	Kerr	Included
18-01-04	332797.0	1046858.0	316.0	320	48.0	82.0	2019	RC	Kerr	Included
18-01A-02	332811.0	1046818.0	318.0	170	59.2	86.1	2019	RC	Kerr	Included
18-01A-03	332811.0	1046818.0	318.0	310	41.9	87.9	2019	RC	Kerr	Included
18-01A-04	332796.1	1046821.6	320.0	220	71.5	241.9	2019	RC	Kerr	Included
18-03-05	332620.0	1047112.4	325.3	60	-7.0	202.0	2019	RC	Kerr	Included
18-03-07	332614.0	1047120.5	321.6	60	-46.3	48.2	2019	RC	Kerr	Included
18-03-08	332612.5	1047122.5	321.8	50	-45.5	0.6	2019	RC	Kerr	Included
18-04-01	332932.5	1047671.6	224.0	130	-6.5	229.1	2019	RC	Kerr	Included
18-04-02	332932.6	1047671.5	223.2	140	-12.4	204.5	2019	RC	Kerr	Included
18-04-03	332932.9	1047672.0	222.6	80	-36.1	231.4	2019	RC	Kerr	Included
18-04A-01	332924.0	1047702.0	226.0	75	-24.8	319.0	2019	RC	Kerr	Included
18-04A-02	332934.0	1047688.0	221.0	110	-59.9	135.0	2019	RC	Kerr	Included
18-04A-03	332934.0	1047688.0	221.0	100	-32.5	168.4	2019	RC	Kerr	Included
18-04A-04	332924.0	1047702.0	226.0	110	-79.9	341.6	2019	RC	Kerr	Included
18-05-01	333333.3	1046723.6	366.8	280	19.0	54.0	2019	RC	Kerr	Included
18-05-02	333333.0	1046719.0	370.0	200	33.0	91.0	2019	RC	Kerr	Included
18-05-03	333333.0	1046719.0	368.0	320	32.0	75.0	2019	RC	Kerr	Included
18-05-04	333333.0	1046719.0	373.0	200	56.3	40.3	2019	RC	Kerr	Included
18-05-05	333327.0	1046715.0	361.0	125	-80.2	317.8	2019	RC	Kerr	Included
18-05-06	333320.0	1046714.9	360.9	215	-15.9	216.3	2019	RC	Kerr	Included
18-05-07	333318.4	1046717.3	361.7	180	-16.1	241.4	2019	RC	Kerr	Included
18-05-08	333322.0	1046710.0	361.0	200	-74.6	79.5	2019	RC	Kerr	Included
18-05A-01	333311.1	1046727.5	361.9	210	0.0	245.0	2019	RC	Kerr	Included
18-05A-04	333316.0	1046730.0	358.0	150	-42.4	227.2	2019	RC	Kerr	Included
18-05A-06	333311.3	1046725.9	362.0	230	-0.7	235.0	2019	RC	Kerr	Included
18-05E-01	333248.3	1046826.2	341.0	155	-65.3	241.5	2019	RC	Kerr	Included
18-05E-07	333248.0	1046825.7	341.0	290	-45.2	244.3	2019	RC	Kerr	Included
18-07-01	333214.0	1046931.0	335.0	160	-89.0	187.4	2019	RC	Kerr	Included
18-07-02	333214.0	1046931.0	335.0	160	-80.0	45.0	2019	RC	Kerr	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
18-07-03	333214.0	1046931.0	335.0	150	-70.4	277.7	2019	RC	Kerr	Included
18-08-01	333129.0	1046988.0	326.0	100	-88.5	44.2	2019	RC	Kerr	Included
18-08-02	333129.0	1046988.0	326.0	155	-64.5	126.6	2019	RC	Kerr	Included
18-08A-01	333090.0	1047017.0	323.0	145	-85.2	353.8	2019	RC	Kerr	Included
18-08A-02	333086.6	1047015.4	324.5	130	-20.5	244.2	2019	RC	Kerr	Included
18-08A-03	333086.9	1047015.6	323.0	120	-39.2	244.9	2019	RC	Kerr	Included
18-08B-01	333051.0	1047068.0	320.0	125	-33.8	224.0	2019	RC	Kerr	Included
18-08B-02	333057.0	1047071.0	317.0	125	-88.6	8.4	2019	RC	Kerr	Included
18-08B-03	333062.0	1047075.0	317.0	245	-59.2	40.8	2019	RC	Kerr	Included
18-08B-04	333051.0	1047068.0	326.0	120	49.9	212.1	2019	RC	Kerr	Included
18-09-01	332910.0	1047233.0	294.0	120	0.0	0.0	2019	RC	Kerr	Excluded
18-09-01B	332910.0	1047233.0	294.0	120	0.0	0.0	2019	RC	Kerr	Excluded
18-09-02	332902.0	1047240.0	294.0	135	-49.6	250.8	2019	RC	Kerr	Included
18-09-03	332902.0	1047240.0	294.0	170	-30.3	252.0	2019	RC	Kerr	Included
18-10-03	332599.0	1047294.0	341.6	75	-63.0	276.1	2019	RC	Kerr	Included
18-10-05	332599.0	1047294.0	346.0	15	27.0	87.0	2019	RC	Kerr	Included
18-12A-01	332782.0	1047232.0	294.0	70	-84.7	281.3	2019	RC	Kerr	Included
18-12A-02	332773.9	1047233.5	295.1	150	-20.9	280.1	2019	RC	Kerr	Included
18-17A-01	332952.0	1047249.0	190.0	80	76.0	359.7	2019	RC	Kerr	Included
18-17A-04	332952.0	1047248.8	190.0	270	34.4	301.5	2019	RC	Kerr	Included
18-17B-01	332997.0	1047252.0	176.0	100	34.8	120.4	2019	RC	Kerr	Included
18-17B-02	332994.0	1047266.0	174.0	150	28.4	349.8	2019	RC	Kerr	Included
18-17B-03	332994.0	1047266.0	174.0	130	42.5	324.9	2019	RC	Kerr	Included
18-17B-04	332994.0	1047266.0	174.0	130	49.0	299.9	2019	RC	Kerr	Included
18-18-01	333227.0	1047337.0	152.0	80	-65.1	272.3	2019	RC	Kerr	Included
18-18-02	333227.0	1047337.0	152.0	60	-64.1	91.8	2019	RC	Kerr	Included
18-18A-01	333264.0	1047336.0	148.0	155	-26.9	166.8	2019	RC	Kerr	Included
18-18A-03	333264.0	1047336.0	148.0	35	-40.0	65.0	2019	RC	Kerr	Included
18-20-01	333169.9	1047565.0	196.0	225	-33.1	108.1	2019	RC	Kerr	Included
18-20-02	333148.9	1047555.8	196.0	175	-31.7	217.9	2019	RC	Kerr	Included
18-20-03	333154.9	1047561.0	193.6	145	-75.3	213.5	2019	RC	Kerr	Included
18-20-05	333147.4	1047555.5	198.2	200	3.9	240.7	2019	RC	Kerr	Included
18-20-06	333170.6	1047563.1	196.7	240	-31.5	90.1	2019	RC	Kerr	Included
18-20-08	333168.6	1047569.6	197.0	210	-33.9	71.5	2019	RC	Kerr	Included
18-20-09	333146.7	1047559.6	197.0	265	6.0	262.0	2019	RC	Kerr	Included
18-20-10	333148.2	1047557.9	195.7	175	-26.0	251.1	2019	RC	Kerr	Included
18-20-11	333148.7	1047553.3	195.8	280	-32.5	207.8	2019	RC	Kerr	Included
18-20-12	333161.9	1047564.4	193.2	250	-73.0	158.7	2019	RC	Kerr	Included
18-20-13	333166.2	1047565.0	193.2	245	-47.1	126.6	2019	RC	Kerr	Included
18-20-14	333152.2	1047559.2	196.1	235	-29.4	190.0	2019	RC	Kerr	Included
18-21-02	333083.4	1047777.4	222.7	190	-15.5	312.6	2019	RC	Kerr	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
18-21-03	333083.9	1047776.9	220.1	170	-43.6	303.9	2019	RC	Kerr	Included
18-21-04	333084.7	1047776.1	220.1	310	-32.2	6.3	2019	RC	Kerr	Included
18-21-05	333084.0	1047776.8	223.1	290	-13.0	321.0	2019	RC	Kerr	Included
18-21-06	333083.8	1047777.0	220.1	270	-28.0	350.1	2019	RC	Kerr	Included
18-21-07	333084.3	1047776.5	222.5	280	-17.0	325.2	2019	RC	Kerr	Included
18-21-08	333084.6	1047776.3	220.1	85	-34.6	4.1	2019	RC	Kerr	Included
18-21-08A	333084.6	1047776.3	220.1	320	-34.7	12.7	2019	RC	Kerr	Included
18-21-09	333090.4	1047770.4	220.3	55	-80.0	100.0	2019	RC	Kerr	Included
18-21-09A	333090.4	1047770.4	220.3	150	-80.7	101.2	2019	RC	Kerr	Included
18-21-10	333083.9	1047777.0	221.5	250	-20.3	341.8	2019	RC	Kerr	Included
18-21-11	333082.8	1047776.0	222.5	265	-13.2	296.8	2019	RC	Kerr	Included
18-21-13	333084.4	1047776.0	224.2	390	-3.5	345.2	2019	RC	Kerr	Included
18-21A-05	333102.1	1047686.0	215.0	160	-89.1	331.9	2019	RC	Kerr	Included
18-34-01	333009.0	1047123.0	307.0	100	-36.0	228.0	2019	RC	Kerr	Included
18-34-02	333016.0	1047129.0	305.0	110	-77.5	41.4	2019	RC	Kerr	Included
18-34-03	333016.0	1047129.0	305.0	135	-62.8	65.0	2019	RC	Kerr	Included
18-34-04	333016.0	1047129.0	305.0	200	-51.6	51.3	2019	RC	Kerr	Included
18-34-05	333009.0	1047123.0	307.0	75	-72.9	230.4	2019	RC	Kerr	Included
18-36-01	332730.4	1047647.3	259.2	120	56.3	1.4	2019	RC	Kerr	Included
18-36-02	332713.0	1047627.0	265.0	140	51.5	163.0	2019	RC	Kerr	Included
18-36-03	332713.0	1047627.0	265.0	140	47.5	185.8	2019	RC	Kerr	Included
18-36-04	332694.8	1047638.0	265.0	130	29.6	334.5	2019	RC	Kerr	Included
18-36-05	332693.8	1047638.1	265.0	160	55.9	315.2	2019	RC	Kerr	Included
18-36-06	332728.8	1047646.1	260.7	160	39.3	12.8	2019	RC	Kerr	Included
18-38-02	332570.0	1047700.0	254.0	180	29.7	27.7	2019	RC	Kerr	Included
18-38-03	332570.0	1047700.0	254.0	170	75.3	53.0	2019	RC	Kerr	Included
18-38A-01	332551.0	1047688.0	257.0	180	69.8	107.7	2019	RC	Kerr	Included
18-38A-02	332551.0	1047688.0	257.0	220	69.2	172.0	2019	RC	Kerr	Included
18-38A-03	332551.0	1047688.0	257.0	220	70.8	350.5	2019	RC	Kerr	Included
AZG-20-21-13	333105.3	1047784.5	232.8	350	40.0	30.8	2020	DDH	Arizona Gold Corp.	Included
AZG-20-21-15	333102.7	1047784.1	230.0	303	30.3	3.0	2020	DDH	Arizona Gold Corp.	Included
AZG-20-21-16	333094.7	1047781.3	222.2	410	-35.9	14.8	2020	DDH	Arizona Gold Corp.	Included
AZG-20S-01	333905.8	1046851.0	879.7	1295	-78.4	218.6	2020	RC	Arizona Gold Corp.	Included
AZG-20S-02	334847.4	1046034.9	893.8	1200	-65.0	265.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-03	332715.3	1045699.5	873.6	750	-89.6	302.3	2020	RC	Arizona Gold Corp.	Included
AZG-20S-04	333225.9	1045328.9	782.7	850	-71.2	235.4	2020	RC	Arizona Gold Corp.	Included
AZG-20S-05	333889.5	1044787.0	612.1	605	-49.9	220.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-06	333878.4	1044784.8	612.1	625	-49.8	255.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-07	334859.8	1045246.5	799.4	765	-72.3	63.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-08	333441.0	1045865.0	483.0	700	-63.0	268.1	2020	RC	Arizona Gold Corp.	Included
AZG-20S-09	333437.3	1045911.8	481.0	800	-60.5	177.9	2020	RC	Arizona Gold Corp.	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
AZG-20S-10	333000.6	1045246.2	888.6	745	-55.2	240.4	2020	RC	Arizona Gold Corp.	Included
AZG-20S-11	333030.4	1045876.0	725.0	750	-71.8	184.5	2020	RC	Arizona Gold Corp.	Included
AZG-20S-12	333030.1	1045933.9	723.8	820	-90.0	0.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-13	333026.3	1045933.2	724.0	725	-70.0	259.2	2020	RC	Arizona Gold Corp.	Included
AZG-20S-14	332970.3	1045368.2	881.1	850	-79.3	241.9	2020	RC	Arizona Gold Corp.	Included
AZG-20S-15	334096.4	1046221.3	877.1	750	-55.4	237.6	2020	RC	Arizona Gold Corp.	Included
AZG-20S-16	334061.3	1046214.7	877.5	1000	-70.0	290.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-17	332872.3	1045542.0	879.3	700	-89.8	221.4	2020	RC	Arizona Gold Corp.	Included
AZG-20S-18	333507.5	1046990.0	874.7	950	-80.0	235.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-21	334199.7	1046164.8	878.4	700	-89.6	131.0	2020	RC	Arizona Gold Corp.	Included
AZG-20S-22	334775.3	1045111.3	808.4	480	-55.5	281.4	2020	RC	Arizona Gold Corp.	Included
AZG-20S-23	334424.0	1045815.8	739.0	565	-65.6	301.0	2021	RC	Arizona Gold Corp.	Included
AZG-21-30-30	332971.7	1046707.8	255.7	250	65.2	155.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-33-31	332902.0	1046964.1	258.9	250	44.7	74.8	2021	DDH	Arizona Gold Corp.	Included
AZG-20-21-01	333096.3	1047782.2	222.3	460	-40.8	29.9	2021	DDH	Arizona Gold Corp.	Included
AZG-20-21-02	333103.2	1047783.6	224.5	370	-25.3	350.0	2021	DDH	Arizona Gold Corp.	Included
AZG-20-21-03	333093.4	1047781.9	221.8	375	-22.6	340.0	2021	DDH	Arizona Gold Corp.	Included
AZG-20-21-14	333087.4	1047779.3	224.2	395	-30.6	3.2	2021	DDH	Arizona Gold Corp.	Included
AZG-21-01-20	332812.3	1046826.4	315.8	223	76.2	282.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-01-21	332812.2	1046826.7	318.0	224	65.4	320.2	2021	DDH	Arizona Gold Corp.	Included
AZG-21-02-27	333050.3	1046169.0	414.9	600	-70.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-10-06	332592.9	1047329.6	347.2	281	-0.3	269.8	2021	DDH	Arizona Gold Corp.	Included
AZG-21-10-09	332609.2	1047331.5	354.9	203	26.9	87.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-10-17	332594.7	1047329.6	355.4	196.5	69.3	315.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-12-28	332682.7	1047803.0	223.6	250	2.2	32.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-12-29	332658.9	1047800.0	224.8	150	30.9	40.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-04	332664.5	1047408.7	288.4	90	-89.3	180.2	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-05	332662.8	1047405.3	290.2	146	-0.8	280.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-07	332674.5	1047409.1	301.3	225	50.7	253.9	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-08	332669.4	1047402.0	293.3	80	-35.5	226.3	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-10	332698.4	1047409.4	288.2	142.5	-64.6	80.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-11	332699.1	1047412.2	286.4	129	-39.7	136.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13-12	332694.3	1047410.2	301.3	124	53.7	60.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13A-18	332682.8	1047479.8	288.3	224	73.3	268.6	2021	DDH	Arizona Gold Corp.	Included
AZG-21-13A-19	332681.9	1047478.2	288.4	389	47.5	254.9	2021	DDH	Arizona Gold Corp.	Included
AZG-21-21-22	333101.6	1047783.5	224.2	310	-33.4	349.8	2021	DDH	Arizona Gold Corp.	Included
AZG-21-21-23	333102.4	1047783.6	224.1	433	-36.2	358.6	2021	DDH	Arizona Gold Corp.	Included
AZG-21-21-24	333103.7	1047783.1	223.7	570	-33.1	1.1	2021	DDH	Arizona Gold Corp.	Included
AZG-21-37-25	332930.0	1046511.0	370.1	194	44.8	64.6	2021	DDH	Arizona Gold Corp.	Included
AZG-21-37-26	332913.9	1046510.8	357.4	209	-61.4	224.3	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2B-01C	333112.5	1047659.6	210.2	112	-13.0	240.0	2021	DDH	Arizona Gold Corp.	Included

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
AZG-21-P2B-02C	333112.0	1047659.2	212.5	79	-43.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2C-03C	333082.9	1047709.1	222.0	94	-28.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2C-04C	333082.9	1047709.1	222.0	80	-90.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2D-05C	333105.2	1047782.5	223.0	95	-85.0	320.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2F-06C	333035.7	1047758.1	219.8	65	-90.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2G-07C	333018.1	1047743.2	219.4	40	-80.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P3A-01C	333309.2	1046732.9	356.9	80	-74.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P3A-02C	333322.2	1046737.9	357.5	103	-84.0	60.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P3B-03C	333283.8	1046770.7	348.8	75	-79.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P3B-04C	333294.2	1046777.2	350.0	110	-75.0	60.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P3C-05C	333258.4	1046804.8	343.4	55	-80.0	240.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P3C-06C	333268.5	1046816.4	342.2	105	-73.0	60.0	2021	DDH	Arizona Gold Corp.	Included
AZG-21-P2A-01	333138.9	1047604.1	200.6	120	-10.6	228.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-02	333138.4	1047605.2	201.0	125	-6.9	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-03	333138.7	1047605.3	198.7	105	-16.6	240.1	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-04	333139.6	1047606.5	197.4	85	-28.6	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-05	333137.2	1047606.0	201.3	135	-2.6	251.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-06	333136.6	1047608.1	198.9	120	-7.7	254.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-07	333137.8	1047608.0	198.6	105	-15.4	258.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-08	333140.7	1047609.5	197.2	85	-27.8	265.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2A-09	333141.8	1047603.3	196.1	75	-48.7	199.9	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-10	333113.0	1047658.0	213.0	140	-6.2	229.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-11	333114.4	1047658.0	208.1	130	-11.8	226.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-12	333114.9	1047657.3	208.0	110	-20.2	223.1	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-13	333114.3	1047656.2	209.3	95	-34.5	216.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-14	333118.1	1047656.0	207.1	85	-64.9	180.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-15	333111.5	1047658.1	211.4	130	-5.2	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-16	333111.9	1047657.7	210.7	110	-13.6	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-17	333112.4	1047657.9	209.7	100	-25.2	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-18	333112.0	1047659.2	209.8	85	-43.6	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-19	333114.4	1047659.4	207.9	80	-67.4	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-20	333115.9	1047655.5	207.2	85	-51.4	204.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-21	333112.2	1047659.7	212.7	120	-7.3	253.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-22	333111.5	1047659.5	211.5	105	-14.1	257.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-23	333111.7	1047659.3	210.2	90	-29.2	263.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-24	333112.9	1047659.9	208.6	85	-48.3	275.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-25	333115.2	1047660.0	207.9	85	-65.7	305.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-76	333120.2	1047664.2	207.5	95	-69.8	174.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-77	333123.1	1047665.5	207.3	85	-74.8	245.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-78	333124.5	1047665.4	207.4	95	-85.0	43.1	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2B-79	333123.6	1047667.5	207.7	90	-72.0	316.0	2021	RC	Sabre Gold Mines Corp.	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
AZG-21-P2B-83	333123.8	1047669.3	208.2	95	-71.0	10.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-26	333084.4	1047705.9	220.1	115	-12.2	225.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-27	333084.9	1047705.5	218.5	105	-24.9	219.9	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-28	333086.2	1047709.9	217.4	90	-41.1	211.9	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-29	333091.6	1047719.6	217.9	85	-55.4	198.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-31	333083.3	1047708.2	221.3	115	-14.7	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-32	333083.8	1047708.6	219.5	100	-27.5	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-33	333085.0	1047711.1	217.8	85	-52.1	240.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-34	333092.2	1047720.0	218.2	80	-74.8	240.1	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-36	333097.6	1047718.0	217.5	100	-76.2	60.1	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-37	333082.3	1047710.3	221.3	115	-11.7	257.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-38	333082.5	1047711.4	219.0	105	-24.7	263.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-39	333083.9	1047713.6	218.3	90	-41.1	276.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2C-42	333097.2	1047719.8	217.9	100	-67.8	5.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2D-44	333090.6	1047772.9	220.8	100	-84.8	319.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2D-45	333096.4	1047779.8	220.2	110	-75.4	42.9	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2D-46	333093.9	1047777.2	220.9	115	-54.1	352.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2D-47	333094.4	1047776.0	220.7	110	-66.6	6.4	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2D-48	333096.2	1047780.9	221.1	125	-51.7	10.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2D-84	333091.7	1047779.6	221.5	130	-46.6	347.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2E-50	333082.5	1047758.7	220.8	100	-86.8	170.6	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2E-52	333074.9	1047767.2	220.0	85	-67.2	249.4	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2E-55	333074.3	1047767.8	220.2	95	-67.9	326.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2E-93	333070.6	1047771.8	220.4	130	-31.8	318.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2F-61	333035.2	1047746.5	219.9	85	-65.9	151.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2F-64	333031.6	1047757.1	219.9	75	-53.1	277.9	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2F-95	333033.1	1047756.1	219.9	85	-22.9	315.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2G-69	333006.9	1047751.3	219.6	75	-58.0	169.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2G-73	333016.6	1047734.1	220.6	60	-64.7	300.1	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2H-101	332993.2	1047719.8	222.6	50	-24.0	180.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P2H-99	332984.8	1047735.3	220.3	45	-40.0	272.0	2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P6A-01	333371.0	1046416.0	250.0	115	*Unknown*		2021	RC	Sabre Gold Mines Corp.	Excluded
AZG-21-P6D-32	333279.0	1046529.0	247.0	120	*Unknown*		2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-40	333359.5	1046622.1	376.7	205	-52.5	68.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-41	333361.1	1046623.0	376.4	185	-60.4	70.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-42	333360.9	1046622.2	376.4	175	-62.1	71.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-43	333360.1	1046621.6	376.3	165	-66.3	74.7	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-44	333360.7	1046623.7	376.4	195	-56.3	58.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-45	333359.9	1046623.1	376.1	185	-61.9	57.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-46	333358.8	1046622.7	376.3	175	-63.9	57.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-47	333359.7	1046624.2	376.5	130	-80.0	54.8	2021	RC	Sabre Gold Mines Corp.	Excluded

Drillhole ID	X Coordinate	Y Coordinate	Elevation	Total Depth	Dip	Azimuth	Year	Type	Company	Use in Model
21-P6E-48	333359.6	1046623.6	376.5	190	-57.3	46.1	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-49	333360.0	1046624.2	376.5	180	-61.1	42.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6E-50	333358.7	1046623.6	376.6	170	-63.4	38.7	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-51	333348.7	1046686.8	367.5	165	-65.6	78.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-52	333348.2	1046686.1	366.8	150	-68.9	84.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-53	333347.9	1046685.4	366.6	120	-79.1	164.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-54	333347.9	1046687.1	366.0	165	-63.9	61.2	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-55	333347.3	1046686.7	366.2	155	-67.1	61.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-56	333348.4	1046687.5	366.3	145	-59.6	62.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-57	333349.0	1046689.2	367.2	170	-60.7	45.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-58	333348.7	1046690.2	366.8	165	-64.3	43.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-59	333348.7	1046689.1	366.2	150	-67.5	38.0	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6F-60	333347.2	1046689.4	366.1	115	-78.5	323.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6G-61	333318.8	1046735.5	356.7	120	-67.5	88.9	2021	RC	Sabre Gold Mines Corp.	Excluded
21-P6G-62	333318.8	1046735.5	356.6	135	-61.7	81.0	2021	RC	Sabre Gold Mines Corp.	Excluded