Initial Assessment of the Granite Creek Mine, Humboldt County, NV

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Date and Signature Page

The undersigned prepared this Technical Report Summary (TRS) report, titled: Initial Assessment of the Granite Creek Mine, Humboldt County, NV, dated the 26th day of March, 2025, with an effective date of December 31, 2024, in support of the public disclosure of Mineral Resource and Mineral Reserve estimates for the Granite Creek Mine. The format and content of the TRS has been prepared in accordance with Securities and Exchange Commission (SEC) S-K regulations (Title 17, Part 229, Items and 1300 through 1305).

Dated this March 26, 2025

/s/ Practical Mining LLC

Practical Mining LLC

/s/ T.R. Raponi Consulting Ltd.

T.R. Raponi Consulting Ltd.

/s/ Global Resources Engineering

Global Resources Engineering

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List of Abbreviations

A	Ampere	kA	kiloamperes
AA	atomic absorption	kCFM	thousand cubic feet per minute
AGP	Acid Generation Potential	Kg	Kilograms
Ag	Silver	km	kilometer
ANFO	ammonium nitrate fuel oil	km2	square kilometer
ANP	Acid Neutralization Potential	kWh/t	kilowatt-hour per ton
Au	Gold	LoM	Life-of-Mine
AuEq	gold equivalent	m	meter
btu	British Thermal Unit	m2	square meter
°C	degrees Celsius	m3	cubic meter
CCD	counter-current decantation	masl	meters above sea level
CIL	carbon-in-leach	mg/L	milligrams/liter
CoG	Cut off grade	mm3	cubic millimeter
cm	centimeter	MME	Mine & Mill Engineering
cm2	square centimeter	Moz	million troy ounces
cm3	cubic centimeter	Mt	million tonnes
cfm	cubic feet per minute	MTW	measured true width
CRec	core recovery	MW	million watts
CSS	closed-side setting	m.y.	million years
CTW	calculated true width	NGO	non-governmental organization
°	degree (degrees)	NI 43-101	Canadian National Instrument 43-101
dia.	diameter	oz	Troy Ounce
EA	Environmental Assesment	opt	Troy Ounce per short ton
EIS	Environmental Impact Statement	oz/ton	Troy Ounce per short ton
EMP	Environmental Management Plan	%	percent
FA	fire assay	PLC	Programmable Logic Controller
Ft	Foot	PLS	Pregnant Leach Solution
Ft2	Square foot	PMF	probable maximum flood
Ft3	Cubic foot	POO	Plan of Operations
g	Gram	ppb	parts per billion
g/L	gram per liter	ppm	parts per million
g-mol	gram-mole	POX	Pressure Oxidation
g/t	grams per metric tonne	QAQC	Quality Assurance/Quality Control
ha	hectares	RC	reverse circulation drilling
HDPE	Height Density Polyethylene	ROM	Run-of-Mine
HTW	horizontal true width	RQD	Rock Quality Description
ICP	induced couple plasma	SEC	U.S. Securities & Exchange Commission
ID2	inverse-distance squared	Sec	second
ID3	inverse-distance cubed	SG	specific gravity
mm	millimeter	SPT	Standard penetration test
mm2	square millimeter	ton	US Short Ton

1 Summary

1.1 Introduction

This Technical Report Summary (TRS) dated the 26th day of March, 2025 with an effective date of December 31, 2024 provides an updated statement of Mineral Reserves and Mineral Resources for Osgood Mining Company's Granite Creek mine. This TRS also provides an Initial Assessment and statement of mineral resources at Granite Creek.

Cautionary Notes:

1. *The financial analysis contains certain information that may constitute "forward-looking information" under applicable Canadian securities legislation. Forward-looking information includes, but is not limited to, statements regarding the Company's achievement of the full-year projections for ounce production, production costs, AISC costs per ounce, cash cost per ounce and realized gold/silver price per ounce, the Company's ability to meet annual operations estimates, and statements about strategic plans, including future operations, future work programs, capital expenditures, discovery and production of minerals, price of gold and currency exchange rates, timing of geological reports and corporate and technical objectives. Forward-looking information is necessarily based upon a number of assumptions that, while considered reasonable, are subject to known and unknown risks, uncertainties, and other factors which may cause the actual results and future events to differ materially from those expressed or implied by such forward looking information, including the risks inherent to the mining industry, adverse economic and market developments and the risks identified in Premier's annual information form under the heading "Risk Factors". There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information. All forward-looking information contained in this Presentation is given as of the date hereof and is based upon the opinions and estimates of management and information available to management as at the date hereof. Premier disclaims any intention or obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, except as required by law.*

1.2 Property Description

The Granite Creek Project is located in Humboldt County, Nevada, 28 miles northeast of the town of Winnemucca, and it is part of the historic Potosi mining district. It is centered at roughly 41° 8' N latitude and 117° 15.5' W longitude. It encompasses about 4,506 acres (1,823.5 hectares) including owned unpatented claims, leased unpatented claims and owned surface fee land. i-80 Gold purchased the Granite Creek property from Waterton Global in June 2020.

1.3 Geology and Mineral Deposits

The Property is located on the eastern flank of the Osgood Mountains within the Basin and Range tectonic province of northern Nevada. The Granite Creek Mine occurs within a northeast-trending structural corridor known as the Getchell gold trend. This trend also encompasses a number of gold deposits located outside the Property including the Preble, Getchell, Turquoise Ridge, and

Twin Creeks. These deposits are hosted in Paleozoic marine sedimentary rocks. Gold mineralization at the Property is described as a Carlin-type, sedimentary-rock hosted system.

The Property geology comprises a sequence of Cambrian to Ordovician sedimentary rocks that form part of the Osgood Mountain Terrane and the Osgood Mountains. Much of the Property comprises shales, hornfels sedimentary rocks and limestone interbeds of the Preble Formation, and an overlying (or juxtaposed), alternating sequence of limestone, shale, and dolomite with tuffaceous shale and intraformational conglomerates belonging to the Comus Formation. The Preble and Comus Formations have been folded into a broad north-plunging anticline and have been intruded by large Cretaceous granodiorite stocks, resulting in irregular contact metamorphism.

Gold mineralization at the Property is strongly structurally controlled, occurring at favorable sites within a fault network occurring around the eastern edge of the Osgood granodiorite and predominantly within Comus Formation host rocks. Mineralization is commonly associated with the decarbonatization of carbonate rocks and the introduction of silica, fine grained pyrite, arsenian pyrite, and remobilized carbon. Continuity of mineralization is highly variable, ranging from 40 to 4,500 feet (12 to 1,372 meters) in strike extent, 250 to 1,800 feet (76 to 550 meters) in down-dip extent and 5 to 400 feet (1.5 to 122 meters) in thickness. The underground mineralization has a variable thickness between 5 and 30 feet (1.5 and 9 meters).

Oxidation reaches depths of up to 1,800 feet (550 meters) within shear zones. Oxide mineralization includes pervasive limonite, hematite, along with other iron and arsenic oxides. Historical production from the open pits was focused on oxidized material.

Underground mineralization displays pervasive argillization and decarbonatization of host lithologies, along with the formation of dissolution collapse breccias and intense shearing. Where the alteration is strongest, the altered zones consist of punky, spongy decarbonatized limestone in an argillically altered fine-grained, carbon-rich matrix (Gustavson, 2012). Silicification is minor and occurs as a broad overprint on the zone. Underground production includes both sulfide and oxide material.

1.4 Metallurgical Testing and Processing

A wide range of metallurgical testing has been conducted on the Granite Creek deposit from 1999 to 2024. These tests were conducted on material from the oxide, transition and sulfide domains (sulfide being primarily underground). Test work focused on the primary variables in a typical gold project including: comminution, heap leaching, tank leaching (direct cyanidation and carbon in leach), solid/liquid separation, cyanide destruction, and refractory ore treatment (underground material).

The results of the test work program showed that the deposit (open pit resource) was amenable to both heap leach and carbon-in-leach (CIL) processes. The heap leach gold extraction is proportional to the cyanide soluble gold content and the CIL extraction is proportional to the feed grade averaging 86.6% over the life of mine. The Granite Creek material contains organic carbon and shows varying degrees of "preg-robbing," as such the CIL process route was selected for the process. The underground material is currently toll treated in an offsite autoclave process.

1.5 Geology and Mineralization

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1.6 History

The Property has been explored by a number of individuals and mining/exploration companies since the late 1930s. The original discovery on the Property was made by Clovis Pinson and Charles Ogee in the mid to late 1930s, but production did not occur until after World War II, when ore from the original discovery was shipped to and processed at the Getchell mine mill. In 1949 and 1950, total production from the Granite Creek Mine amounted to approximately 10,000 short tons (9,071 tonnes) grading approximately 0.14 ounces per ton (opt) (4.8 g/t).

1.7 Exploration, Drilling, and Sampling

Cordex completed ground-based magnetics over the CX Zone in 1970. In 1983, Cordex conducted a 1:6000-scale mapping program of the Property. In 2016, OMC contracted Mr. Robert Leonardson to complete a geological study on the Property that focused on advancing OMC's understanding of the structural framework and providing guidance on exploration targeting. By the time of preparation of this technical report, i-80 had drilled 17 surface holes within the open pit areas.

The Property has been historically drilled using a combination of reverse circulation (RC) and diamond drilling. The majority of drilling was completed from surface. More recent drilling was completed as underground diamond core drilling. Sampling protocols adopted by former operators were similar and generally followed industry best practices of the time.

RC samples were collected from the drill cyclone in 5-foot (1.5-meter) intervals. Diamond core was sampled predominantly as 5-foot (1.5-meter) intervals but were locally adjusted based on geological alteration and oxidation contacts. RC and core recovery were recorded and considered to be excellent.

Samples were prepared and analyzed by a number of accredited laboratories throughout the Project history, including ALS Chemex, Inspectorate American Laboratories (IAL), and American Assay Laboratories (AAL).

1.8 Data Verification

Data validation has been completed by various operators throughout the Project's history. This process comprised the checking of original assay certificates and drillhole records against the digital database. This was completed most recently in April 2019 by OMC.

Quality assurance/quality control (QA/QC) samples including Certified Reference Materials (CRMs), coarse blanks, and field duplicate samples were included regularly with samples submitted between 2005 and 2008. A limited number of CRMs were included with drilling completed in 2012.

In 2021, GRE reviewed all prior work on available QA/QC data between 2005 and 2015. GRE also reviewed and checked QA/QC procedures and the database provided by i-80 Gold Corp.

In general, the QA/QC sample insertion rates used fall below general accepted industry standards. For future exploration campaigns, standards, blanks, and duplicates, including one standard, one duplicate, and one blank sample, should be inserted every 20 interval samples, as is common within industry standards.

CRM samples show a reasonable level of accuracy, but poor to moderate precision when using standard deviations provided by the CRM supplier. A maximum of three to five different CRM samples would be adequate to monitor laboratory performance at the approximate cutoff grades, average grades, and higher grades of the deposits.

Blank sample results are considered acceptable and suggest no systematic contamination has occurred throughout the analytical process.

Duplicate sample results show suboptimal performance, which may be a result of the heterogenous nature of mineralization, uncrushed samples, and sampling variance. Overall, duplicate samples appear to be positively biased, with duplicate results returning higher grade than original samples.

Based on the review of the project database and all existing project documents and the author’s observations of the geology and mineralization at the project during the site visit, GRE’s QP considers the lithology, mineralization, and assay data contained in the project database to be reasonably accurate and suitable for use in estimating mineral resources.

1.9 Granite Creek Underground

1.9.1 Mineral Resources

Table 1-1 Summary of Mineral Resources at the End of the Fiscal Year Ended December 31, 2024

Zone	ktons	ktonnes	Au opt	Au g/t	Au koz
			Measured		
Ogee	88	80	0.244	8.4	22
Otto	59	53	0.256	8.8	15
Meas Total	147	133	0.249	8.5	37
			Indicated		
CX	8	7	0.391	13.4	3
Ogee	181	164	0.352	12.1	64
Otto	295	268	0.316	10.8	93
South Pacific	223	203	0.286	9.8	64
Ind Total	707	641	0.317	10.9	224
			Measured and Indicated		
CX	8	7	0.391	13.4	3
Ogee	269	244	0.317	10.9	85
Otto	354	321	0.306	10.5	108
South Pacific	223	203	0.286	9.8	64
M&I Total	854	775	0.305	10.5	261
			Inferred		
CX	97	88	0.351	12.0	34
Ogee	42	38	0.563	19.3	24
Otto	187	170	0.401	13.7	75
South Pacific	536	486	0.361	12.4	194
Inf Total	862	782	0.378	13.0	326

Notes Pertaining to Underground Mineral Resources:

1. Mineral Resources have been estimated at a gold price of $2,175 per troy ounce and a silver price of $27.25 per ounce. Refer to section 16.1 for price selection details.
2. Mineral Resources have been estimated using gold metallurgical recoveries of 85.2% to 94.2% for pressure oxidation. Payment for refractory mineralization sold to a third party is 58%. Oxide CIL mineralization payments vary from 40% to 70% based upon the grade of the mineralization.
3. The cutoff grade for refractory Mineral Resources varies from 0.151 to 0.184 opt. for acidic conditions. The cutoff grade for oxide mineral resources is 0.075 opt;
4. The contained gold estimates in the Mineral Resource table have not been adjusted for metallurgical recoveries.
5. Numbers have been rounded as required by reporting guidelines and may result in apparent summation differences.
6. A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or

quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling;

7. An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
8. Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant factors,
9. Mineral Resources have an effective date of December 31, 2024, and

- The reference point for mineral resources is in situ.

1.9.2 Mining, Infrastructure, and Project Schedule

The Granite Creek Underground mine is fully permitted and has entered the production phase of operations. All infrastructure is in place to support the anticipated production rate and duration of mine operations.

1.9.3 Economic Analysis

The mineral resource at the Granite Creek Underground Mine contains 50% by weight inferred mineral resources. On a contained ounce basis, inferred mineral resources account for 56% of the contained gold ounces. The without inferred case presented is a gross factorization of the mine plan and has not been modified to reflect accompanying changes to capital development, productivities or unit operating costs. The results of a constant dollar cash flow analysis of the planned underground mining operation are shown in Table 1-2.

Only the case that includes inferred mineral resources provides a positive cash flow with a NPV 5% of $155M and 84% IRR. The high IRR is attributed to the fact that all mine infrastructure has been completed and production ramp up has started.

Table 1-2 Underground Mine Financial Statistics

	With Inferred	Without Inferred
Gold price (US$/oz)	$2,175	
Silver price (US$/oz)	$27.25	
Mine life (years)	8	
Average mineralized mining rate (tons/day)	435	225
Average grade (oz/t Au)	0.339	0.292

Average gold recovery (autoclave %)	78%	78%
Average annual gold production (koz)	52	23
Total recovered gold (koz)	418	186
Sustaining capital (M$)	$88.8	$88.8
Cash cost (US$/oz) [1]	$1,366	$1,699
All-in sustaining cost (US$/oz) [1,2]	$1,597	$2,217
Project after-tax $NPV_{5\%}$ (M$)	$155	($30)
Project after-tax $NPV_{8\%}$ (M$)	$135	($33)
Project after-tax IRR	84%	-12.7%
Payback Period	3.2 Years	NA
Profitability Index $_{5\%}$ [3]	12.6	0.7

Notes:

1. *Net of byproduct sales;*
2. *Excluding income taxes, resource conversion drilling, corporate G&A, corporate taxes and interest on debt;*
3. *Profitability index (PI), is the ratio of payoff to investment of a proposed project. It is a useful tool for ranking projects because it allows you to quantify the amount of value created per unit of investment. A profitability index of 1 indicates breakeven;*
4. *This IA is preliminary in nature, it 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 IA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability;*
5. *Inferred mineral resources constitute 50% of mass and 56% of gold ounces of all mineral resources. The "Without Inferred" statistics presented are a gross factorization of the mine plan without any redesign of mine excavations or recalculation of productivities and costs. Capital costs are the same for the "With Inferred" and "Without Inferred" scenarios. The "Without Inferred" scenario is presented solely to illustrate the project's dependence on inferred mineral resources.*
6. *The financial analysis contains certain information that may constitute "forward-looking information" under applicable Canadian and United States securities regulations. Forward-looking information includes, but is not limited to, statements regarding the Company's achievement of the full-year projections for ounce production, production costs, AISC costs per ounce, cash cost per ounce and realized gold/silver price per ounce, the Company's ability to meet annual operations estimates, and statements about strategic plans, including future operations, future work programs, capital expenditures, discovery and production of minerals, price of gold and currency exchange rates, timing of geological reports and corporate and technical objectives. Forward-looking information is necessarily based upon a number of assumptions that, while considered reasonable, are subject to known and unknown risks, uncertainties, and other factors which may cause the actual results and future events to differ materially from those expressed or implied by such forward looking information, including the risks inherent to the mining industry, adverse economic and market developments and the risks identified in Premier's annual information form under the heading "Risk Factors". There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information. All forward-looking information contained in this Presentation is given as of the date hereof and is based upon the opinions and estimates of management and information available to management as at the date hereof. Premier disclaims any intention or obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, except as required by law;*

1.9.4 Conclusions

Metallurgy

1. Granite Creek underground samples were refractory with baseline CIL gold recoveries ranging from 9% to 46%, averaging 31%;
2. Shake flask tests with gold cyanide spikes were used to determine preg robbing index. The average preg-robbing index was 17.9%, ranging from 4.4% to 54.1%;
3. Bench top autoclave batch pressure oxidation tests were completed on all samples with 2 sets of acid conditions and four sets of alkaline conditions. Acid conditions resulted in resulted in higher sulfur oxidations and higher gold recoveries;
4. Three continuous pressure oxidation runs were completed with two acid and one alkaline sets of conditions based on the batch results. The continuous results followed the results of the batch tests with the acid conditions producing the higher sulfur oxidations and gold recoveries;
5. Overall gold recoveries increased with increasing sulfur oxidation;
6. Cyanide destruction tests on CIL tailings using the SO2/air process reduced weak acid dissociable cyanide concentrations to below 50 ppm using established reagent addition rates and retention time;
7. Thickening and filtration tests on CIL tailings showed unacceptable thickening properties and filtration rates. Thickening and filtration of pressure oxidation streams is not recommended.
8. Arsenic concentrations in the samples averaged 0.29%, largely occurring as arsenian pyrite with only trace amounts of arsenopyrite.
9. Sulfide minerals were predominantly pyrite with some marcasite.
10. Mercury concentrations ranged from 31 ppm to 138 ppm, averaging 81 ppm. These concentrations will require mercury capture and abatement equipment in the process flowsheet.

Mining and Infrastructure

1. The mine infrastructure has been completed.
2. Production ramp up has reached approximately 400 tons per day.
3. The mining contractor is in place with the full complement of equipment and personnel.
4. Decline development has accessed 700 vertical feet of mineralization of the Otto and Ogee zones. Development has reached the top of the South Pacific zone allowing additional active production stopes.
5. The drill lateral drift over the South Pacific zone has been completed.
6. Reconciliation of the model to mill indicates process head ounces exceed model by 19%. This appears to be from mining in a larger low grade halo around high grade core.

7. Processed grade is lower than the life-of-mine planned grade due to extensive mining of marginal mineralization below the economic cutoff grade.

1.9.5 Recommendations

Metallurgical Testing

1. Establish sampling using the most recent mine plan to select samples to evaluate pressure oxidation with CIL cyanidation under Lone Tree conditions. Testing should also include baseline CIL tests and roasting testing as a comparison.
2. Testing should attempt to establish head grade and extraction relationships for use in more detailed resource modelling.
3. Mineralogy impacts need to be established and geologic domains within each resource need to be determined.
4. Additional comminution data should be collected to assess hardness variability within the zones and any potential impacts on throughput in the Lone Tree process plant.
5. The resource model should be advanced to include arsenic, TCM, TOC, mercury, as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources.
6. The estimated cost for the suggested next phase metallurgical program is to $350,000 based on current market pricing.

Resource Conversion and Exploration Drilling

1. Initiate and complete the infill drilling program in the South Pacific zone.
2. Update the mineralization model with the new drilling results.

Dewatering

1. Complete the planned dewatering well and re-evaluate the ground water model and inflow into the underground workings.

Mining

1. Stopes should be designed to mine parallel to strike wherever possible. Mining across strike results in additional planned dilution.

The width of stope production drifts in narrow mineralized areas should be decreased to 13 feet and smaller equipment mobilized to accommodate mineral resources.

1.10 Open Pit

1.10.1 Mineral Resources

Table 1-3 shows the pit-constrained open pit Mineral Resource at a gold grade cutoff of 0.30 g/t.

Table 1-3: Granite Creek Mine Project Open Pit Mineral Resource

Class	Zone	Total Process Material (1000s Tonnes)	Total Process Material (1000s Tons)	Au Grade (g/t)	Au Grade (opt)	Total Contained Au (1000s t. oz)
Measured	Pit B	2,910	3,207	1.32	0.042	123.41
	Pit A	563	620	1.07	0.034	19.30
	CX	10,889	12,003	1.30	0.042	455.27
	MAG	12,000	13,228	1.21	0.039	467.97
	Total	**26,362**	**29,059**	**1.26**	**0.040**	**1,065.95**
Indicated	Pit B	360	397	1.10	0.035	12.73
	Pit A	689	760	0.80	0.026	17.78
	CX	2,973	3,277	1.25	0.040	119.62
	MAG	7,317	8,066	0.93	0.030	219.16
	Total	**11,339**	**12,499**	**1.01**	**0.033**	**369.29**
Measured + Indicated	Pit B	3,270	3,604	1.29	0.042	136.14
	Pit A	1,252	1,380	0.92	0.030	37.08
	CX	13,862	15,280	1.29	0.041	574.89
	MAG	19,317	21,293	1.11	0.036	687.13
	Total	**37,701**	**41,558**	**1.18**	**0.038**	**1,435.24**
Inferred	Pit B	32	36	0.64	0.021	0.67
	Pit A	205	226	0.59	0.019	3.88
	CX	1,347	1,485	1.16	0.037	50.24
	MAG	563	620	1.11	0.036	20.17
	Total	**2,148**	2,367	**1.09**	**0.0**35	**74.95**

Please note that mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.10.2 Mining Methods

Mine plans for the resource areas were designed and planned using conventional open pit mining method for the low grade, widely distributed gold. The open pit areas are suitable for phased designs.

1.10.3 Economic Analysis

GRE performed an economic analysis of the project by building an economic model based on the following assumptions:

- Federal corporate income tax rate of 21%
- Nevada taxes:
  - Proceeds of Minerals Tax – variable, with a maximum of 5% of Net Proceeds
  - Property tax – 2.5605%
  - Nevada gold and silver mine royalty – variable, with a maximum of 1.1% of gross revenue
- Sales and use taxes are not included in the model
- Equipment depreciated over a straight 7 or 15 years and has no salvage value at the end of mine life
- Loss carried forward
- Depletion allowance, lesser of 15% of net revenue or 50% of operating costs
- Gold price of $2,175 per troy ounce
- Gold recovery calculated per block as detailed in Section 13
- Royalties on individual claims calculated by block, ranging from 0.02% to 7.5%, averaging 5.7%. There also is a 10% royalty applied to net profit.

1.10.3.1 Base Case

After analyzing the economic results of all cases considered, GRE selected the CIL only case with 0.85 g/t high grade cutoff, contractor operation, conventional tailings, and 133-tonne haul trucks and 21.9-tonne loaders as the base case as it results in the best overall economic results.

The economic model assumes a 1-year construction period. The time for permitting has not been included in the economic model, but the permitting for the open pit mine is likely to take three to five years and occur during the underground mining portion of the project.

Table 1-4 lists the key economic results for the selected scenario.

Table 1-4 Key Economic Indicators

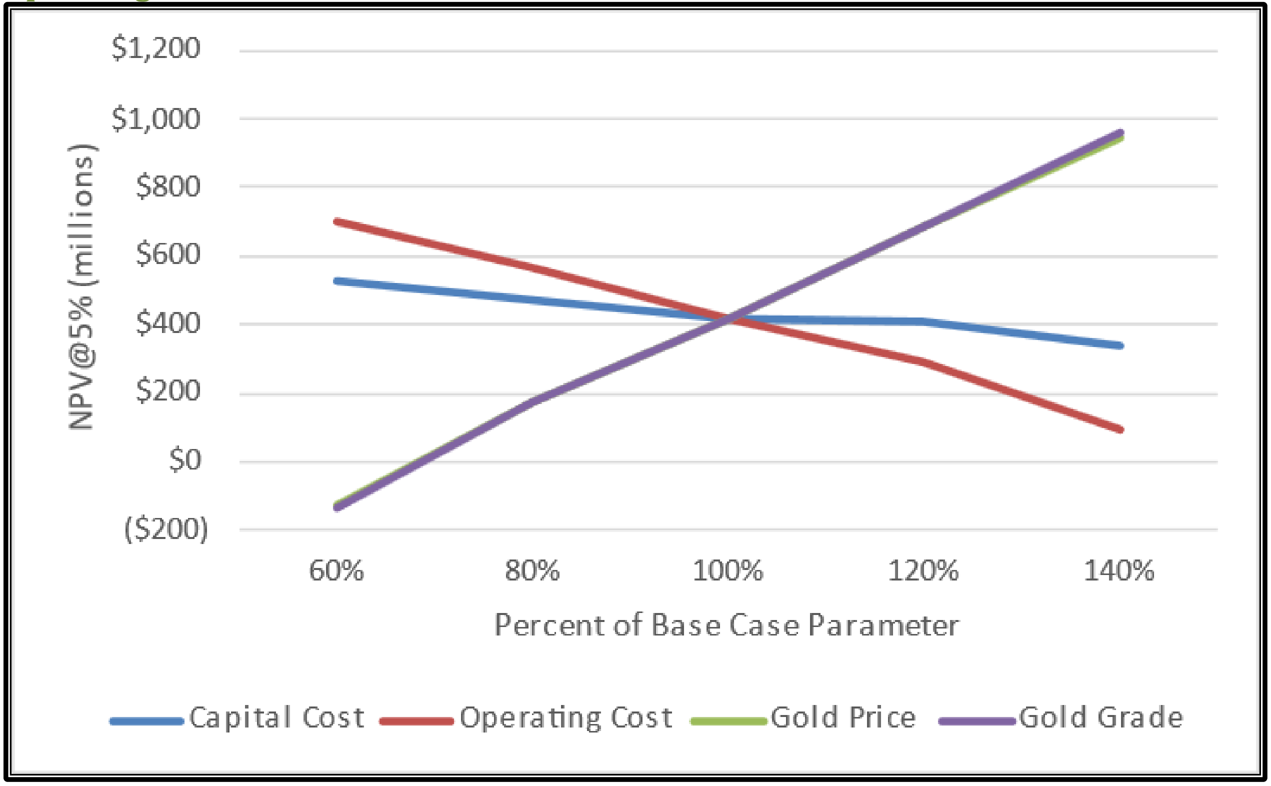
After Tax Economic Measure	Value
After Tax NPV@5% (millions)	$417.2
After Tax IRR	28.7%
Initial Capital (millions)	$254.7
Payback Period (years)	3.72
All-in Sustaining Cost ($/oz Au Produced)	$1,227.4
Cash Cost ($/oz Au Produced)	$1,180.5

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under S-K 1300. This IA is preliminary in nature and 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

1.10.3.2 Sensitivity Analyses

GRE evaluated the after-tax NPV@5% sensitivity to changes in gold price, gold grade, capital costs, and operating costs. The results indicate that the after-tax NPV@5% is most sensitive to gold price, moderately sensitive to operating cost, and least sensitive to capital cost (see Figure 1-1).

Figure 1-1 NPV@5% Sensitivity to Varying Gold Price, Gold Grade, Capital Costs, and Operating Costs



1.10.3.3 Conclusions of Economic Model

The open pit project economics shown in the IA are favorable, providing positive NPV values at varying gold prices, gold grades, capital costs, and operating costs.

2 Introduction

2.1 Registrant for Whom the Technical Report Summary was Prepared

This TRS was prepared for i-80 Gold Corporation and its subsidiaries Premier Gold Mines USA, Inc. and Osgood Mining Company (collectively i-80) in accordance with the requirements of the Securities and Exchange Commission (SEC) S-K regulations (Title 17, Part 229, Items 601 and 1300 through 1305) for i-80.

2.2Terms of Reference and Purpose of this Technical Report

This TRS provides an initial statement of Mineral Resources for the Granite Creek Mine under regulation S-K 1300 and presents an Initial Assessment (IA) of the indicated and inferred mineral resources.

The quality of information, conclusions, and estimates contained herein are based on: i) information available at the time of preparation and ii) the assumptions, conditions, and qualifications set forth in this report. This IA is a preliminary technical and economic study of the economic potential of all or parts of mineralization to support the disclosure of mineral resources. This IA is preliminary in nature. It 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 Initial Assessment will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

2.3Details of Inspection

This TR includes technical evaluations from three independent consultants. The consultants are specialists in the fields of geology, exploration, metallurgy, open pit and underground mining.

None of the Qualified Professionals (QPs) has any beneficial interest in i-80 or any of its subsidiaries, or in the assets of i-80 or any of its subsidiaries or in any property near the Granite Creek Project. The QPs will be paid a fee for this work in accordance with normal professional consulting practices.

The QP's and the sections of this report each contributed to are listed in Table 2-2.

Table 2-2Table 2-1 summarizes the details of the personal inspections on the property by each qualified firm or, if applicable, the reason why a personal inspection has not been completed.

Table 2-1 Personal Inspections by Qualified Professionals

Company	Discipline	Dates of Personal Inspection	Details of Inspection
Practical Mining	Mining, Mineral Resources and Mineral Reserves	October 10, 2024	Site specific hazard training, examined core and core logging procedures, examined underground mine workings, observed core drilling operations, observed mining operations.
Raponi Engineering	Metallurgical Testing and Mineral Processing	None	The Granite Creek Mine does not have facilities for mineral processing.
Global Resource Engineering	Geology	April 20, 2021	General geological inspection of the Granite Creek area, including visual inspection of key geologic formations, lithologies, structural geology, and mineralization.
Global Resource Engineering	Mining, Mineral Resources	April 20, 2021	Examined infrastructure, pit walls, haul roads, examined core storage; examined conditions of underground workings
Global Resource Engineering	Metallurgical Testing and Mineral Processing	April 20, 2021	Examined infrastructure, pit walls, haul roads, examined core storage; examined conditions of underground workings

2.4 Sources of Information

This report is based in part on internal Company technical reports, previous studies, maps, published government reports, Company letters and memoranda, and public information as cited throughout this report and listed in 24 References.

2.5Report Version

This TRS presents the inaugural statement of the Granite Creek Project mineral resources by i-80 under 17 CFR § 229.1300. The Company has most recently disclosed mineral resources for the project under Canadian Securities NI 43-101 regulations with the report Titled “Preliminary Economic Assessment NI 43-101 Technical Report Granite Creek Mine Project Humboldt County, Nevada, USA” dated September 27,2021.

2.6 Qualification of the Authors

This TR includes technical evaluations from three independent consultants. The consultants are specialists in the fields of geology, exploration, metallurgy, open pit and underground mining.

None of the Qualified Professionals (QPs) has any beneficial interest in i-80 or any of its subsidiaries, or in the assets of i-80 or any of its subsidiaries or in any property near the Granite Creek Project. The QPs will be paid a fee for this work in accordance with normal professional consulting practices.

The QP's and the sections of this report each contributed to are listed in Table 2-2.

Table 2-2 QP Section Responsibility

Section No.	Title	Responsible Firm
1.1.	Introduction	Practical
1.2.	Property Description	Practical
1.3.	Geology and Mineral Deposits	Practical
1.4.	Metallurgical Testing and Processing	Raponi
1.5	Geology and Mineralization	GRE
1.6	History	GRE
1.7	Exploration, Drilling, and Sampling	GRE
1.8	Data Verification	GRE
1.9	Granite Creek Underground	Practical
1.10	Granite Creek Open Pit	GRE
2	Introduction	Practical, Raponi, GRE
3	Property Description and Location	Practical
4	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Practical
5	History	Practical
6	Geologic Setting, Mineralization and Deposit	Practical
7.1.	Exploration	GRE
7.2.	Drilling	GRE
7.3	Update to Drilling Statistics to Include i-80 Drilling and Land Package Expansion	Practical
7.4.	Hydrogeology	Practical
8.1 – 8.9	Sample Preparation, Analysis and Security	GRE

Section No.	Title	Responsible Firm
8.10 & 8.11	PM Discussions on QA/QC 2021 & 2022	Practical
8.12	Conclusions	GRE
9.1.	GRE Site Inspection (2021)	GRE
9.2.	Visual Sample Inspection and Check Sampling	GRE
9.3.	Database Audits	GRE
9.4.	QP Opinions on Adequacy	GRE
9.5.	Practical Mining Drillhole Database Verification	Practical
10.1.	Introduction	Raponi
10.2.	Metallurgical Test Work	Raponi
10.3.	Sample Representativity	Raponi
10.4.	Deleterious Elements	Raponi
10.5.	Geometallurgical Modeling	GRE
10.6.	Conclusions	GRE
10.7.	Recommendations	GRE
11.1.	Introduction	GRE
11.2.	Drill Hole Database	GRE
11.3.	Topography	GRE
11.4.	Geologic Model	GRE
11.5.	Open Pit Estimation	GRE
11.6.	Open Pit Resource	GRE
11.7.	Underground Mineral Resources	Practical
13.1.	Open Pit	GRE
13.2.	Underground	Practical
14.1.	Introduction	GRE
14.2.	Process Description	GRE
14.3.	Refractory Processing	Raponi
15	Infrastructure	Practical
16	Market Studies and Contracts	Practical
17.1.	Environmental Setting	GRE
17.2.	Environmental Studies and Issues	Practical
17.3.	Social or Community Impacts	Practical
17.4.	Permits	Practical
17.5.	Water Use Permits	Practical
17.6.	Environmental Permits	GRE

Section No.	Title	Responsible Firm
17.7.	Mine Closure	GRE
17.8.	Local Procurement and Hiring	GRE
18.1	Open Pit Capital Cost Estimate	GRE
18.2	Open Pit Operating Cost Estimate	GRE
18.3.	Granite Creek Underground	Practical
19.1.	Taxes	Practical
19.2.	Granite Creek Underground	Practical
19.3.	Open Pit	GRE
20	Adjacent Properties	Practical
21	Other Relevant Data and Information	Practical, Raponi, GRE
22	Interpretation and Conclusions	Practical, Raponi, GRE
23	Recommendations	Practical, Raponi, GRE
24	References	Practical, Raponi, GRE
25	Reliance on Information Provided by the Registrant	Practical, Raponi, GRE
25.1.1	Environmental Recommendations	GRE

2.7Sources of Information

Information sources are documented either within the text and cited in references or are cited in references only. The authors believe the information provided by i-80 and to be accurate based on their work on the project.

2.8 Units of Measure

US Imperial units of measure are used throughout this document unless otherwise noted. US/Metric conversion factors are listed in Table 2-3. Currency is expressed as United States Dollars unless otherwise noted.

Table 2-3 Units of Measure Conversion Factors

US Imperial to Metric Conversions	
Linear Measure	**Weight**
1 inch = 2.54 cm	1 short ton = 2,000 lbs = 0.9071 metric tons
1 foot = 0.3048 m	1 lb = 0.454 kg = 14.5833 troy oz.
1 mile = 1.6 km	**Assay values**
Area Measure	1 ounce per short ton = 34.2857 g/t
1 acre = 0.4047 ha	1 part per million = 0.0292 opt

1 square mile = 640 acres = 259 ha	1 troy oz. = 31.10348 g
Density	
1 tonne/m^3 = 0.0312 tons per ft^3	

2.9 Coordinate Datum

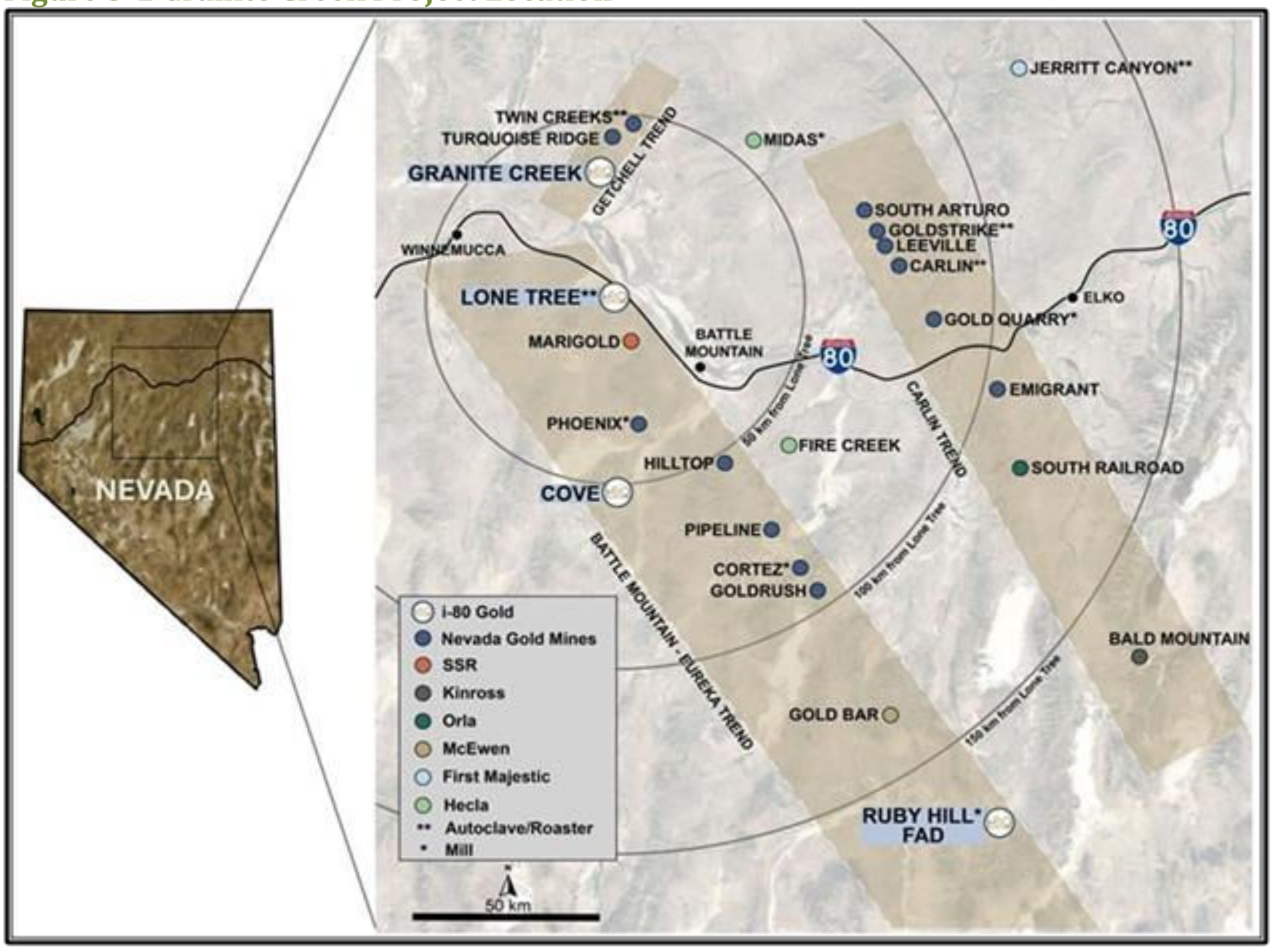
Spatial data utilized in analysis presented in this TR are projected in the Granite Creek local mine grid unless noted otherwise.

3 Property Description and Location

3.1Property Description

The Granite Creek Project is located in Humboldt County, Nevada, 28 miles northeast of the town of Winnemucca, and it is part of the historic Potosi mining district. It is centered at roughly 41° 8' N latitude and 117° 15.5' W longitude. It encompasses about 4,506 acres (1,823.5 hectares) including owned unpatented claims, leased unpatented claims and owned surface fee land. The federal land is administered by the BLM. Figure 3-1 shows the location of the Granite Creek Project.

Figure 3-1 Granite Creek Project Location



3.2Status of Mineral Titles

Ownership of the Granite Creek Project land position comprises various forms of title. Figure 3-2 shows the Granite Creek land position. i-80 owns 48 unpatented lode claims covering about 897 acres (Table 3-2), and leases 56 unpatented lode claims covering about 1,007 acres (Table 3-3).

The lease expires May 9, 2040. i-80 also owns, through its subsidiaries, fee surface land parcels covering about 2,602 acres.

Unpatented claims have annual maintenance fees of $200 per claim payable to the BLM and a notice of intent to hold (NIH) in the amount of $12 per claim plus $12 filing fee per document payable to Humboldt County. Claim maintenance fees are paid through September 2025 with the BLM. The NIH was paid to Humboldt County on July 9, 2024; payments are current at the time of this report. Fee land is subject to Nevada state real property tax, and certain mine infrastructure is subject to Nevada state personal property tax. Leased unpatented claims are subject to yearly lease fees. Holding costs for 2025 are listed in Table 3-1.

Table 3-1 Holding Costs

Description	Payee	Quantity	Amount
Unpatented Claim Maintenance Fee	BLM	104	$20,800.00
Notice of Intent to Hold Unpatented Claims	Humboldt County	104	$1,260.00
Real Property Taxes	Humboldt County	5 parcels	$7,234.63
Personal Property Taxes	Humboldt County	various infrastructure	$94,594.15
Lease fees, annually adjusted by CPI	Lease Holders	56 unpatented claims	$122,495.24
Total			**$246,384.02**

Figure 3-2 Granite Creek Land Position Map

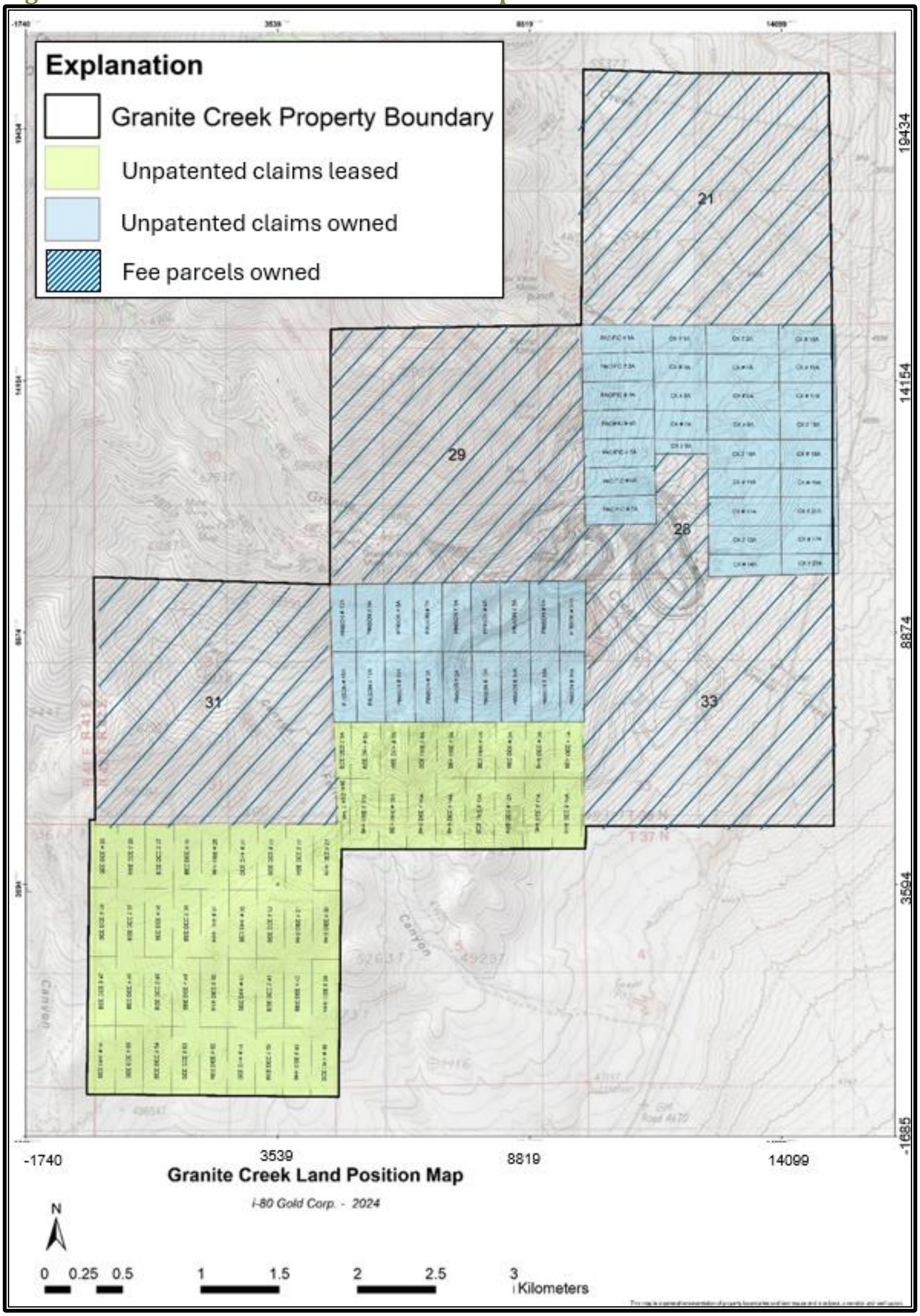


Table 3-2 Granite Creek Owned Unpatented Claims

Claim Name	BLM Legacy Number	Claim Type	Number of Claims
PACIFIC # 1A - PACIFIC # 7A	NMC319814 - NMC319820	Lode	7
CX # 1A - CX # 23A	NMC319833 - NMC319855	Lode	23
PINSON # 1A - PINSON # 18A	NMC319856 - NMC319873	Lode	18
Total Owned Patented Claims			**48**

Table 3-3 Granite Creek Leased Unpatented Claims

Claim Name	BLM Legacy Number	Claim Type	Number of Claims
NEW BEE DEE FRAC. #1, #2	NMC282121, NMC282122	Lode	2
BEE DEE # 21 - BEE DEE # 56	NMC282123 - NMC282158	Lode	36
BEE DEE # 1A	NMC319892 - NMC319909	Lode	18
Total Leased Patented Claims			**56**

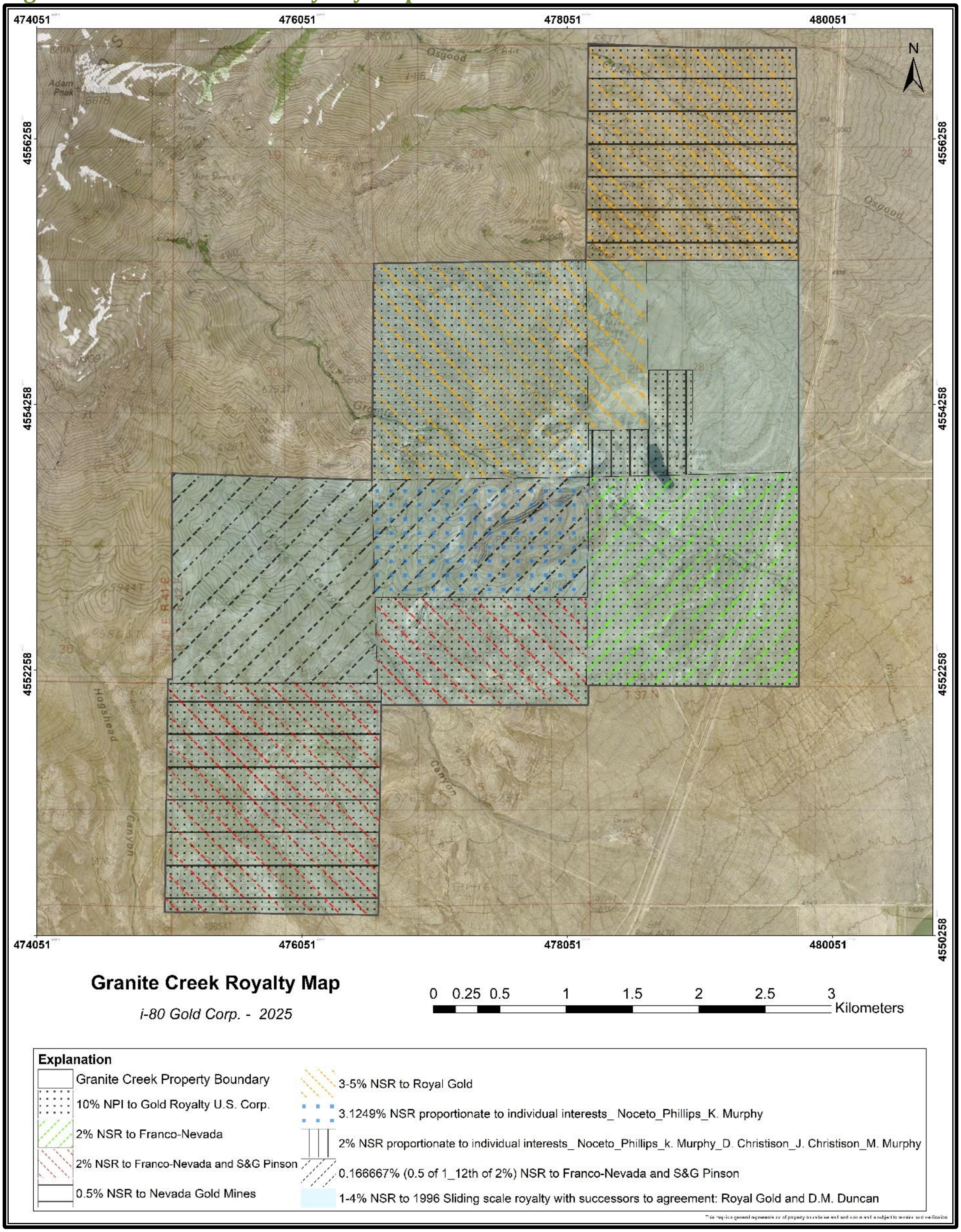
3.2.1 Royalties

Several royalties are in effect on various areas of the property. Table 3-4 lists the royalties and Figure 3-3 shows the royalty areas. Some royalties were retained by previous owners upon sale of the property while others were negotiated as lease agreements with claim holders.

Table 3-4 Granite Creek Royalties

Lessor/Grantor	Lease Type
Gold Royalty U.S. Corp.	10% NPI
Franco-Nevada	2% NSR
Franco-Nevada and S&G Pinson	2% NSR
proportionate to individual interests Noceto/Phillips/K. Murphy	3.1249% NSR
proportionate to individual interests Noceto/Phillips/K. Murphy/D. Christison/J. Christison/M. Murphy	2% NSR
0.166667% (0.5 of 1/12th of 2%) NSR to Franco-Nevada and S&G Pinson	0.166667% NSR
Nevada Gold Mines	0.5% NSR
Royal Gold	3-5% NSR
1996 Sliding scale royalty with successors to agreement: Royal Gold and D.M. Duncan	1-4% NSR

Figure 3-3 Granite Creek Royalty Map



3.3 Environmental Liabilities

The reclamation and closure cost for the Granite Creek Mine surface and underground is currently estimated to be approximately $3 million (Osgood Mining Company LLC [Osgood], 2023). A bond in the amount of approximately $2 million is held by the Bureau of Land Management (BLM) to address reclamation activities associated with the underground mine while an additional approximately $1 million bond is held for surface reclamation activities. There are no other known environmental liabilities associated with pre-Project operations (Osgood, 2024).

3.4 Permits/Licenses

The existing permits for the Granite Creek Mine are outlined in Section 17.3 and 17.4. Surface Reclamation Permit #0047 authorizes 448.6 acres of disturbance on private land and 490.4 acres of disturbance on public land while Underground Reclamation Permit #0242 authorizes 88.9 acres of disturbance on private land and 0.6 acres of disturbance on public land. These authorized disturbance acres allow Osgood to conduct the exploration, geotechnical and metallurgical field work to support the study work recommended in this Report as long as the amount of new surface disturbance remains less than that available under the existing authorizations.

Section 17 describes the permits required to execute the mine plan discussed in this report.

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

4.1 Accessibility

The project can be reached by traveling east on US Interstate Highway 80 from Winnemucca to the Golconda exit, about 15 miles, then following Nevada State Route 789 and the Getchell Mine Road northeast about 20.5 miles to the Granite Creek Mine access road. The last 4.5 miles is unpaved, well maintained gravel road. Areas of Route 789 are designated open range, and travelers must watch for cattle. The Granite Creek security facility is located about 0.2 miles west of the intersection with Route 789. Traveling from the east, the Golconda exit lies west of Battle Mountain about 36.5 miles on Interstate Hwy 80. The route from Battle Mountain passes i-80's Lone Tree facility, which lies south of the interstate about 19 miles west of Battle Mountain.

4.2 Climate

The climate in Humboldt County is typical of the high-desert environment. Typical summer temperatures average roughly 75°F with occasional day/night extremes of 105°F/40°F. Winter temperatures average roughly 30°F with occasional day/night extremes ranging 60°F/-10°F. Average annual precipitation is about 8 inches, the majority of which accumulates as snowfall during the winter months. Typical snow accumulation is roughly 3 inches on average at lower elevations, although occasional large storms may accumulate significantly more for short durations.

Mining operations are able to continue year-round with brief pauses for summer lightning storms or unusually heavy winter snowstorms.

4.3 Local Resources

The town of Winnemucca has a population of about 8,400. Basic services are available. Local mining districts have been active since the 1980's, and mining suppliers and contractors are accustomed to working in the area. Some experienced and general labor is available locally and from other small towns in the region. There are a number of mining operations in the region and as such there is always competition for employees.

4.4 Infrastructure

Existing infrastructure at the Project includes an office building, dry and warehouse facilities, and a lined stockpile area on the surface. Over 9,000 feet (2,743 meters) of underground workings have been completed, and four deep dewatering wells were drilled and cased, two of which are currently being operated.

Electrical infrastructure suitable for mine operations is installed, and two re-infiltration basins and associated pipelines have been constructed to re-infiltrate water produced in mine dewatering into the valley aquifer.

The mine is accessed through either of two portals, and dual egress has been established for most areas of the mine. Where dual egress is not possible, rescue chambers have been installed. Equipment is repaired in an underground mine shop. Air doors and a ventilation fan provide required air supply to the workings in compliance with Mine Safety and Health Administration (MSHA) standards.

Landline telephone and digital subscriber line service are available at the Project site. Cellular phone service is also available, but is dependent on the strength of receiving antennas, topography, and lines of sight.

4.5Physiography

The Project lies in the Basin and Range Province, a structural and physiographic province comprised of generally north to north-northeast trending, fault bounded mountain ranges separated by alluvial filled valleys.

The Project is located on the eastern flank of the Osgood Mountains. Topography is gentle to moderate at the Project, ranging from an elevation of 4,840 feet at the mine offices to 5,500 feet at the crest of the historic CX pit highwall, and rises steeply to the west over 3,800 feet to Adam Peak at the top of the Granite Creek drainage. Vegetation is typical of the high desert with sagebrush on the alluvial fans, and juniper on the mountain slopes.

5 History

The Property has been explored by a number of individuals and mining/exploration companies since the late 1930s. The original discovery on the Property was made by Clovis Pinson and Charles Ogee in the mid to late-1930s, but production did not occur until after World War II, when ore from the original discovery was shipped to and processed at the Getchell mine mill. In 1949 and 1950, total production from the Granite Creek mine amounted to approximately 10,000 short tons (9,071 tonnes) grading approximately 0.14 ounces per ton (opt) (4.8 g/t).

5.1 Historic Ownership

5.1.1 Cordex I Syndicate

The Property remained functionally dormant from 1950 until 1970, when an exploration group known as the Cordex I Syndicate (John Livermore, Peter Galli, Don Duncan, and Rayrock Resources) leased the Property from the Christison Family (descendants of Mr. Pinson and Property owners), on the strength of its similarity to the Getchell Property and structural position along the range-front fault zone bordering the Osgood Mountains. Following a surface mapping and sampling program in 1971, 17 reverse circulation (RC) drillholes were completed in and around the 1940s era Granite Creek Mine pit, confirming low- grade gold values. An 18th step-out hole encountered a 90-foot (27.4-meter) intercept of 0.17 opt (5.8 g/t) gold (Au). This intercept was interpreted as a subcropping extension of known mineralization northeast of the original pit and was the basis for delineation of what would become the “A” Zone at the Property, a 60-foot (18-meter) by 1,000-foot (305-meter) shear zone. During the late 1970s, the Cordex I Syndicate reorganized into a Nevada Partnership known as PMC, with Rayrock Resources as the Project operator, and began production at the Property.

Cordex Syndicate (Cordex), and its successor, PMC, explored the Property largely through mapping and geochemical sampling. There are three known mapping programs:

- A regional mapping program from Preble to Getchell by Pete Chapman in the late 1970s
- A 1:6000-scale mapping program of the Property in 1983
- A 1:2400-scale mapping program of the Pit areas through the active life of the mine

5.1.2 Pinson Mining Company

PMC began developing the A Pit in 1980 and produced gold the following year. Production from the B Pit began in 1982. Step-out drilling in 1982 to 1983 to the northeast of the A Zone intersected two more discrete zones: the C Zone extending east-northeast from the A Zone and the CX Zone extending northeast from the C Zone. Step-out drilling northeast of the CX Zone in 1984 located an apparently independent fault system (striking north-northwest), dipping steeply east that became the core of the Mag deposit, which went into production in 1987. PMC produced from the CX, CX West and Mag Pits into the mid to late 1990s, until a combination of falling gold prices and erratic mill feed forced closure of the oxide mill in early 1998. Continued attempts to expand production of oxide ore failed, and all active mining ceased on 28 January 1999 (McLachlan, et al., 2000). The project was officially closed in May 2000.

5.1.3 Homestake – Barrick

In the 1990s, Homestake and Barrick became 50/50 partners in PMC through purchase of minority interests (McLachlan, et al., 2000). Homestake and Barrick conducted an exploration program from 1996 to 2000 through PMC, expending some $12M on the Project. The joint venture explored the deeper feeder fault zones of the Property, exploring for a large, high-grade gold system that would support a refractory mill complex. This work, while successful in identifying gold mineralization with underground grades, failed to identify a deposit of sufficient size to be of development interest to Homestake or Barrick, and the partners concluded the exploration program. Subsequent to that decision, in 2003, Barrick acquired Homestake and drilled an additional three exploration drillholes.

5.1.4 Atna Resources Ltd. Earn-in and PMC Back-in

In August 2004, Atna acquired an option to earn 70% Joint Venture interest in the Property from PMC, a wholly owned subsidiary of Barrick, and commenced additional follow-up exploration and development of the Property. Atna completed its earn-in in 2006 and vested in its 70% interest in the Project after expending the required $12M in exploration and development expenditures. PMC elected to back-in to the Project and re-earn an additional 40% interest (bringing PMC's interest to 70% and Atna's to 30%) on 5 April 2006. PMC spent over $30M on the Project during the next three-year period and completed its "claw-back" in early 2009. Their work included surface and underground diamond core drilling, RC drilling, underground drifting, and surface infrastructure construction (rapid infiltration basins, mineralized material stockpile pad, underground electrical service upgrades, etc.). A new mining joint venture was formed in 2009 reflecting the Project's ownership, with PMC owning a 70% interest in the venture and Atna owning a 30% interest. PMC, as the majority interest owner, was the operator of the joint venture.

5.1.5 Atna 2011 – 2013 Underground Development

In September 2011, Atna negotiated the acquisition of PMC's 70% joint venture interest in the core property position at the Granite Creek Mine Project. The asset purchase and sale agreement included all right title and interest to the core property described above as well as an evergreen processing agreement with Barrick for the processing of underground refractory ores from Granite Creek Mine at Barrick's Goldstrike facilities.

Development of the Granite Creek Mine underground commenced in early 2012, and mine ramp-up began in late 2012. In total, 6,011 feet (1,832 meters) of primary and secondary development were completed during 2012 and 2013. The primary spiral ramp was driven to the 4530 level from the 4650 adit level, and both top cut and underhand ore mining occurred in three Ogee-zone stope blocks during development. Additional secondary access drifts were in progress when the mine was placed on care and maintenance to access the Range Front and Adams Peak mineral zones but were not completed prior to cessation of underground work. Mining was performed by contract miners using underground mining equipment owned by the contractor. Approximately 30,000 short tons (27,216 tonnes) of ore containing 7,900 oz of gold were mined and shipped to off-site processing facilities.

Work on the Project continued until June of 2013, when the mine was placed on care and maintenance. This decision was driven by a number of factors, including the steep decline in the gold prices in 2013.

In May 2014, the status of the underground mine was changed to an intermittent production status. Under this status, periodic mining of ores from stoping areas developed in 2013 was conducted to develop and test revised stoping methods for the underground and to prove mining economics at small production rates.

5.1.6 Osgood Mining Company LLC Acquisition

In 2016, OMC, a wholly owned subsidiary of Waterton Global Resources Management, acquired the Project. OMC completed numerous drillhole database compilation and verification campaigns, beginning with migration of the ATNA database to Maxwell Datashed Database software in 2017 and database verification and improvement efforts in 2018. In 2016, OMC, with an external consultant, completed a project-scale structural geology study that included surface and underground mapping, historical data review, and cross section interpretation aimed at defining the main structural architecture at Granite Creek Mine and developing exploration and resource drilling targets. This work formed the basis of an updated 3-dimensional (3D) litho-structural model that was used for the 2020 Mineral Resource estimation (AMC, 2020). From 2017 to 2018, OMC also completed an extensive drill material inventory and salvage program that secured the available drill core and RC chips on the property.

OMC continued to maintain compliance and keep all environmental permits for the site in good standing. This included performing permit-related sampling and reporting, as well as renewing permits. In addition, OMC performed regular inspections of the site. During the ownership period, OMC worked with the State of Nevada to close out a Water Pollution Control Permit for a reclaimed portion of the mine, reducing the overall compliance monitoring and reporting liabilities for the operator. In addition, OMC received approval from the State to remove portions of the reclaimed site from the bond.

In addition to these geology and compliance activities, OMC continued to maintain and improve site infrastructure, including a third-party review of hydrology and dewatering requirements that resulted in the replacement of pumps (2019) and the upgrading of two dewatering well process controls. Rapid infiltration basins have been maintained as needed, with water flows being tracked and monitored.

5.1.7 i-80

In April 2021, i-80 Gold Corp was created as a spinout of Premier Gold Mines Limited's assets located in Nevada concurrent to Equinox Gold Corp's acquisition of most of the balance of Premier Gold Mines Limited's assets in Canada and Mexico. The same month, the newly created i-80 Gold Corp completed acquisition of OMC from Waterton Global Resources Management. In May 2021, additional land was purchased by i-80, further increasing the size and ownership in the land package.

In June 2021, Section 31 fee land was acquired by Premier Gold Mines USA, Inc. from Seven Dot Cattle Co., LLC., as well as Christison interest in Section 28 fee lands, and in the PINSON unpatented claims. (Note these are still held in the Premier name, not Osgood.) In 2022, Section 21 fee land T. 38 N., R. 42 E. was acquired by Osgood Mining Company from Nevada Gold Mines, and lessee interest on the BEE DEE unpatented claims in Section 6, T. 37 N., R. 42 E.

5.2 Historical Mine Production

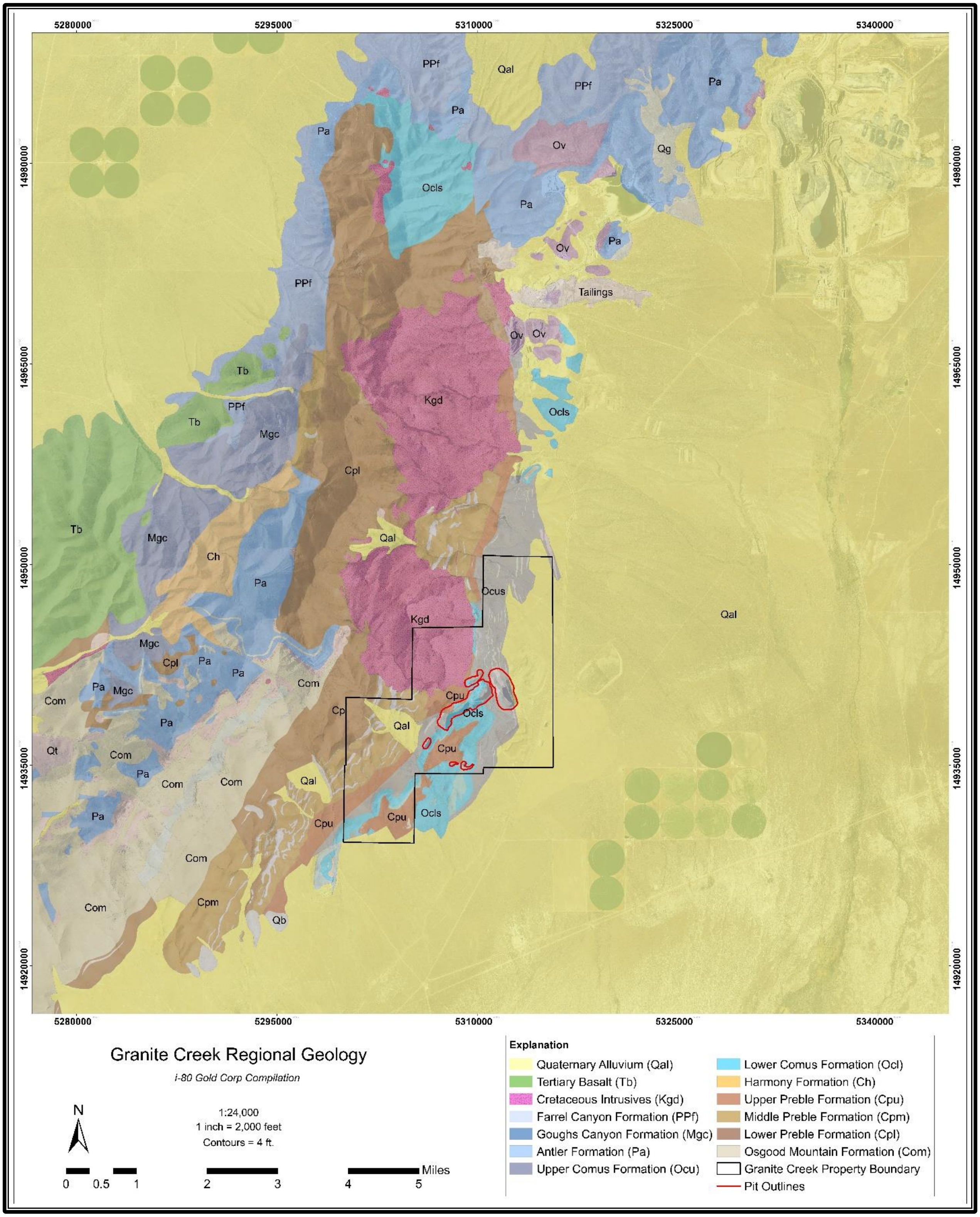
Historically, the Granite Creek Mine Project, with small additions from the nearby Preble and Kramer Hill mines, was credited with gold production in excess of 1 million ounces and less than 100,000 oz of silver (Tingley, 1998). PMC independently compiled a record of production and credited the Granite Creek Mine Property with production of 986,000 oz of gold through 1999.

6 Geologic Setting, Mineralization and Deposit

6.1 Regional Geology

The Property is located on the eastern flank of the Osgood Mountains within the Basin and Range tectonic province of northern Nevada. The Granite Creek Mine, together with the Preble, Getchell, Turquoise Ridge, and Twin Creeks mines, are on what is referred to as the Getchell gold trend (Getchell trend). The main Getchell trend generally strikes northeast-southwest and has been cross-cut by secondary north-south and northwest-southeast-trending structures. The deposits are hosted in Paleozoic marine sedimentary rocks. The rocks are exposed in the Osgood Mountains and have been complexly thrust faulted (Hotz, et al., 1964) and intruded by the Cretaceous-aged (92 Ma) (Silberman, et al., 1974) Osgood Mountains granodiorite stock. These units are unconformably overlain by Miocene volcanic rocks. Figure 6-1 is a regional geologic map of the Osgood Range.

Figure 6-1 Regional Geologic Map of a Portion of the Osgood Mountains including Granite Creek



The Osgood Mountains Range is underlain by Cambrian Osgood Mountain Quartzite, Cambrian Preble Formation, Ordovician "Comus" Formation and the "upper plate" Valmy Formation. These units are unconformably overlain by the Permian Etchart Formation (Antler Peak Equivalent) of the Roberts Mountains overlap assemblage, and by the Triassic Golconda allochthon. These uppermost units form a belt of outcrops flanking the western and northern sides of the Osgood Range. These rocks have been intruded by the Cretaceous-aged Osgood Mountains granodiorite stock, which forms the core of the Osgood Mountains. Stratigraphy throughout the Osgood Mountains plunges north (Chevillon, et al., 2000). A significant thermal metamorphic aureole surrounds the stock. At least four Paleozoic units, defined by structure, lithology, and age comprise the Osgood Mountains (McLachlan, et al., 2000). These include the:

- Autochthonous Cambrian Osgood Mountains Quartzite and Preble Formation and Cambrian to Ordovician Comus Formation
- Allochthonous Ordovician Valmy Formation, part of the Roberts Mountains allochthon
- Antler overlap sequence including the Mississippian Goughs Canyon Formation, Pennsylvanian Battle Formation, and Pennsylvanian-Permian Etchart Limestone
- Allochthonous Pennsylvanian-Permian Farrel Canyon Formation, part of the Golconda allochthon

The autochthonous Cambrian-Ordovician package has been described by Jones (1991, cited in McLachlan et al. 2000) and is comprised of the Osgood Mountains Quartzite, Preble Formation, and Comus Formation. All of these units have undergone regional metamorphism and intense, northwest-directed folding (McLachlan, et al., 2000). At the Getchell Project, these two units are folded together to form the northwest-verging Pinson anticline. The Comus and Preble Formations show distinct facies changes across the district. These units at Turquoise Ridge and Twin Creeks contain tuffs, pillow basalts, and mafic sills, none of which are present in the same units at Granite Creek.

The Roberts Mountains allochthon described by Stenger et al. (1998) is exposed at the Turquoise Ridge and Twin Creeks mines where it has been mapped as the Valmy Formation. The Roberts Mountains allochthon is composed of a thick (>980-foot [299-meter]) sequence of mid-ocean ridge basalts and intercalated pelagic sediments that have been thrust over the Twin Creeks member of the Comus Formation (Stenger, et al., 1998). This sequence has not been identified at the Granite Creek Mine but was likely present and eroded prior to the present day.

The Antler overlap sequence in the Osgood Mountains consists of the Pennsylvanian Battle Formation, and Pennsylvanian-Permian Etchart and Adam Peak formations (McLachlan, et al., 2000). The Battle conglomerate consists of cobbles and pebbles of quartzite. The Etchart lies conformably on the Battle and consists of calcareous sandstone underlying fossiliferous limestone. South of the Getchell Project, the Battle and Etchart lie unconformably on the Preble Formation and Osgood Mountain Quartzite (McLachlan, et al., 2000). These units are not present at the Granite Creek Mine Project.

The Golconda allochthon comprises the Mississippian Goughs Canyon and the Pennsylvanian Permian Farrel Canyon formations present along the northwest flank of the Osgood Mountains. The thrust strikes north to northeast from the central part of the range to the Dry Hills in the north (McLachlan, et al., 2000). These units are not present at the Granite Creek Mine Project.

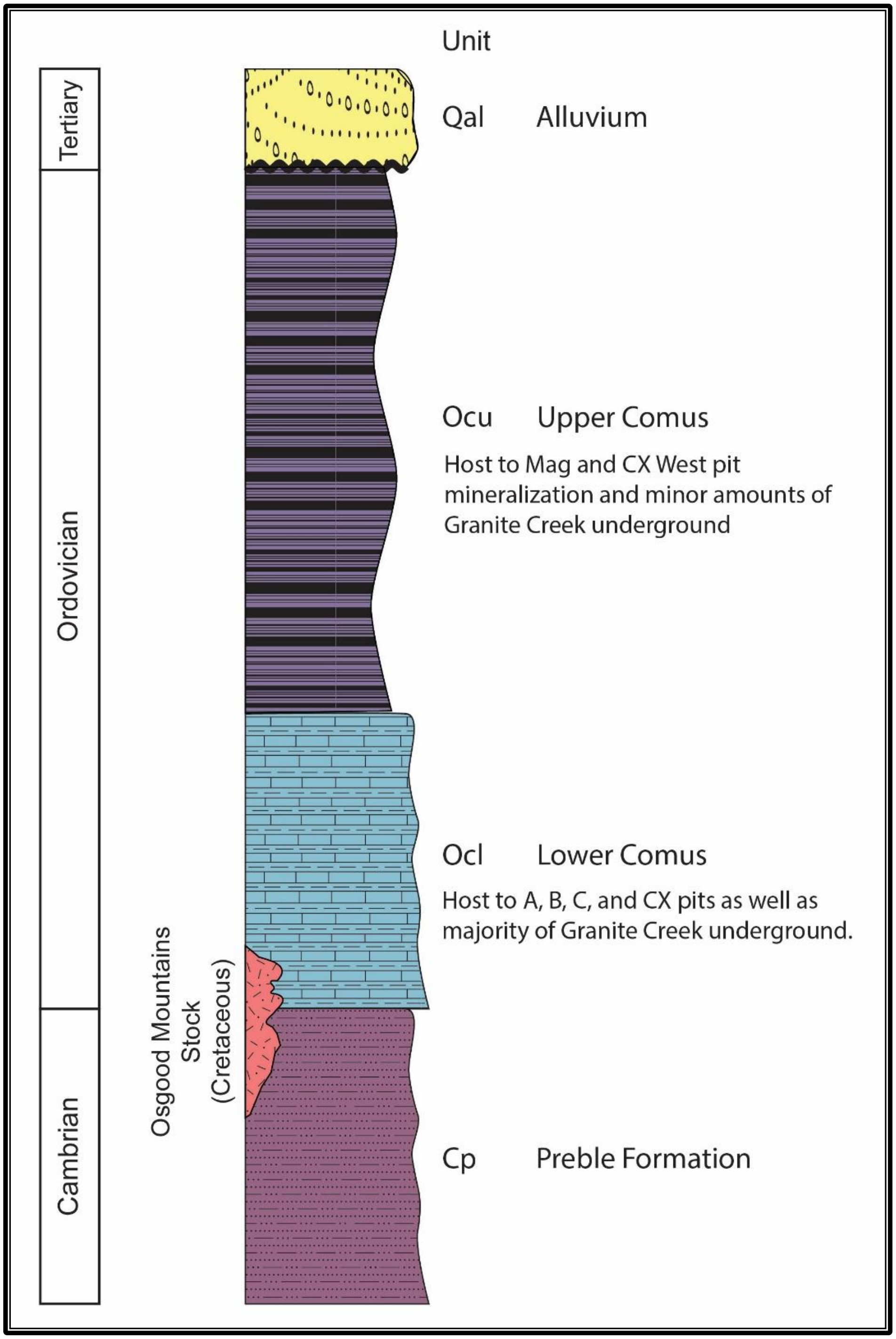
6.2Local and Property Geology

The geology throughout the Osgood Mountains is typified by folded Cambrian to Ordovician sedimentary rocks that have been intruded by Cretaceous stocks, which have been cross-cut by later high-angle structural deformation. Hotz and Willden (Geology and Mineral Deposits of the Osgood Mountains Quadrangle, Humboldt County, Nevada: U.S. Geological Survey Professional Paper 431, 1964) suggest the high angle faulting is related to the Basin and Range extension. The older rocks are overlain by Miocene andesitic basalt and the surrounding fault bounded basins are filled with quaternary alluvial (Qal) gravel. The Osgood Mountains have a general northeast trend, although, at a structural hinge in the vicinity of the Granite Creek Mine, the east flank of the range rotates and trends north towards the Getchell mine. Gold mineralization is primarily hosted by fine-grained marine sedimentary rocks that overlie a large stock of Cretaceous granodiorite.

Throughout the district Cambrian to Ordovician siliciclastic and carbonate rocks have been intruded by the Cretaceous Osgood Mountains granodiorite, resulting in the formation of large, metamorphosed aureoles with development of several tungsten-bearing skarns. The lowest stratigraphic units recognized locally are the Cambrian Osgood Mountains Quartzite, which is overlain by phyllitic shales, limestone interbeds, and various hornfelsed sedimentary rocks of the Preble Formation. The Preble is overlain by Ordovician sedimentary rocks of the Comus Formation, both of which have been folded into a broad, north-plunging anticline. The west flank of the anticline has been over-thrust by the Ordovician Valmy Formation, which consists of deep-water siliceous shales and cherts. The core of the anticline and scattered localities along the east side of the Osgood Mountains are unconformably overlain or in fault contact with sandstones and conglomerates of the Battle Formation and limestones of the Etchart Formation. The Golconda and Humboldt thrusts displaced Mississippian volcanics and Pennsylvanian shales eastward along the northwest and southern flanks of the Osgood Mountains. Extension during the Tertiary resulted in outflows of Miocene rhyolitic tuffs, basalts, andesite flows, and younger Quaternary basalt flows.

Gold mineralization at the Property is primarily hosted in the Comus Formation, as shown in Figure 6-2.

Figure 6-2 Granite Creek Stratigraphic Column



The stratigraphy of the Osgood Mountains from youngest to oldest is:

- Quaternary: Qal / Qb – Alluvium and basalt
- Tertiary:

  - Tba – Andesite and basalt flows. Dark green to black aphanitic and weakly porphyritic flows, flow breccia
  - Tr – Rhyolitic tuffs. Pumice, welded, reworked, tan to white
  - Tcg – Chert, shale, rhyolite clasts in a sandy matrix
  - Tbi - Dacite and andesite dikes
- Cretaceous: Kgd – Granodiorite, quartz diorite. Equigranular, medium grain intergrowths of feldspar, quartz, biotite, and hornblende.
- Permian / Pennsylvanian:
  - PPmh – Havallah Formation. Interbedded sandstone, chert, shale, siltstone with minor volcanic flows and pyroclastics. Chert, interbedded with sandstone composes up to 50% of the unit.
  - PPe – Etchart Limestone. Limestone, sandy limestone, dolomite. Lower portion is sandy limestone with local pebble conglomerate. Upper portion is pure limestone with interbedded dolomite and sandy dolomite. Minor calcareous shale.
- Pennsylvanian: Pb – Battle Formation. Poorly bedded, poorly sorted boulder and pebble conglomerate with coarse-grained sandstone and minor limestone clasts composed of Osgood Quartzite and chert in a shaley to sandy matrix.
- Ordovician:
  - Ov – Valmy Formation – Chert, shale, quartzite, volcanics (greenstone). Interbedded chert and shale with quartzite greenstone bed on the east side of the Osgood Mountains. Quartzite is dominant in the lower portion and chert and shale in upper portion.
  - Oc – Comus Formation – Upper unit composed of black argillite generally lacking bedding. Lower units composed of alternating thin to medium beds of limestone and argillite. In the Twin Creeks mine area, pillow basalts, mafic igneous sills and dikes exist within the sequence. Mafic igneous rocks are not present in the Comus Formation at the Granite Creek Mine.
- Cambrian:
  - Cp – Preble Formation – Dominantly sandstone phyllitic shale. Maroon, light olive, and brown. Upper part contains thin interbeds of limestone rhythmically bedded with shale.
  - Com – Osgood Mountain Quartzite. White, gray, light brown, purple-brown to green-gray, medium to thick-bedded quartzite. Impure quartzite, silty sandstone, phyllitic shale.

The Granite Creek Mine is located on the eastern flank of a large Cretaceous granodiorite stock that forms the southern core of the Osgood Mountains. Rocks adjacent to the eastern side of the stock have a general east dip and strike sub-parallel to the trend of the Osgood Mountains. The oldest units exposed against the granodiorite are Cambrian Preble sandstone, phyllitic shales, and interbedded limestones all of which are often metamorphosed. Overlying the Preble is a thick package limestone and argillite of the Ordovician Comus Formation. The Lower Comus is composed of thin to medium interbeds of limestone and argillite. The Upper Comus consists of black argillite typically lacking bedding.

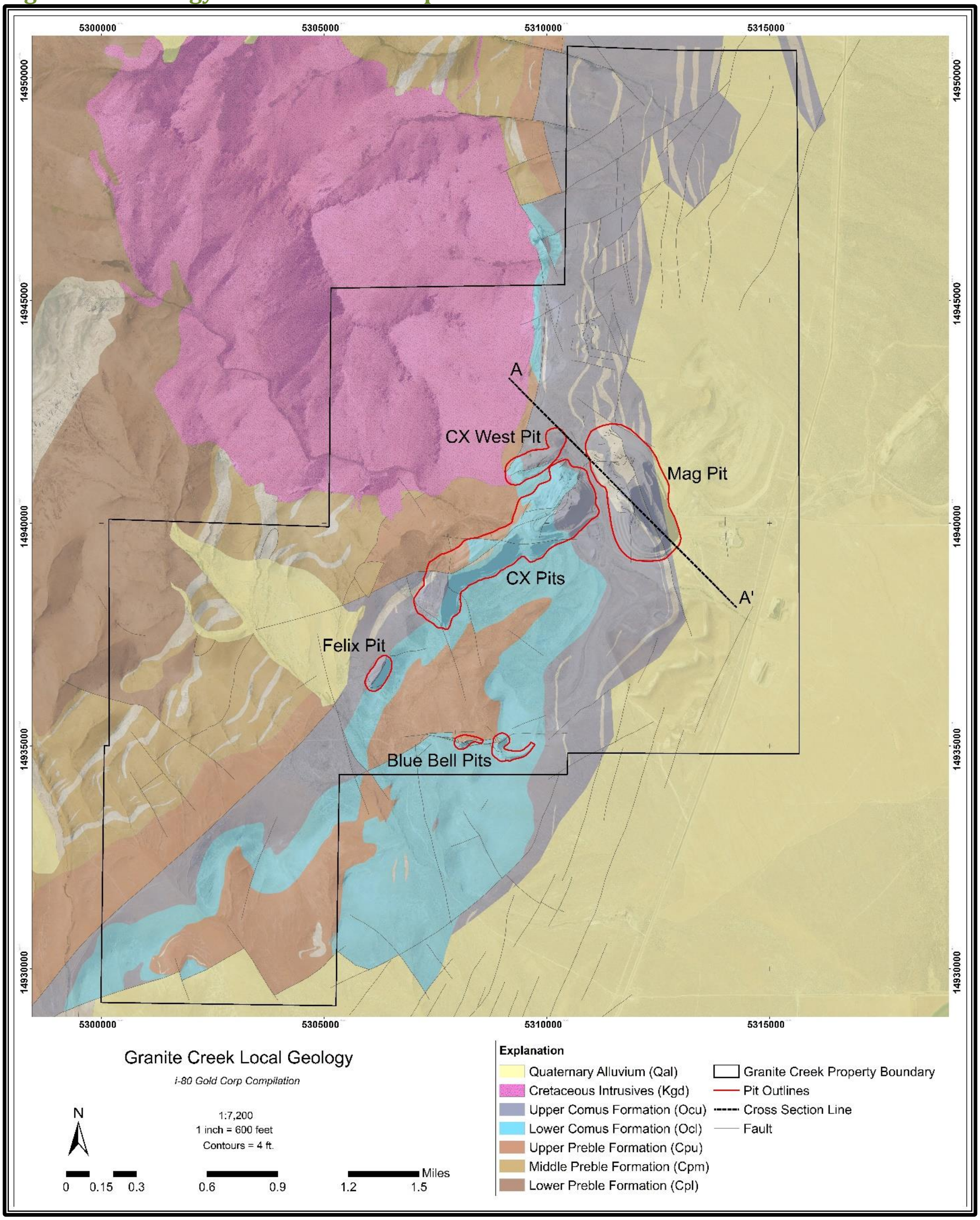
A Cretaceous aged (90 – 92 million years [Ma]) (Silberman, Berger, & Koski, 1974) granodiorite stock intrudes the Paleozoic section in the southern half of the Osgood Mountains. Emplacement of the stock resulted in the formation of an irregular contact metamorphic aureole, which extends as much as 10,000 feet (3,048 meters) from the intrusive contact. The metamorphic event resulted in the formation of maroon-colored, biotite-cordierite- hornfels in the Preble Formation and chiastolite hornfels in the Upper Comus Formation within much of the Property area (McLachlan, Struhsacker, & Thompson, 2000). In addition, carbonate rocks were metamorphosed to marble and calc-silicates (wollastonite, garnet, diopside, and vesuvianite). Several tungsten-bearing skarn deposits were also formed along the margins of the stock (Silberman, Berger, & Koski, 1974). Two tungsten skarns are on the Property.

Outcrop mapping and historic drilling has revealed the presence of extensive folding of the Paleozoic section in the Osgood Mountains. The most prominent of these folds is the Pinson Anticline. The fold is northeast-plunging and northwest-verging and extends for a distance of approximately three miles southwest from the Granite Creek Mine (McLachlan, Struhsacker, & Thompson, 2000). Numerous parasitic folds have also been noted along the limbs of the anticline. Where exposed, the Pinson Anticline is cored by the Cambrian Preble Formation and flanked on the northwest and southeast by sediments of the Ordovician Comus Formation.

Mineralization on the Property exhibits strong structural control. A wide variety of mineralized structural orientations have been documented. The most important structural feature on the Property is the network of faults that border the escarpment marking the southern and eastern edge of the Osgood granodiorite (Sim, 2005). This fault system has been variably interpreted as a single master fault (RFF) (McLachlan, Struhsacker, & Thompson, 2000) that curves around the stock, or more likely, a network of shorter, straighter segments that collectively accommodate several thousand feet of displacement while making a 50° bend around the southeast corner of the stock (Sim, 2005). The fault system can be divided into three structural and stratigraphically mineralized zones, with each mineralized zone defined by one or more major structural elements. These are referred to as the Rangefront, CX, and Mag Zones. Sedimentary rocks in the vicinity of this system generally dip steeply (easterly) away from the contacts of the granodiorite (Sim, 2005).

Figure 6-3 Geology and Structural Map shows the structural and geology map of the Property with the mined-out pits outlined for reference.

Figure 6-3 Geology and Structural Map



In addition to this large-scale fault system, there are numerous northwest and a few east-west structures that have been identified by past mapping and drilling (McLachlan, Struhsacker, & Thompson, 2000). In general, these appear to be mostly older than, and truncated by, the main system. Some of these faults have been re-activated and disrupt the continuity of the main Granite Creek system (Sim, 2005).

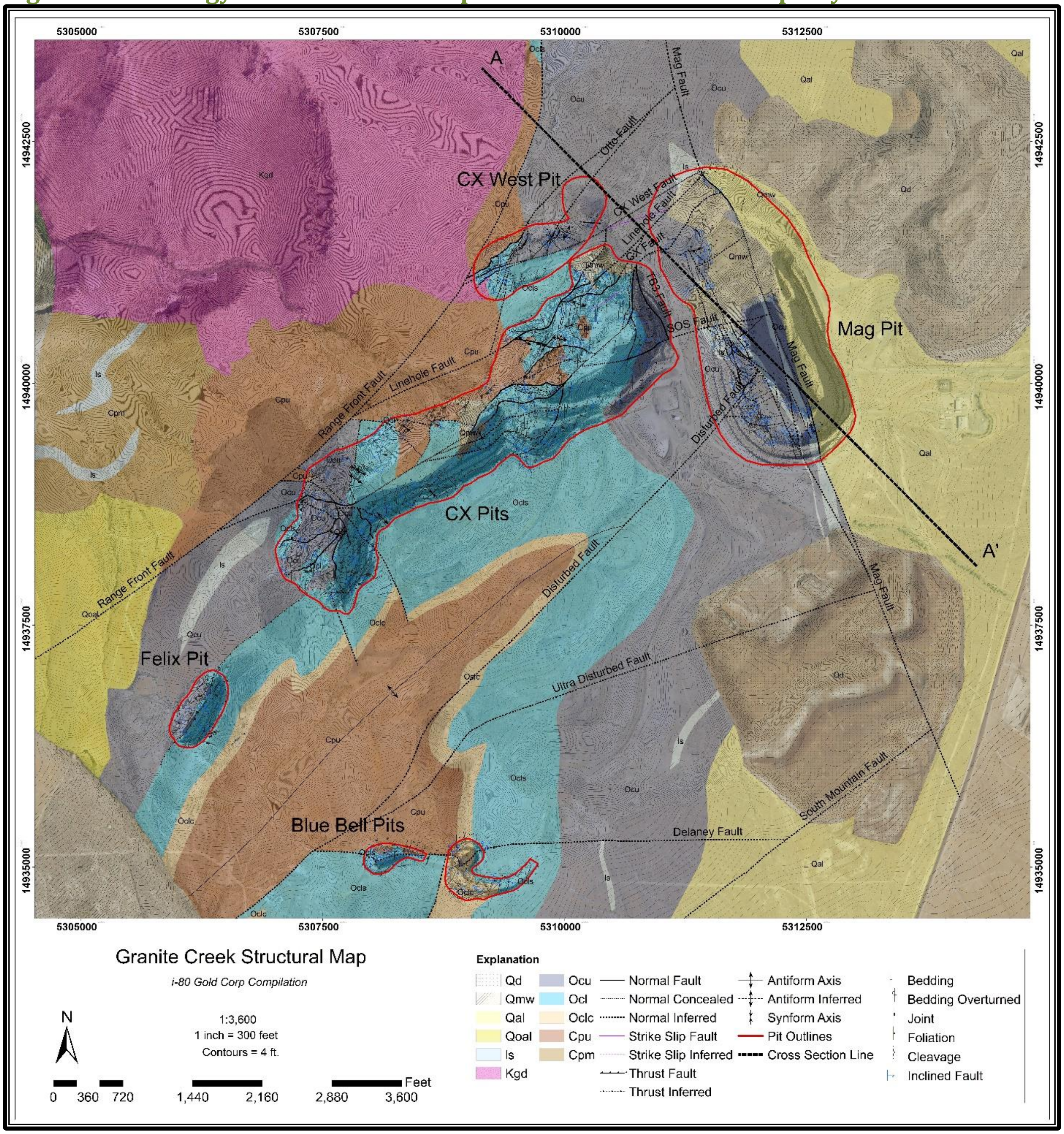
6.3 Structural Framework

6.3.1 Structural Overview

In 2022, i-80 Gold Corp. created an updated comprehensive geologic model for the Granite Creek Mine Project. This work included surface and underground mapping, structural analyses, geologic interpretation, and the creation of a complete 3D geologic model using Leapfrog Geo. The 3D model utilizes all available data including: drillholes (lithology, assays, etc.), surface mapping (pit and regional), underground mapping (current and historic), televiewer data (interpreted in-house), and structural analyses (stereonet, cross section, etc.).

Structure at the Granite Creek Project is highly complex and indicative of multiple deformation events. Regional deformation events, such as the Antler, Sonoma, and Elko orogenies, are likely responsible for the numerous overprinted fabrics and compressional structures observed at Granite Creek. These compressional structures appear to have been dissected and/or reactivated by subsequent Basin and Range extension. In 2016, Robert Leonardson generated a geologic model for the property that describes a west-northwest-verging imbricate thrust system deflected around the Osgood Stock. i-80 Gold Corp recognizes this thrust system comprised of the Rangefront, Adam Peak, Otto, and CX faults but interprets most of the compression and westward transport along these faults to have pre-dated the emplacement of the stock. The main structural element on the property is the Rangefront fault with a variable strike of 045º in the south to 010º in the north, and a dip of 60º near-surface that shallows with depth. The Adam Peak and Otto faults act as hanging wall splays off the northern extent of the Rangefront fault. The CX fault has a strike of 050º with a 55º near-surface dip that shallows with depth. Bedding generally dips steeply to the northeast, however fold geometries in the hanging wall of the Rangefront fault are complex and polyphase resulting in non-cylindrical interference folds. A property scale northeast trending doubly plunging upright anticline in the hanging wall of the CX fault is interpreted as a fault propagation fold related to west-northwestward compression along the Rangefront and CX faults. The current fault and fold geometries are interpreted to be the result of displacement and top-to-the-east rotation of the country rock by the emplacement of the Osgood Stock. Rotating the main thrust faults from their current ~60º dips back to an assumed syn-compressional dip of ~30º rotates the upright anticline's axial plane orientation to ~045º, 60º, resulting in typically observed compressional geometries. Reactivation of these thrust faults in their current orientation during Basin and Range extension has resulted in normal-sense down-to-the-east displacement across the property. The Mag fault system on the eastern portion of the property trends 335° and appears to be a younger fault system associated with Tertiary extension. The two main faults of the system are the Mag and Mag West faults that form a horst, with the Mag fault having significant down-to-the-east displacement.

Figure 6-4 Geology and Structural Map of the Granite Creek Property



The following subsections give details on significant structural features observed across the Property. Pit structural mapping by Chadwick (2002) collected orientation data and cross-cutting relationships.

7.3.2 Faults

Rangefront Zone

The Rangefront Zone (RFZ) is a northeast trending fault zone that forms a broad persistent zone of shearing and brecciation along the RFF that bounds the eastern margin of the Osgood Mountains. The RFZ involves the entire stratigraphic sequence at the Property, including the Cambrian Preble, Ordovician Comus, and Cretaceous granodiorite.

Rangefront Fault

The RFF is a prominent 010° to 045° striking normal fault that defines the eastern front of the Osgood Mountains. For much of its length, the fault juxtaposes the Comus Formation in the hanging wall against the Preble Formation in the footwall (McLachlan, et al., 2000). The fault originated as a west-verging thrust fault and has since been reactivated as a down-to-the-east normal fault of significant yet unknown displacement. The hanging wall zone is intensely brecciated and pervasively argillized. The fault has a near-surface dip of 60° that shallows with depth. The hanging wall delineates the lower boundary of the RFZ.

Adam Peak Fault

The Adam Peak fault is a 048° striking hanging wall splay off the Rangefront fault with a near-surface dip of 72° that shallows with depth. The fault has been reactivated as a normal fault with unknown displacement. The footwall delineates the upper boundary of the RFZ.

Otto Fault

The Otto fault is a 040° striking hanging wall splay off the Rangefront fault with a near-surface dip of 80° that shallows with depth. The fault has been reactivated as a normal fault with unknown displacement. The fault is defined by a zone of discrete anastomosing splays. The on-strike and down-dip extent of the Otto fault defines the South Pacific Zone (SPZ) fault system.

CX West Fault

The CX West fault is a younger offsetting fault with a strike of 245° and dip of 70°. This normal fault has a displacement of approximately 150 ft and offsets stratigraphy as well as the Adam Peak and Otto faults in a down-to-the-northwest direction.

Ogee Fault

The Ogee fault is a 10-100 ft wide fault zone with a strike of 070° and dip of 85° that juxtaposes the Upper and Lower Comus formations. The fault is defined by an anastomosing system of splays along strike and down dip. The fault is interpreted as a long-lived accommodation zone that has experienced multiple phases of reactivation. Recent underground mapping data indicates recent right-lateral oblique-normal motion of unknown displacement.

Linehole Fault

The Linehole fault is a through-going southwest trending normal fault that dips 85° to the northwest. The fault has displacement of approximately 100 ft and appears to act as a structural and mineralization boundary on the property.

LH Fault

The LH fault is a splay off the Linehole fault with a strike of 030° and dip of 80°. The fault has normal sense offset with approximately 50 ft of displacement. The intersection of the LH and Ogee faults is one of the most important and prolific structural intersections on the property.

CX Fault

The CX Fault is a complex zone of brittle fracturing that juxtaposes Upper Comus argillite against limestone beds of the Lower Comus. The fault strikes approximately 035° to 045° and dips 55° to 65° southeast Chadwick (2002), as shown in Figure 6-5. The fault originated as a west-verging thrust and has since been reactivated as a normal fault with an unknown amount of down-to-the-southeast displacement.

SPZ Fault System

The SPZ fault system is comprised of the along-strike and down-dip extent of the Otto fault and its associated splays (SPZ and Otto suite of faults). The zone trends northeast with a dip of approximately 50° to the southeast. The Upper and Lower Comus formations are often juxtaposed along the Otto suite of faults in this zone. However, the defining characteristic of this fault zone is a transition from weakly metamorphosed rock in the hanging-wall to more strongly metamorphosed rock in the footwall.

Mag Fault

The Mag fault is a younger through-going normal fault with a strike of 340° and dip of 75°. The fault appears to be related to Basin and Range extension. Displacement is unknown but interpreted to be significant in a down-to-the-east direction.

Mag Fault System

The Mag Fault system is a northwest trending suite of brittle faults that define the Mag pit. The two main faults, the east-northeast dipping Mag fault and the west-southwest dipping Mag West fault, form a horst block within which mineralization is concentrated.

6.4Mineralization

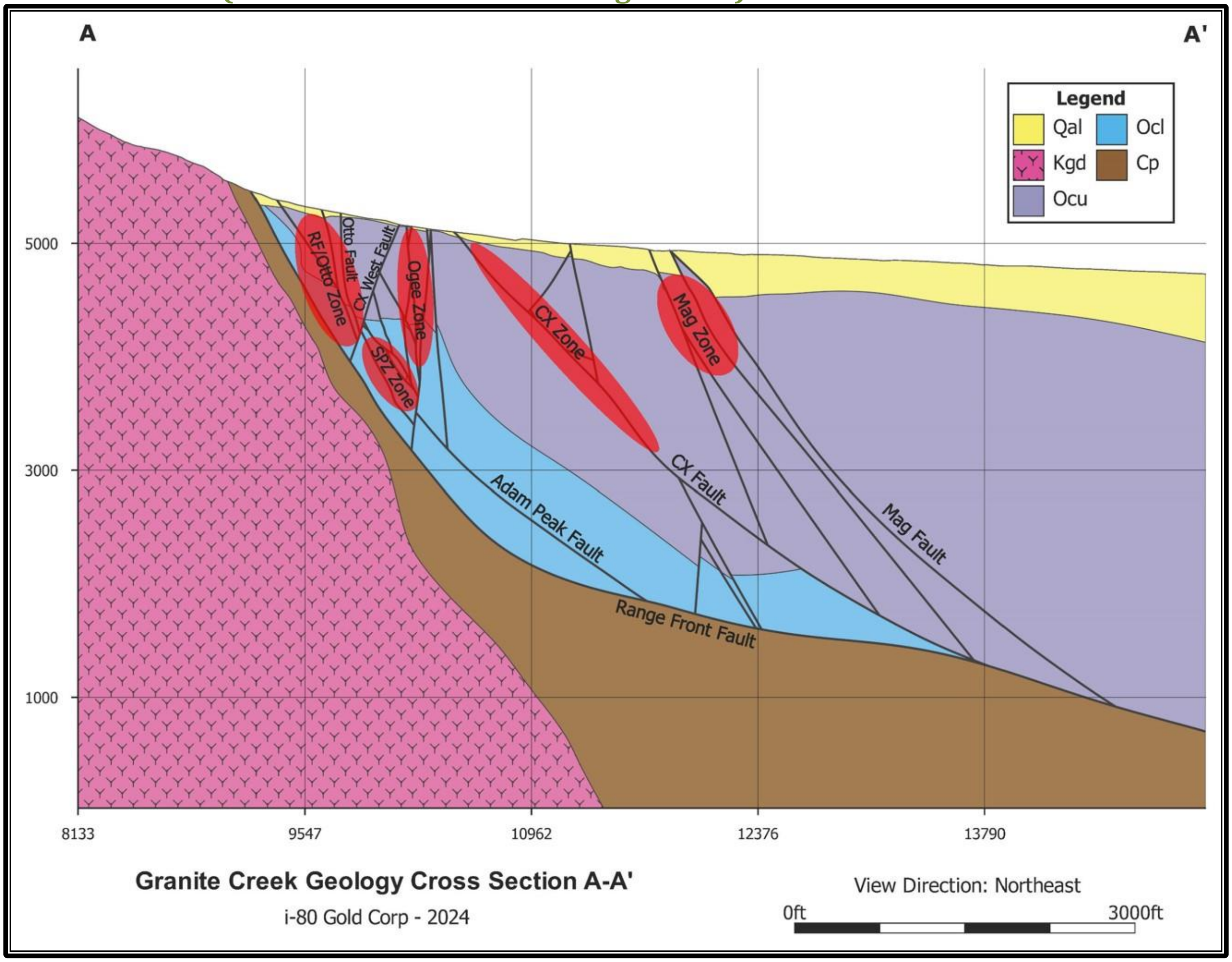
Mineralization at Granite Creek is structurally controlled. Faults are the primary control of mineralization, especially in high-grade underground zones. Lithologic contacts, bedding, and folds also play an important role, especially in near surface (open pit) mineralization. High-grade mineralized zones are moderately continuous along faults with the most prolific zones occurring at structural intersections. Gold mineralization is found within pyrite that consists of two stages of development, an early non-ore- pyrite stage and a gold-bearing arsenian pyrite stage (Ridgley, Edmondo, MacKerrow, & Stanley, 2005). Megascopically, the gold-bearing pyrite is typically dull brassy to black in color and very fine-grained. Pyrite may also be associated with remobilized carbon, imparting a “sooty” appearance to the pyrite. Gold is primarily contained in pyrite as microscopic inclusions or found in solid solution within arsenian-pyrite rims around fine pyrite grains (Wallace & Wittkopp, 1983; Foster, Gold in Arsenian Framboidal Pyrite in Deep CX Core Hole DDH-1541: Unpublished Pinson Mining Company Report, 1994; Ridgley, Edmondo, MacKerrow, & Stanley, 2005). Gold mineralization shows a correlation with arsenic, antimony, mercury, and thallium.

Gold mineralization at the Property is primarily hosted by the Upper and Lower Comus Formations, which consist of argillite and interbedded argillite and limestone, respectively. The Upper Comus is the primary host lithology in the Mag Zone and currently is host to the majority of surface resources at the Pinson (Granite Creek) deposit (Gustavson, 2012). The Upper Comus is also locally mineralized within the B, C, CX, CX-West, and portions of the RFZ. The Lower Comus hosts the majority of the high-grade underground resources. In areas proximal to the Osgood Mountains stock including the underground resources, most of the host rock has been metamorphosed. In these areas argillite has been metamorphosed to hornfels with limestone altered to garnet, pyroxene, wollastonite, and marble. Higher gold grades are typically located in these metamorphosed rocks along fault zones due to the lack of wall rock permeability.

Rocks of the Preble Formation are a poor host for gold mineralization but do contain localized gold concentrations where they have been brecciated and are adjacent to major fluid conduits.

Figure 6-5 is a representative cross section of the property illustrating the geometry of mineralization controlling structural features, such as faults and the lithologic contact between the Upper and Lower Comus Formation.

Figure 6-5 Cross-section A-A' looking Northeast showing Structure, Lithology and Mineralization (Section Line is shown on Figure 6-4)



Oxide mineralization includes pervasive limonite and hematite, along with other iron and arsenic oxides. Oxidation is extensive in the Ogee Zone and CX Fault system, occurring along the entire length of the zones and penetrating to a depth of 1,500 feet (457 meters). Within the RFF system, oxidation is more variable. In some fault and shear zones, oxidation may be present to depths of 1,800 feet (549 meters), whereas in others it may only reach to depths of < 500 feet (152 meters) (Ridgley, Edmondo, MacKerrow, & Stanley, 2005).

6.4.1 Mag Pit Mineralization

Gold mineralization within the Mag Pit is hosted by argillite of the Upper Comus Formation. The mineralized zone has a north-northwest orientation, sub-parallel to the Mag Fault, dips to the east-northeast and plunges to the south-southeast (McLachlan, Struhsacker, & Thompson, 2000). The mineralized body is tabular, has a strike length of approximately 4,000 feet (1,219 meters), varies from 200 to 400 feet (61 to 122 meters) in width, and has an average down dip extent of 450 feet (137 meters) (Kretschmer, 1985; Foster & Kretschmer, Geology of the Mag Deposit, Pinson Mine, Humboldt County, Nevada, 1991). Bedding within the Upper Comus Formation is the primary control of mineralization. High-grade zones in the southern portion of the deposit are localized along northwest trending faults within the Mag horst block. Mineralization within the Mag deposit is more disseminated and lower grade than the Rangefront, CX, and Ogee zones (Gustavson, 2012). Gold mineralization is spatially associated with decarbonatization, kaolinization, white kaolinite fracture filling, silicification, and quartz veinlets (McLachlan, Struhsacker, & Thompson, 2000).

6.4.2 Underground Mineralized Zones

Multiple areas of high-grade gold mineralization at the Granite Creek deposit are amenable to underground mining methods, as shown by previous operators. These include the Rangefront, Otto-Adam Peak, Ogee CX, and South Pacific Zones. All of these zones show strong structural control.

6.4.3 Rangefront Zone

The RFZ consists of pervasive argillization and decarbonatization with intense brecciation along the lower bounding RFF. Structural/mineralization trends are difficult to discern in this zone with mineralization occurring as discontinuous amorphous bodies within the Comus Formation. High-grade zones are concentrated in the Lower Comus with anomalous mineralization present in the Preble Formation, proximal to the RFF. Silicification is minor, with calcite veins occurring along the margins of fault zones. Structural and dissolution breccias that occur along bedding and structural intersections within the Lower Comus Formation are particularly receptive to mineralization. The zone has a strike length of approximately 950 feet (290 meters), a down dip extent of 1,100 feet (335 meters), and an average width of 100 feet (30 meters).

6.4.3.1 Otto-Adam Peak Zone

The Otto-Adam Peak zone is defined by the Otto and Adam Peak faults and their associated splays. The zone trends northeast, dips southeast, plunges to east-northeast and is offset down-dip by the CX West fault. The zone is pervasively argillized with intense brecciation occurring along faults. Mineralization is moderately continuous, controlled by a network of discrete anastomosing faults and splays within the Lower Comus Formation. High grade mineralization occurs along fault intersections throughout the zone. The mineralization has a strike length of approximately 500 feet (152 meters), a vertical extent of 700 feet (213 meters), and an average width of 75 feet (23 meters).

6.4.3.2 Ogee Zone

The Ogee zone is an east-northeast trending near vertical mineralized zone controlled by the Ogee Fault and associated splays. The zone is argillized, decarbonatized, and intensely brecciated along faults. The upper portion, defined by the intersection of the Ogee Fault and the contact between the Upper and Lower Comus Formation, plunges to the east-northeast at 55°. The lower portion is near vertical and controlled by faults and structural intersections. The upper portion is strongly oxidized while the lower portion is mostly oxidized but contains more comingled sulfide. The mineralization has a strike length of 400 feet (122 meters), a vertical extent of 1,500 feet (457 meters), and an average width of 75 feet (23 meters).

6.4.4 CX Zone

The CX zone consists of both near-surface (open pit) and higher-grade underground mineralization. The lower-grade open pit mineralization is controlled by the through-going CX fault and its associated hanging wall and footwall splays. Mineralization is discontinuous and associated with pervasive argillization and decarbonatization within structural and dissolution breccias in the Lower Comus Formation. The near-surface portion of the zone has a strike length of 3,500 feet (1,066 meters), a down dip extent of 400 feet (122 meters), and an average width of 75 feet (23 meters). The higher-grade underground portion of the mineralization is more tightly structurally controlled along the down-dip section of the CX Fault with an average width of 40 feet (12 meters). The underground portion of the zone has a strike length of 1,000 feet (305 meters) and a down dip extent of 1,200 feet (366 meters).

6.4.5 South Pacific Zone

The South Pacific Zone (SPZ) is a northeast trending and southeast dipping zone of high-grade fault-bound mineralization with a northeast plunge of 45°. The mineralization is controlled by the along-strike and down-dip extent of the Otto fault. The zone is defined by a suite of northeast striking moderately southeast dipping anastomosing fault splays with the highest grades concentrated along faults that juxtapose the Upper and Lower Comus Formations. The mineralization has a strike length of 1,250 feet (381 meters), a down dip extent of 900 feet (274 meters), and average fault-bound mineralization widths of 25 feet (7.6 meters).

6.5Alteration

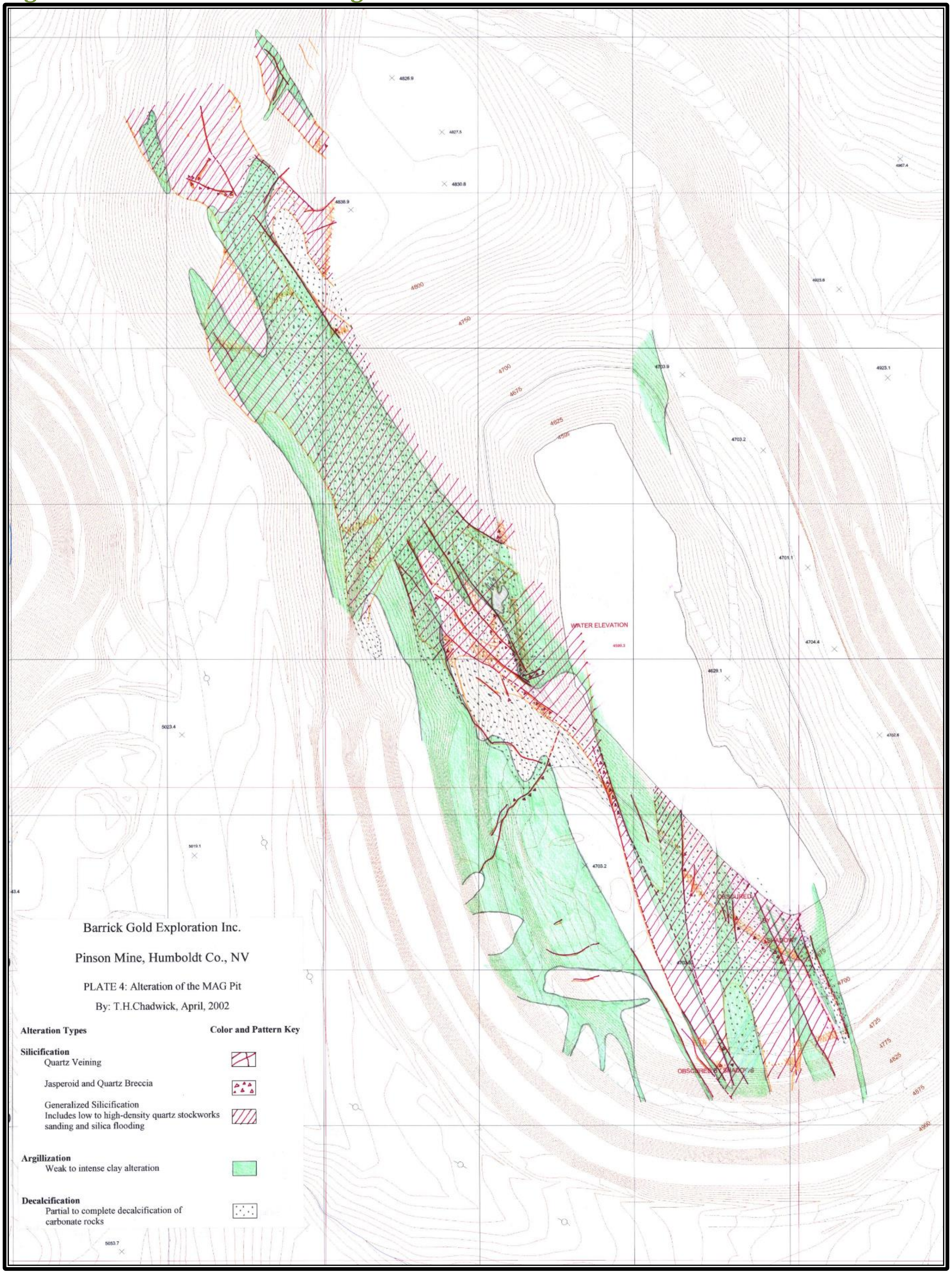
Alteration assemblages observed at the Granite Creek include silicification, decarbonatization, pyrite, and remobilization of carbon. Alteration mapping by Chadwick outlined the distribution of these assemblages within the pits.

In the CX Zone, which follows the strike of the CX Fault and includes the A, B, C, CX, and CX-West pits, McLachlan et al. (The Gold Deposits of Pinson Mining Company: A Review of the Geology and Mining History through 1999, Humboldt County, Nevada, 2000) documented gradational changes in the style and intensity of observed alteration. In the southwest, within the B Pit, gold mineralization occurs in strongly fractured shale and silty carbonate that has been weakly silicified and clay altered. In the nearby A Pit, alteration consists of intense silicification of carbonate lithologies and formation of gold-rich jasperoid along structures. Gold grains within the jasperoid are typically <5 microns in size and are found as inclusions in arsenian pyrite (McLachlan, Struhsacker, & Thompson, 2000). Within the C Pit, located northeast of the A Pit, high-grade material is hosted in decarbonatized carbonates that have been crosscut by small faults.

Within the CX Pit, mineralization consisted of silica and pyrite replacing carbonate along narrow structures, resulting in the formation of intermittent jasperoid and locally silicified wallrock. A large volume of the adjacent hanging wall carbonate-bearing siltstone is decarbonatized, but barren. Within the CX-West Pit, mineralization is hosted in strongly calc-silicate carbonates, which exhibit strong argillic alteration.

Mineralization in the Mag Pit is associated with decarbonatization, kaolinization, white kaolinite fracture filling, silicification and quartz veining (McLachlan, Struhsacker, & Thompson, 2000). Except for some massive limestone units, the original carbonate content of the calcareous host lithologies was removed during decarbonatization, resulting in a porous silty textured rock. Silicification occurs as replacement of the decalcified lithologies and healing fault gouge and breccia. Quartz veining and drusy open space coatings are common throughout the deposit. White kaolinite is commonly formed along fractures within the central portion of the deposit and elsewhere occurs as an argillic replacement of the host lithologies (McLachlan, Struhsacker, & Thompson, 2000). Lithology and alteration relationships can be observed in Chadwick's 2002 pit maps. Chadwick's alteration map of the Mag Pit is shown in Figure 6-6.

Figure 6-6 Alteration of the Mag Pit



Source: Chadwick 2002

The RFF Zone displays pervasive argillization and decarbonatization of host lithologies along with the formation of dissolution collapse breccias and intense shearing. Where the alteration is strongest, the altered zones consist of punky, spongy decarbonatized limestone in an argillically altered fine grained, carbon-rich matrix (Gustavson, 2012). Silicification is minor and occurs as a broad overprint on the zone. Calcite veining is also prevalent along the margins of the RFF.

6.6Deposit Types

The structural setting, alteration mineralogy, and mineralization characteristics of Granite Creek are consistent with Carlin-type deposits as defined in Radtke (Geology of the Carlin Gold Deposit, Nevada, 1985: USGS Professional Paper 1267, 1985) and Hofstra and Cline (Characteristics and Models for Carlin-Type Gold Deposits, 2000).

Carlin-type deposits formed in the mid-Tertiary after the onset of extension in an east-west-trending, subduction-related magmatic belt. The deposits are located along long-lived, deep crustal structures inherited from Late Proterozoic rifting and the formation of a passive margin within Paleozoic carbonate sequences composed of silty limestone to calcareous siltstone. The carbonate sequences are overlain by either structurally controlled siliciclastic sequences controlled by the Early Mississippian-aged Roberts Mountain allochthon or by stratigraphically controlled siliciclastic sequences. The siliciclastic rocks are less permeable than the underlying carbonate rocks, which traps fluids along major structures, causing them to flow laterally into the permeable and reactive carbonate sequences.

Alteration of host carbonate sequences consists of decarbonatization, argillization, and selective silicification, forming jasperoid and causing carbon flooding. Gangue minerals in Carlin-type deposits consist of calcite, siderite, and ferroan dolomites that can occur as geochemical fronts beyond the mineralized zones.

Gold deposition occurs in arsenian pyrite, is hosted within carbonaceous sequences near major high angle structural zones, and is concentrated in structural traps and/or replacement horizons of reactive and permeable sedimentary beds.

The Carlin-type deposits typically show enrichment in antimony, arsenic, mercury, and thallium, caused by hydrothermal fluids with temperatures ranging from 180-230°C. The source of fluids is likely deep-seated magmas that released gold bearing fluids at depths of 10 to 12 km. These magmas formed during Eocene slab-rollback of the Farallon plate as upwelling asthenosphere impinged on a strongly metasomatized sub-continental lithospheric mantle (Muntean, Cline, Simon, & Longo, 2011). Tertiary dikes associated with mineralization and radiometric age dates between 39 to 42 Ma along with isotopic data provide evidence toward the above hypothesis.

Structural pathways, reactive rocks, and sources of heat, gold, sulfur, and iron are required for Carlin-type deposits to form. Large regional structures transecting reactive rocks create contacts, faults, and shears. These secondary structures create pathways and traps for hydrothermal and metalliferous fluids.

7 Exploration

No exploration has been conducted at Granite Creek by i-80. Granite Creek is a production stage project and as such the geologic focus is on drilling to convert resources and extend known mineralization trends.

7.1Exploration

No exploration work has been conducted by i-80. This section discusses exploration undertaken by previous owners.

Exploration techniques employed on the Property to define additional gold resources have consisted primarily of mapping, geochemical sampling, and drilling. Use of these methods has resulted in the discovery of approximately one million ounces of gold in several open pit deposits. Several geophysical techniques have also been used to aid in the delineation of gold resources, albeit with limited success. The geophysical programs have mostly been applied to exploration programs along strike of known mineralization and as grass-roots applications to locate additional mineralized zones.

Atna became involved in Project planning in July 2004 and began drilling the Property in August 2004 after execution of the earn-in agreement with PMC on 12 August 2004. Atna continued work through April 2006. Atna vested a 70% interest by completing $12M in exploration and development expenditures and completing an NI 43-101 Technical Report of the Project's resources (Atna Resources Ltd., 2007).

7.1.1 Geologic Mapping and Geochemical Sampling

Cordex, and its successor, PMC, explored the Property through geologic mapping and geochemical sampling. There are three known mapping programs:

- A regional mapping program from the Preble to the Getchell mines conducted in the late 1970s
- A 1:6000-scale mapping program of the Property in 1983
- A 1:2400-scale mapping program of the Pinson pit area through the active life of the mine

Bench mapping in the pits occurred during mining and was followed up by detailed 1:1200-scale mapping of the A, B, C, CX, MAG, CXW, and Blue Bell pits by Tom Chadwick starting in 2000, after mining ceased. These maps were completed under the Homestake/Barrick partnership agreement.

Several geochemical programs were also completed by Cordex and PMC during the active mine life of the Granite Creek Mine, and by Homestake. These included programs:

- Cordex took rock chip samples in conjunction with mapping programs. A total of 737 rock chip samples were collected. Samples were assayed for gold, silver, arsenic, antimony, and mercury. Select samples were also analyzed for lead, zinc, copper, and manganese. The combined mapping/sampling programs were responsible for the discoveries of the Blue Bell and Felix Canyon deposits (Sim, 2005).
- PMC completed six float chip geochemical grids consisting of 8,756 samples. These grids covered the MAG deposit and along strike south of the A and B Pits.
- A biogeochemical sagebrush sampling program was conducted in the 1990s with inconclusive results.
- Under the Homestake/Barrick JV, an additional 312 rock samples and 273 soil samples were collected. These programs were completed on strike south of the existing pit areas and west of the A, B, C, and CX Pits.

7.1.2 Osgood Mining Geologic/Structural Mapping

In 2016, OMC contracted Mr. Robert Leonardson to complete a geological study on the Property that focused on advancing OMC's understanding of the structural framework and on providing guidance on exploration targeting. This work included structural and geologic mapping of the open pits and underground exposures, construction of Property-wide cross-sections, and report writing that included the identification of exploration targets on the Project.

Mr. Leonardson concluded that potential targets to discover additional gold mineralization are at intersections of the east-dipping, north–south faults (Rangefront/Mag) with the southeast-dipping CX-type faults. Other areas include the intersection of the sub-vertical northwest-striking faults with the CX-type faults. Examples of the first type are the CX hanging wall splays where they intersect the Mag Fault in the north half of the Mag Pit. The second example is exemplified by the intersection of the Bluebird Fault Zone with the Delaney thrust in the Blue Bell East pit and the intersection of the Bluebell 2 Fault with the CX thrust in the CX B Pit. Zones of limestone decarbonization such as seen in the CX Pit are also potential hosts for gold mineralization. These zones indicate strong fluid/vapor flow through the rock mass. Specific areas for exploration include:

- The intersection of the SOS and JP dikes on the south wall of the CX Pit. This area contains the largest block of decarbonization on the Property, and the hydrothermal alteration may represent an "exhaust plume" emanating from depth.
- The Ogee pipe extension is located between 1,500 feet and 1,800 feet below the CX-C Pit. A historical hole, HPC-070A intersected a 760-foot interval of low to moderate gold grades

above 3,160 feet and high-grade mineralization from 3,160 feet to 3,130 feet near, and just south of, the proposed Ogee high-grade down-dip extension.

- The northern continuation of the fault-propagated anticline in the western portion of the Mag Pit between the Mag Fault and CX Fault and to the north of the Mag Pit. The anticline steepens to the south, and the best chance to intersect high-grade mineralization would be at the intersection with the Disturbed Fault.
- The intersection of the Adam Peak Fault and the Mag Fault suite north of the Mag Pit.
- The CX-B Pit decarbonatization zone at the intersection of the CX and Bluebell 2 faults on the west limb of the Pinson anticline.
- The Mag Pit decarbonatization on the west wall along a section of the Mag Fault intersection with the CX and HW faults and the Disturbed Fault.
- The Mag Pit decarbonatization on the west wall along a portion of the Mag Fault intersections with the Disturbed Fault.
- Bluebell east pit decarbonatization at the intersection of the Bluebell and Delaney faults.
- Traps and fault intersections along the north-northwest-trending Mag Fault suite and the northeast-trending CX type faults.
- Flat to ramp traps down dip extension of fluids that mineralized the Bluebell, CX(?) between South Mountain Fault, and the southern Mag suite of faults. Flat to ramp traps along the Adam Peak detachment and subsequent faults (CX, Disturbed, and South Mountain).

7.1.3 Geophysical Surveys

Numerous geophysical surveys have been conducted on the Property. These include both regional and detailed surveys. The regional surveys included gravity and aeromagnetics. Detailed surveys involved mostly electromagnetic techniques and included Induced Polarization (IP), Electromagnetics (EM), Magnetotellurics (MT), and Controlled Source Audio-frequency Magneto Tellurics (CSAMT) surveys. A summary of these techniques includes:

- Airborne EM and magnetics by the U.S. Geological Survey (USGS) at quarter-mile line spacing throughout much of the Getchell Trend
- Ground-based magnetics over the CX Zone completed in 1970 by Cordex
- Regional gravity surveys, both public and private, compiled by Homestake in 1997
- Ground-based magnetic survey at the north edge of the Mag Pit completed in 1998 by Homestake
- Several generations of AMT (EM, IP, CSAMT) completed by PMC
- Several CSAMT lines completed by Homestake between 1998 and 2000
- Several EM lines completed by Homestake in 2000

- A detailed gravity survey over the Property conducted by Magee Geophysical Services, LLC of Reno, Nevada in October 2006 (Magee Geophysical Services, 2006), during which a total of 2,587 gravity readings were acquired using a 100-meter (328-foot) station spacing covering approximately 27 square km (10 square miles) (Figure 7-1). The results were interpreted by Fritz Geophysics in 2007 (Fritz Geophysics, 2007). The existence of about 1,700 drill holes within the gravity survey area allowed a novel approach to be attempted for the detailed gravity data. Typically, gravity surveys are conducted to attempt to determine the thickness of alluvial cover over bedrock, as well as structures, etc., for possible targets of interest. The final gravity response on the bedrock surface is shown in Figure 7-2, overlain on the topography, with interpreted structures, bedrock rock types and drill hole collar locations. Also included are plots of the original surface gravity field measurements and the thickness of alluvium from drill data. The basement rock types defined by the basement gravity response correlate with the general mapped geology. To the north-west the high density is related to the large intrusive, TKg. This intrusive is magnetic as well. To the south-west is a lower density unit that correlates with a mapped Oc, probably Valmy. Through the center of the survey area there is an even lower density unit that trends reasonably north south and is defined by northerly, northwesterly, and northeasterly structures. Further to the east, there is another higher density unit, possibly Valmy again. This unit is at an alluvial thickness of greater than 1,500 feet (457 meters) and is not as well defined as the other units. As the thickness of alluvium increases, the resolution of the surface gravity data decreases. Finally, at the southeast edge of the survey, there is a large basin fault that appears to drop the bedrock to depths greater than 3,000 feet (914 meters).

Figure 7-1: Gravity Survey, 2,587 Stations, Magee Geophysical Services, 2006

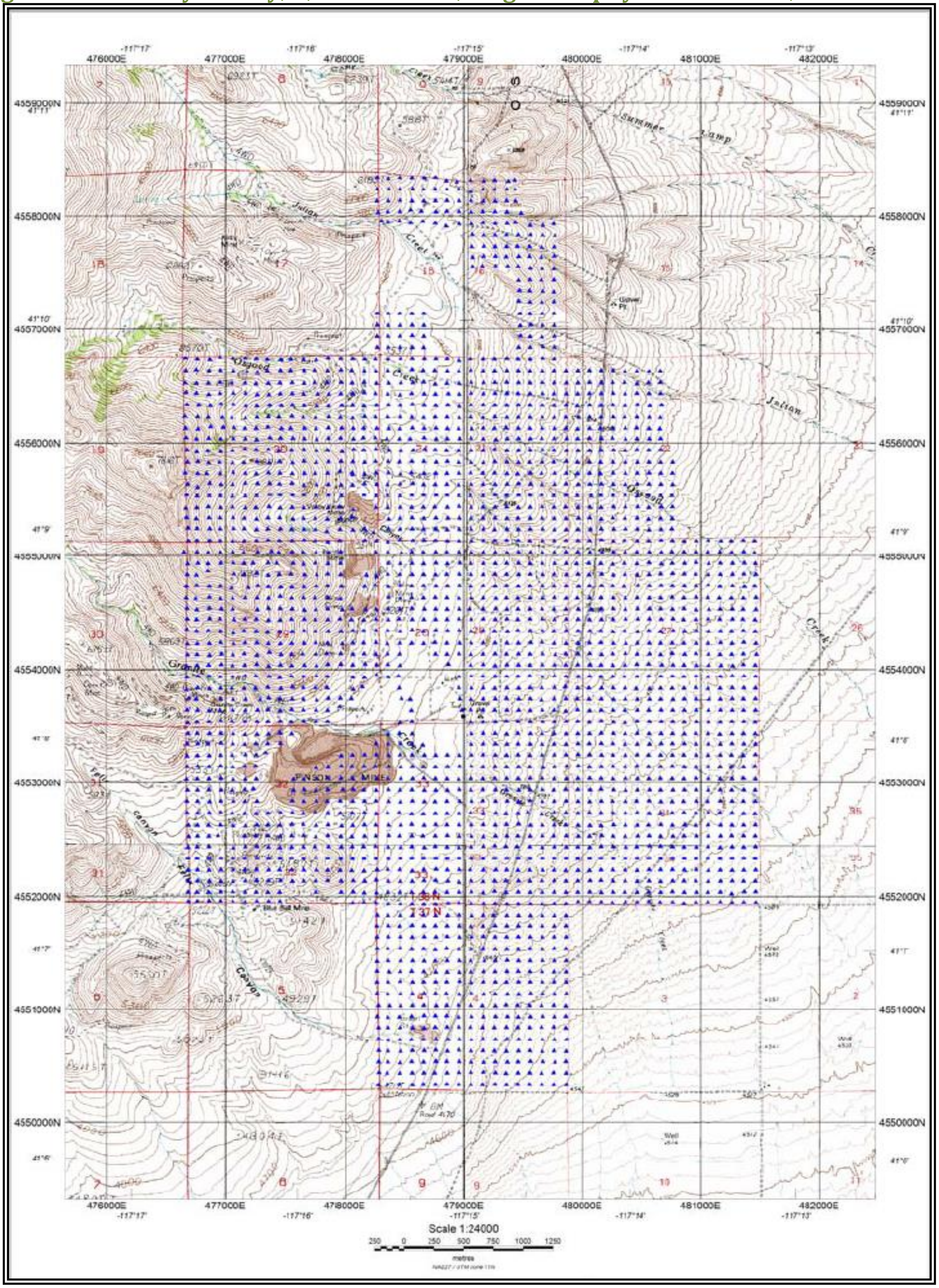
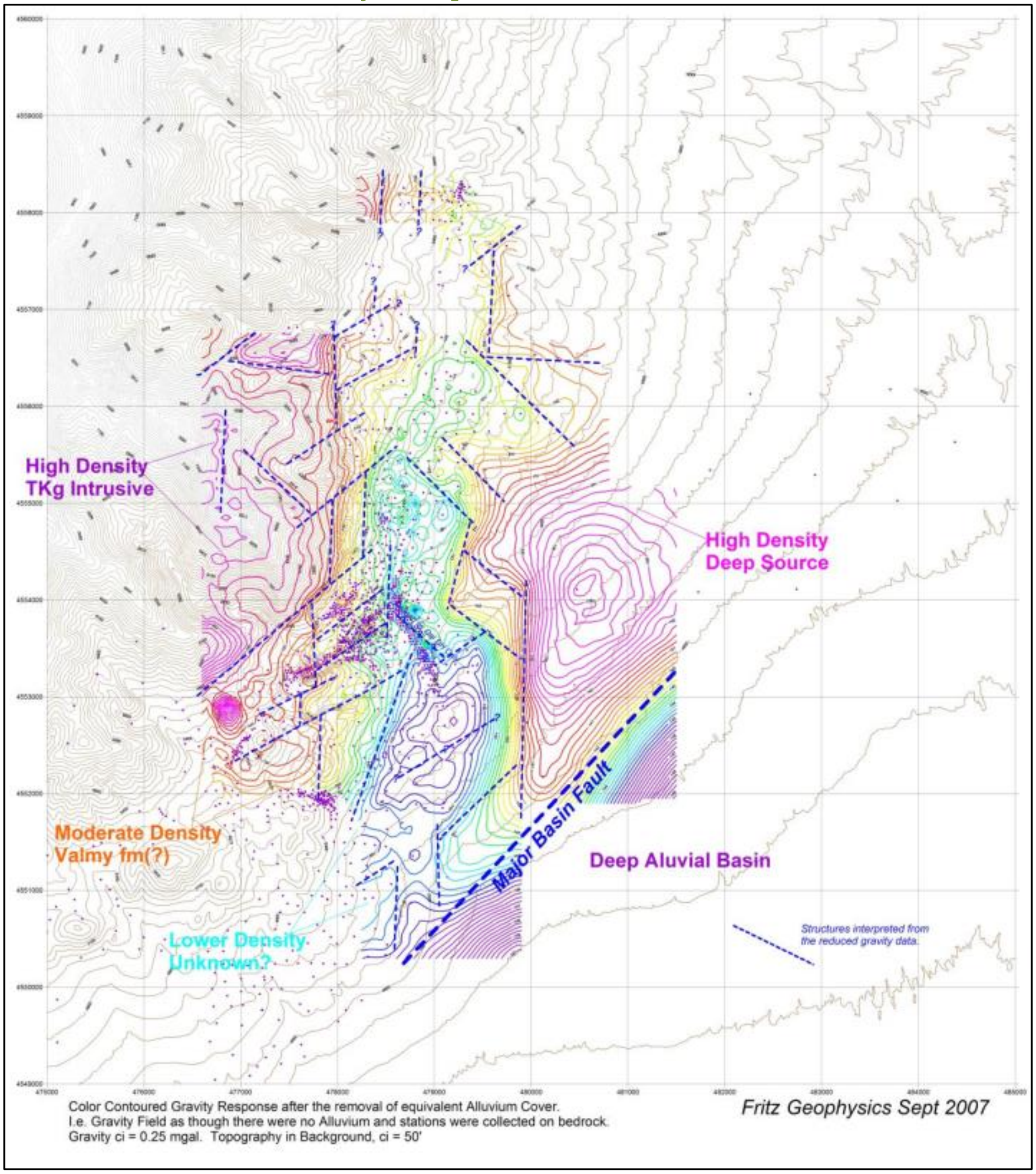


Figure 7-2: Pinson Local Gravity Interpretation



- In 2008, Barrick interpreted the geophysical survey data at Pinson (Barrick, 2008). For that work, the 2002 MT survey and 2006 gravity survey and all available geological/geochemical information were combined, and a couple of target areas defined requiring a drillhole test. In 2002, Quantec Geoscience were contracted to acquire TITAN 24 MT data over the Pinson property. Six east-west lines were collected along the Rangefront, spaced on average around 2,000 feet (610 meters) apart. The dipole spacing along line was 300 feet (91 meters). Quantec ran regular 2 dimensional (2D) inversions on the MT data to create resistivity depth sections which have been deemed sufficient for this targeting exercise.

The location of the MT survey lines has been plotted on the geology and pit locations and on the residual gravity with Bourne's structural interpretation and targets annotated (Figure 7-3). Figure 7-4 to Figure 7-9 show MT resistivity depth inversions for each of the survey lines, with all drilling, surface geological mapping Chadwick's interpreted sectional geology, and Bourne's structural targets annotated (Barrick, 2008).

Figure 7-3: Location of the MT Survey Lines on the Geology and Pit locations (Left) and on the Residual Gravity (Right)

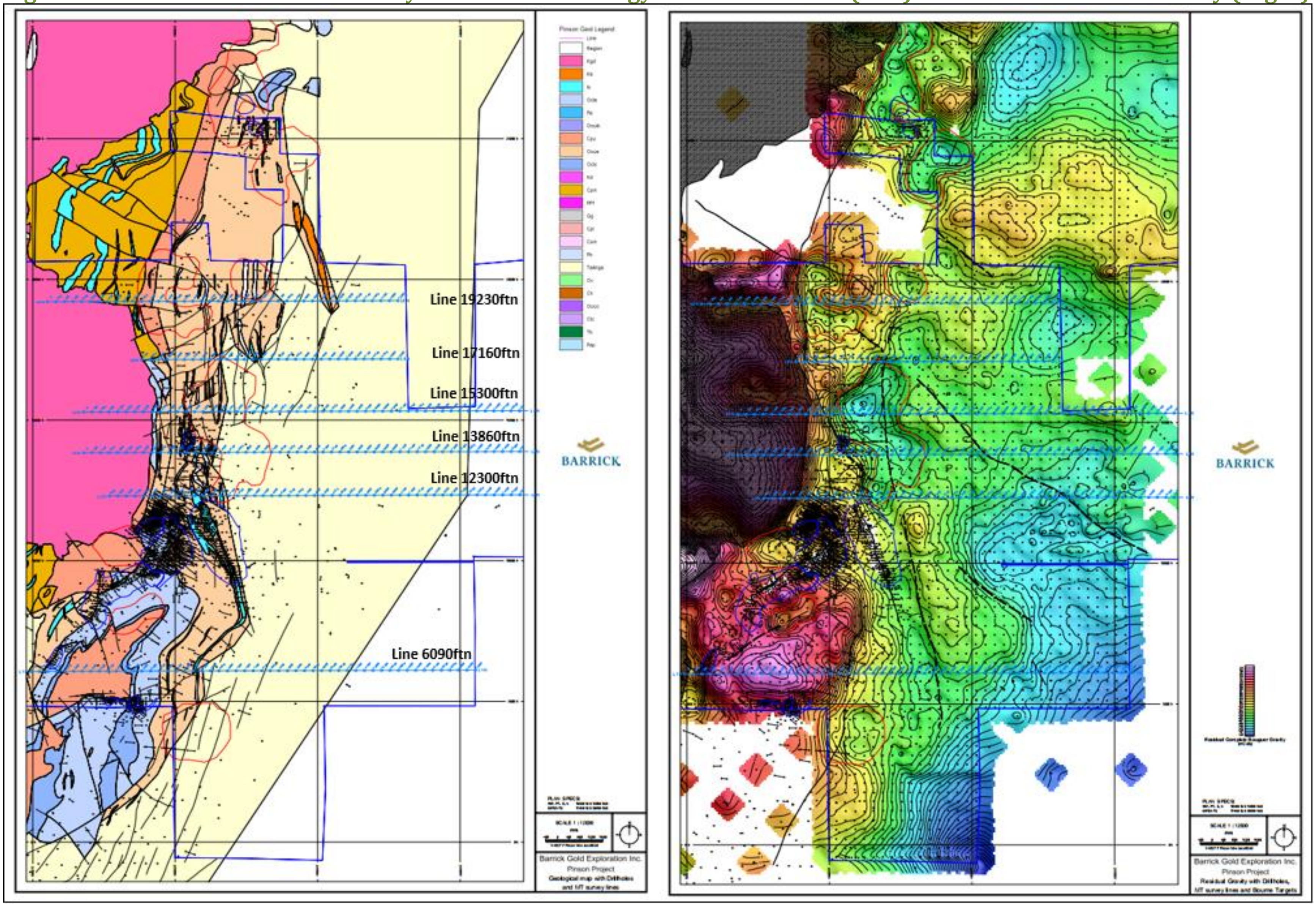


Figure 7-4: MT Resistivity Depth Inversion for Line 6090

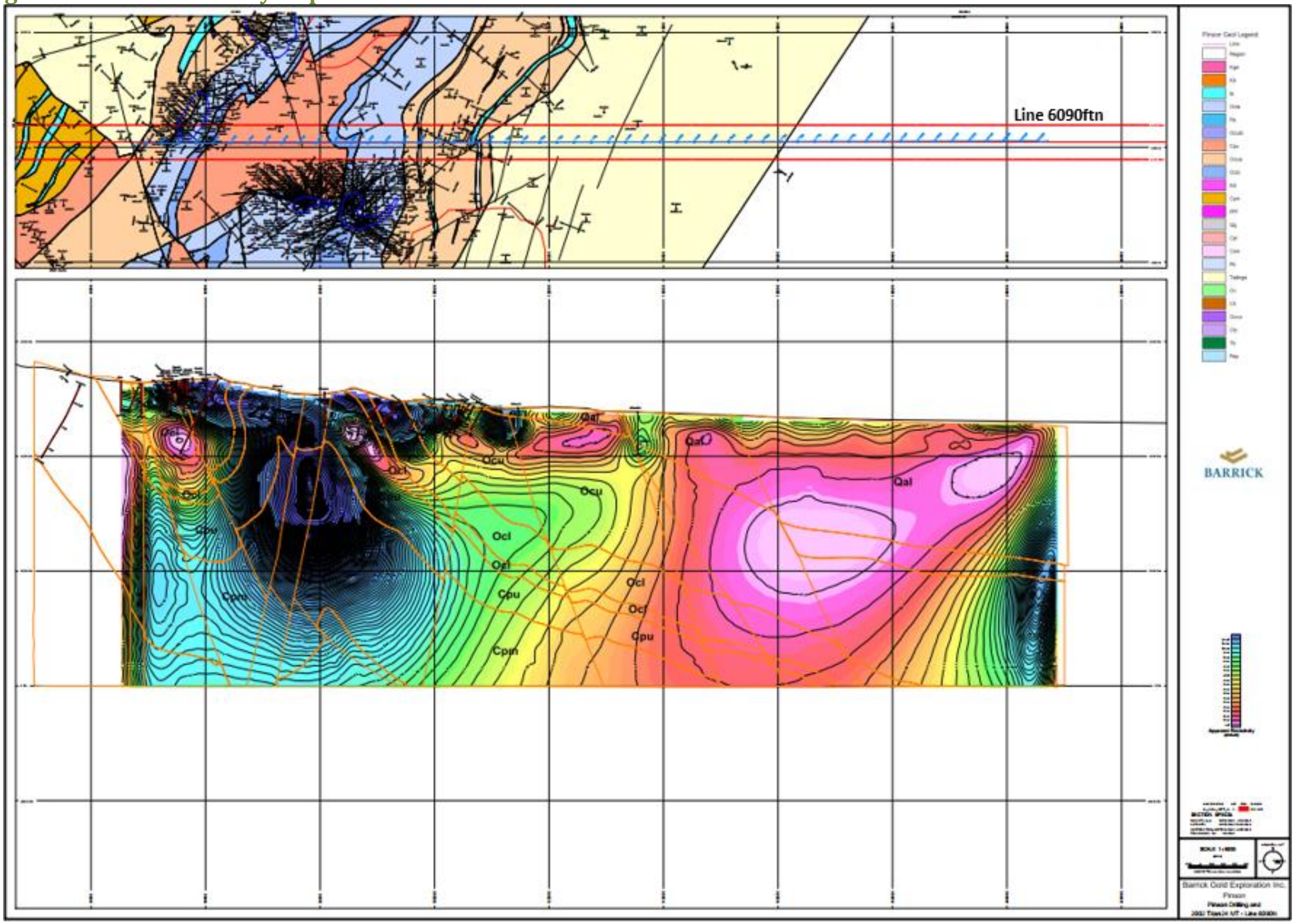


Figure 7-5: MT Resistivity Depth Inversion for Line 12300

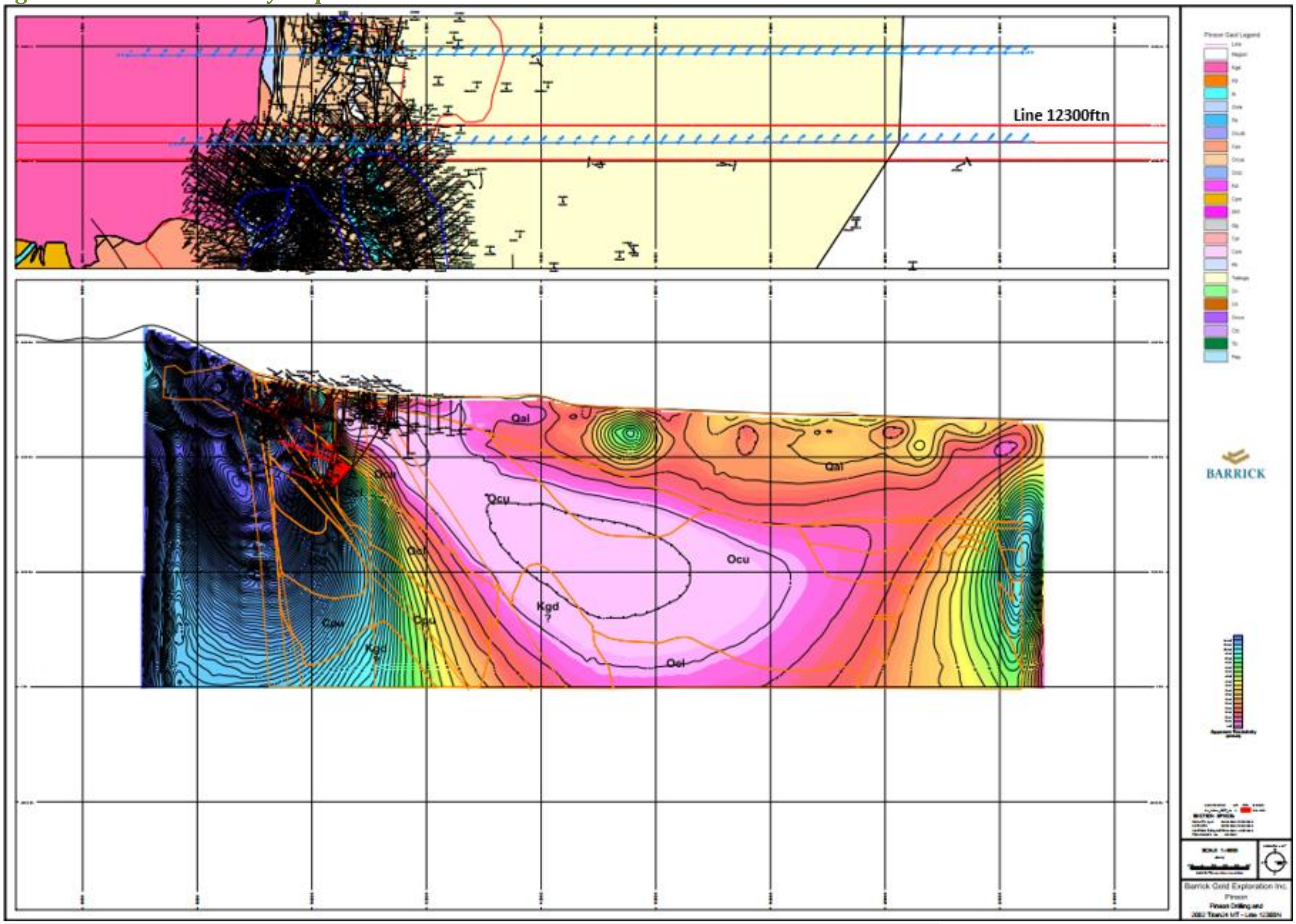


Figure 7-6: MT Resistivity Depth Inversion for Line 13860

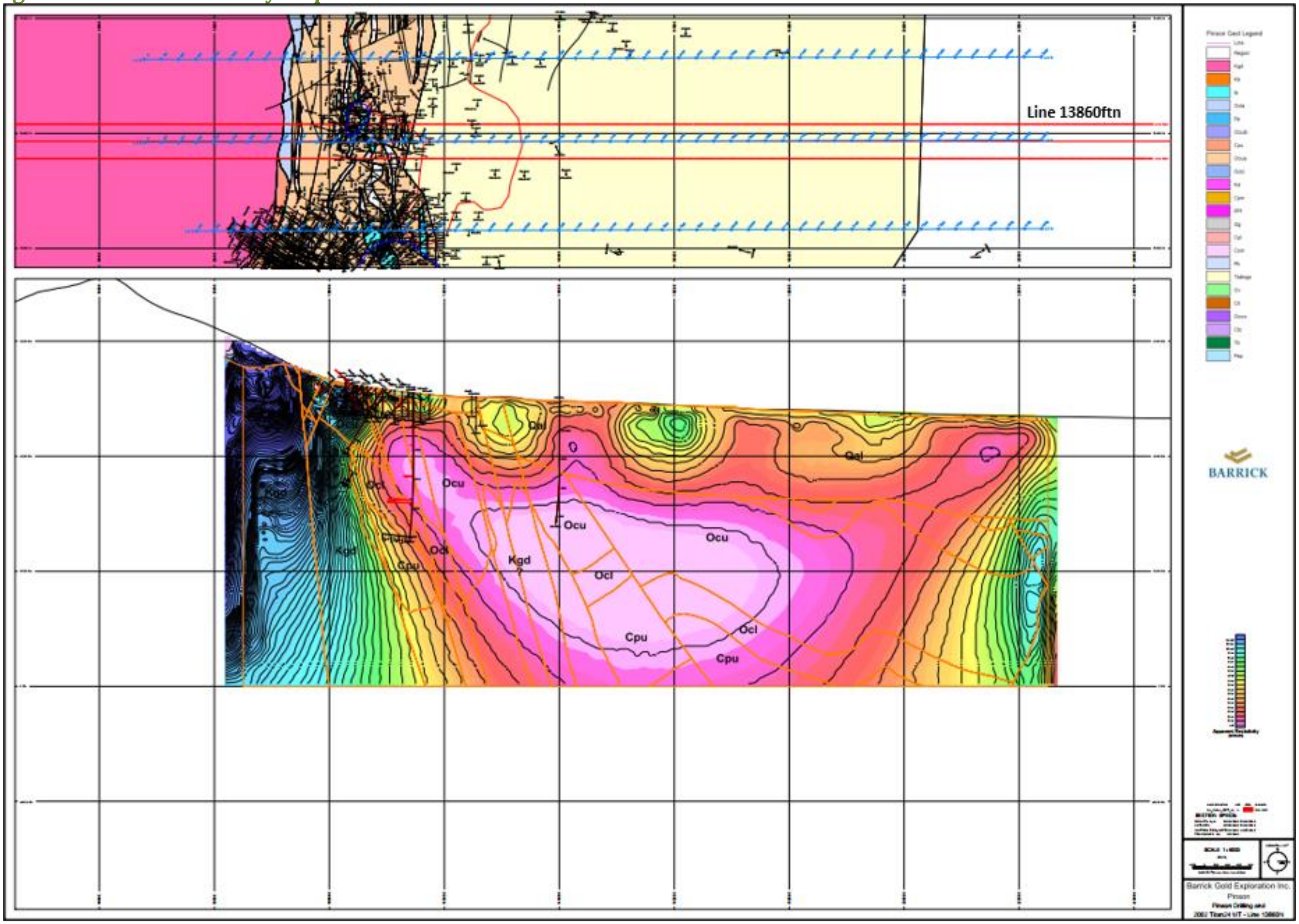


Figure 7-7: MT Resistivity Depth Inversion for Line 15300

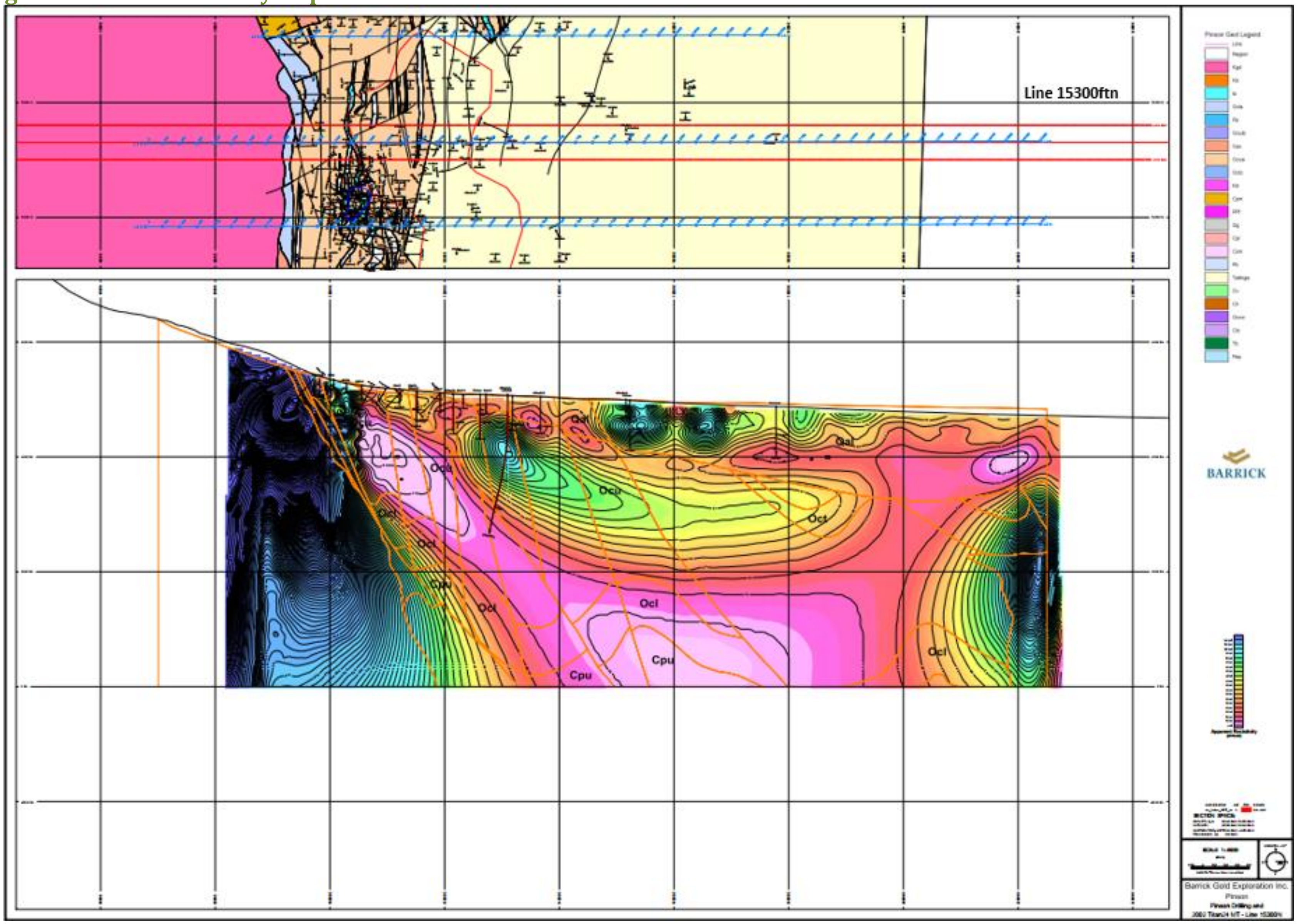


Figure 7-8: MT Resistivity Depth Inversion for Line 17160

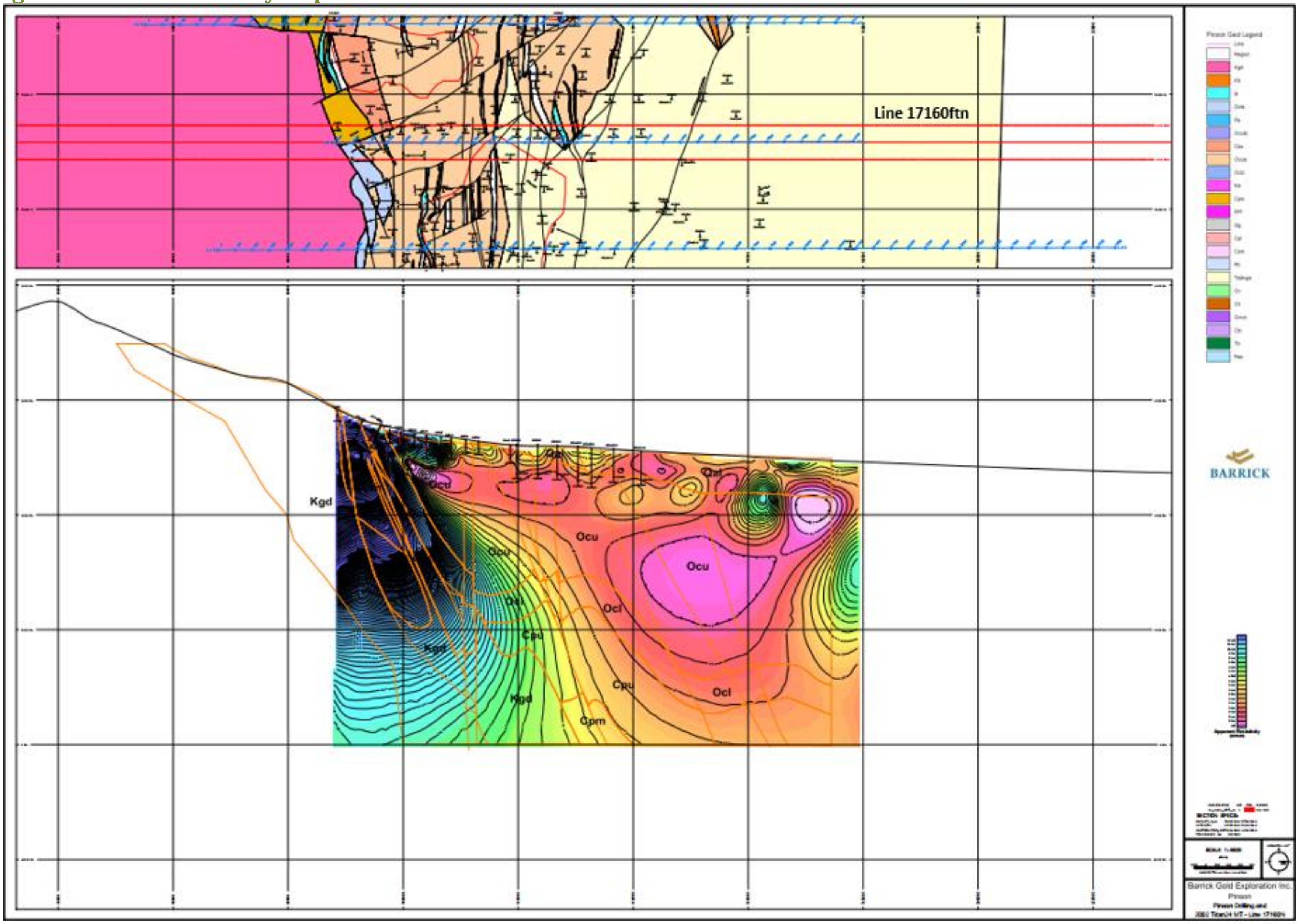
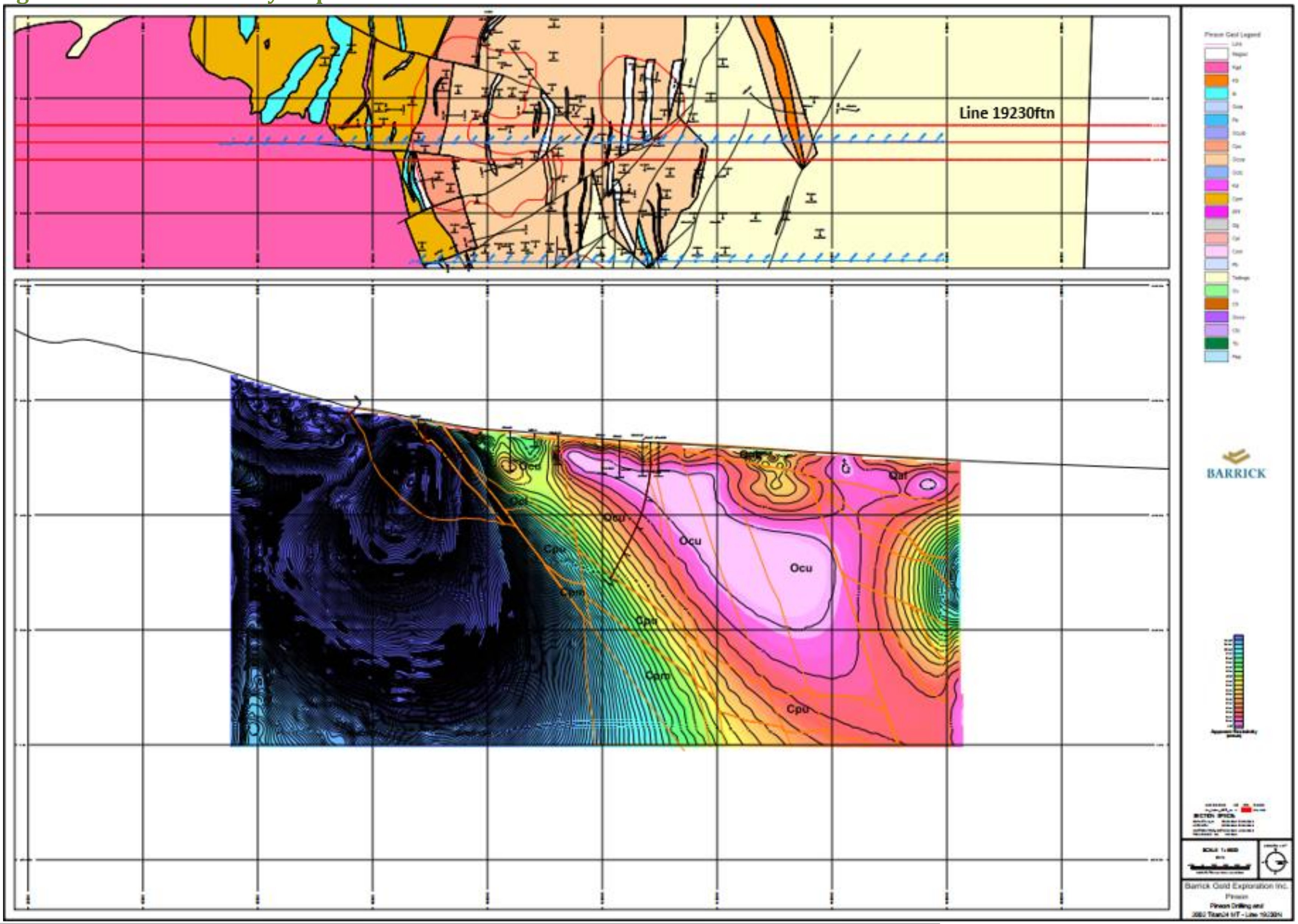


Figure 7-9: MT Resistivity Depth Inversion for Line 19230



7.1.4 Underground Drifting/Evaluation

A small exploration drifting program was conducted on the upper "B" zone by Cordex in the 1970s to conduct bulk testing. Results from this program are unavailable.

In May of 2005, Small Mine Development (SMD) of Boise, Idaho, was contracted by Atna to drive exploration drifts, crosscuts, and develop drill stations to complete Atna's evaluation of the Range Front resource area. Both the Range Front and CX resource areas were of interest in Atna's program.

The underground development work completed 1,988 feet (606 meters) of 14-foot (4.3-meter) by 16-foot (4.9 meter) adit, 378 feet (115 meters) of decline, and six diamond drill stations (Gustavson, 2012). A small mineability test was also carried out on the newly defined Ogee Zone to evaluate the potential conditions for future stoping. Approximately 400 short tons (363 tonnes) of material were extracted during this test. The results indicated the possibility of drift and fill as a potential mining method.

During 2008, approximately 693 feet (211 meters) of development drifting was completed, and significant geological data was recorded in the RFZ. However, no data on ground conditions was acquired. This data was not collected because it was anticipated that ground conditions would be similar to those encountered at the Getchell Mine, and mineralization would be exploitable by underhand drift and fill stoping methods (Gustavson, 2012).

7.1.5 Trenching and Sampling

Atna channel sampled 14 ribs in the Ogee Zone and sent 74 rib and face samples out for assay (Edmondo, McDonald, & Stanley, 2007). Salient results are summarized in Table 7-1. Assays from the samples indicated that no high-grade mineralization was encountered except where the main drift intersected the Ogee Zone on the 4770 elevation.

Table 7-1: Salient Results of the Ogee Zone Channel Sample Assays

Sample No.	From feet (meters)	To feet (meters)	Length feet (meters)	Gold Grade opt (g/t)
North Rib				
RFUG-055	76 (23.1)	81 (24.7)	5 (1.5)	0.144 (4.94)
RFUG-056	81 (24.7)	85 (25.9)	4 (1.2)	0.445 (15.26)
RFUG-059	85 (25.9)	88 (26.8 (	3 (0.9)	0.274 (9.39)
RFUG-061	88 (26.8)	93 (28.3)	5 (1.5)	1.448 (49.65)
RFUG-063	93 (28.3)	97 (29.6)	4 (1.2)	0.176 (6.03)
RFUG-064	97 (29.6)	101 (30.8)	4 (1.2)	0.739 (25.34
RFUG-067	101 (30.8)	110 (33.5)	9 (2.7)	0.996 (34.15)
Weighted Average			**34 (10.4)**	**0.682 (23.38)**
South Rib				

Sample No.	From feet (meters)	To feet (meters)	Length feet (meters)	Gold Grade opt (g/t)
RFUG-081	77 (23.5)	80 (24.4	3 (0.9)	0.106 (3.63)
RFUG-082	80 (24.4)	83 (25.3)	3 (0.9)	0.065 (2.23)
RFUG-083	83 (25.3)	93 (28.3)	10 (3)	1.082 (37.10)
RFUG-084	93 (28.3)	96 (29.3)	3 (0.9)	0.894 (30.65)
RFUG-086	96 (29.3)	99 (30.2)	3 (0.9)	0.355 (12.17)
RFUG-087	99 (30.2)	107 (32.6)	8 (2.4)	0.028 (0.96)
RFUG-088	107 (32.6)	112 (34.1)	5 (1.5)	0.228 (7.82)
Weighted Average			**35 (10.7)**	

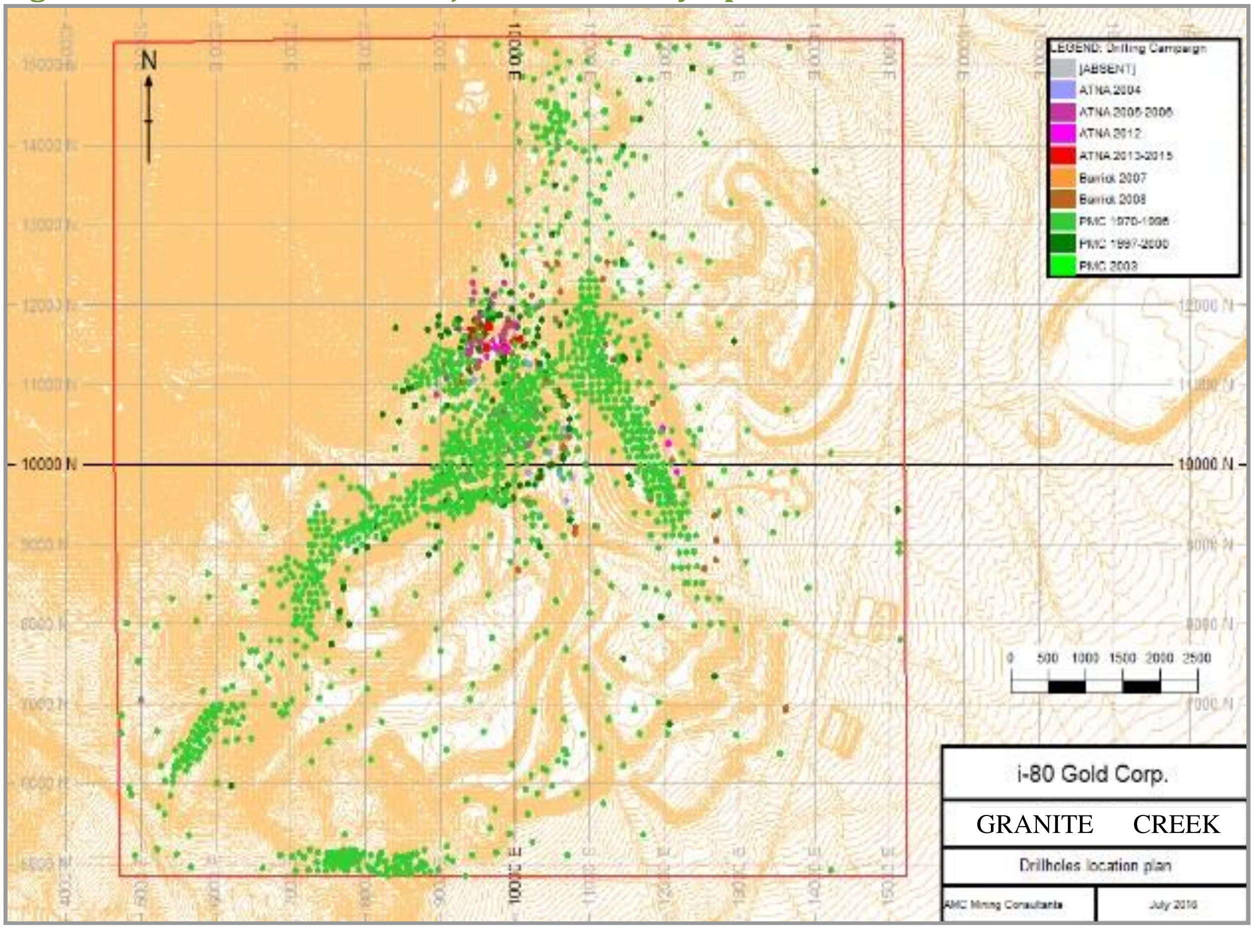
7.2 Drilling

7.2.1 Drilling Campaigns Overview

Numerous holes have been drilled in and around the Property prior to 1970. Unfortunately, this drillhole data is no longer available. Since 1970, a total of 2,083 drillholes totaling 955,747.9 feet (291,312 meters) have been drilled within the Property area. Figure 7-10 shows the drilling by each operator and significant time period. PMC and its predecessors, Rayrock Mines and the Cordex Syndicate, account for most of these holes: 1,434 holes totaling 554,435 feet (168,991.8 meters). Homestake drilled 165 holes totaling 160,207.7 feet (48,831.3 meters), and Barrick drilled 106 holes totaling 101,345.1 feet (30,890 meters). Both companies acted as operators for PMC. Atna, the last company to operate at the Granite Creek Mine, drilled 318 holes totaling 119,074.1 feet (36,293.8 meters).

Table 7-2 presents a summary of the drilling at the Property.

Figure 7-10: Granite Creek Project Drill Plan by Operator



Source: AMC Mining Consultants (Canada) Ltd, 2019

Table 7-2: Summary of Drilling on the Granite Creek Property Since 1970

Company	Surface RC		Surface Core		UG RC		UG Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)	# Holes	Footage (feet)	# Holes	Footage (feet)		
PMC	1,426	546,313.0	8	8,122.0					1,434	554,435.0
PMC (Homestake)	136	108,335.0	29	51,872.7					165	160,207.7
PMC (Barrick)	39	35,645.0	67	65,700.1	4	930.0	56	19,756.0	106	101,345.1
Atna	29	18,672.0	65	52,847.6	176	32,068.0	48	15,486.5	318	119,074.1
Total	**1,630**	**708,965.0**	**169**	**178,542.4**	**180**	**32,998.0**	**104**	**35,242.5**	**2,083**	**955,747.9**

Note: RC=reverse circulation, UG=underground.
Source: Osgood Mining Company LLC.

Each period of drilling is described in further detail in Sections 7.2.1.1 to 7.2.1.9.

7.2.1.1 PMC Drilling 1970 to 1996

Many holes drilled by PMC during this time period were development holes drilled in and adjacent to existing pits. Over 1,400 holes were drilled within the A, B, C, CX, Mag, CX-West, Felix, and

Blue Bell pit areas. Many of these holes were drilled vertically, and all but eight were either conventional rotary or RC. The eight core holes that were drilled (8,122 feet [2,475.6 meters]) were in the B, C, CX, and Mag Pit areas to test stratigraphy, metallurgy, or deep mineralized structures (Golder Associates, 2014). Table 7-3 summarizes the drilling PMC conducted through 1996.

Table 7-3: PMC Drilling through 1996

Company	Surface RC		Surface Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)		
PMC	1,426	546,313.0	8	8,122.0	1,434	554,435.0

Source: Osgood Mining Company LLC

7.2.1.2 PMC – Homestake Drilling 1997 to 2000

Between 1997 and 2000, Homestake, as the operator for PMC, drilled 165 holes, as shown in Table 7-4. Of the 165 holes drilled, 136 (108,335 feet [33,020.5 meters]) were directed into the CX and RFF system.

Table 7-4: Homestake Drilling

Company	Surface RC		Surface Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)		
PMC (Homestake)	136	108,335.0	29	51,872.7	165	160,207.7

Source: Osgood Mining Company LLC

7.2.1.3 PMC – Barrick Drilling 2003

Four exploration holes were drilled by Barrick, operator at the time for PMC, to test extensions of the CX Fault Zone near its projected intersection with the Mag Pit fault system. The drilling did not identify significant mineralized zones, and no additional work was conducted by Barrick (Golder Associates, 2014). Table 7-5 shows a summary of the Barrick drilling.

Table 7-5: Barrick Drilling 2003

Company	Surface RC		Surface Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)		
PMC (Barrick)	3	3,340.0	1	3,003.3	4	6,343.3

Source: Golder Associates 2014.

7.2.1.4 Atna Drilling 2004

The drilling by Atna in 2004 followed up on mineralized zones previously identified by PMC and Homestake. Thirty-one holes totaling 29,739.5 feet (9,064.6 meters) were drilled. These holes were comprised of four RC holes totaling 2,217 feet (675.7 meters) and 27 core holes totaling 27,522.5 feet (8,388.9 meters) (Table 7-6). This drilling program had five objectives:

- Improve the grade and thickness of mineralized zones, especially in areas where drilling consisted of only RC drilling.
- Infill drilling, especially where previous drill spacing was greater than 400 feet (121.9 meters).
- Expand mineralized zones both laterally and down-dip.
- Obtain rock quality data on hanging wall, footwall, and mineralized zones.
- Evaluate previously identified targets.

Table 7-6 shows a summary of the Atna drilling.

Table 7-6: Atna Drilling 2004

Company	Surface RC		Surface Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)		
Atna	4	2,217.0	27	27,522.5	31	29,739.5

Source: Osgood Mining Company LLC.

Of the 31 holes drilled, 13 holes (13,000 feet [3,962.4 meters]) were drilled into the CX Fault Zone and 18 holes (16,739.5 feet [5,102.2 meters]) were drilled into the RFF Zone (Golder Associates, 2014).

7.2.1.5 Atna Drilling 2005 – 2006

The objective of the 2005 to 2006 drilling program was to define and delineate Measured and Indicated gold Mineral Resources in the upper portions of the RFF Zone where Atna had outlined a 1,000-foot (305-meter) long by 200- to 500-foot (61- to 152.4-meter) thick mineralized zone during its 2004 drilling program. The drilling program was designed to test the upper RFZ between the 5,000 and 4,400 feet (1,524 and 1,341 meters) amsl (Golder Associates, 2014). The program used both surface and underground drilling to delineate the zone. A total of 107 drillholes (55,180.1 feet [16,818.9 meters]) were drilled between 2005 and 2006 (Table 7-7).

Table 7-7: Atna Drilling 2005-2006

Company	Surface RC		Surface core		UG Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)	# Holes	Footage (feet)		
Atna	25	16,455.0	34	23,238.6	48	15,486.5	107	55,180.1

Source: Osgood Mining Company LLC

Surface drilling began in May of 2005. The majority of these holes were core holes, which were pre-collared via RC drilling and completed with core drilling. Fifty-nine (59) drillholes, totaling 39,693.6 feet (12,098.6 meters) of drilling, were completed from surface.

Underground drilling began in September of 2005 after drifting was completed and underground drill rigs became available. In total, 48 holes aggregating 15,486.5 feet (4,720.3 meters) of underground drilling were completed in the Ogee, CX West, and Range Front targets.

7.2.1.6 PMC (Barrick) Drilling 2007

In August of 2007, surface exploration and development drilling began using an Eklund RC drill rig and a Major Drilling core rig. Targets tested included portions of the CX and RFF, Ogee Zone, and the HPR104 area. The HPR104 area is north of the Granite Creek Mine.

Twenty-three (23) surface holes (18,916.2 feet [5,765.7 meters]) were completed during the latter part of 2007 as shown in Table 7-8. The results of the drilling were disappointing in that only thin, sub-economic zones of underground mining gold grades were intersected.

Table 7-8: PMC - Barrick Drilling 2007

Company	Surface RC		Surface Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)		
PMC (Barrick)	7	4,935.0	16	13,981.2	23	18,916.2

Source: Osgood Mining Company LLC.

7.2.1.7 PMC (Barrick) 2008 Drilling

Surface drilling began in January of 2008 with three core drills and one RC drill testing areas north of the CX West pit. The core drilling was focused on completing holes pre-collared by RC drilling in 2007 and testing the deep potential of the Getchell Fault system north of the Granite Creek Mine, which had associated gravity and MT anomalies (Golder Associates, 2014). RC drilling was primarily focused on pre-collaring holes for follow-up core drilling north of the CX/CX West pits. Surface core drilling was completed in April of 2008. RC drilling continued throughout 2008, with the focus on drilling pilot holes for potential dewatering well locations.

Underground exploration began in April 2008 as discussed in Section 7.1.4. SMD was contracted to rehabilitate existing underground workings and drive exploration headings into the Ogee and CX zones. SMD supplied an underground RC drill for closely spaced definition drilling, and Connors Drilling was contracted to conduct underground core drilling. The SMD contract was terminated in May of 2008. Connors Drilling remained on site and brought in a second underground core rig in mid-July. Both core rigs continued operation through mid-December, testing the Ogee Zone and conducting widely spaced drilling within the RFZ.

In August 2008, a second surface drilling program was initiated to twin RC holes in key areas of the resource suspected of having downhole contamination. Two core rigs and one RC rig (to pre-collar holes) were used. A third surface core rig was also brought in to complete one deep hole to test the Mag fault-Delaney fault intersection south of the resource area. The drilling program was completed in mid-December and all drilling equipment removed from site.

During 2008, total surface drilling included 29 RC holes totaling 27,370 feet (8,342.4 meters) and 50 core holes totaling 48,715.6 feet (14,848.5 meters). Underground drilling included 4 RC holes for 930 feet (283.5 meters) and 56 core holes totaling 19,756 feet (6,021.6 meters) (Table 7-9).

Table 7-9: PMC – Barrick Drilling 2008

Company	Surface RC		Surface Core		UG RC		UG Core		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)	# Holes	Footage (ft)	# Holes	Footage (feet)		
PMC (Barrick)	29	27,370.0	50	48,715.6	4	930.0	56	19,756.0	139	96,771.6

Source: Osgood Mining Company LLC

HPR104

During the 2008 drilling program, eight holes were drilled north of the Pinson deposit resource area. These holes were designed to twin earlier PMC drilling that were drilled to test the intersection of the Range Front and Linehole Faults. The results of the initial drilling could not reproduce the thick low -grade intercept identified in an earlier hole, hole HPR104. This was considered to constitute downhole contamination in hole HPR104, and the hole was removed from the database. A second round of core drilling did intersect thin, higher-grade mineralization. Hole BPIN-008 intercepted 21.5 feet grading 0.620 opt at a depth of 1,378 feet (Golder Associates, 2014). This mineralization appeared to be structurally controlled by the intersection of the Linehole Fault and the Upper/Lower Comus contact 900 feet northeast of the main portal.

Deep Exploration Targets

Two deep drillholes, BPIN-010C and BPIN-011A, were drilled in 2008. Hole BPIN-010C was drilled to a depth of 2,845.5 feet (867.3 meters) and was designed to test the Lower Comus Formation adjacent to structures identified from a 2006 gravity survey (Golder Associates, 2014). The hole bottomed in Upper Preble Formation, and assay results proved negative. Hole BPIN-011A was drilled to a depth of 2,778 feet (846.7 meters) and ended in argillite and shale of the Upper Comus (Golder Associates, 2014). The hole was designed to test the projected intersection of the Mag and Delaney faults. Analyses of chip samples indicated a 60-foot (18.3-meter) zone of low grade- gold (0.029 opt [0.99 g/t]) at 1,440 feet (438.9 meters) hosted in silicified Upper Comus claystone and shale (Golder Associates, 2014). Subsequent analyses of core from the entire hole indicated narrow zones of mineralization associated with decarbonatization and pyritized sediments.

7.2.1.8 2012 Atna Mag Pit Core Drilling

In 2012, Atna completed four PQ-size core holes, totaling 2,086.5 feet (636 meters), to acquire samples for column leach testing from mineralized material within the Mag Pit resource area. The holes were drilled along strike of the known mineralized zone, with each hole intersecting potential high-grade material. In addition to the metallurgical holes an additional 56 underground exploration RC holes totaling 7,495 feet (2,284.5 meters) were drilled in the Ogee Zone. Table 7-10 summarizes the drilling conducted by Atna in 2012.

Table 7-10: Atna Drilling 2012

Company	Surface Core		UG RC		Total Holes	Total Footage
	# Holes	Footage (feet)	# Holes	Footage (feet)		
Atna	4	2,086.5	56	7,495.0	60	9,581.5

Source: Osgood Mining Company LLC

7.2.1.9 2013 – 2015 Atna Underground Development RC Drilling

Between 2012 and 2015, Atna completed 120 underground RC holes totaling 24,573 feet (7,489.9 meters) (Table 7-11). These holes were designed to confirm continuity of mineralization and to delineate stope configuration within the Ogee Zone for mining.

Table 7-11: Atna Drilling 2013 – 2015

Company	UG RC		Total Holes	Total Footage
	# Holes	Footage (feet)		
Atna	120	24,573.0	120	24,573.0

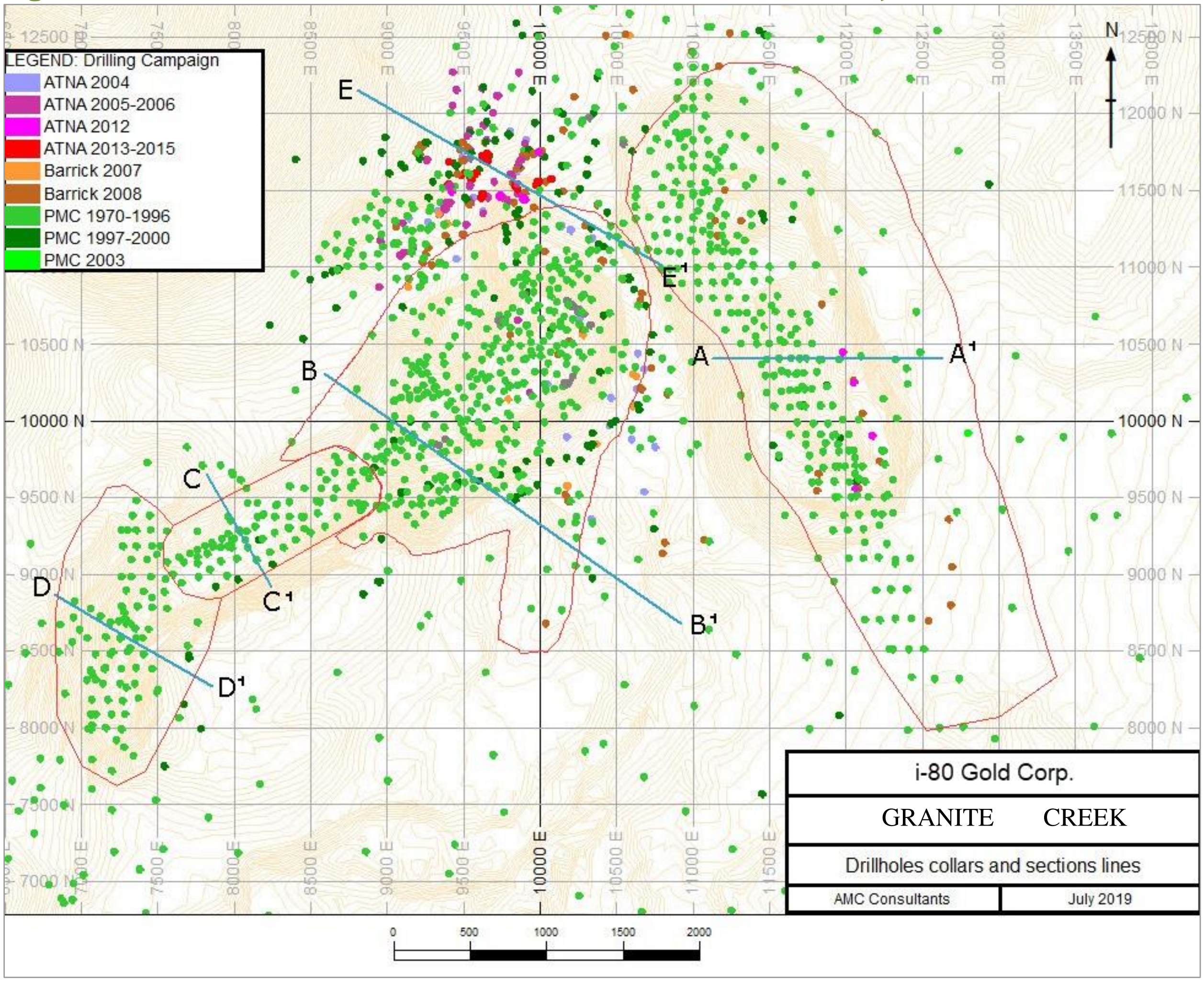
Source: Osgood Mining Company LLC

7.2.2 Representative Drill Sections and Plan

Figure 7-10 shows the drill plan of the Property in the area of the 2021 Mineral Resource, shown by a red outline. The drillholes are coded by operator and significant time periods. Figure 7-11 shows a plan view with section lines of the Open Pit area. Figure 7-12 to Figure 7-15 show representative vertical sections through the four Open Pit areas. Figure 7-16 shows a vertical section through the underground resource area.

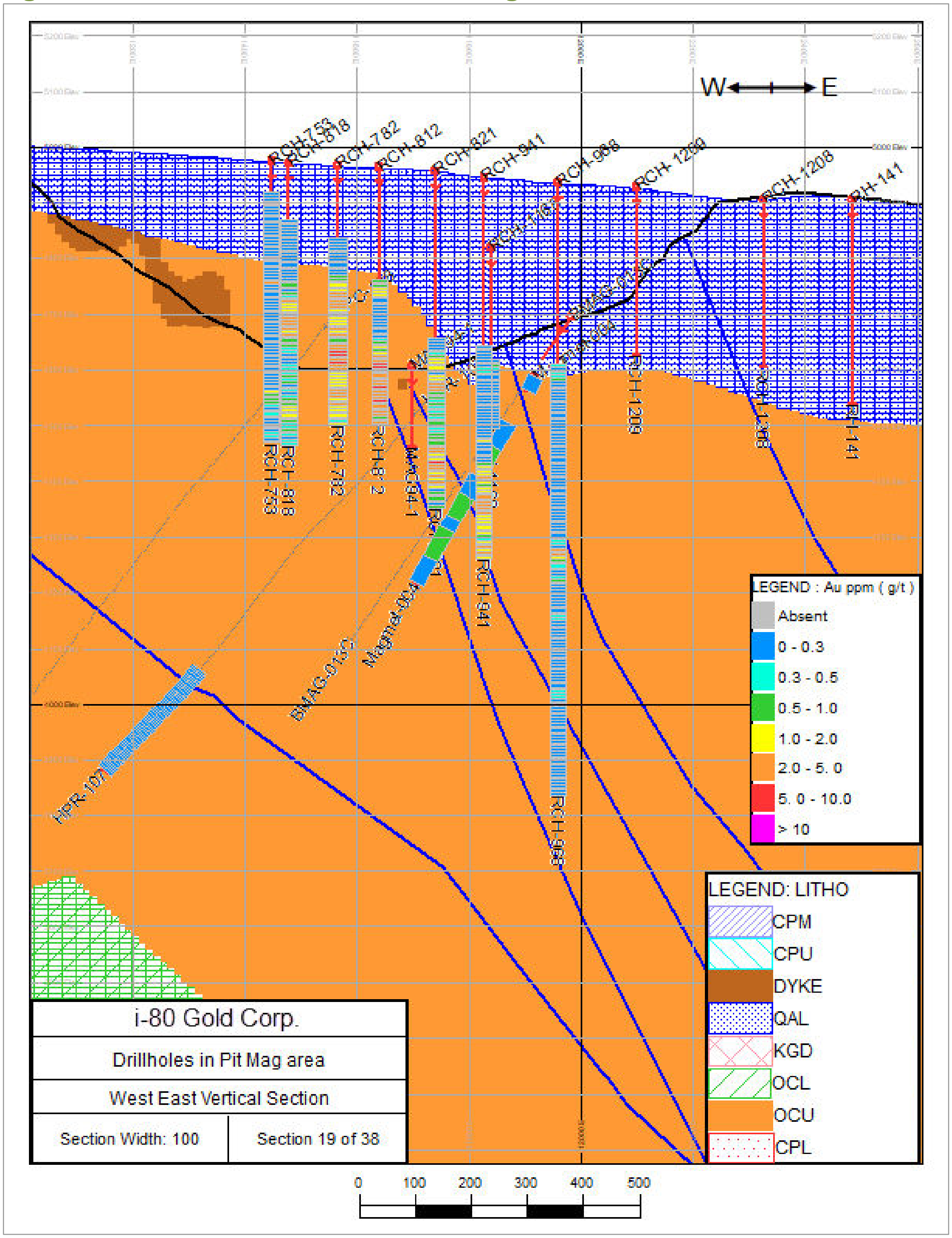
All drill results presented in Section 7.2 are from previous operators. i-80 had not conducted drilling on the Property at the time of the 2021 resource model.

Figure 7-11: Plan View Section Lines of Granite Creek Mine Project



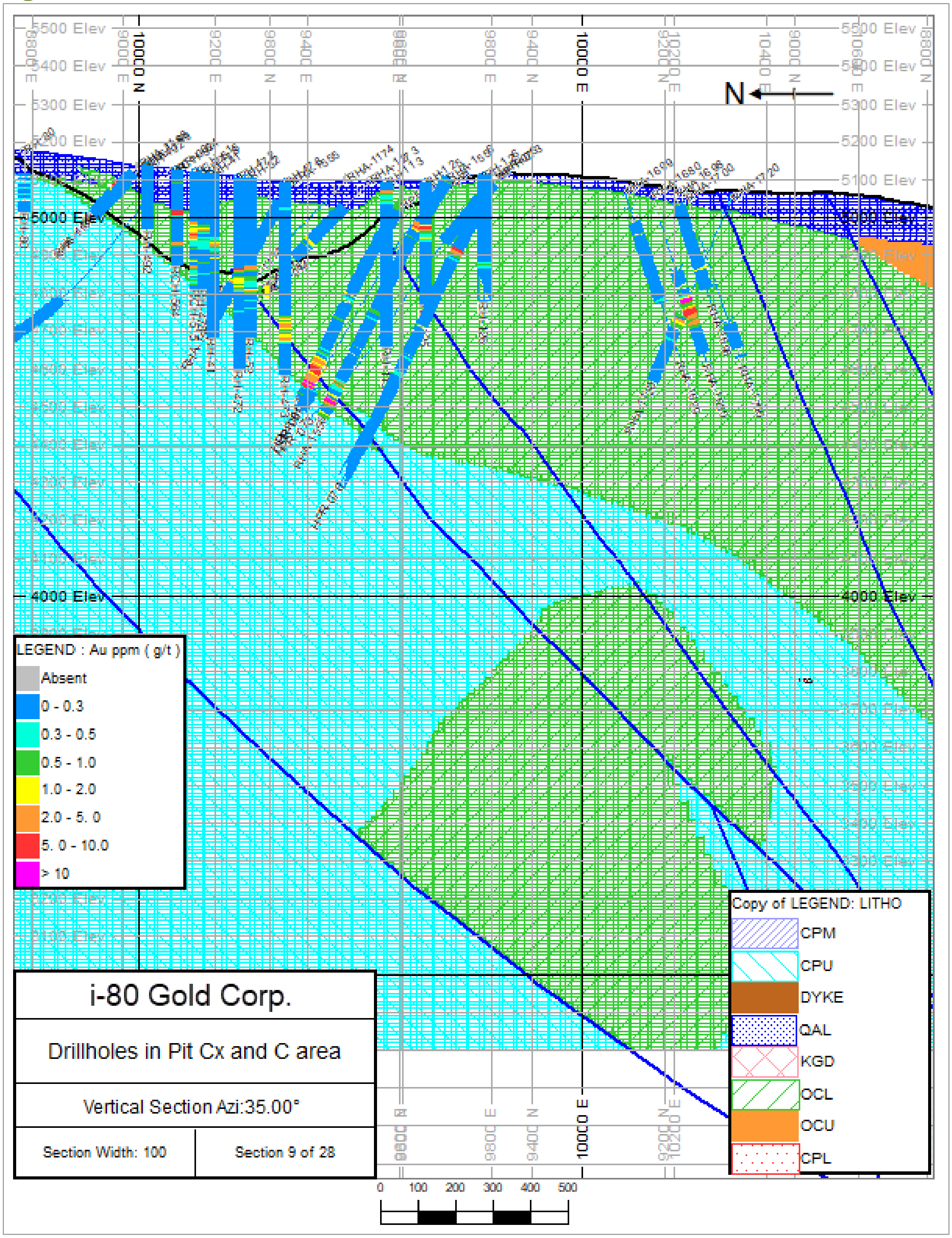
Note: Red outlines show the outline of the open pits.
Source: AMC Mining Consultants (Canada) Ltd 2019

Figure 7-12: Vertical Section A-A[1] of the Mag Pit Area



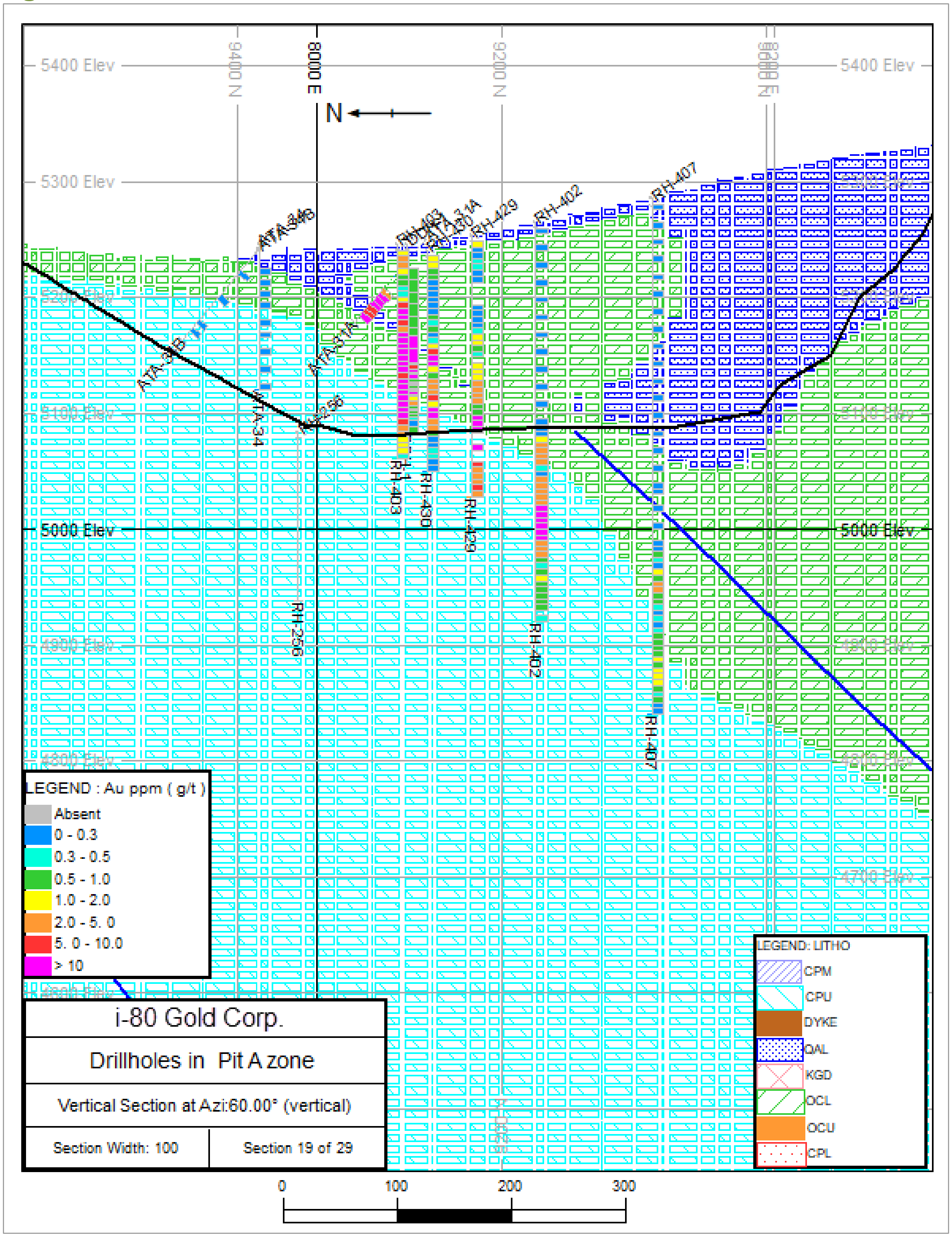
Notes: Blue lines are faults. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd 2019

Figure 7-13: Vertical Section B-B[1] of the Pit CX and C Area



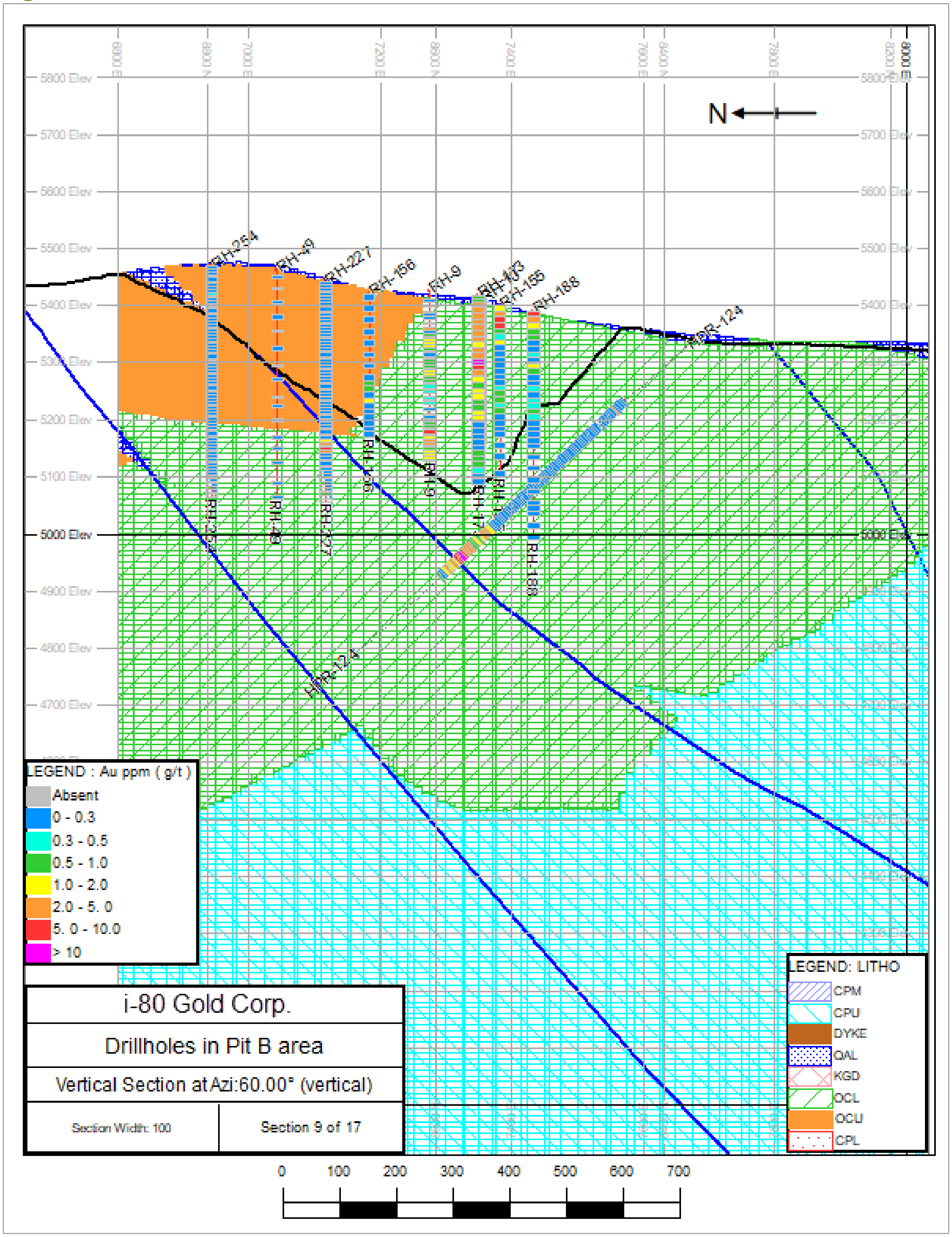
Notes: Blue lines are faults. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd 2019

Figure 7-14: Vertical Section C-C[1] of the Pit A Area



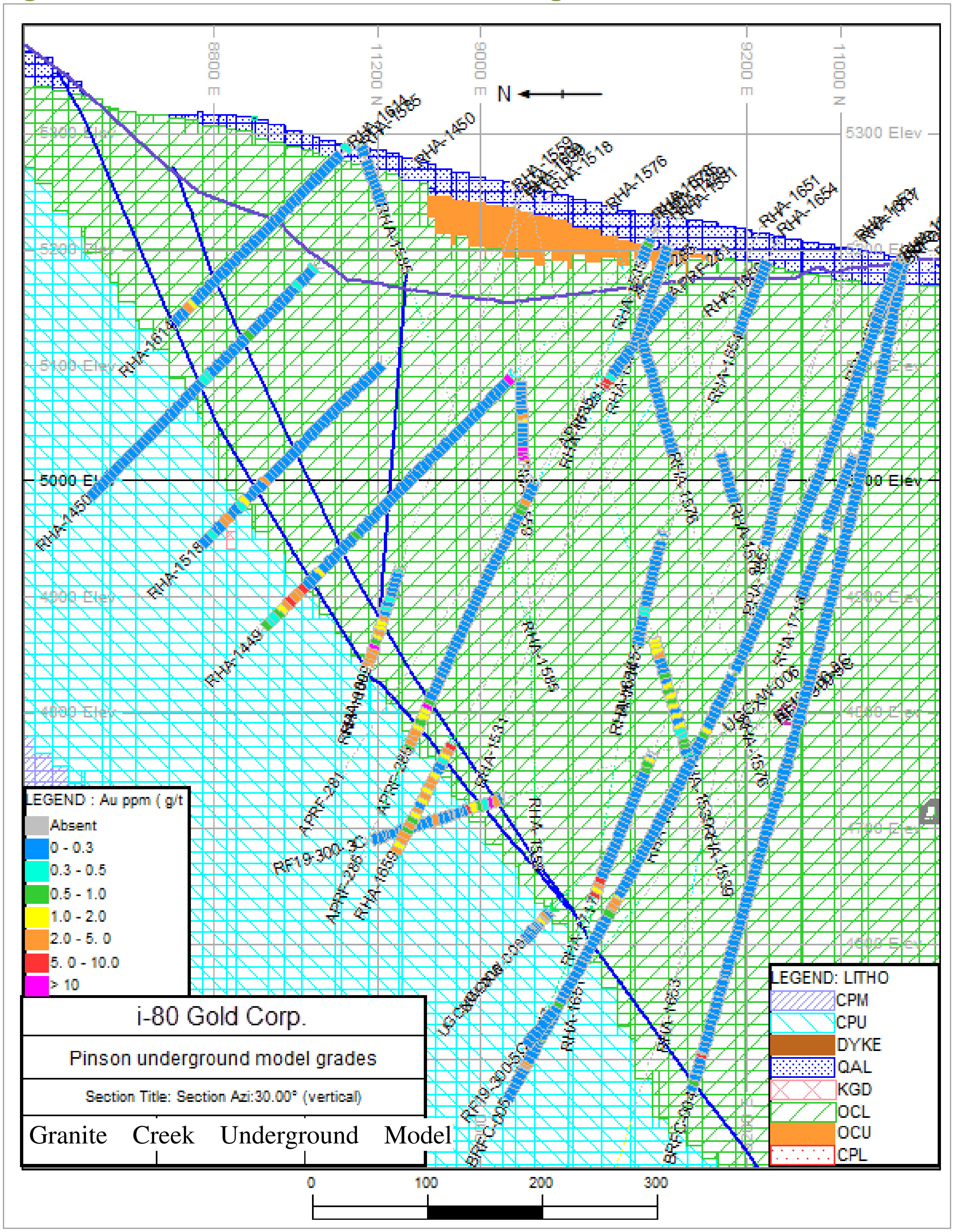
Notes: Blue line is a fault. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd 2019

Figure 7-15: Vertical Section D-D¹ of the Pit B Area



Notes: Blue lines are faults. Black line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd 2019

Figure 7-16: Vertical Section E-E¹ of the Underground Resource Area



Notes: Blue lines are faults. Top blue line is a topographic surface. Not all items listed in the legend are on all sections.
Source: AMC Mining Consultants (Canada) Ltd 2019

7.2.3 Drilling, Sampling, and Recovery Factors

There are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results. Drilling has been discussed in this section and sampling and recovery factors are discussed in Section 7.

7.3Update to Drilling Statistics to Include i-80 Drilling and Land Package Expansion

The Granite Creek land position has expanded since the previous resource estimate was published in 2021. This sub-section contains a summary of all holes drilled within the current land package, and the previous sub-sections describe in greater detail holes drilled within the core land package. No discoveries have been made beyond the core land package, and all holes outside the core land package were drilled by previous operator PMC. Holes without significant mineralized intercepts serve primarily to augment geological knowledge of the property and do not contribute to the resource estimation beyond defining where no resource exists. This section also details drilling completed by i-80, which had recently acquired the property and had not yet commenced drilling when the previous resource estimate was published. Table 7-12 lists number of holes drilled within the current property boundary by type and operator (includes holes drilled from surface and underground), and Figure 7-17 through Figure 7-19 show drilling by previous operators.

Table 7-12 Drillholes Within the Current Property Boundary by Type and Operator

Company	Core Holes (includes RC pre-collar with Core Tail)	Core Footage	RC Holes	RC Footage	Rotary Holes	Rotary Footage	Total Holes	Total Footage
i-80	225	152,941	1	2,100	0	0	226	155,041
Atna	113	68,334	201	49,920	0	0	314	118,254
PMC-Barrick	123	91,006	41	33,375	0	0	164	124,381
PMC	46	76,297	1,369	631,061	387	124,298	1,802	831,656
Totals	507	388,578	1,612	716,456	387	124,298	2,506	1,229,332

Figure 7-17 Drilling Completed by PMC

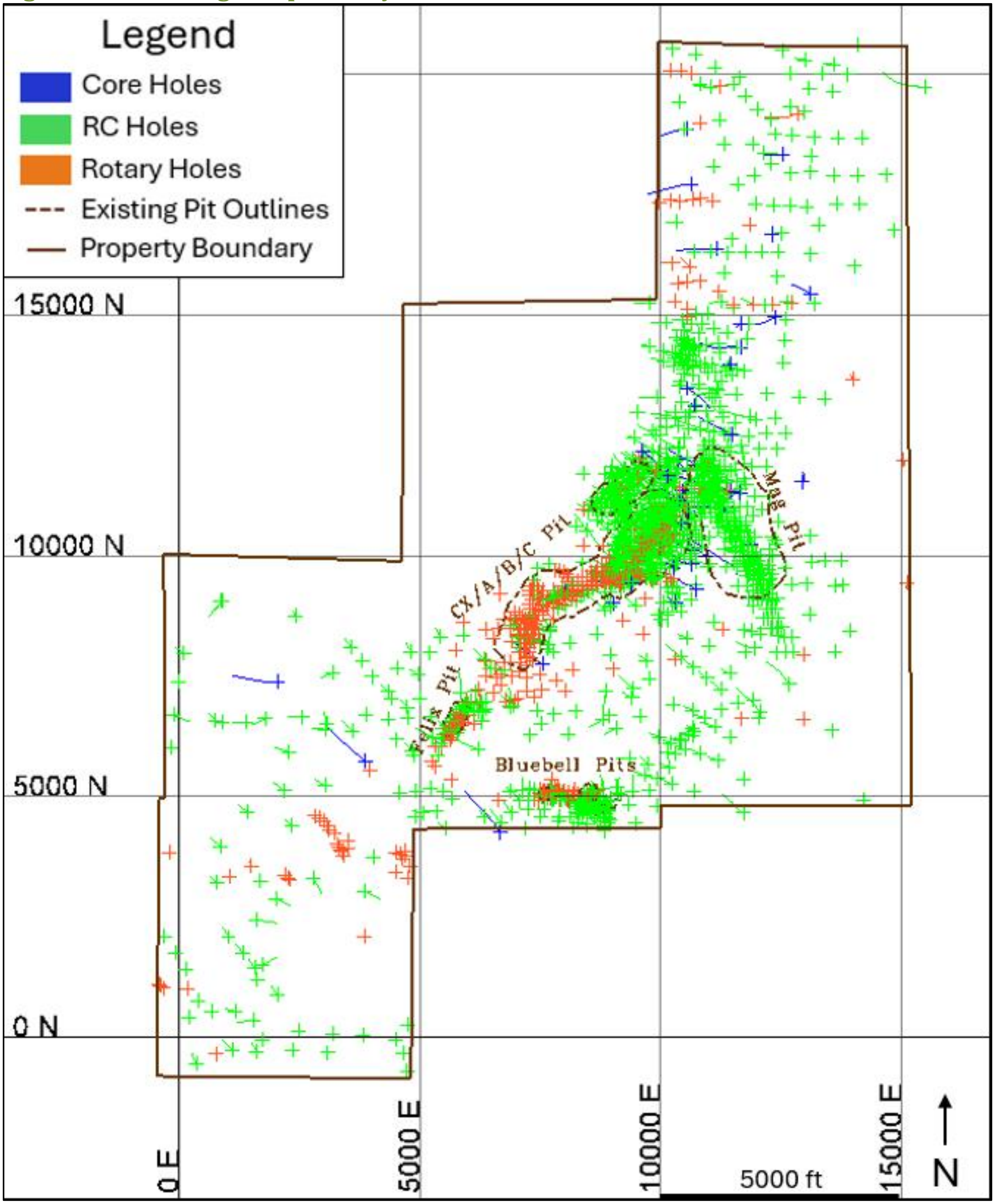


Figure 7-18 Drilling Completed by PMC with Barrick as Operator

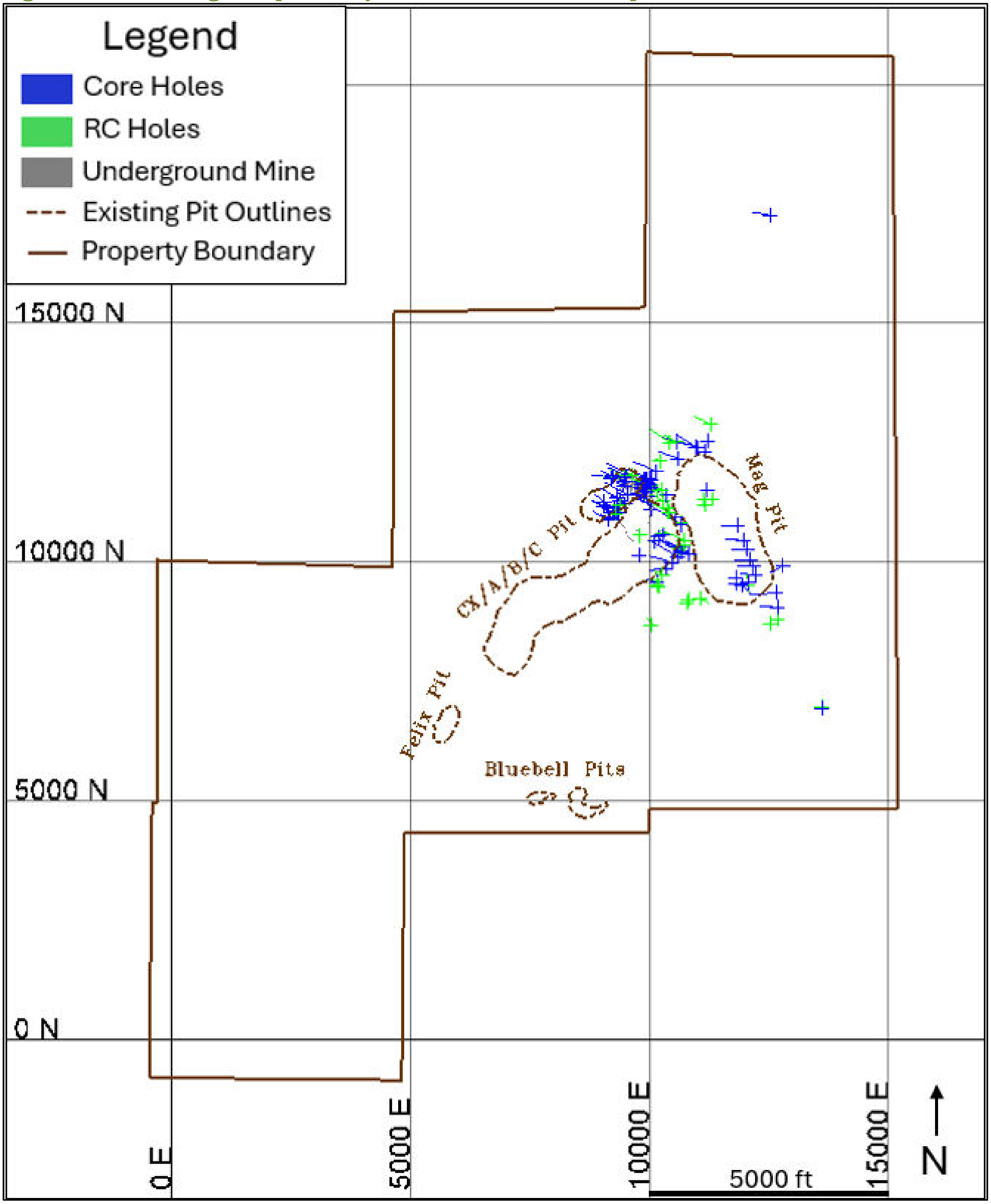
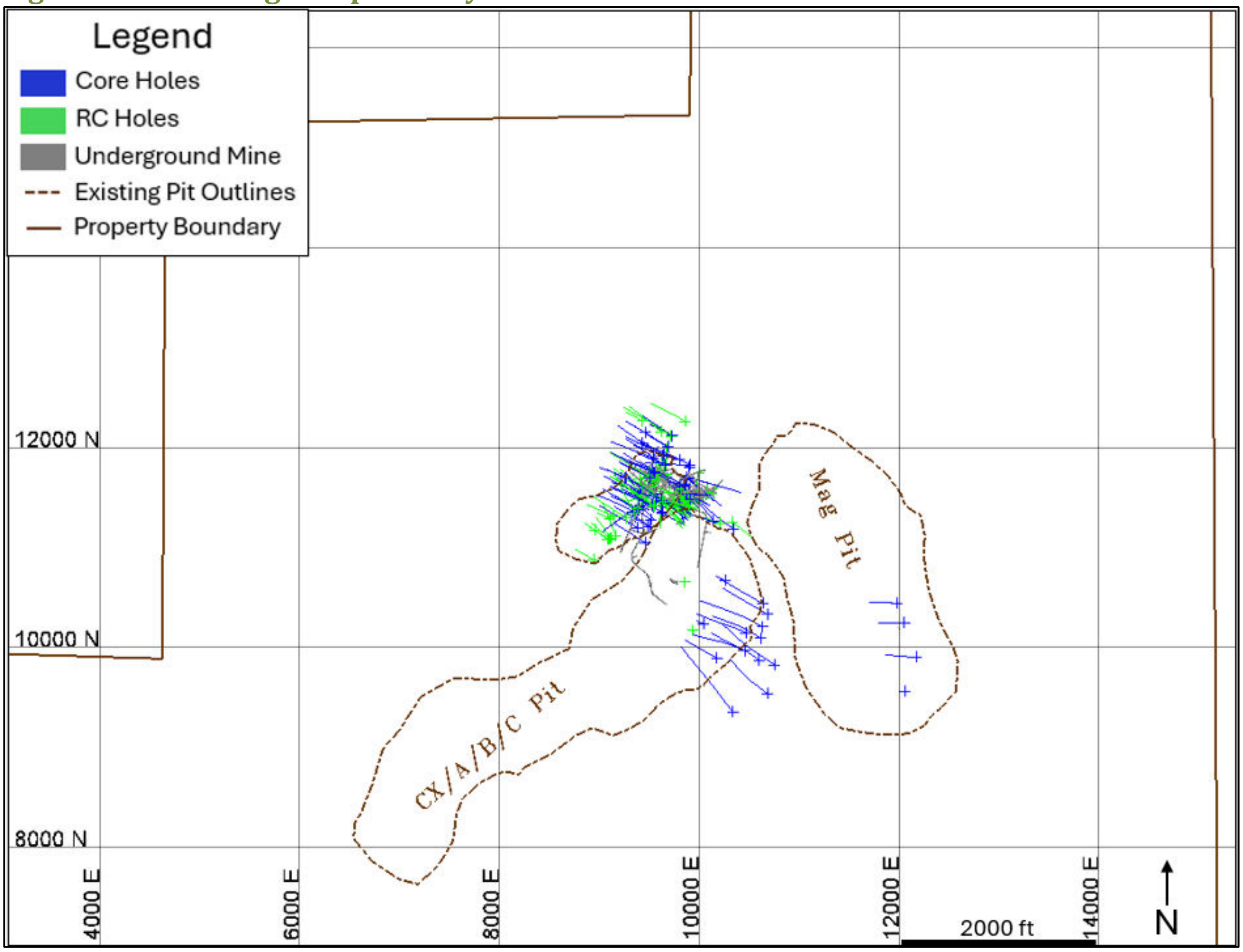


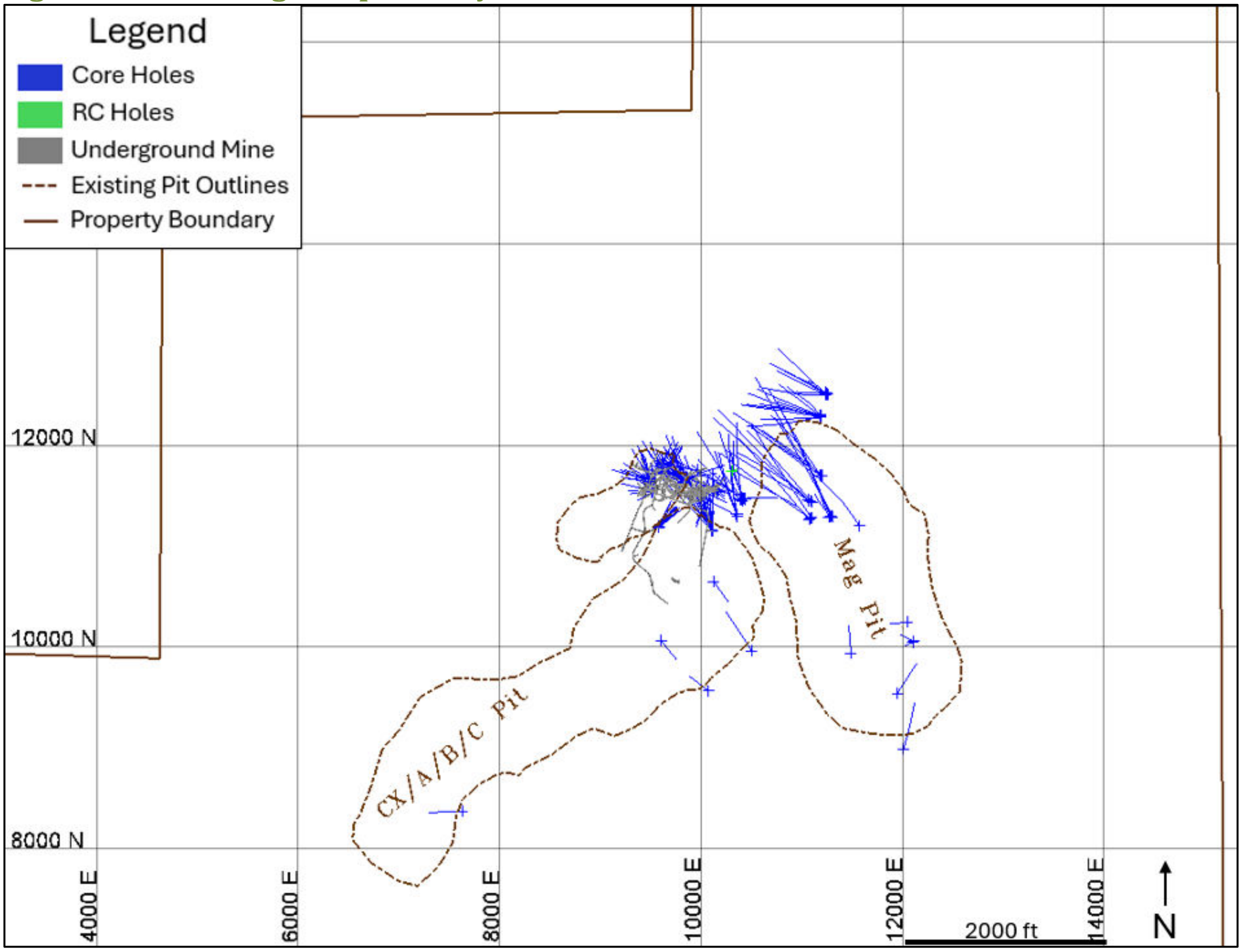
Figure 7-19 Drilling Completed by Atna



7.3.1 i-80 Drilling

Drilling at Granite Creek is ongoing. The holes drilled by i-80 presented in this section were drilled from April 2021 through December 2022 and had complete assay results by March 2023, which was the cutoff date for data to be included in the current underground resource estimate. i-80 primarily uses core drilling for sample collection, although one RC water well was drilled and sampled. Most surface holes are pre-collared using RC down to the water table, then completed with HQ size core. Most surface drilling has focused on the Ogee and South Pacific Zones. Underground holes were all drilled as HQ size core and focused on the Otto, Rangefront, Ogee, and South Pacific zones near the existing workings. A Cubex RC rig is used by ore control geologists to assist with short term mining decisions, but the RC holes are not merged into the resource database. Practical Mining recommends managing the underground RC drilling more attentively to ensure the results are of suitable quality for use in resource estimation. Figure 7-20 shows holes drilled by i-80.

Figure 7-20 Drilling Completed by i-80



7.3.2 Representative Cross Sections

Example sections showing drilling in the underground resource area are shown in Figure 7-22 through Figure 7-24. Holes drilled by i-80 are labeled with hole name and shown with thicker traces. Faults and mineralized envelopes modeled at 0.1 oz Au (3-gram) cutoff grade are shown for reference. Figure 7-21 shows the section locations.

Figure 7-21 Plan View Showing Section Locations through the Underground Resource Area

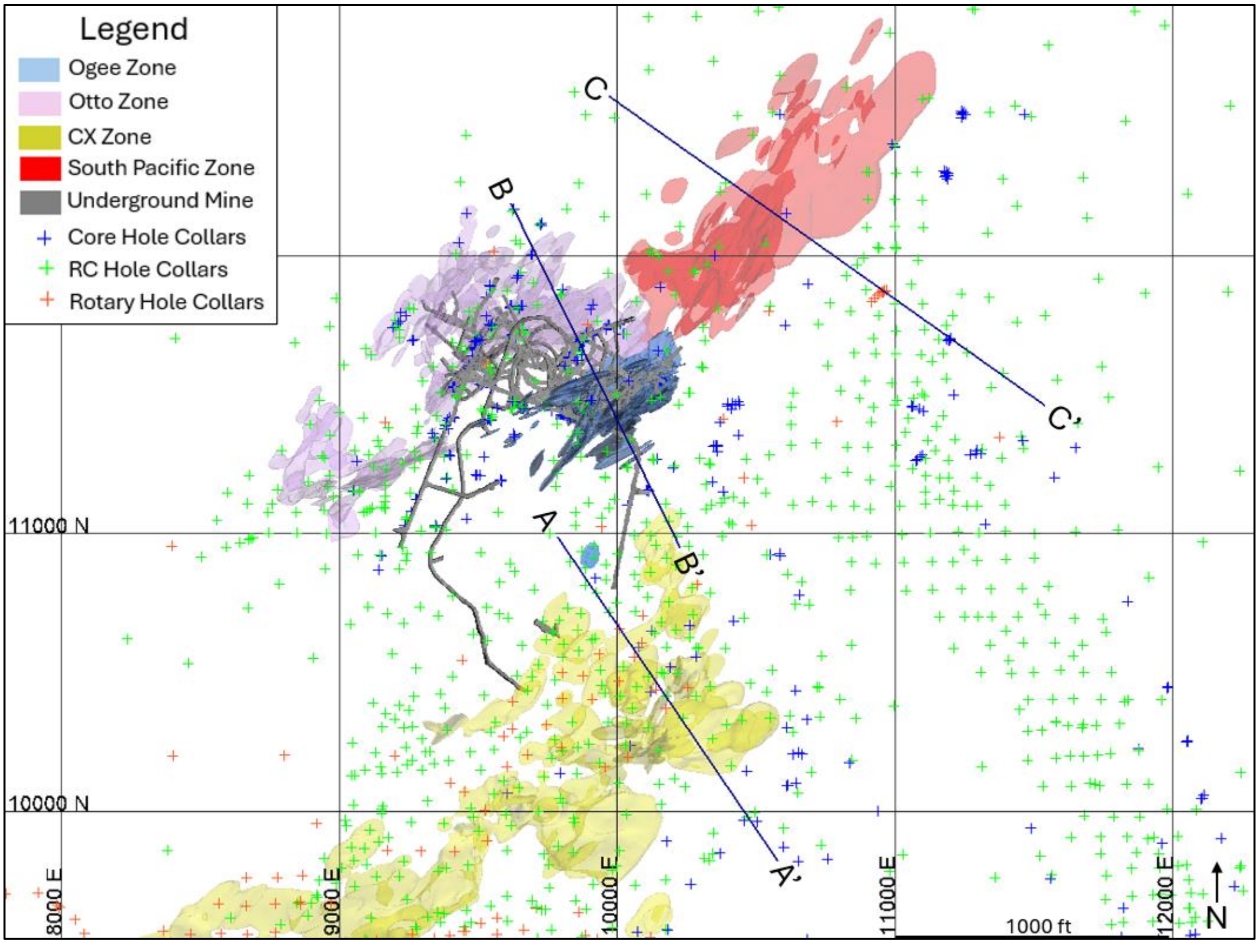


Figure 7-22 Section A-A' Showing Drilling in the CX Zone, 100 ft thick, looking North

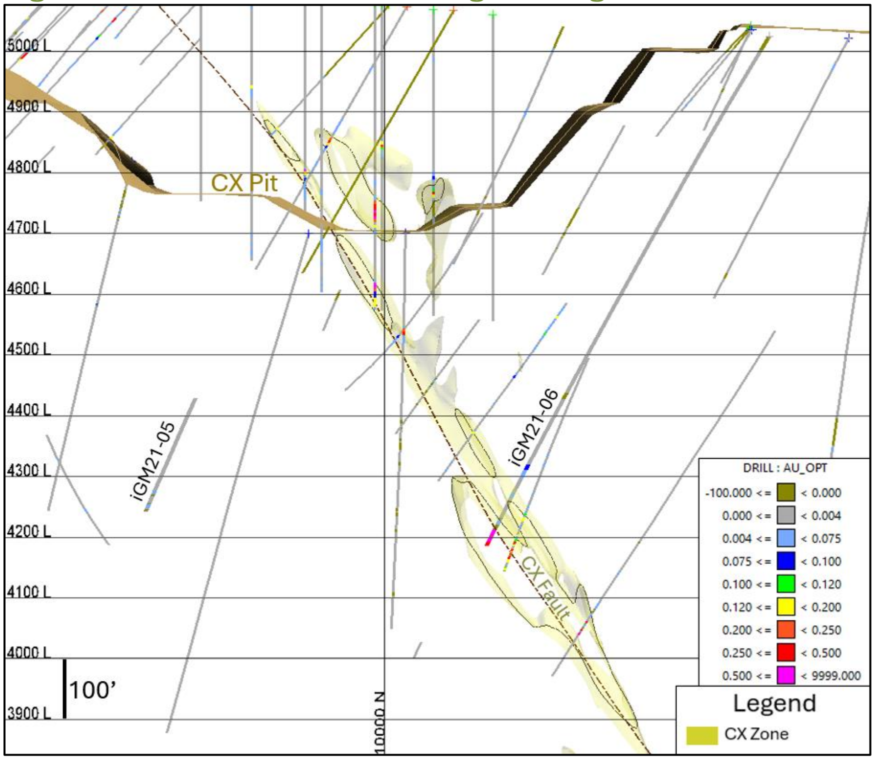


Figure 7-23 Section B-B' Showing Drilling in the Otto and Ogee Zones, 25 ft thick, looking North

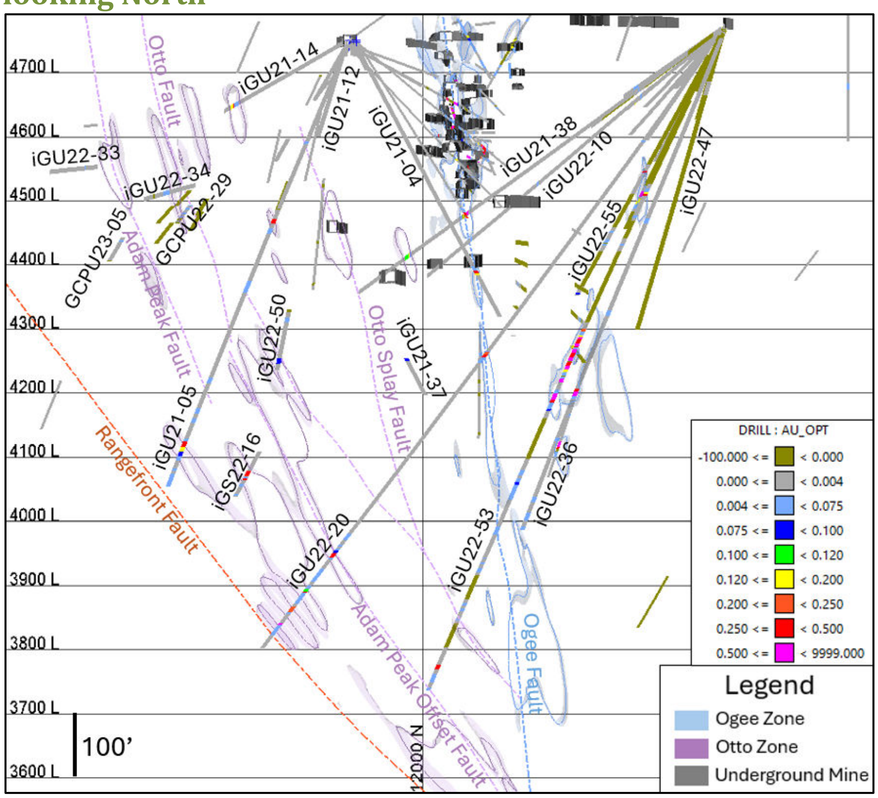
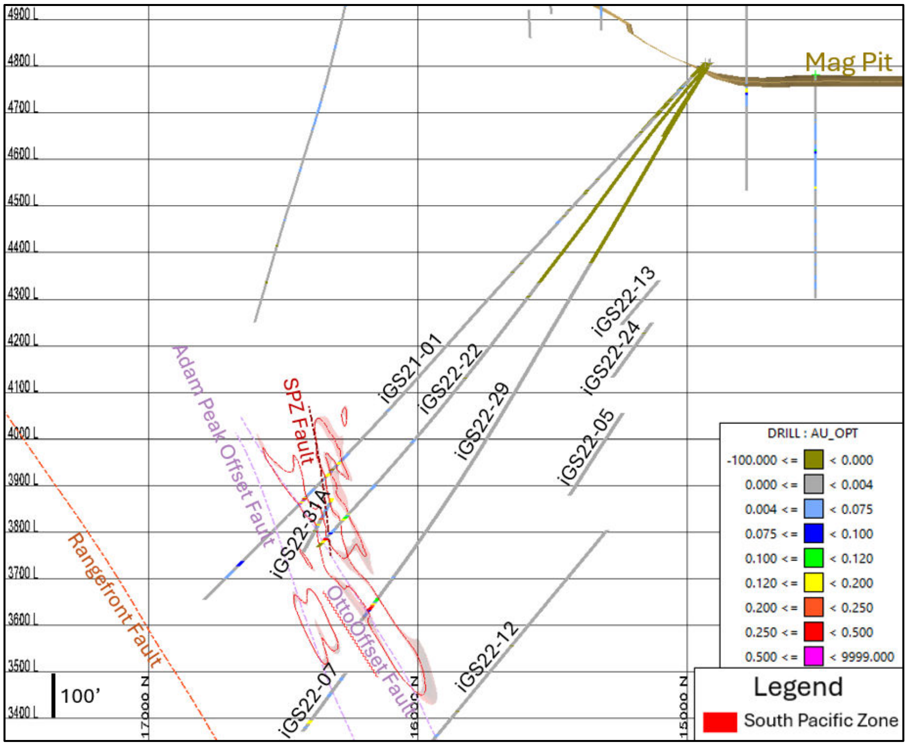


Figure 7-24 Section C-C Showing Drilling in the South Pacific Zone, 50 ft thick, looking North



7.4 Hydrogeology

7.4.1 Sampling Methods and Laboratory Determinations

Hydrogeological data, including water table measurements, pore pressure distribution and direction of groundwater flow, were normally collected in conjunction with exploration and geotechnical investigations in pre-construction studies and later from hydrogeological studies for on-going programs in pit and underground mining areas.

Groundwater dewatering and monitoring wells are the primary method of collecting hydrogeological data in support of mining operations, as well as the collection of pore pressure data which can be converted to groundwater level elevations from a network of vibrating wire piezometers (VWPs). Another source of data is hydrologic testing. Most wells that are drilled undergo hydrologic testing to establish aquifer parameters. These tests range from injection (slug) tests, air-lift tests, short-term and long-term pumping tests, and spinner logging. Data obtained from testing operations are analyzed using industry standard analytical methods. Analytical and numerical groundwater flow models have been developed using hydraulic parameters using testing results, in addition to 3D geological modeling.

From approximately 1980 through 2024 a total of 14 dewatering wells and 42 monitoring wells were completed in the Project area. In 2005, rapid infiltration basins (RIBs) were constructed east of the Project area to infiltrate groundwater pumped from dewatering operations into downgradient, permeable alluvial sediments. During 2022 and 2023, 19 vibrating wire piezometers were installed in the area underground mining operations (HGL, 2022 and 2024). To further assist in underground dewatering operations, one dewatering well was completed in 2023 (HGL, 2023) and another dewatering well was deepened in 2024 (LRE, in preparation) to capture additional groundwater yield. Currently, there are four active dewatering wells, 41 active monitoring wells, and 15 active vibrating wire piezometers across 5 locations (Figure 7-25). Current dewatering pumping rates range from 100 gpm to 750 gpm at the four dewatering wells. All dewatering wells are monitored, controlled and data are logged using a supervisory control data acquisition system (SCADA).

According to permitting requirements, 11 monitoring wells are sampled on a routine basis and analyses run for the State of Nevada Profile I suite at a certified analytical laboratory, currently Western Environmental Testing Laboratory (WETLAB), Reno, NV. Monitor wells and exploration drill holes that have piezometers installed are monitored for water levels and piezometric heads. Surface water is also measured and sampled on a routine basis as required by various permits.

7.4.2 Hydrogeology Investigations

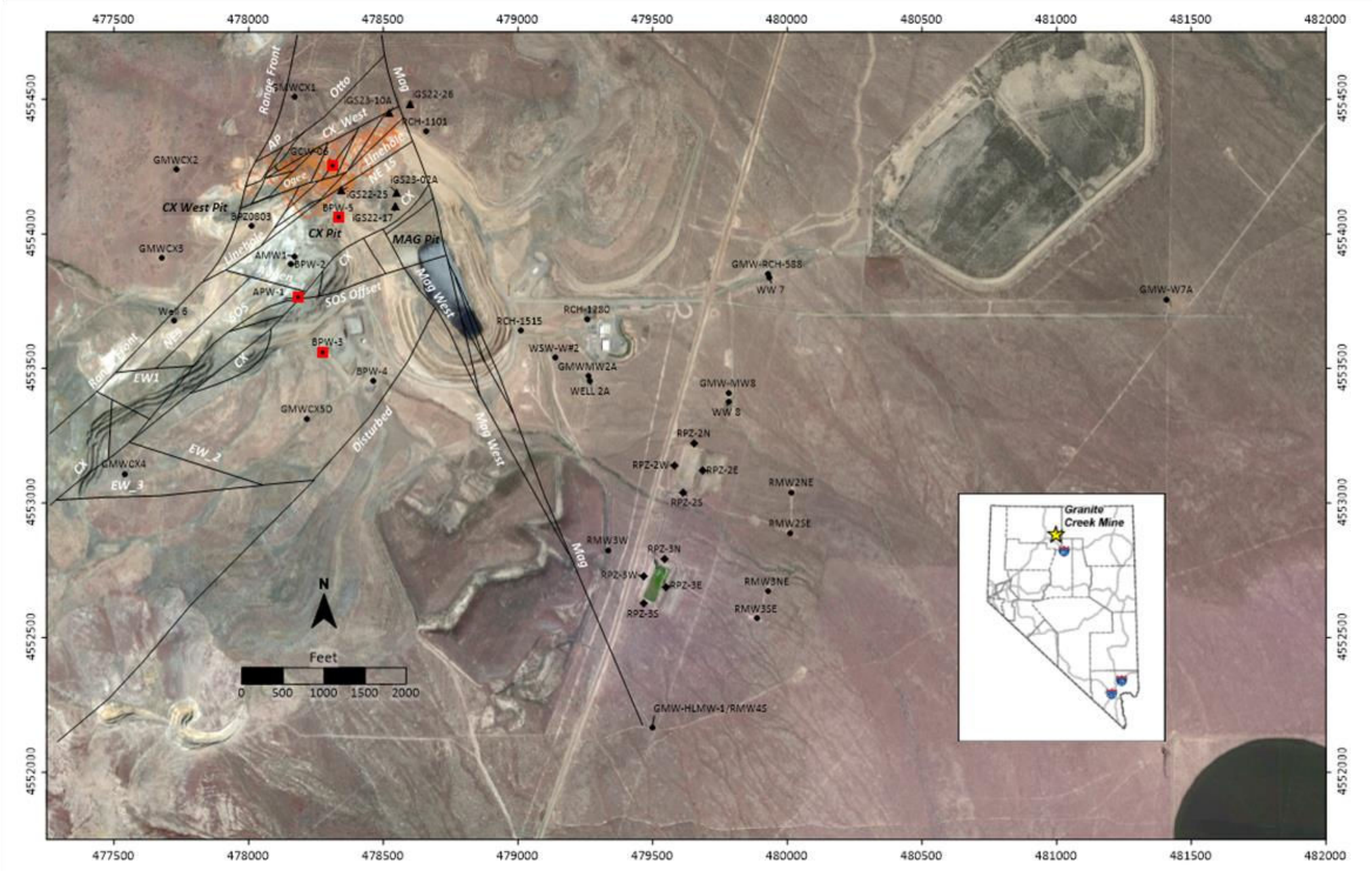
Throughout the span of various mine property owners and operators, the Project area has been the subject of multiple studies aimed at characterizing the hydrogeologic properties of the stratigraphy within the Project area and the surrounding region (Table 7-13). WMC (1998) established an early conceptual hydrogeological model and characterized the physical properties of major water bearing geologic units in the Mag and CX pit areas. Continuing in the early 2000s through 2018, additional hydrogeologic studies were completed by WMC, SWS, and Piteau Associates in support of groundwater monitoring, dewatering operations, water balances, and RIB design (WMC 1998, 2000, 2002, and 2005; SWS 2014; and Piteau Associates 2018).

More recently, i80 contracted HydroGeoLogica (HGL), now part of LRE Water, to conduct operations for monitoring of groundwater levels and pore pressures, plan and oversee operations of dewatering wells, and groundwater flow modeling for local-scale dewatering and regional scale permitting.

7.4.3 Hydrogeologic Description

The Granite Creek Mine is in the Great Basin region of the Basin and Range Physiographic Province. Mountain ranges trending north-south with parallel intermontane basins characterize the

Figure 7-25 Well Locations



Note: Active dewatering wells identified with red squares. Major structural faults are shown as black lines representing intersections at an elevation of 4,500 ft amsl. Current and planned underground workings are shown in orange

Table 7-13: Timeline for Hydrogeologic Characterization with Relationship to Mining

Year	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003
	A	A	A	A	A	A																		
			B	B	B	B	B	B	B															
									C	C	C	C	C	C	C	C	C							
								M	M	M	M	M	M	M	M	M	M	M	M					
											CX	CX	CX	CX	CX	CX	CX	CX	CX					
														CXW	CXW	CXW	CXW	CXW	CXW					
								~	~	~	~	~	~/	~/	~/	~/	~/	~/			\	\	\	\
																*	*	*	*	*	*	*=	*=	*=

Year	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	###	###	###
		UG	UG		UG	UG	UG	UG	UG	UG									UG	UG	UG	UG
	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\	\
	*=	*=																				
		>	>	>>	>>	>>	>>	>>	>>	>>	>>>	>>>	>>>	>>>	>>>	>>>	>>>	>>>	>>>>	>>>>	>>>>	>>>>
		RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB	RIB

Explanation

A Pinson Mining Company (PMC) began developing the A Pit in 1980 and produced gold in the following year; production years 1980 - 1985 (GRE, 2021).

B Production from the B Pit began in 1982; step-out drilling in 1982-1983 to the northeast of the A zone intersected 2 more discrete zones: the C zone extending ENE from the A zone, and the CX zone extending NE from the C zone; production years 1982 - 1988 (GRE, 2021).

C C zone production replaced the final B zone production in 1988; production years in 1988 - 1996 (GRE, 2021).

M Step-out drilling NE of the CX zone in 1984 located an apparently independent fault system (striking NNW, dipping steeply E) that became the core of the Mag Deposit; Mag Pit production years 1987 - 1999 (GRE, 2021)

CX CX production was delayed until 1990, owing to the slightly greater stripping requirement to reach ore; CX Pit production years 1990 - 1999 (GRE, 2021)

CXW Step-out drilling NW of the CX zone established a CX West zone subparallel to, and closer to, the Osgood Mountain front; CX West production years 1994 - 1999 (GRE, 2021)

Note: Pinson Mining Company produced from the CX, CX West and Mag pits as long as possible in the mid to late 1990s, until a combination of falling gold prices and erratic mill feed forced closure of its oxide millin early 1998. Continued attempts to expand production of oxide ore failed, and all active mining ceased on January 28,1999 (Sim, 2005). Development of A, B, C, and CX West pits occurred above the water table.

UG Underground exploration activities, 2005 through 2006: In 2005, building on prior exploration activities conducted by PMC and its partners, exploration activities were initiated by Atna in the vicinity of the CX Pit. As part of the exploration project, an exploration adit, and incline/decline, was driven from within the CX Pit. The CX portal was constructed through the northwest pit highwall on the 4,760 ft amsl bench. The CX incline/decline was driven toward the north-northwest for about 500 ft, after which it was continued for about 700 ft toward the west-northwest, intersecting the Line Hole/Ogee target zone. The incline/decline extended a total of 1,200 ft to a terminal elevation of about 4,785 ft (2% incline) with subsequent decline development originating in the Ogee zone (Piteau, 2018).

UG Active underground exploration activity was resumed by PMC in February 2008. Underground drifts were advanced from the existing 2005 incline/decline and a second portal was collared along the CX Pit west highwall south of the original portal. During 2008, approximately 12,000 ft of exploration drifting occurred through unmineralized zones for the purpose of providing access for underground exploration drilling. Underground exploration and mining operations, and backfilling of CX incline/decline non-ore rock material to the CX Pit resumed and continued intermittently from mid-February 2008. Underground mining operations were suspended at the end of June 2013 pending development of a revised operating plan commensurate with the current lower gold price (WMC, 2013). Production occurred from 2012 - 2013 consisting of 6,011 feet (1,832 meters) of primary and secondary development were completed during 2012-2013. The primary spiral ramp was driven to the 4530 level from the 4650 adit level, and both top cut and underhand ore mining occurred in three Ogee-zone stope blocks during development. Additional secondary access drifts were in progress when the mine was placed on care and maintenance to access the Range Front and Adams Peak mineral zones but were not completed prior to cessation of underground work. In May 2014, the status of the underground mine was changed to an intermittent production status. Under this status, periodic mining of ores from stoping areas developed in 2013 was conducted to develop and test revised stoping methods for the underground and to prove mining economics at small production rates (GRE, 2021).

UG i-80 development of UG.

~ During the period between 1987 and 1991, only sumps were used to dewater Mag pit. The pumping rate was small and included only minor intermittent pumping from bedrock sumps in the pit floor (WMC, 1998 and 2005).

~/ Continuous pumping using dewatering wells commenced in late 1991 when Well #12 was brought on-line and later replaced by Well #13 (WMC, 2000). Several west-northwest and east-southeast cross faults were exposed within the pit. These faults created low permeability barriers to groundwater flow across the main structural trend. The dewatering rate for the pit was typically within the range 400 to 700 gpm, with much of the water derived as a result of seepage faces from the alluvium.seepage faces from the alluvium. The maximum sustained dewatering rate (wells and sumps) was less than 1,000 gpm. The Mag Pit dewatering system was shut down during October 1997 (WMC, 2000 and 2005).

\ The Mag Pit lake was rapidly filled during 2000 using a combination of water from the CX pumping system (which was operational at the time) and water from alluvial water supply wells #7 and #8. Rapid filling of the lake was initiated on February 24, 2000, and completed on August 17, 2000 (175 days). A total of 200.4 million gallons was pumped into the lake, which raised the lake elevation from 4,606 ft amsl to approximately 4,671 ft amsl. Current (10/2025) pit lake level is approximately 4631 ft amsl.

* The CX Pit was mined at various rates between 1995 and January 1999. The lowest point on the pit rim is about 5,030 ft. The lowest elevation of mining in the CX Pit was 4,686 ft. During mining, groundwater inflows to the CX Pit were first encountered during late 1997 while mining the 4,720 ft bench. A dewatering program for the pit was commenced in April 1998. Dewatering was initially carried out using sumps. Subsequently, a combination of sumps and borehole drains installed in the pit floor were used. Most sumps were about 10-15 ft deep. Although mining was suspended in January 1999, the dewatering system remained operational through August 2000 to provide access for equipment, for mineral exploration drilling activities from the pit floor, and to provide water supply for the Mag Pit rapid filling operations. Most borehole drains were 170 ft deep. The dewatering rate was typically within the range 300-1,200 gpm (WMC, 2005)

*= The dewatering system for the CX Pit was shut down in August 2000 and water levels began to rise beneath the excavated CX Pit. By December 2005, the CX pit lake in the adjacent groundwater block reached an elevation of 4,713 ft (WMC, 2013).

> Dewatering operations to lower the groundwater elevation below the anticipated underground exploration and development commenced in December 2005 (Well APW-1). Groundwater levels in the bedrock hydrologic block containing the CX pit were lowered to an elevation below the pit floor and the small pit lake in the lower pit was dewatered. Dewatering operations were suspended in April 2006 and the water level in the Pinson Fault Shear Zone hydrologic unit recovered naturally to a point where water was again impounded in the CX Pit beginning in May 2006 (WMC, 2013).

>> Dewatering operations resumed in August 2007 (Well APW-1) in support of underground exploration activities resumed in February 2008. Over the next year, approximately 12,000 ft of exploration drifting occurred for the purpose of providing access for underground exploration drilling (WMC, 2013).

>>> Care and maintance dewatering of UG (APW-1, BPW-3, and BPW-5)

>>>> Dewatering well upgrades (APW-1, BPW-3, BPW-5), drilling and construction of APW-1, and deepening of BPW-5.

RIB Rapid Infiltration Basins became effective October 2005 under Water Pollution Control Permit (WPCP) NEV2005102. Well APW-1 became operational December 13, 2005 discharging to RIB. Four RIBs are permitted for a total of 6,900 gpm. To date, two RIBs (RIB2 and RIB3) have been constructed and placed into operation.

terrain. The entire region is a closed drainage system with all the permanent streams flowing to interior "sinks" such as the Carson and Humboldt sinks, or interior lakes such as Pyramid and Walker. Elevations in the area range from about 4,000 ft amsl in the basins, to over 9,000 ft amsl in the surrounding ranges. The local terrain near the mine area is in the transition from the bedrock, mountain front zone to the alluvial, basin-fill zone.

7.4.3.1 Surface Water

The Granite Creek Mine is in the Kelly Creek Hydrographic Area of the Middle Humboldt Watershed that lies within the Humboldt Basin. The Middle Humboldt has a catchment area of approximately 3,200 square miles draining to the Humboldt River to the southwest.

Granite Creek is an ephemeral stream sourced from seasonal snow melt originating in a bowl below the crest of the Osgood Mountains above Granite Creek Canyon, immediately west of the Project. Stream flow, when present, is currently diverted and routed through a series of pipes and culverts above the south part of the CX Pit and rejoins the original stream channel about 1,100 feet southeast of the CX Pit. The water in Granite Creek typically infiltrates into the permeable alluvial deposits of the middle and lower pediment slopes within 1,500 to 2,500 feet down gradient of the site. Annual stream flow generally occurs from February through July. Generally, no flow is observed in the channel west of the project by the end of June. Based on previous estimates, mean average annual flow is estimated at 0.28 cubic feet per second (cfs) (200 acre-ft/yr or 125 gpm) with springtime flows averaging about 1.9 cfs.

High flow rates in Granite Creek occur during the spring runoff period. When peak seasonal stream flow exceeds the diversion capacity, the overflow has historically been routed via a pipeline to the floor of the mined-out A and B pits. The surface water has infiltrated rapidly through the pit floors and into the CX shear zone, which is hydrologically connected to the CX Pit and the underground development. During spring 1998, a period of record precipitation in northern Nevada, and following peak runoff events, a rise in groundwater levels was noted beneath the floor of the CX Pit necessitating an increase in the dewatering rate from the CX Pit.

Permitting requires annual sampling of surface water at two stations in the Granite Creek channel, when present during the first or second quarters; one station upgradient from the mine site and the other station downgradient of the mine site at the eastern property boundary. Results of hydrochemical analysis of Granite Creek samples indicate that the average chemistry in Granite Creek is similar to that of the CX shear zone bedrock groundwater hydrologic unit (also referred to as the CX hydrologic block) as discussed further in the following section. Collected samples generally report constituent concentrations that meet NDEP RVs.

7.4.3.2 Groundwater

Groundwater generally moves from recharge areas along the range front of the Osgood Mountains to the east-southeast, towards the central part of the Kelly Creek Valley basin. Two main groundwater systems are recognized in the mine area: 1) alluvial units and 2) and sedimentary and metamorphic bedrock units:

- Alluvial Groundwater System: groundwater moves south – southeast in the alluvial deposits throughout the entire Kelly Creek Valley basin towards the Humboldt River. The alluvial groundwater system in the mine area has been historically monitored by numerous monitoring wells and is well understood.

  Saturated alluvium exists on the east and southeast sides of Mag Pit and its thickness increases significantly to the east of Mag Pit toward the lower elevation alluvial basin. There is no saturated alluvium to the north and west of the Mag Pit or in the immediate vicinity of the CX Pit. Groundwater flow within the alluvium is generally characterized as porous media flow. Hydraulic layering of the alluvium is known to occur, controlled in part by the presence of lower permeability horizontal lenses of fine-grained materials. There is a general increase in the proportion of fine-grained materials closer to Mag Pit. The saturated thickness of the alluvium increases to the east as the underlying bedrock surface dips beneath the valley floor. The pre-mining alluvial water table in the area of Mag Pit was 4,654 ft amsl. The lowest occurrence of alluvium in the east wall of the Mag Pit is about 4,590 ft amsl. The maximum saturated alluvial thickness in the east wall of the Mag Pit was therefore about 65 ft prior to mining Mag Pit.

- Bedrock Groundwater System - groundwater flow within the bedrock units in both the Mag and CX pit areas is predominately controlled by stratigraphy and geologic structure. The bedrock hydraulic properties are therefore highly variable, and flow is dependent on the frequency and alignment of open fracture sets. Major faults, some displaying individual offsets on the order of several hundred feet, function as significant hydrologic features by: providing offsets, juxtaposing geologic units of differing hydrologic characteristics, providing preferred pathways for groundwater flow in fracture zones parallel to the fault plane, and forming gouge filled barriers to horizontal flow perpendicular to the fault plane. These characteristics favor the formation of hydraulically isolated blocks of bedrock bounded by the high angle faults in the Project mine area.

Regionally, groundwater recharge occurs to both the alluvium and the bedrock of the upper piedmont slope elevations and, during years of high run-off, to the alluvium at middle and lower piedmont slope elevations. Groundwater moves towards the center of the basin in the thickening sequences of alluvial deposits. Most natural discharge from the basin occurs through evapotranspiration from the alluvial deposits beneath the valley floor. Locally, historical and current dewatering of the Mag Pit and CX Pit, formation of the Mag Pit lake, as well as the underground mine workings exploration and development to facilitate mining of deeper ore reserves north of CX Pit, has influenced direction of groundwater movement in the vicinity of the mine. Local groundwater movement is also influenced by delivery of water from dewatering operations to a rapid infiltration basin (RIBs) constructed in the alluvial aquifer system for downgradient recharge to the basin.

7.4.4 Mine Dewatering

7.4.4.1 Mag Pit

The Mag Pit was mostly excavated in calcareous mudstones, siltstones, carbonaceous shales with interbedded limestone units exposed in the lower east and southeast walls. The rocks are mostly fine grained and belong to the Upper Comus formation, with locally silicified zones that are present along many of the cross faults. Intrusive rocks and breccias are also common in the pit area and exhibit a high degree of argillization. The main geological structures in the Mag Pit show a strong north northwest-south southeast alignment. During dewatering operations for mining of the pit, groundwater level drawdown was seen to extend along this trend to the northwest and southeast of the pit, but the extent of drawdown along strike was limited by the presence of cross faults.

The Mag Pit area is overlain by basin-fill alluvium, which increases from a thin veneer (less than 30 ft thick) above the west wall to over 200 ft thick along much of the east wall. The alluvial-bedrock contact is down-dropped to the to the east immediately behind the east pit wall. Locally, in the pit area, the alluvium was reported to be fine grained and low permeability close to the east pit wall, which was notably different to the high permeability alluvium penetrated by the mine-water supply wells located in deeper basin sediments east of Mag Pit. Reporting postulated that the alluvium close to the Mag Pit area was locally influenced by debris flow material resulting in lower permeability.

Dewatering the Mag Pit occurred during the period 1987 through April 1998. During the initial period between 1987 and 1991, only sumps were used to dewater the pit. The pumping rate was small and included only minor intermittent pumping from bedrock sumps in the pit floor. In 1991, in pit mining in the lower pit became increasingly difficult because of fracturing and water strikes in the central pit area, consequently dewatering wells #12 and #13 were installed to the north and south of the central pit area, along the main strike of the north northwest-south southeast structures. Several west-northwest and east-southeast cross faults were exposed within the pit. These faults created low-permeability barriers to groundwater flow across the main structural trend. The dewatering rate for the pit was typically within the range 400 to 700 gpm, with much of the water derived as a result of seepage faces from the alluvium. The dewatering of the Mag Pit ceased in April 1998 and water was allowed to start accumulating in the base of the pit.

In June 1998, the standing water in the base of the pit was 4,550 ft amsl, about 117 ft lower than a 2002 stabilized elevation of the lake of 4,667 ft amsl (WMC, 2002). The Mag Pit lake was rapidly filled during 2000 using a combination of water from the CX pumping system (which was operational at the time) and water from alluvial water supply wells #7 and #8. Rapid filling of the lake was initiated on February 24, 2000, and completed on August 17, 2000 (175 days). A total of 200.4 million gallons was pumped into the lake, which raised the lake elevation from 4,606 ft amsl to approximately 4,671 ft amsl. In December 2024, i80 reported a lake level of 4632.78 ft amsl, about 34- ft below the measurement reported in 2002.

Prior to the start of rapid filling activities, approximately 1,620,000 yd3 of mainly alluvial material was backfilled into the northern end of the pit. The material was placed up to an elevation of approximately 4,625-4,640 ft amsl. The goal of the backfilling was to 1) buttress the unstable northwest and north pit wall, 2) isolate part of the geochemically reactive subunits of Upper Comus in the lower part of the pit, and 3) provide a source of additional alkalinity for the juvenile lake waters. The placed material reduced the area of the lake from approximately 15.8 acres to 12.2 acres.

In summary, the Mag Pit is hydraulically connected with both the local bedrock and regional alluvial groundwater system. During the past 15 years, the Mag Pit lake level has declined consistently with the alluvial groundwater levels, potentially caused by a combination of factors: 1) increased irrigation pumping from the central part of the Kelly Creek groundwater basin to the east and southeast, 2) lower than average precipitation and reduced recharge along the east side of the Osgood Range mountain front, and 3) leakage of groundwater into the highly fractured rock of the CX faulted zones due to dewatering operations for the CX Pit through low-permeable, Mag Pit fault barriers. Based on available data and analyses, the cause of lowering of the Mag Pit lake is likely due to a combination of these factors.

7.4.4.2 CX Pit and Underground Mine Workings

The structural alignment and geometry of the CX Pit differs from the Mag Pit area. The CX Pit lies along the line of the east-northeast-trending, steeply southeast dipping faulted shear zones where the Lower Comus Formation consists of variably metamorphosed, inter-bedded carbonaceous shale and limestone. These sedimentary rocks have been altered by metamorphic processes, producing calc-silicate marble zones within the limestone and argillite. A later period of alteration, associated with mineralization events, has caused localized decalcification, kaolinization and silicification, but there remains a considerable about of unaltered limestone within the middle and upper walls of the pit. After all alteration events, the rocks in the CX pit have been pervasively oxidized by meteoric waters.

Alteration and mineralization associated with the CX pit occurs along and adjacent to the shear zone. The shear zone is an important structural element that allows groundwater flow along strike within the CX bedrock block and contains the bulk of CX pit.

The adit and decline in the CX Pit was collared in 2005. The decline passes through steeply-dipping, tightly folded rocks of the Lower Comus formation and penetrates the faulted axis of an upright anticline along the axis of the Line Hole target area. This structure controls the location of a strongly altered and mineralized zone containing abundant iron oxide. The fault controlling the Line Hole Extension mineralization dips steeply to the northwest and has a reverse (southeast down) sense of movement. This zone is now referred to as the Ogee Zone with respect to the underground workings. Bedding on the south side of the fault typically dips steeply towards the southeast, whereas bedding on the north side of the fault dips steeply towards the northwest. Most underground development and mining in the CX pit has occurred within unmineralized rocks of the Upper and Lower Comus formation.

Early dewatering of the CX pit was also carried out using a system of sumps down to the final floor elevation of approximately 4689 ft amsl. In 2005, underground exploration activities were initiated by Atna Resources in the vicinity of the CX pit under an exploration agreement with Pinson Mining Company (PMC). An exploration incline and decline were driven from within the CX pit, with the portal collared in June 2005 on the northwest portion of the 4,760 ft amsl bench. Atna's exploration program was suspended in April 2006. In February 2008, PMC resumed underground exploration activities from the 2005 decline.

Dewatering operations to lower the groundwater elevation below the anticipated underground exploration and development commenced in December 2005. Groundwater levels in the bedrock hydrologic block containing the CX pit were lowered to an elevation below the pit floor and the small pit lake in the lower pit was dewatered.

The CX block has subsequently been dewatered from 2007 through present to facilitate underground exploration of deeper ore zones. There are currently four operating dewatering wells APW-1, BPW-3, BPW-5 and GCW-6 (Figure 7-25) pumping from the CX block hydrogeologic unit at a combined average rate of approximately 1,450 gpm, with an additional pumping of approximately 1,000 gpm collected from sumps in the underground mine workings. This combined pumping rate has caused groundwater levels to decline to an elevation of approximately 4,310 ft amsl at the face of current decline mining operation.

7.4.5 Dewatering Treatment and Discharge

Water from the dewatering wells that is not utilized for operations is currently discharged to Rapid Infiltration Basins (RIBs) on the east side of Getchel Mine Road through HDPE pipelines. Two of the four permitted RIBs (NEV2005102) have been constructed to date, with discharge to one of the two cells at any given time. When RIB maintenance is required, discharge is routed to the dormant cell. Current dewatering efforts are well under the permitted 6,900 GPM threshold of the RIBs and the RIB infiltration is sufficiently limiting surface ponding in the active cell.

Due to arsenic levels that are above Nevada Profile 1 standards, BPW-5 and GCW-6 require treatment prior to discharge to the RIB's. Currently, the Water Treatment Plant (WTP) processes 380 GPM from BPW-5 and 110 GPM from GCW-6.

7.4.6 Groundwater Flow Model

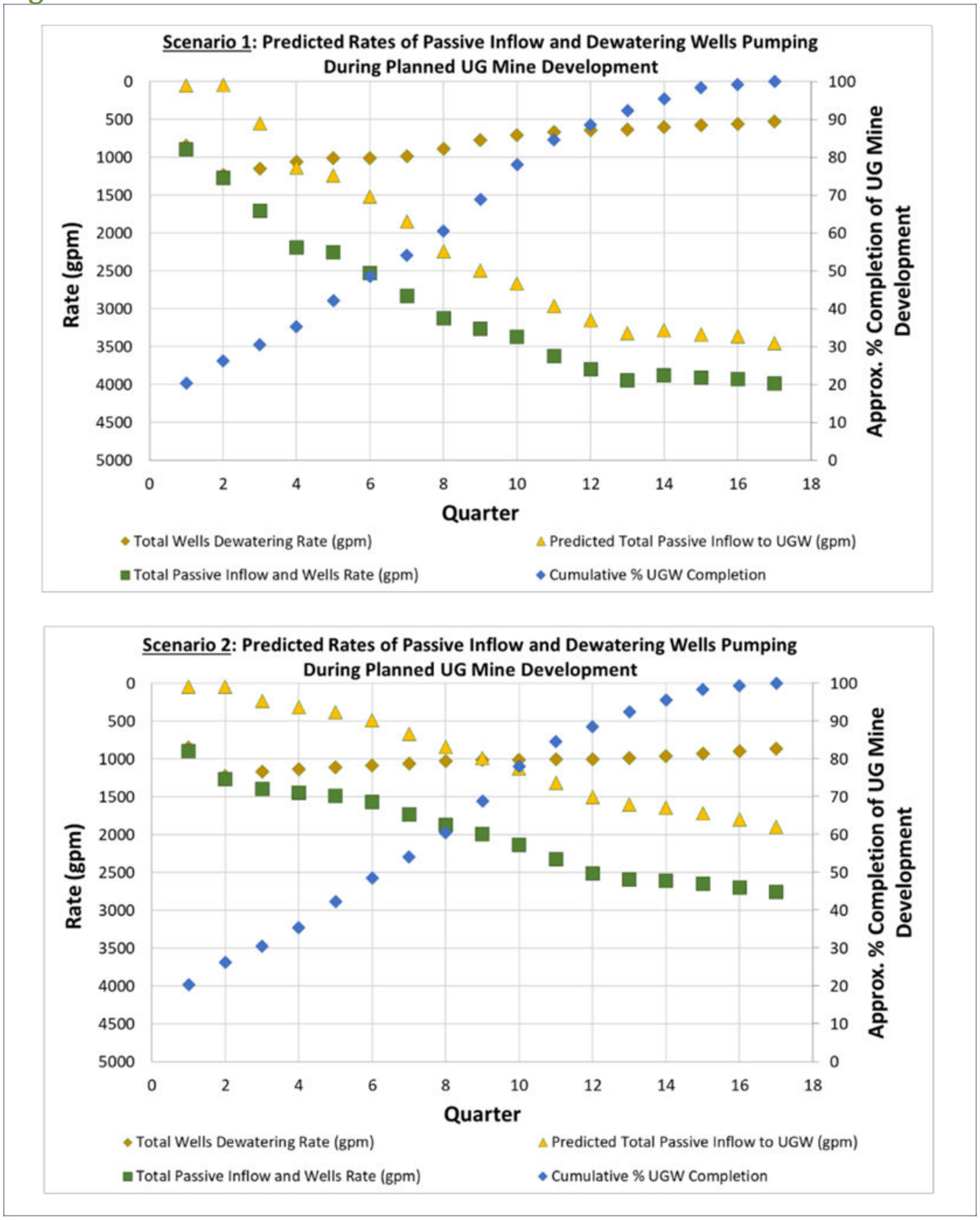
Supporting tasks for the Project include groundwater flow model development for dewatering assessment, predictions for passive inflow related to planned underground mine development and workings (UGWs), and regional-scale permitting requirements for potential effects on Kelly Creek Basin. To conduct this work, HGL (2023) subcontracted and supervised INTERA Incorporated (INTERA) to construct the groundwater flow model, to calibrate the model using historical records, and finally to use the model to assess effectiveness of current production wells for dewatering the planned UGWs and predict passive inflow to UGWs during development.

Two endmember predictive model scenarios (Scenario 1 and Scenario 2) were developed to estimate the upper and lower limits of passive inflow to UGWs that may result from the UG mine plan and current dewatering infrastructure. The range of results reflects the current uncertainty in hydraulic parameters in the proposed mine area. Quarterly stress periods are used to represent the predictive period spanning 17 quarters (approximately 2024 through 2028), when the planned UGWs are developed in accordance with the July 2023 Granite Creek underground mine plan provided by Practical Mining LLC (2023). Figure 7-26 provides a graphical summary showing passive inflow rates during UGWs build-out on a quarterly basis, as well as total dewatering rates from the production wells and combined dewatering rates from passive inflow and production well dewatering rates.

Passive inflow for Scenario 1 was predicted at 50 gallons per minute (gpm) in early time, then increasing to 3,450 gpm through the predictive period, whereas Scenario 2 predicted approximately 50 gpm initially, and then increasing to 1,900 gpm. Differences between model scenarios were largely controlled by the conductance parameter defined in the model, which is linked to the hydraulic conductivity value that represents drain cells in the model representing UGWs development. Total dewatering rates from wells APW-1, BPW-3, BPW-5, and GCW-6 were predicted to decrease with the progression of the planned UGWs due to the decline hydraulic head in the underground workings. The inability of dewatering wells to sufficiently depressurize the aquifer ahead of planned underground mining results in increasing passive inflows during UGWs build-out.

The range in model results is reasonable given the current understanding of the groundwater system, observed water level data, and responses to pumping in nearby wells (both during the pumping test and operational pumping). Based on historical dewatering rates required to dewater Mag and CX Pits and likely compartmentalized hard-rock groundwater flow regime of the UGWs, the actual passive inflow rates during UGW development may trend toward the lower limit endmember of the predicted range at the end of UG mining. Results of the modeling work are preliminary and updates are underway to refine the work with updated calibration both at local and regional scales.

Figure 7-26 Predictive and Passive Inflows from Scenarios One and Two



8 Sample Preparation, Analysis and Security

8.1 Sampling Methods and Approach

Drilling at the Property used both surface RC and core drilling along with underground core and RC drilling. The RC drilling was used primarily to pre-collar holes to bedrock followed by core drilling. This was done to minimize costs by not core drilling through unmineralized material overlying the mineralized fault zones. Core drilling provides a higher confidence in sample quality versus RC drilling along with providing additional data for engineering studies and detailed geologic definition of structurally controlled high grade mineralized zones.

The primary objective of the drilling programs was to collect clean, uncontaminated representative samples that were correctly labeled when drilled and logged, and that could be accurately tracked from the drill rig to the assay laboratory. Atna, PMC (Barrick) Exploration, and i-80 Gold used similar sampling and analytical protocols.

8.1.1 Reverse Circulation Drilling

8.1.2 Sampling Methods

In this drilling method, cuttings produced by the bit are sent up the drill pipe into a cyclone at surface, where the sample is homogenized prior to collection. From the cyclone, the sample is processed through a rotary splitter that takes a representative split of the sample (usually a quarter split), sending a split portion to the sample port, with the remainder to the reject port. Samples are placed into 10-by-17-inch sample bags that have been clearly labeled with the drillhole number and a unique numbering sequence prepared beforehand using a spreadsheet. This spreadsheet helps in tracking bag numbers, feet drilled, and quality control samples. A representative sample of each interval drilled is also preserved in chip trays that are clearly labeled with the hole number and drill interval for future reference.

8.1.3 Recovery

Sample recovery for RC drilling is measured by weight of material collected, which is usually eight to ten pounds of material from the quarter split in a typical six-inch diameter hole. Historical RC sample recovery was excellent. Full five to ten-pound bags of sample were collected from every interval. The only exception were 15 samples out of 6,100 that were collected by Atna. The missing samples occurred in an isolated zone of badly broken ground.

8.1.4 Sample Intervals

Typical truck-mounted RC drill rigs use 20-foot drill rods, with samples collected in five-foot intervals. Atna, PMC, and i-80 used this sampling procedure in their drilling programs.

For each RC hole drilled, the drill crew was provided with a sequentially numbered set of sample bags. The outsides of the bags were marked with the drillhole number and a sample number.

To ensure that blanks and standards were inserted into the sample stream correctly (every tenth sample), several steps were taken. First, the sampler was provided with chip trays that were labelled with both the true footage and the corresponding bag number. Second, an incompletely labeled set of sample bags was provided that did not include bags for the standards or blanks. Third, since the total depth of the hole was not known prior to drilling, bags for duplicate samples (collected every 100 feet) were labeled with the letters “A”, “B”, “C”, etc. and flagged with a tear-off paper tag. For i-80 Gold, duplicate samples were not labeled with a letter but rather were kept in the same number sequence and noted as a duplicate on the sample sheet for the driller.

Samples were allowed to drain/dry at the sample site, which was routinely visited by the geologist in charge of the drill program to ensure accurate numbering of the sample suite. Once drained and/or dried, the samples were re-located from the drill site to the shipment staging area, where personnel relabeled the bags containing the duplicate samples by assigning the correct sequential number in the case of Atna and PMC. This ensured that they were “blind” to the laboratory personnel. The samples were then loaded into 4 x 4 x 3-foot wooden or plastic crates in preparation for pickup by the assay lab.

8.1.5 Logging

Representative rock chips for each 5-foot run were collected in clearly labeled 20 compartment plastic chip trays. These trays were taken to the logging facility, where the geologist logged the chips with the aid of a binocular microscope. For Atna and PMC, the geologist recorded lithology, mineralization, alteration, and other pertinent features on a paper drill log. A schematic graphic log was also produced to aid in interpretation of the stratigraphic sequence. Geologists with i-80 Gold recorded geologic information directly into the acQuire database.

8.2Diamond Drilling

8.2.1 Sampling Methods

At the drill site, the drill crew was responsible for obtaining a complete and representative sample of the cored interval. This interval is usually five feet in length but may be shorter depending on how difficult the ground conditions are. Core is recovered from the core barrel via a wire line core tube, which may be outfitted with an inner “triple-tube.”

For core obtained using a triple-tube system, the core was placed on a rack, and the drill crew recorded rock quality determination (RQD) values on a worksheet and photographed the core. For holes drilled with conventional core barrels, RQD values were recorded later by a geologist from the core in the box.

At the drill site, once the RQD values were recorded and the core photographed, the drill crew placed the core in waxed cardboard boxes that were labeled with the company name, Property, hole ID, box number, and from-to footage. Core boxes were partitioned in five, two-foot-long sections totaling 10 feet in length. As core was drilled, it was placed in the core boxes in sequential order from top of the run to bottom of the run. A wooden block was inserted at the end of each run, and at the driller's discretion, to indicate problems with drilling, such as caving, voids, or core tube mismatches. The last block of each run was marked with the ending footage on the thin edge of the block and two numbers on the larger surface.

If the core was not photographed for RQD purposes, the drillers marked the breaks they made to fit the core into the core boxes with the letter "M or X" on each side of the break, so it was not counted in the RQD analysis. After boxing, each core box was securely closed with elastic banding and loaded into the driller's vehicle for transport to the logging area, at which point it was unloaded and logged. i-80 Gold transports all core from a staging area at the mine site to the Lone Tree mine site logging facility with a third party contractor on a flat-bed trailer once per day.

8.2.2 Recovery

Core recovery is measured by the ratio of the length of drill core recovered versus the length of the drilled run and is expressed in percent. Core recovery was excellent with greater than or equal to 99% core recovered (Golder Associates, 2014). Where core loss was recorded, it amounted to less than two feet in zones where voids were present in the stratigraphy.

8.2.3 Sample Intervals

Once the core was logged, the geologist determined the sample intervals to be sent to the laboratory. The geologist adhered to a set of guidelines to better define boundaries between mineralized material and barren samples. Original core blocks, inserted by the driller to mark the end of a drill run, served as the primary sample boundary, subject to the rules below; where a conflict existed between the inserted core blocks and the guidelines, the guidelines prevailed, and extra blocks were inserted by the geologist to compensate:

- A sample must not cross a geologic contact.

- A sample must not cross an obvious alteration boundary, including oxidation.

- A sample must not exceed seven feet long for Atna/PMC and ten feet long for i-80 and only be that long if it occurred in barren material, with 5-foot (1.5-meter) samples being the optimum.

- Any core blocks that do not mark a sample boundary, for whatever reason (such as "cave," "loss," "void," etc.) must be labeled in black marker for photographic visibility.

Each block that marked a sample boundary was outlined or highlighted in red marker, and the interval boundaries were entered into a sample sequence log. Sample intervals generally ranged from one to six feet in length and averaged 4.6 feet.

During the core sampling process, the sampler was provided with the geologic core log and the sample sequence to allow the sampler to have a better understanding of why and how the sample boundaries were picked and to act as a check on the geologist's accuracy.

For Atna and PMC, the condition of the rock and whether it was mineralized or not dictated the splitting method of the core. Unmineralized rock was split with a hydraulic splitter. Mineralized and silicified intervals were sawn with a water-cooled diamond-bladed rock saw. Mineralized un-silicified intervals were also typically sawn, but in some instances split with the hydraulic splitter. For i-80 Gold, all competent samples were sawn with a water-cooled diamond-bladed rock saw. Broken mineralized core was separated and divided into two equal portions by all companies.

To avoid sampling bias, the core was sawn or split parallel to the vertical axis of the core. The portion of the core to be saved was placed in the core box in its original position with the core blocks in place, and the box was rubber banded for additional security. The sampled half of the split core was bagged, and the bags were placed in 4- x 4- x 3-foot (1.2- x 1.2- x 0.9-meter) wooden or plastic crates for shipment to the laboratory. For Atna and PMC, the remaining core was palletized, covered with tarps, and moved to an outdoor cement pad for storage and reference. It is unknown if this storage facility was secure. i-80 Gold palletized the core, covered it with tarps, and moved it to a lay-down yard near the cutting facility at Lone Tree. The facility at Lone Tree is secured by a locked gate at all times.

8.2.4 Logging

Once the core was received at the logging facility, it was arranged sequentially from top of the hole to bottom of the hole.

For Atna and PMC, data was captured on paper drill logs including footage of the core runs, lithology, alteration, major structural features, bedding dips, and fractures. A horizontal line was drawn across the log, indicating footage where core blocks were present within the drilled core. Footage of core drilled and recovery were also recorded. Intervals with no recovery were indicated on the drill log by horizontal lines crossing the entire page, with a blanked-out zone of "no information," making it readily apparent where information was missing. For i-80 Gold all geologic data was logged directly into the acQuire database.

Any discrepancies in the footage shown on the core blocks or in core recovery were noted by the logging geologist on the log. Where there was missing core, additional core blocks were inserted by the geologist reflecting the missing interval and a cursory explanation written on the core block stating why the interval was missing.

Additionally, for Atna and PMC, graphic logs of the lithology were also produced to reflect the major rock types using conventional or agreed upon symbols. Major structural features including contact relationships, dips and fractures, bedding, and veins were plotted on the log and described as angle from core axis. Alteration and mineralization styles were also recorded along with a description of the lithology.

8.3Sample Security

Methods for securing samples by companies conducting work at the Property prior to the formation of PMC are unknown. Between 1970 and 1996, during which time PMC was actively mining at the Property, samples were sent to the mine laboratory for analyses. It is not known what provisions PMC employed for sample security.

When Homestake operated PMC, samples were picked up and transported to the laboratory by ALS Chemex as part of the chain of custody. In 2003 and from 2007 to 2008, Barrick, as operator of PMC, conducted drilling programs. It is uncertain what protocols were employed by Barrick to ensure sample security.

Atna conducted exploration and development drilling between 2004 and 2006 and from 2012 to 2015. Once a set of samples was ready for shipment to the laboratory, the laboratory was contacted for a job number and a pickup time by the laboratory scheduler. It is unknown if samples were stored onsite or whether the sample storage area was secured. Both RC chips and core samples were placed in numbered bags and the bags placed in 4- x 4- x 3-foot wooden crates for shipping, along with a transmittal sheet indicating whether the samples were core or RC cuttings, the range of sample numbers, and the total number of samples. In some instances, an Atna geologist travelling to Reno delivered samples to the lab.

During the control of the property by i-80 Gold, samples are secured at all times. At both the Granite Creek and Lone Tree mine sites all samples are kept behind a locked and controlled gate on the property until pick-up by a third party contractor or assay lab.

8.4Sample Preparation and Analysis

8.4.1 PMC 1970 – 1996

Sample preparation procedures for the Granite Creek Mine were not recorded.

PMC's standard assaying practice was to run assays using atomic absorption (AA) methods. For all assays, this was generally done on a cyanide leach to aid in identifying leachable material (Sim, 2005). At some unknown point, PMC changed this to only run fire assay with AA finish on samples over 0.01 opt (0.34 g/t). Check assays were performed on high-grade zone samples at third-party laboratories. Detection limits for the PMC samples varied from <0.003 to <0.001 opt (<0.1 to <0.03 g/t), depending on the age of the assay.

8.4.2 PMC - Homestake 1997 – 2000

When Homestake operated PMC, assays were analyzed by ALS Chemex in Reno, NV (ALS). Samples were prepared at ALS as follows:

- Primary crush and mill to 80% passing -10 mesh
- 300-gram split of material for pulverization to 90% passing -150 mesh
- 30-gram split for digestion and assay

Samples were assayed using the Au-AA23 fire assay method with AA finish. Analyses were reported in parts per billion (ppb). Samples reporting Au values greater than 10,000 ppb were re-assayed by fire assay with a gravimetric finish.

Detection limits for gold analyses performed by ALS Chemex were 5 ppb and 0.0005 opt (0.017 g/t). For statistical purposes, most of the Homestake holes that reported "detection limit" gold were converted to 2.5 ppb and 0.0003 opt. (These values were subsequently converted back to -5 ppb and -0.0005 opt in the current database).

8.4.3 PMC Barrick 2000 – 2008

American Assay Laboratories (AAL) located in Sparks, Nevada was used by PMC (Barrick) to prepare and analyze samples generated from its drilling programs.

Samples were dried, weighed, and crushed using either a roll or jaw crusher. A split of crushed material was pulverized for further analytical work. Samples were analyzed for gold using a one assay ton (29.116 gram) fire assay with AA finish (Fire AA). Samples with a fire assay greater than 0.005 opt (0.17 g/t) were subject to a cyanide soluble leach assay by AA spectroscopy to determine gold recovery and carbon and sulfur analysis for metallurgical evaluation. Samples returning an initial gold assay >5 parts per million (ppm) were subject to fire assay with a gravimetric finish.

In addition to gold, PMC (Barrick) also had the samples analyzed for an additional 69 elements using an aqua regia digestion with an Induced Coupled Plasma Atomic Emission Spectroscopy (ICP-AES finish). PMC (Barrick) employed its own internal quality assurance/quality control (QA/QC) protocols. Once the assay results were received via email, the exploration database manager loaded the assay data into AcQuire database management software (ACQ). The ACQ software evaluated the gold values of the standards and flagged any standards that performed outside of acceptable limits. Failed standards were documented and reviewed by the geologist in charge of the project. Depending on the rate of failure, a selection of samples, or possibly the entire batch, was rejected and another round of analyses requested by the geologist.

When samples needed re-assaying, the lab was notified of the failures, and a list of samples to be re-assayed were sent to the lab. Upon receipt of the results of the re-assayed samples by the database supervisor, they were loaded into ACQ, and XY-scatter plots were generated for the geologist to review for approval or rejection. Should the second round of analyses be rejected, a third round would ensue until acceptable results were achieved. Check samples were also collected and sent to a second lab to evaluate potential laboratory bias. It is unknown which laboratories were used to analyze the check samples.

8.4.4 Atna 2004 – 2013

Atna used Inspectorate American Laboratories (IAL), an ISO 9002-accredited facility located in Reno, Nevada, as their primary analytical lab for the Granite Creek Mine Project. Sample preparation procedures used by IAL follow.

The samples were dried and weighed prior to crushing. Crushing used a two-stage process. Once the sample was dried, it was passed through a jaw crusher to reduce it to a uniform size. It then passed through a roll mill to reduce the sample to >80% passing -10 mesh. A 300-gram split of this material was obtained using a Jones riffle splitter. The split material was further reduced to >90% passing -150 mesh using a ring and puck pulverizer.

After pulverization, a 30-gram sample of pulp was taken and digested and analyzed for gold using standard fire assay with AA finish. Samples returning gold values greater than 3 g/t were subjected to gravimetric analyses.

8.4.5 Atna Underground 2011 – 2016

The new mine lab constructed adjacent to the administration building in 2012 was in operation from 2012 to 2016.

Underground samples were transported to the on-site laboratory by Atna personnel. Samples were logged in and checked against sample transmittal sheets. Samples were then dried and weighed before being passed through a small jaw crusher to minus 3/8-inch (0.95-centimeter) passing. Crushed material was then passed through a Jones splitter, multiple times if necessary, to produce a 200-gram to 300-gram sample split for pulverization. The pulp split was then transferred to the ring and puck pulverizer for grinding to 80% passing 150 mesh. Pulverized material was weighed out to a 30-gram fire assay sample charge (Pinson Mine, 2015).

8.4.6 i-80 Gold 2021 – 2025

Underground production samples are transported daily to the Lone Tree mine site assay lab by a third party contractor. Samples are logged in and checked against sample transmittal sheets. Samples are then dried in an oven at 80°C before being passed through a large jaw crusher reducing size to 2 inch and then a small jaw crusher passing 1 ½ inch. A 250g split is then taken and the sample dried again. The samples are then pulverized to 85% passing 150 mesh creating a 250g pulp. Pulverized material is weighed out to a 30-gram fire assay sample charge. All samples are also assayed by a 10g gold cyanide shake method for an hour.

Exploration samples of both core and RC from underground and surface are assayed at a third party laboratory. These assay labs have included ALS Minerals, American Assay Laboratories, and Paragon Geochemical, all located in Sparks, Nevada with their respective procedures listed below.

Samples submitted to ALS Minerals (ALS) of Sparks, NV, an ISO 9001 and 17025 certified and accredited laboratory were assayed for gold and multi-element. Samples submitted through ALS are dried, crushed, and pulverized to 85% passing -200 mesh, creating a 250g pulp. Samples are then analyzed using Au-AA23 (Au; 30g fire assay) and ME-ICP41 (35 element suite; 0.5g Aqua Regia/ICP-AES). Samples containing greater than 10 g/t gold are analyzed by fire assay with a gravimetric finish (Au-GRA21). ALS also undertakes their own internal coarse and pulp duplicate analysis to ensure proper sample preparation and equipment calibration.

Samples submitted to American Assay Laboratories (AAL) of Sparks, NV, an ISO 9001 and 17025 certified and accredited laboratory were assayed for gold and multi-element. Samples submitted through AAL are dried, crushed, and pulverized to 85% passing -200 mesh, creating a 300g pulp. Samples are then analyzed using FA-PB30-ICP (Au; 30g fire assay) and ICP-2OA36 (36 element suite; 0.5g Aqua Regia ICP-OES+MS). Samples containing greater than 10 g/t gold are analyzed by fire assay with a gravimetric finish (G-FAAU). AAL undertakes their own internal coarse and pulp duplicate analysis to ensure proper sample preparation and equipment calibration.

8.5Data Validation

8.5.1 Summary

The Property database has been subjected to three major campaigns of data validation by Atna, Barrick, and most recently OMC. The details of data validation completed by Atna and Barrick are described in detail in previous Technical Reports (Sim, 2005; Atna Resources Ltd., 2007; Gustavson, 2012; Golder Associates, 2014; AMC, 2020), Atna (2007), Gustavson (2012), and Golder (2014). A summary of this work is described herein.

8.5.2 Atna Review of Prior Data

Atna completed a detailed review of historic data as part of due diligence studies, upon acquiring the Property. This process involved comparing data stored within a historic Microsoft Access database with digital files, databases, Vulcan files, and records stored onsite. Errors were corrected based on a “well maintained filing system containing most, if not all, drill logs, downhole surveys and Homestake assays” (Atna Resources Ltd., 2007). Validation errors such as overlapping samples and length discrepancies (i.e., surveys beyond hole depth) were investigated and corrected as appropriate.

Atna was unable to verify PMC analytical results because much of the historical analysis had been completed using the mine laboratory, and original certificates were not available. To assess historical analytical results, Atna reanalyzed 652 drill sample pulps from mineralized intercepts within the CX and Range Front target area. The pulps were sourced from the onsite pulp library maintained by PMC. Check pulp samples were submitted with Certified Reference Materials (CRMs). Atna concluded that re-assay results confirmed the accuracy of original Homestake and PMC assay results.

Atna subsequently completed two separate database audits. The first audit involved the selection of 20% of the 370 holes within the database, extracting assays greater than 0.08 opt (2.74 g/t) and checking assays. Out of 216 assays, 16 errors were noted and corrected. A second audit was completed by checking 15% of the drillholes completed by Atna in the Phase 2 program of 2006. Out of 1,653 assays, a total of 12 errors were identified.

8.5.3 Barrick Review of Prior Data

On exercising their earn-back option with Atna, Barrick conducted a detailed verification review of the historical drillhole database. This included reviewing the use of standards, blanks, and duplicates along with a second round of checks on the data entry and database maintenance. The results of the verification program are documented in an internal Barrick report that concluded that, "…10% of the database was checked, and it was considered adequate for use in a Scoping Level study…" (Golder Associates, 2014).

Barrick broadened the scope of their investigation of potential Mineral Resources at the Granite Creek Mine Project to include open pit potential and initiated a check of the accuracy of the historical database within an area of interest, which included checks on drillhole collars for 2,014 holes.

Barrick contracted Geostrata LLC of Bluffdale, Utah, to complete data verification checks on historical data. Collar coordinates, downhole surveys, from and to intervals, and assay values were reviewed. Six errors were identified out of 208 collars checked. Errors comprised transcription errors, where the collar coordinates or hole length was incorrect, and field errors, where data had been entered into the incorrect field. Out of a total of 18,013 assays, a total of 184 errors were identified (1%). Errors comprised:

- Data in the database but not in the drill log and vice versa
- Incorrect numbers in the database according to the drill log
- Discrepancy transcribing nil, trace, no sample, or detection limit values
- Sample type is recorded in the drill log but not in the database
- No assay data is available via certificate or drill log, but there was data in the database

Table 8-1 provides a summary of the errors.

Table 8-1 Summary of Errors Within the Granite Creek Project Database

Company	Total Assays Reviewed	Missing Data	Incorrect Numbers	Discrepancies Nil, Trace, No Sample, Detection Limit	Sample Type Errors	No Certificate or Drill Log	Total Number of Errors
Atna	1867	3 (0.16%)	0	0	0	0	3 (0.1%)
PMC (Barrick)							
Cordex	179	0	0	7 (3.9%)	0	0	7 (3.9%)
Cordilleran	435	4 (0.9%)	0	2 (0.4%)	0	1 (0.2%)	7 (1.6%)
PMC (Homestake)	3319	5 (0.15%)	3 (0.09%)	11 (0.3%)	11 (0.3%)	2 (0.06%)	32 (0.9%)
Pinson Mine Co.	12,392	71 (0.57%)	47 (0.3%)	16 (0.1%)	0	1 (0.008%)	135 (1.0%)
Total	**18,013**	**83 (0.46%)**	**50 (0.27%)**	**36 (0.19%)**	**11 (0.06%)**	**4 (0.02%)**	**184 (1.0%)**

8.5.4 OMC Data Compilation and Validation

8.5.4.1 Database Compilation

In January of 2017, OMC contracted Maxwell Resources (Maxwell) to perform data migration of the drillhole database into their proprietary DataShed database software. Maxwell was supplied with collar, downhole survey, lithology, and original assay files.

While in operation, both mine labs used a digital assay file management system to keep track of assay and other data generated from drilling programs. Only raw digital assay files were located for assays generated by the new mine lab. The new mine lab used an Excel file with multiple tabs to record assay data throughout the assaying process. Only the tab marked as "final assay" was used by Maxwell and OMC for data uploads into DataShed. Assay data from the old mine lab was only available as paper copies with hand-written assays on the form. These paper copies were used to validate assay data in the DataShed database.

Maxwell supplied OMC with an SQL database in February of 2017. During the process of migrating the database into the new software, Maxwell noted that assay files were in various formats and that there were multiple errors in collar information.

All gold assays, including Cyanide Au and calculated values, were uploaded into one Au field. There were also a significant number of generic methods that had unknown ("UN_UN") listed for the analytical method. The new data uploaded from the various labs added more analytical methods. After reviewing the database, it was determined that additional Au fields were needed to separate out the various analytical methods, i.e., Cyanide Au (Au_CN field), along with a field for calculating ounces per ton (opt) (AU_CALC field). It is important to be able to specify the analytical method used for Au analyses since DataShed automatically ranks the methods from most reliable method to least reliable method.

8.5.4.2 Database Corrections

In 2018, OMC corrected the errors found by Maxwell during their data migration process. Errors that were corrected included duplicate holes, core recovery issues, and interval data that went past total hole depth.

In addition, assay batches that were not uploaded correctly were flagged with a “NOCERT” or “assay method unknown” identifier.

In April of 2019, OMC contracted AMC and CSA Global to perform separate Mineral Resource updates on the Pinson underground mineralized zone. After detailed review of the drillhole database, AMC and CSA Global separately expressed concern with the number of “NO CERT” and “assay method unknown” assays. An area of interest surrounding the underground mineralized zone was subsequently defined, and original assay certificates were sourced and reloaded where possible. Analytical methods associated with assay data were updated during this process. Standards and blanks were also compiled and uploaded.

Details of assays reloaded are presented in Table 8-2 and Table 8-3.

Table 8-2 Initial Data Set and 18 April 2019 Data Subset

	Starting Database	18 April 2019 Database
Samples	77,475	77,660
Number of samples with "NOCERT"	58,740	48,498
Percentage of database with “NOCERT”	75.80%	62.40%

Table 8-3 Assay Certificates and Samples Uploaded by Laboratory

Laboratory	Number of Batches	Number of Samples
American assay lab	66	9,098
Inspectorate	164	13,626
Pinson Mine	132	2,921
Total	**362**	**25,645**

Notes: Numbers are from the defined area of interest

Certificate headers contain the certificate identification, analyte, laboratory method, and assay unit. The raw assay headers from all the labs had to be re-formatted to facilitate direct import to DataShed. All certificates, regardless of the lab of origin, had the identifier “_2019” added to the end of the certificate number to aid in separating assays from the same certificate but which had different loading parameters. The 18 April 2019 database described in this section was used in the Mineral Resource estimate.

Significant work has been completed on the transfer of the old database into the new DataShed database and additional clean-up work still needs to be performed on the DataShed database to ensure its completeness and increase confidence in the data.

8.6 Quality Assurance/Quality Control Overview

QA/QC data has been compiled from available databases for all drilling activities completed since 2005. No QA/QC data is available for work occurring prior to this time.

Drilling programs completed at the Property between 2005 and 2015 included QA/QC monitoring programs, which comprised the insertion of CRMs, blanks, and duplicates into the sample streams on a batch by batch basis. Table 8-4 provides a summary of QA/QC samples included during this period.

Table 8-4 QA/QC 2005 – 2015

Year	Company	Drill samples	CRMs	Blanks	Field duplicates
2005	Atna	7,330	267	289	23
2006		4,859	265	263	39
2007	Barrick	3,644	123	107	2
2008		17,661	403	265	197
2012		1,515	0	0	0
2013	Atna	3,360	0	0	0
2015		1,320	23	0	0
Total		**39,689**	**1,081**	**924**	**261**

Notes:

- Counts of individual samples. Multiple analyses types per sample (i.e., fire assay and gravimetric).
- Based on year drilled.

Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC

Table 8-5 shows the insertion rates of QA/QC samples between 2005-2015.

Table 8-5 QA/QC 2005 – 2015 Insertion Rates

Year	Company	CRM's	Blanks	Field duplicates	QA/QC [1]
2005	Atna	3.6%	3.9%	0.3%	7.9%
2006		5.5%	5.4%	0.8%	11.7%
2007	Barrick	3.4%	2.9%	0.1%	6.4%
2008		2.3%	1.5%	1.1%	4.9%
2012		0.0%	0.0%	0.0%	0.0%
2013	Atna	0.0%	0.0%	0.0%	0.0%
2015		1.7%	0.0%	0.0%	1.7%

Total		2.7%	2.3%	0.7%	5.7%

Notes:

- Counts of individual samples. Multiple types of analyses per sample (i.e., fire assay and gravimetric).
- [1] Insertion rate for CRM, Blanks and Field Duplicates combined.
- Based on year drilled.

Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC

8.7Certified Reference Materials

A total of 37 different CRMs were used at the Property between 2005 and 2015. CRMs were supplied by Rocklabs of New Zealand.

CRMs comprised on average 2.7% (and up to 5.5%) of samples submitted to the laboratory. CRM insertion formed part of the QA/QC program consistently in the period between 2005 and 2008. CRMs, during this time, were generally included systematically at a rate of 1 in 20 to 1 in 25 samples. CRMs do not appear to have been consistently used since 2008.

CRMs used in the 2005 and 2006 programs are discussed in the 2007 NI 43-101 Technical Report titled "Technical Report Update Pinson Gold Property, Humboldt County, Nevada, USA" effective 1 June 2007 (Atna Resources Ltd., 2007). There is no documentation available regarding CRM procedures for programs after 2006.

Rocklabs CRMs were stored in bulk in plastic bin in the logging trailer. Individual CRMs were created by measuring 100 grams of the appropriate CRM into kraft envelopes. Packaged CRMs were then stored in separate labeled bins and inserted regularly into the sample stream.

Table 8-6 and Table 8-7 summarize CRMs by year and company.

Table 8-6: CRMs used in each year

Period	Company	# CRMs	CRMs Used
2005	Atna	16	OxA45, OxE21, OxH29, OxK18, OXL25, OxN33, OxP32, SF12, SG31, SI15, SJ10, SK11, SN16, SP17, SQ18, UNKNOWN
2006		16	OxA45, OxE21, OxH29, OxI54, OxJ36, OxK18, OXL25, OxN33, OxP32, SF12, SI15, SJ10, SK11, SN16, SP17, SQ18
2007	Barrick	15	OxA59, OxC58, OxD57, OxF53, OxG60, OxH52, OxI54, OxK48, OxN49, OxP50, SF23, SG31, SJ32, SK33, SN26
2008		18	OxA59, OxC58, OxD57, OxF53, OxG60, OxH52, OxI54, OxJ36, OxK48, OxN49, OxP50, SF23, SG31, SI25, SJ32, SK33, SN26, UNKNOWN
2012	Atna	0	
2013		0	
2015		6	OxK119, OxN117, OxP91, SK78, SN75, SP73

Table 8-7: CRMs Used by Year and Company (2005 – 2015)

CRM ID	Expected Au Value (ppm)	Stand Dev	Number of CRMs used [1]					
			Atna		Barrick		Atna	Total
			2005	2006	2007	2008	2015	
OxA45	0.081	0.0069	2	13				15
OxA59	0.082	0.0052			3	37		40
OxC58	0.201	0.007			7	30		37
OxD57	0.413	0.012			13	42		55
OxE21	0.651	0.026	30	26				56
OxF53	0.810	0.029			4	30		34
SF12	0.819	0.028	36	18				54
SF23	0.831	0.027			7	38		45
SG31	0.996	0.028	1		4	36		41
OxG60	1.025	0.028			10	27		37
OxH52	1.291	0.025			18	28		46
OxH29	1.298	0.033	24	21				45
SI25	1.801	0.044				22		22
SI15	1.805	0.067	1	4				5
OxI54	1.868	0.066		1	6	33		40
OxJ36	2.398	0.073		3		1		4
SJ10	2.643	0.06	2	16				18
SJ32	2.645	0.068			5	30		35
OxK18	3.463	0.132	21	2				23
OxK48	3.557	0.042			10	22		32
OxK119	3.604	0.105					3	3
SK33	4.041	0.103			9	15		24
SK78	4.134	0.138					4	4
SK11	4.823	0.11	21	26				47
OXL25	5.852	0.105	29	25				54
OxN33	7.378	0.208	33	28				61
OxN49	7.635	0.189			19	6		25
OxN117	7.679	0.207					2	2
SN16	8.367	0.217	17	21				38
SN26	8.543	0.175			2	3		5
SN75	8.671	0.199					4	4
OxP91	14.820	0.3					3	3
OxP50	14.890	0.493			6	3		9
OxP32	14.990	0.44	3	15				18
SP17	18.125	0.434	25	32				57
SP73	18.170	0.42					7	7
SQ18	30.490	0.88	22	14				36

Notes:

- [1] Counts of individual samples. Multiple analyses types per sample (i.e., fire assay and gravimetric).
- Based on year drilled.

Source: AMC Mining Consultants (Canada) Ltd. using data provided by Osgood Mining Company LLC

8.8GRE Discussion on QA/QC

The in-house QA/QC procedures for Granite Creek Mine Project (between 2005 and 2015) were reviewed. This review included:

- a considerable quality of data analysis and validation work performed by AMC in prior technical reports (AMC, 2020), (AMC, 2019)

- a review of available data, checked against the Granite Creek Mine Project database.

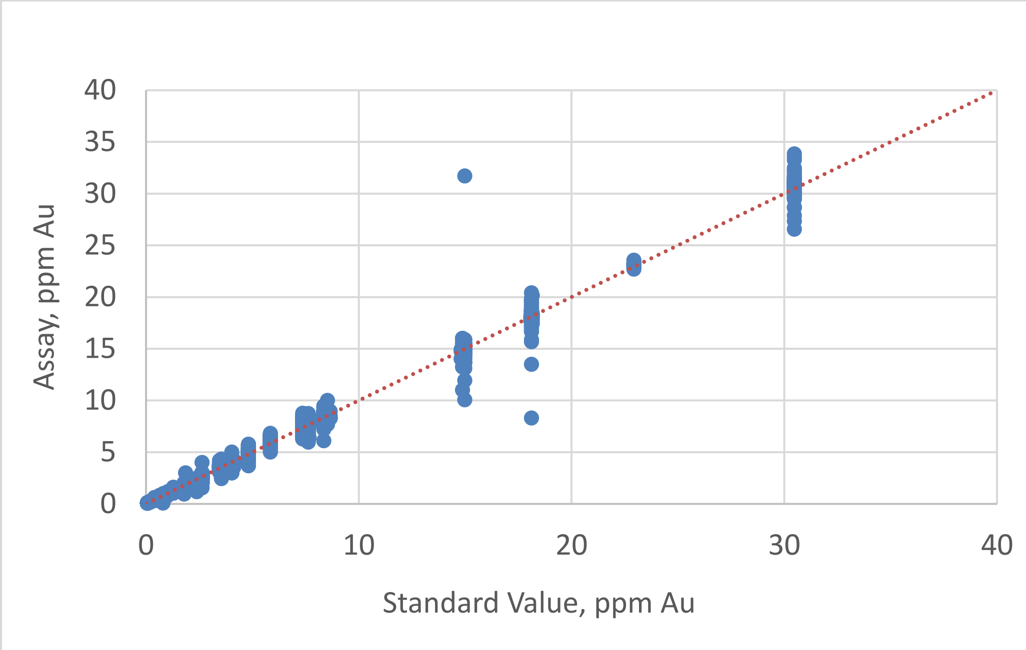
This review generated the following discussion and analysis.

8.8.1 GRE Discussion on CRMs

A total of 1,081 CRMs were inserted into the sample stream from 2005 to 2015 drilling campaigns program, including 555 CRMs by Atna in 2005, 2006, 2013, and 2015 and 526 CRMs by Barrick in 2007, 2008, and 2012. A total of 37 different CRMs were used at the property between 2005 and 2015 (Table 8-7).

Figure 8-1 shows a scatter plot of the certified value for each assay standard compared to the assay value obtained. The laboratory's analytical results generally correlate well with the standard values with few outliers. A 45-degree line represents an excellent correlation between the standard assay certified value and actual assay results. This line passes through all of the sample sets, with the majority of the points directly adjacent to the line, indicating acceptable accuracy performance for the standards. Larger scatter is seen as the grade of the samples increases. The increase in scatter is within an acceptable range in the opinion of the QP.

Figure 8-1: Assay Standard Results (2005-2015)



In addition to control charts contained in (AMC, 2020), GRE selected some additional control charts to monitor the analytical performance of an individual CRM over time and to validate prior conclusions Control lines are also plotted on the chart for the expected value of the CRM, two standard deviations above and below the expected value, and three standard deviations above and below the expected value. CRM assay results are plotted in order of analysis. Control charts at various grades for the two main campaigns of work are presented for select CRMs (outlined in Table 8-8) in Figure 8-2 to Figure 8-7.

Table 8-8: CRMs Selected by GRE for Control Charts

CRM	Au Value (ppm)	No. CRMs	Campaign
OxG60	1.025	36	2007-2009
OxI54	1.868	40	2007-2009
OXL25	5.852	54	2005-2006
SG31	0.996	41	2007-2009
SJ32	2.645	36	2007-2009
SQ18	30.49	36	2005-2006

Figure 8-2: CRM OxG60 (2007 – 2009) FA-ICP-ES

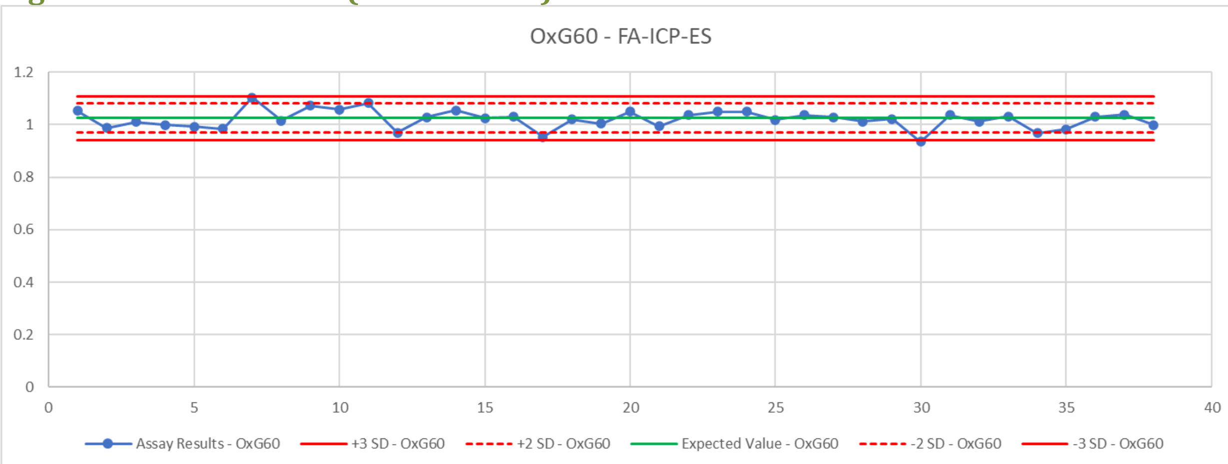


Figure 8-3: CRM OxI54 (2007 – 2009) FA-ICP-ES

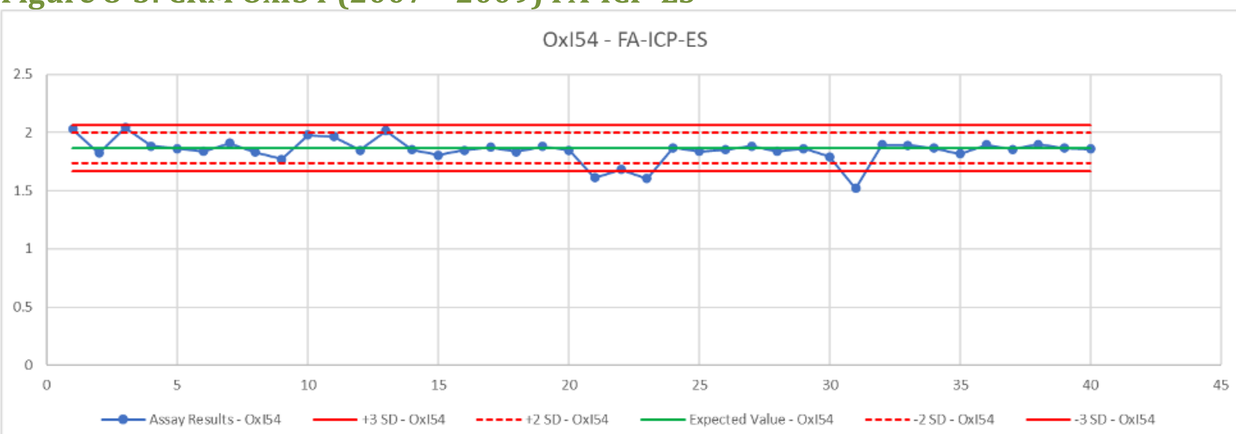


Figure 8-4: CRM OXL25 (2005 – 2006) FA-GRAV

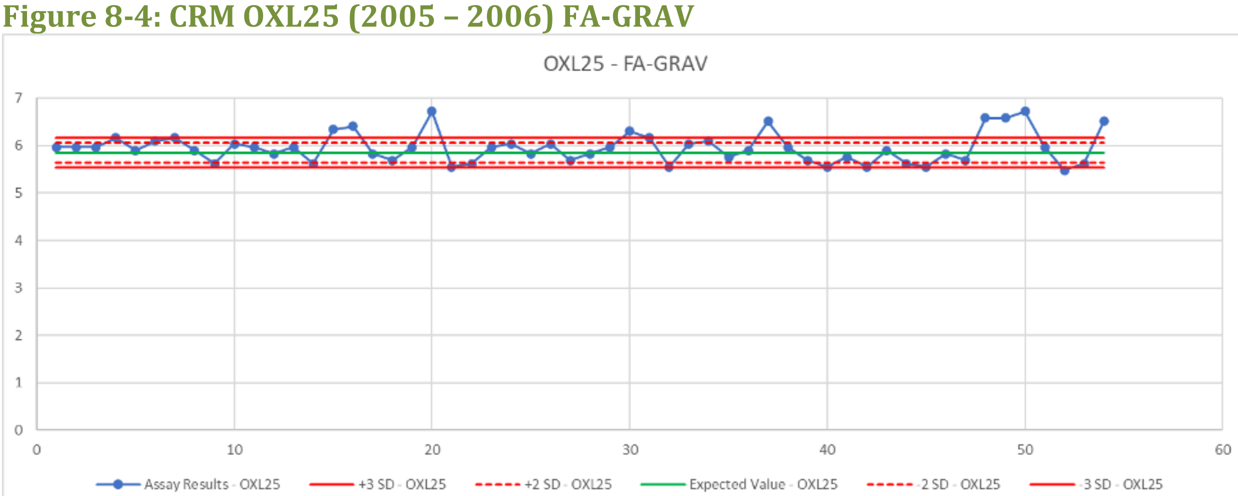


Figure 8-5: CRM SG31 (2007 – 2009) FA-ICP-ES

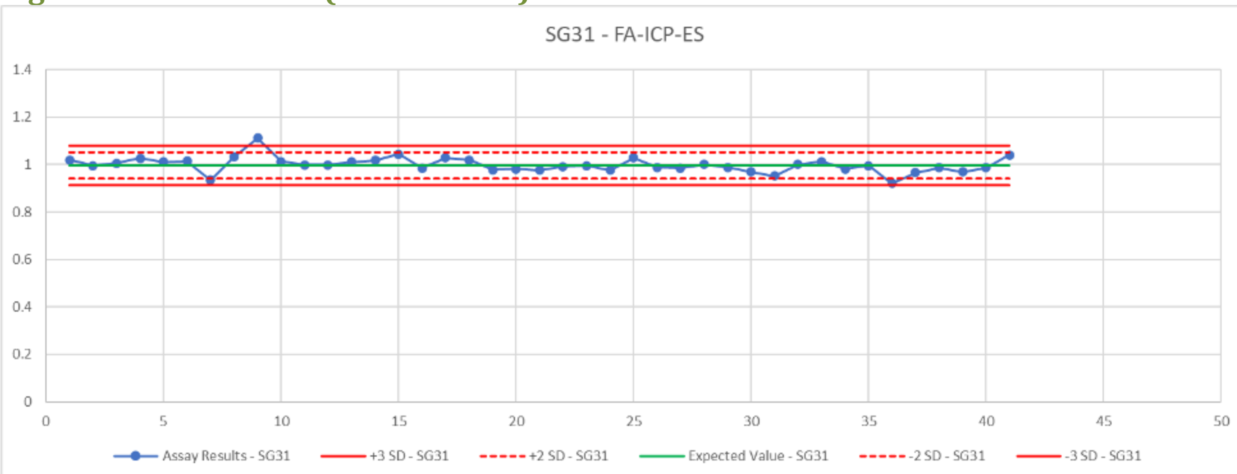


Figure 8-6: CRM SJ32 (2007 – 2009) FA-ICP-ES

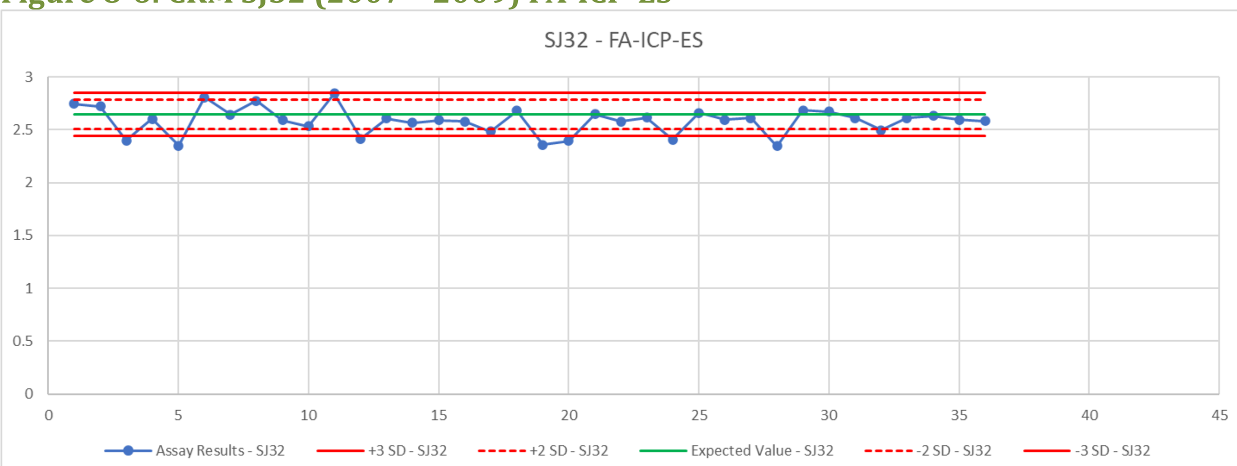
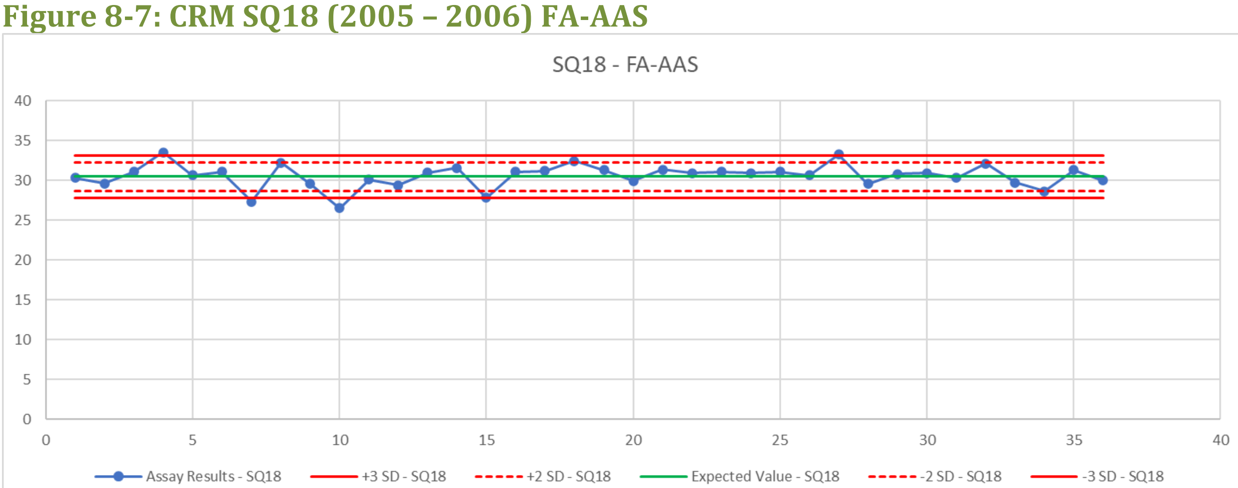


Figure 8-7: CRM SQ18 (2005 – 2006) FA-AAS

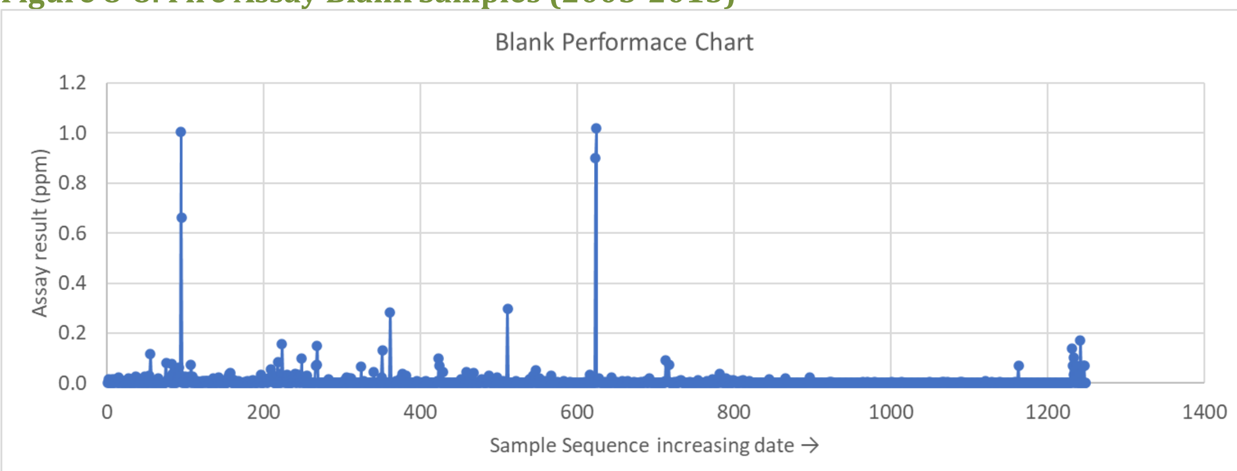


In general, CRMs show reasonable analytical accuracy but relatively poor precision when compared against the certified standard deviation. This poor precision occurs in a number of CRMs from two laboratories over a period of four years. At this time, it is not possible to definitely determine the cause of CRM high failure rate.

8.8.2 GRE Discussion on Blanks

GRE reviewed and checked all blank samples in the database provided by i-80. Figure 8-8 shows the assay results of the blanks used in the QA/QC program between 2005 and 2008. A total of 1,249 blanks returned 270 excursion values, with a maximum value of 1.02 ppm Au. Apart from four samples, the remaining samples are below the probable lower limit of the cutoff grade. 78.4% of the samples are below the detection limit. GRE believes the results indicate there is no artificially introduced contamination in the sampling preparation process that would materially affect the mineral resource estimate.

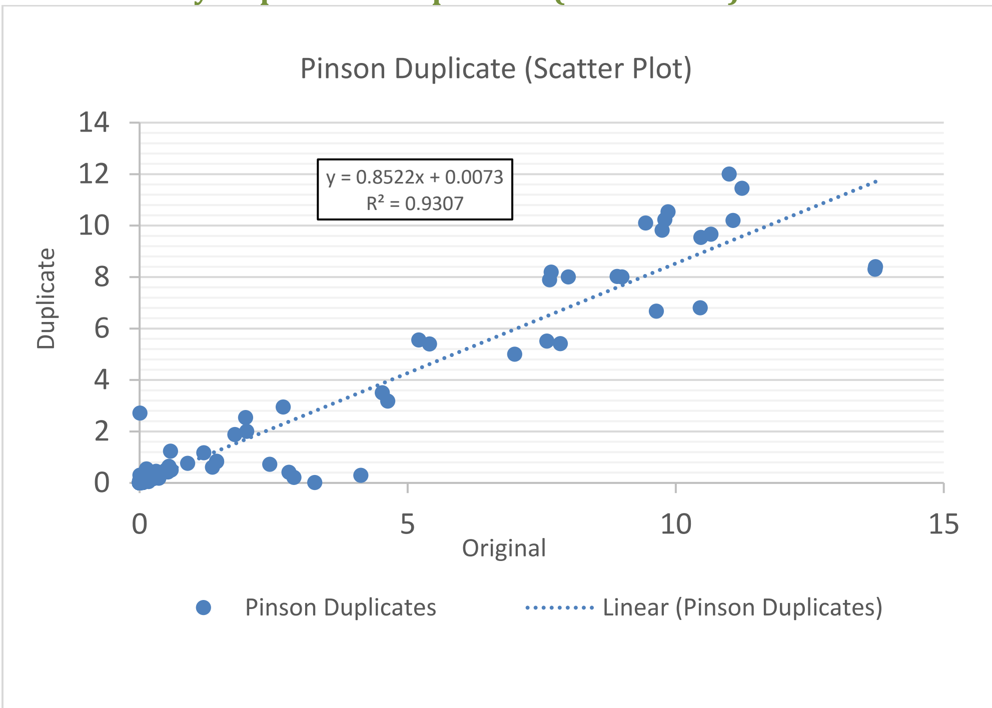
Figure 8-8: Fire Assay Blank Samples (2005-2015)



8.8.3 GRE Discussion on Duplicates

A total of 287 duplicate samples in the database provided by i-80 were checked by the QP. Figure 8-9 shows a comparison graph of the field duplicates. The scatter plots indicate some scatter in the data, with R2 values of 0.93. The scatter increases as the grade values increase but are still within acceptable ranges in the opinion of the QP.

Figure 8-9: Laboratory Duplicate Comparison (2005-2015)



8.9Quality Assurance/Quality Control Overview by PM (2021-2025)

i-80 collected samples from 14 surface exploration holes drilled in 2021 and three exploration holes drilled in 2022 on the Property for the design of pen pits. Drilling programs completed at the Property in 2021 and 2022 included QA/QC monitoring programs, which involved the insertion of CRMs, blanks, and duplicates into the sample streams on a batch-by-batch basis. Table 8-9 provides a summary of QA/QC samples included during this period, while Table 8-10 shows the insertion rates of QA/QC samples for the 2021 and 2022 drilling programs.

Table 8-9: QA/QC 2021 and 2022

Year	Company	Drill samples	CRMs	Blanks	Field duplicates	Preparation duplicates
2021	i80	1,395	55	51	34	38
2022		800	47	51	48	24
Total		**2,195**	**102**	**102**	**82**	**62**

Table 8-10: QA/QC 2021 and 2022 Insertion Rates

Year	Company	CRM's	Blanks	Field duplicates	Preparation duplicates	QA/QC
2021	i80	3.9%	3.7%	2.4%	2.7%	12.8%

2022		5.9%	6.4%	6.0%	3.0%	21.3%
Total		4.6%	4.6%	3.7%	2.8%	15.9%

8.10 PM Discussion on QA/QC 2021

i-80's in-house QA/QC procedures in 2021 involved submitting 55 Certificate Reference Materials, 51 blank samples, 34 field duplicates, and 38 pulp duplicates to the laboratory for the 1,395 drill samples. The standards were sourced from CDN Resource Laboratories, and the blanks were purchased from Ron's Seed and Supply in Winnemucca. The blank material consisted of 50 bags of Vigoro marble chips, which are gravel-sized. Field duplicates were ¼ core samples, while RC duplicates were created by placing a splitter on the cyclone of the RC rig. As the rock emerged from the cyclone, it was evenly distributed between two sample bags.

i-80 geologists routinely reviewed their assay results. The results fall within the anticipated range of variability as described by the standards' manufacturer, and as a result, the QP is of the opinion that there is no indication of systematic errors that might be due to sample collection or assay procedures.

8.10.1 PM Discussion on CRMs

i-80 used CRMs CDN-GS-7J, CDN-GS-8C, CDN-GS-30C, CDN-GS-P1A, and CDN-GSP6E for the 2021 drilling program. In total, CRMs for gold were inserted into the sample stream at a rate of four standards per 100 sample assays for all 1,395 core and RC samples for the 2021 drilling program.

Analysis of CRM charts for the high and lower gold grades showed no obvious errors or bias (see Figure 8-10 through Figure 8-14).

Figure 8-10: CRM CDN-GS-7J for the 2021 Drilling Program

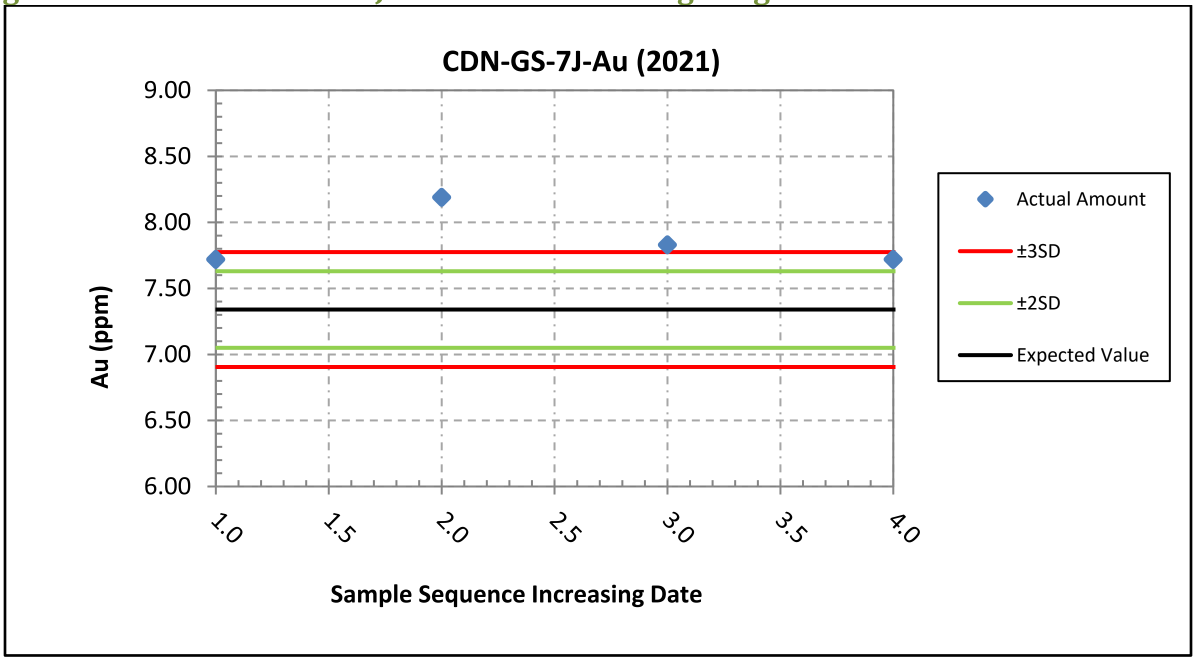


Figure 8-11: CRM CDN-GS-8C for the 2021 Drilling Program

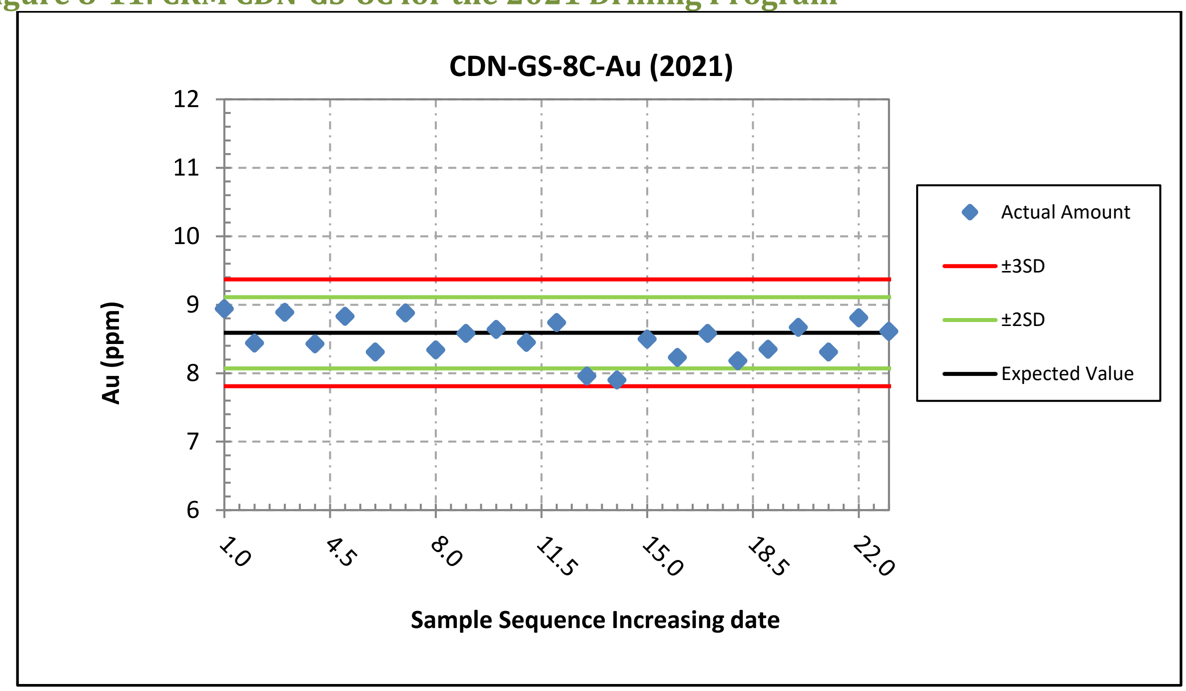


Figure 8-12: CRM CDN-GS-30C for the 2021 Drilling Program

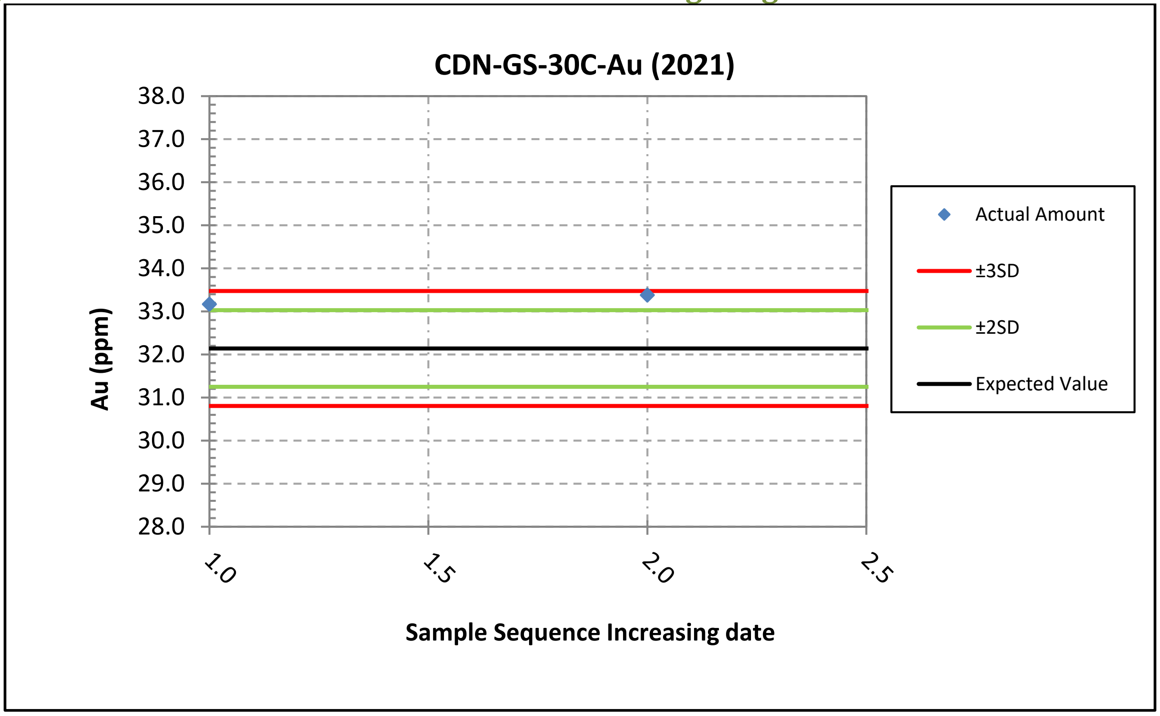


Figure 8-13: CRM CDN-GS-P1A for the 2021 Drilling Program

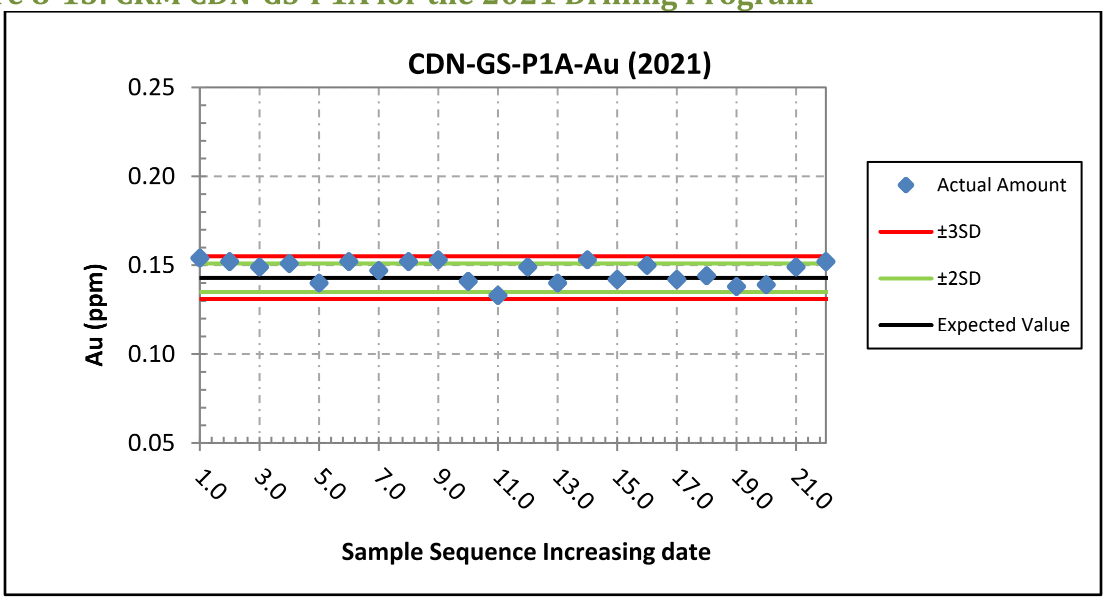
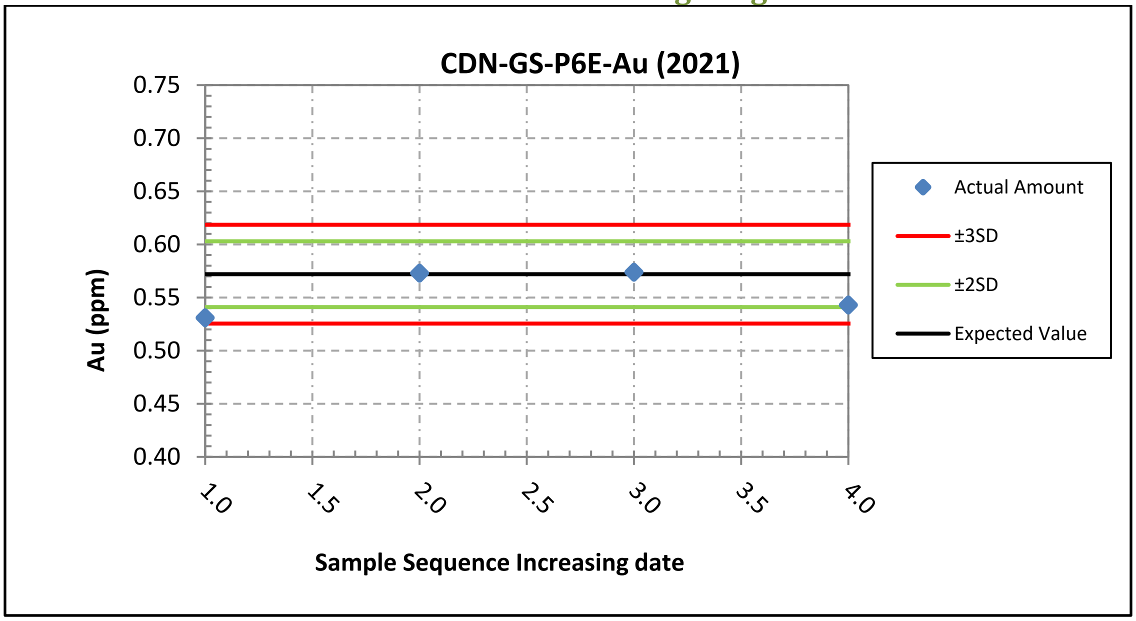


Figure 8-14: CRM CDN-GS-P6E for the 2021 Drilling Program

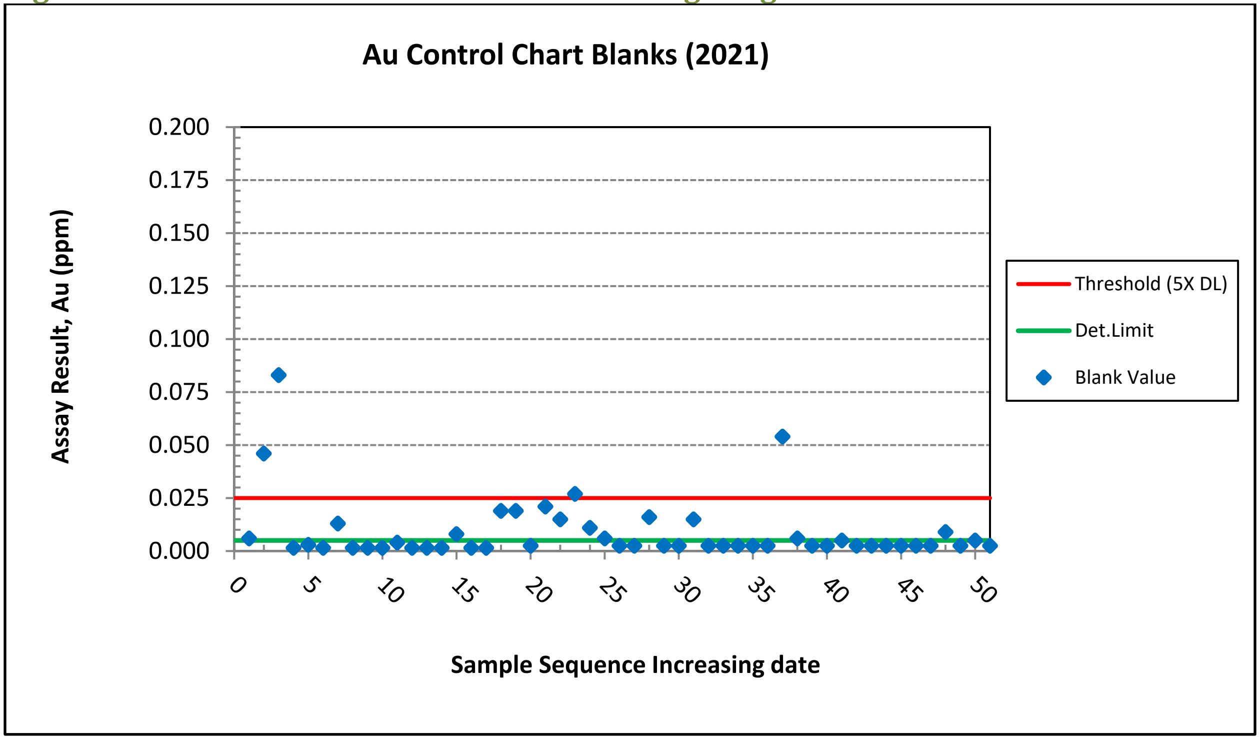


8.10.2 GRE Discussion on Blanks

GRE's QP reviewed and checked all blank samples in the database provided by i-80 for the 2021 drilling program. For all 1,395 drill samples, 51 blank samples were inserted in the sample stream at a rate of three and a half blank samples per 100 rock drill samples. Figure 8-15 shows the assay results of the blanks used in the QA/QC program in the 2021 drilling program.

The remaining samples, except four, are below the threshold, which is five times more than the detection limit. GRE believes the results indicate no artificially introduced contamination in the sampling preparation process that would materially affect the mineral resource estimate.

Figure 8-15: Blank Results for the 2021 Drilling Program



8.10.3 GRE Discussion on Duplicates

For the 2021 drilling program, i-80 considered 34 field duplicates and 38 preparation duplicates for all 1,395 core and RC, at a rate of 2.4 and 2.7 for field and preparation samples per 100 sample intervals. Field duplicate samples were prepared the same way as all assay samples and assayed at the laboratories.

The Q-Q plot for field duplicates shows a few scatters that are acceptable for field duplicates, confirming that high-grade mineralization zones are mainly associated with an inhomogeneous distribution of mineralization along samples (Figure 8-16).

The Q-Q plot for Preparation duplicates effectively indicates that there is no scatter, with R2 values of 0.99 (Figure 8-17).

Figure 8-16: Field Duplicate Samples for the 2021 Drilling Program

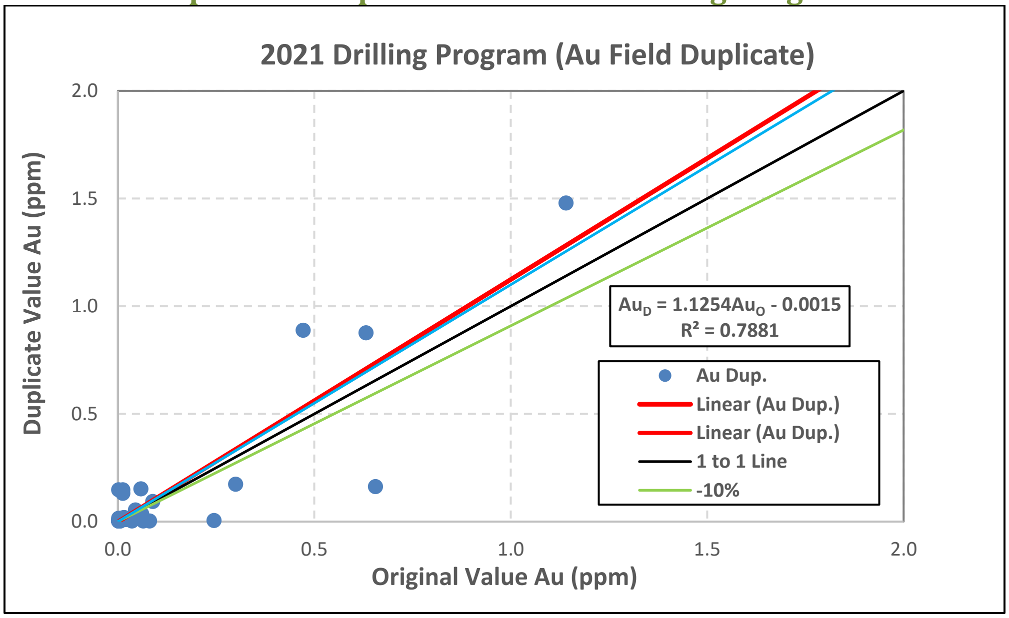
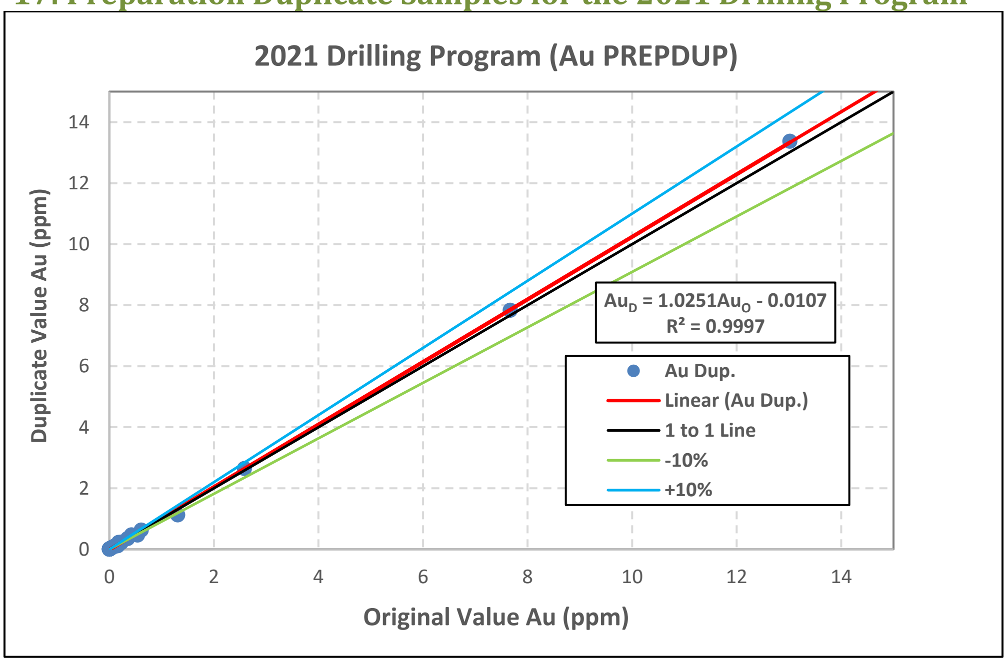


Figure 8-17: Preparation Duplicate Samples for the 2021 Drilling Program



8.11 PM Discussion on QA/QC 2022

i-80's in-house QA/QC procedures in 2022 were limited to submitting 47 Certificate Reference Materials, 51 blank samples, 48 field duplicates, and 24 pulp duplicates to the laboratory for all 800 drill samples. The standards were purchased from CDN Resource Laboratories, and the blanks were purchased from Ron's Seed and Supply in Winnemucca the same as the material used in the 2022 drilling program.

Field duplicates are ¼ core samples. RC duplicates were prepared the same as the 2022 drilling program.

The results fall within the anticipated range of variability as described by the standards' manufacturer, and as a result, the QP is of the opinion that there is no indication of systematic errors that might be due to sample collection or assay procedures.

8.11.1 PM Discussion on CRMs

i-80 used CRMs CDN-GS-7J, CDN-GS-30C, and CDN-GSP6E for the 2022 drilling program. In total, CRMs for gold were inserted into the sample stream at a rate of 5.8 standards per 100 sample assays for all 800 core and RC samples for the 2022 drilling program.

Analysis of CRM charts for the high and lower gold grades showed no obvious errors or bias (see Figure 8-18 through Figure 8-20).

Figure 8-18: CRM CDN-GS-7J for the 2022 Drilling Program

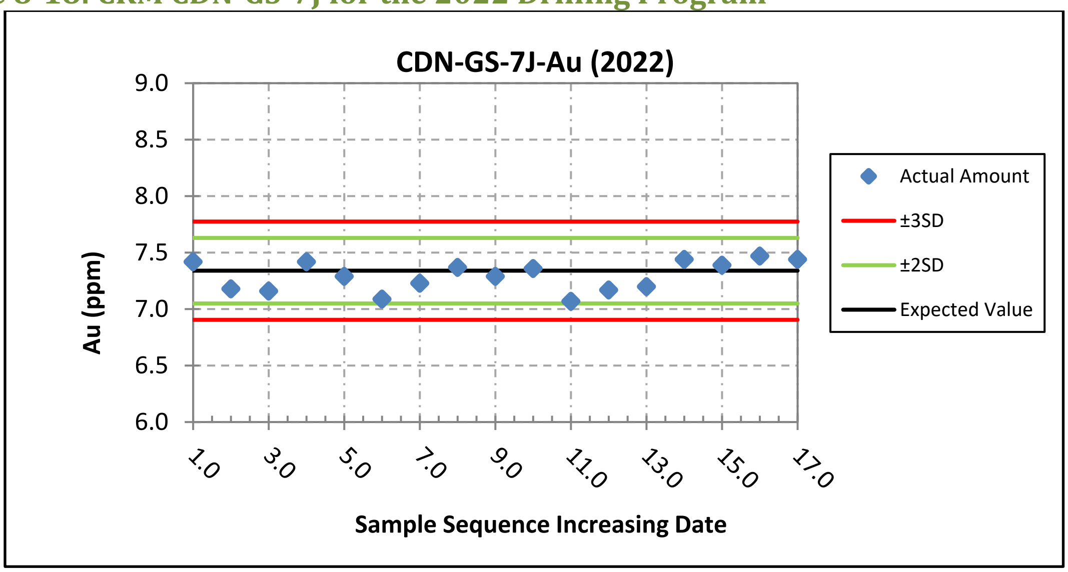


Figure 8-19: CRM CDN-GS-30C for the 2022 Drilling Program

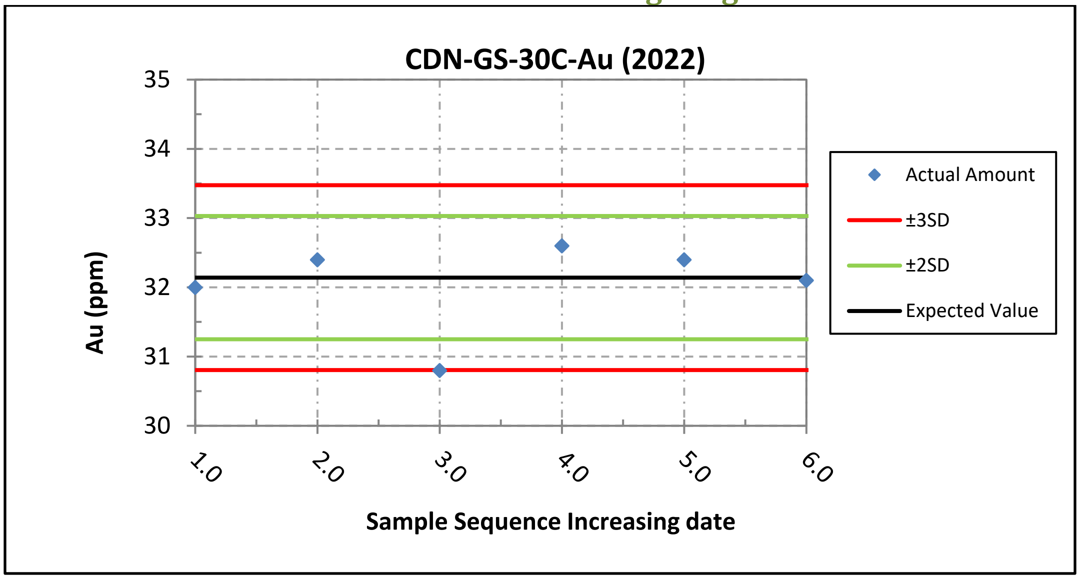
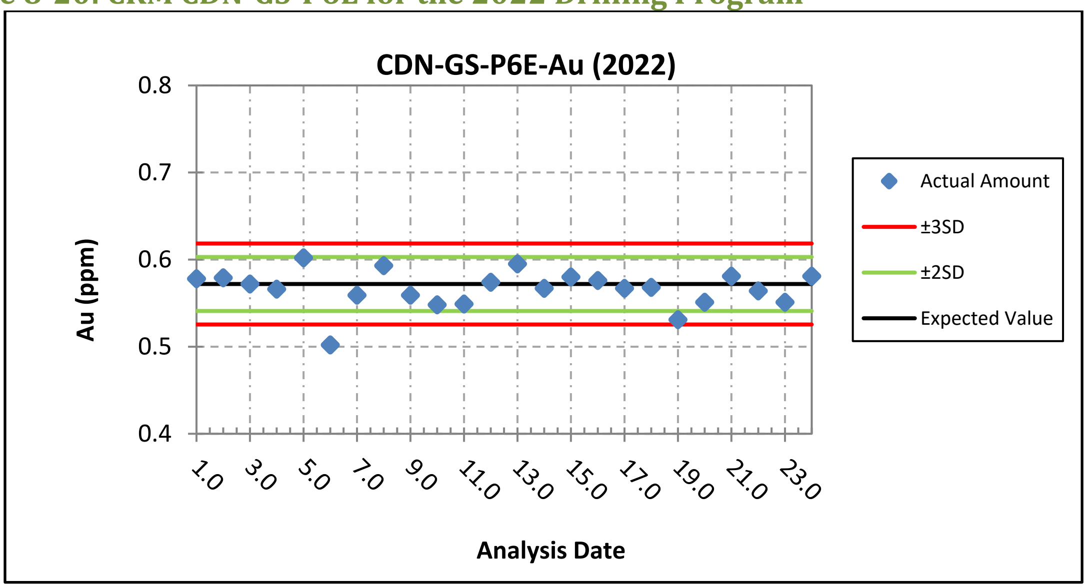


Figure 8-20: CRM CDN-GS-P6E for the 2022 Drilling Program

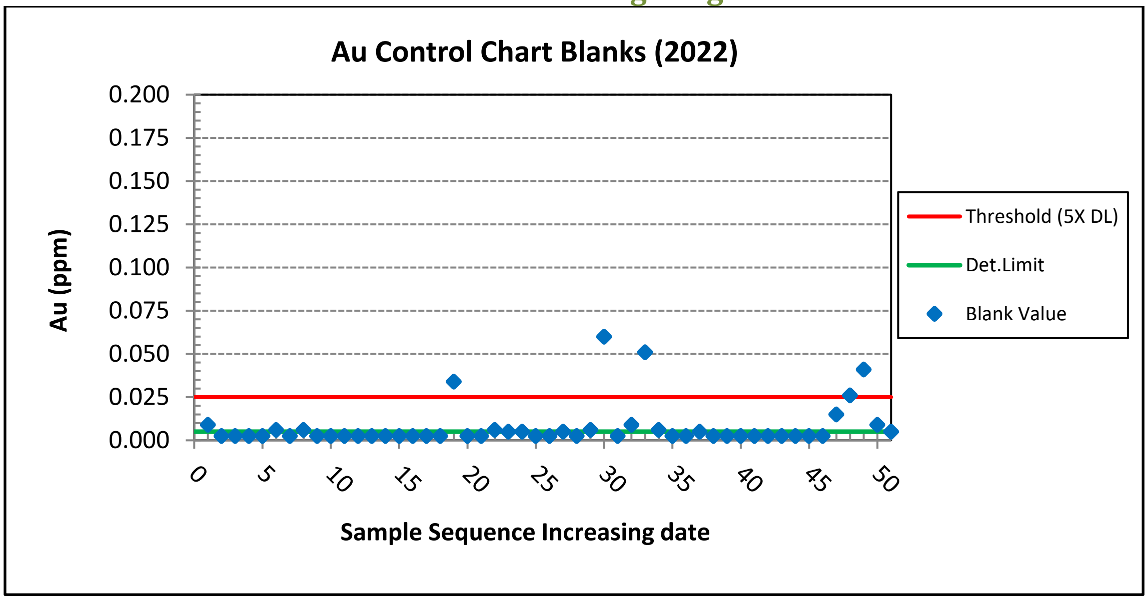


8.11.2 GRE Discussion on Blanks

GRE's QP reviewed and checked all blank samples in the database provided by i-80 for the 2022 drilling program. For all 800 drill samples, 51 blank samples were inserted in the sample stream at a rate of 6.3 blank samples per 100 drill samples. Figure 8-21 shows the assay results of the blanks used in the QA/QC program in the 2022 drilling program.

The remaining samples, apart from five, are below the threshold, which is five times more than the detection limit. GRE believes the results indicate there is no artificially introduced contamination in the sampling preparation process that would materially affect the mineral resource estimate.

Figure 8-21: Blank Results for the 2022 Drilling Program



8.11.3 PM Discussion on Duplicates

For the 2022 drilling program, i-80 considered 48 field duplicates and 24 preparation duplicates for all 800 core and RC, at a rate of six and three for field and preparation samples per 100 sample intervals. Field duplicate samples were prepared the same way as all assay samples and assayed at the laboratories.

The Q-Q plot for field duplicates shows one scatter that is acceptable for field duplicates, confirming that high-grade mineralization zones are mainly associated with an inhomogeneous distribution of mineralization along the samples (Figure 8-22).

The Q-Q plot for Preparation duplicates effectively indicates that there is no scatter, with R2 values of 0.99 (Figure 8-23).

Figure 8-22 Field Duplicate Samples for the 2022 Drilling Program

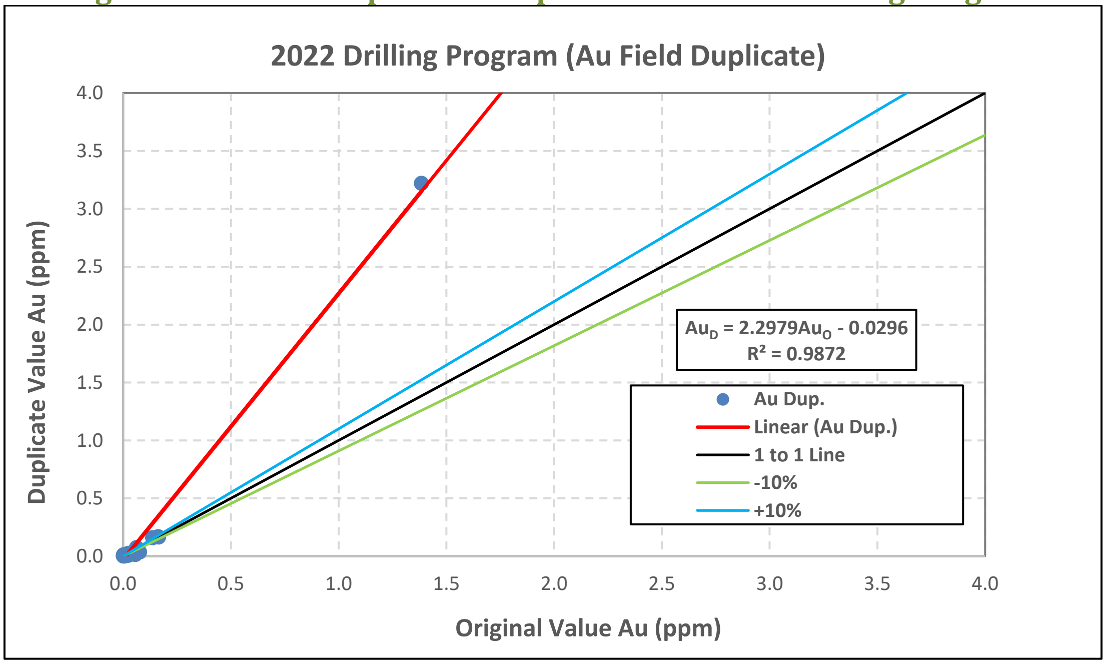
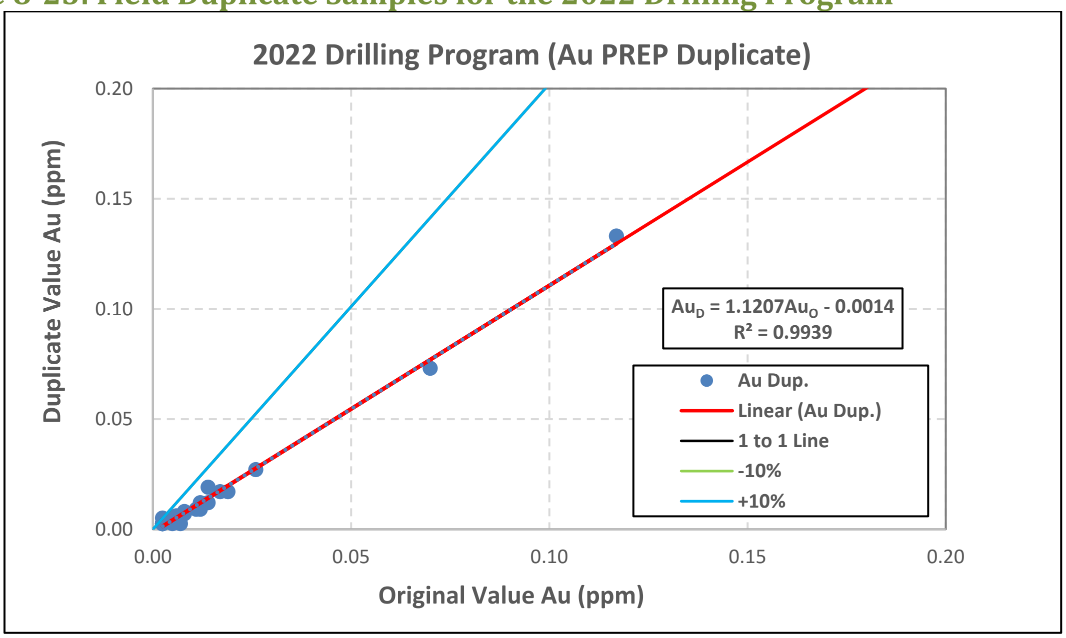


Figure 8-23: Field Duplicate Samples for the 2022 Drilling Program



8.12 Conclusions

Drilling programs completed at the Property between 2005 and 2015 have included QA/QC monitoring programs that have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams.

In 2021, GRE QP reviewed all of AMC's work on available QA/QC data between 2005 and 2015 (AMC, 2020). In 2025, i-80 provided all QA/QC data from surface exploration holes drilled in 2021 and 2022 to GRE, and GRE reviewed all of them and found no material errors. GRE also reviewed and checked QA/QC Procedures and the database provided by i-80. GRE confirmed discussions and recommendations made in prior technical reports and noted the following:

Formal, written procedures for data collection and handling should be developed and made available to PMC field personnel. These should include procedures and protocols for fieldwork, logging, database construction, sample chain of custody, and documentation trail. These procedures should also include detailed and specific QA/QC procedures for analytical work, including acceptance/rejection criteria for batches of samples.

- A detailed review of field practices and sample collection procedures should be performed on a regular basis to ensure that the correct procedures and protocols are being followed.

- Review and evaluation of laboratory work should be an on-going process, including occasional visits to the laboratories involved.

In general, the QA/QC sample insertion rates used fall below general accepted industry standards. For future exploration campaigns, standards, blanks, and duplicates including one standard, one duplicate, and one blank sample should be inserted every 20 interval samples, as is common within industry standards.

CRM samples show a reasonable level of accuracy but poor to moderate precision when using standard deviations provided by the CRM supplier. A maximum of three to five different CRM samples would be adequate to monitor laboratory performance at the approximate cut-off grades, average grades, and higher grades of the deposits.

Blank sample results are considered acceptable and suggest no systematic contamination has occurred throughout the analytical process.

Duplicate sample results show suboptimal performance, which may be a result of the heterogenous nature of mineralization, uncrushed samples, and sampling variance. Overall duplicate samples appear to be positively biased, with duplicate results returning higher grade than original samples.

Previous reporting suggests that umpire sampling has been completed at the Property. The results of this sampling were not available in the drillhole database and therefore the QP was not able to assess accuracy of the primary laboratory.

Although it is not possible to guarantee that there are no material impacts on the local scale, overall, based on the checking and reviewing the previous technical report dated 2020, GRE considers the assay database to be acceptable for Mineral Resource estimation.

9 Data Verification

The following text describes the activities performed and methods employed to personally verify the data that forms the foundation of the report. In summary, these methods included an on-site inspection of the project site and chip tray storage facility, check sampling, and manual auditing of the project database. The QPs noted no limitations nor failures to verify data.

9.1GRE Site Inspection (2021)

GRE's QPs Dr. H. Samari, T. Lane, L. Breckenridge, and R. Mortiz conducted an on-site inspection of the project on April 20, 2021.

While on-site, Dr. Samari conducted a general geological inspection of the Pinson area, including visual inspection of key geologic formations, lithologies, structural geology, and mineralization. Dr. Samari checked all lithologies on the ground with the latest prepared geologic maps prepared by Osgood (2016).

At the time of site visit, entire core boxes of four holes, BMAG-019C, BMAG-020C, UGOG-017, and UGOG-034, were ready to be inspected by Dr. Samari. Historic RC and core samples were stored at the Pinson site in the open space with thick water-resistant covers (see Photo 9-1).

Photo 9-1 Core Boxes Are Stored at the Granite Creek Site

9.2Visual Sample Inspection and Check Sampling

During the site visit on April 20, 2021, about 752 RC and core sample intervals from four separate drill holes, BMAG-019C, BMAG-020C, UGOG-017, and UGOG-034, were selected for visual inspection based on a review of the drill hole logs. The samples inspected accurately reflect the lithologies and sample descriptions recorded on the associated drill hole logs and within the project database.

In 2021, to verify the assay results, GRE collected a total of seven check samples (from four separate drill holes: BMAG-019C, BMAG-020C, UGOG-017, and UGOG-034) and two surface samples. All samples were bagged and labeled by GRE. Samples were packed and delivered by GRE the QP to Hazen Research Inc. (Hazen) in Golden, Colorado, USA (Photo 9-2).

On May 03, 2021, GRE received Hazen's analytical report on nine selected samples by fire assay method for both gold and silver. The certificate of analysis from Hazen is given in Table 9-1. Except for sample UGOC-034-528-531, with an amount of 20 ppm silver, other samples showed less than three ppm of Ag.

A comparison of the original versus check assay values for the seven core samples shows good correlation between the results, with an R2 of 0.9944 (Figure 9-1). Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

Photo 9-2 Sample Intervals Selected for Check Assay

UGOG-017:414-419

UGOG-034:489-491.1

UGOG-034:498-502

UGOG-034:528-531

BMAG-019C:727-733.5

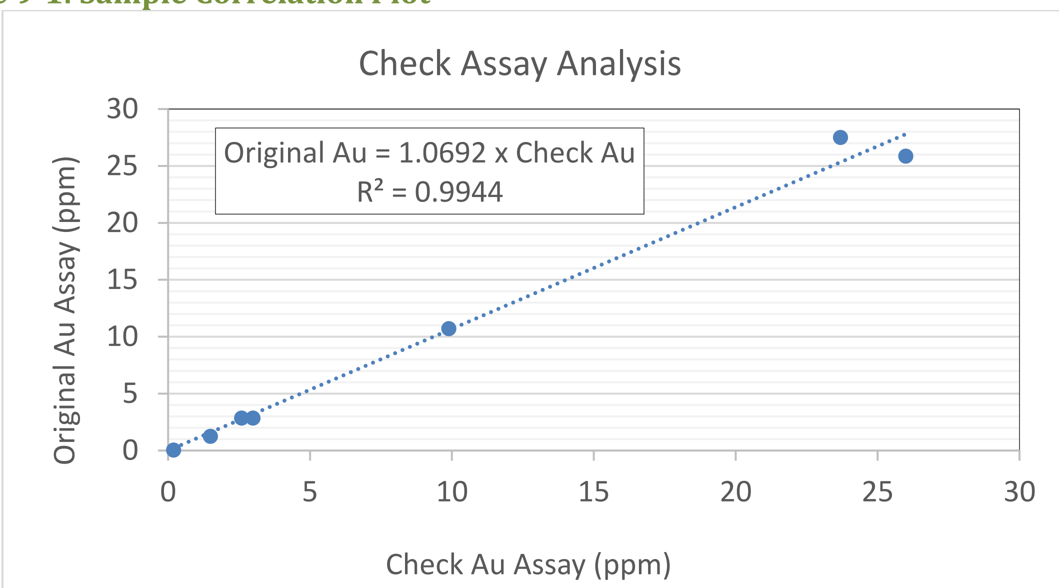
BMAG-019C:829-835

BMAG-020C:781.4-785

Table 9-1: Summary Table of Hazen Results with Original Assays

No.	Sample No.	Type of Sample	Original Au Assay ppm	Hazen Au Assay ppm
1	UGOC-034-489-491.1	Core	27.497	23.7
2	UGOC-034-498-502	Core	10.697	9.9
3	UGOC-034-528-531	Core	25.851	26.0
4	UGOC-017-414-419	Core	1.24	1.5
5	BMAG-019-727-733.5	Chip	2.8389	2.6
6	BMAG-019-829-835	Chip	0.0309	<0.2
7	BMAG-020-781.4-785	Chip	2.844	3.0
8	GRE-R.S.S.1-St.3	Surface sample (Chip)	-	<0.2
9	GRE-R.S.S.2-St.6	Surface sample (Chip)	-	<0.2

Figure 9-1: Sample Correlation Plot



Two surface rock chip samples, GRE-R.S.S.1 and GRE-R.S.S.2, were taken by GRE from the upper Comus formation (gray shale) and lower Comus formation (gray limestone), respectively (Photo 12 3). The assays show that when these two formations, which are the main gold deposit targets within the property, are not affected by faults, alterations, and mineralization conditions, they are barren (Table 9-1). The result emphasizes that mineralization on the Property exhibits strong structural control.

Photo 9-3 The Location of Two Surface Rock Chip Samples, the Upper Comus Formation (GRE-R.S.S.1-St.3, left) and the Lower Comus Formation (GRE-R.S.S.2-St.6, right)

9.3 Database Audits

The manual audit of the database by GRE compared approximately 10% of the original assay certificates with the database for the 2021 and 2022 drill campaigns intended for open pit design and found no material errors. GRE recommends that i-80 establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, or any missing information in the database. After any significant update to the database, an internal mechanical audit should be conducted. The results of each audit, including any corrective actions taken, should be documented to provide a running log of the database validation.

9.4 QP Opinions on Adequacy

Based on their area of expertise, the QPs present the following opinions on data verification and adequacy.

Based on the review of the project database and all existing project documents, and GRE's observations of the geology and mineralization at the project during the site visit, GRE considers the lithology, mineralization, and assay data contained in the project database to be reasonably accurate and suitable for use in estimating mineral resources.

GRE believes that the metallurgical testing was completed for the Granite Creek project by a number of well-known commercial metallurgical laboratories. GRE reviewed the sample selection and compositing used in the metallurgical test work and found that the selection of samples was representative for this type of deposit and geology. GRE performed several mathematical tests to validate the metallurgical balances presented in the test work and found the data presented in the metallurgical reports to be consistent with practices performed by reputable independent test laboratories. A complete discussion of the test work is provided in Section 10. Though much of the work is historical in nature, the work appears to be professionally completed and is well documented, is supported by production data, and is suitable for estimation of CIL recovery calculations in this IA.

Mining and processing methods, costs, and infrastructure needs were verified by comparison to other similar sized open pit gold mines operating in the western USA and experience of GRE. Cost data used in the report was sourced from the most recent Infomine cost data report. All costs used in the analysis were verified and reviewed by GRE and were assessed to be current and appropriate for use. Finally, after the economic study was performed, the overall operating costs for different aspects of the operation (mining, process, and general & admin) were benchmarked against similar sized mines and recent technical reports to determine if they were similar; the results did benchmark well to other operations and economic studies.

The taxation rates used and applied were values available from US government sources at the time of the economic analysis.

9.5 Practical Mining Drillhole Database Verification

Practical Mining performed an initial inspection of the drillhole database to identify drillholes not aligning with surface topography or underground mine workings, as well as holes with excessive downhole survey deviation. i-80 staff performed statistical analysis on assays to identify potential downhole sample contamination in RC holes. Potential issues were identified in 62 holes, which were excluded from the database pending further review. These holes are listed in Table 9-2.

Table 9-2 Excluded Drillholes

Sample Contamination	Excessive Downhole Survey Deviation	Assay Issue	Collar elevation doesn't correlate with topo	Multiple Conflicting Collar Survey Records
AMW-002	HPC-075	PRC-13-065	APRF-241	BRFC-002
AP4665-001	HPC-127		APRF-254	BRFC-003
APRF-284	Magmet-004		APRF-263	BRFC-007
BRFC-036	OCR-29		HPR-021	HPR-005
HPR-004	PRC-13-105		HPR-053	HPR-008
HPR-050	RH-27A		HPR-058	HPR-012
HPR-070			HPR-087	HPR-013
HPR-104			HPR-122	HPR-015
RH-27B			HPR-123	HPR-038
RHA-552A			HPR-124	HPR-041
			RCH-516	HPR-047
			RCH-659	HPR-059
			RH-130A	HPR-071
			RH-130B	OG2-155-1C
			RH-145	PM9214-1
			RH-146	RCH-1366
			RH-147B	RCH-1725
			RH-199	RCH-1727
			RH-200B	RCH-1730
			RH-210	RCH-1731
			RHA-1659	RCH-1732
			ATA-40	RH-345
				RH-346

851 drillholes were flagged for use in the estimate, and 59 holes (representing about 7% of the data set) were selected for detailed review. The holes selected for review were chosen to represent the area of interest in an even spatial distribution as well as represent different operators over time (PMC, Barrick, Atna and i-80.) Table 9-3 summarizes holes drilled by type and operator.

Table 9-3 Drill Holes Selected for Review by Type and Operator

Company	Core (or RC pre-collar with Core Tail)	RC	Rotary	Type Requested	Unavailable
i80	131			21 core	
Barrick	67	7		4 core	
Atna	73	140		5 core, 8 RC	6 RC
PMC	18	337	78	1 core, 17 RC, 3 rotary	1 core, 9 RC, 2 rotary
Totals	289	484	78	59	18

Practical Mining requested original hardcopy data records for the selected holes including collar location surveys, downhole deviation surveys, geology logs, and assay certificates. Records were unavailable for 18 of the historically drilled holes, leaving 41 holes to be reviewed, which represent about 5% of the drillholes used in the estimation. The detailed data review demonstrated good overall correlation between the database and the original hardcopy data.

78 rotary holes are included in the database. These were drilled historically in the area of the CX pit prior to mining and have predominantly been mined out by the CX pit. For the current analysis, they were used primarily for modeling the location of the mineralized structures, which were depleted by the pit topography in the model. Intercept locations of rotary holes were corroborated by viewing with blast hole data where available (blast holes were not used in the model.)

Collar survey records were available for 38 holes. Mismatches were identified in collar locations for two of the selected holes. It was determined that the database was exporting planned locations instead of surveyed locations for a series of nine holes drilled in 2021, and the error was corrected. This does not affect the current analysis because the holes are outside the underground resource area. Practical Mining viewed all holes in Vulcan to confirm collars coincide with topography or underground mine workings, and 22 holes were excluded. Some of the excluded holes may become acceptable for use if their locations can be confirmed. 23 holes were found to have multiple conflicting collar survey records, and further attempts should be made to identify the correct location surveys; those 23 holes were excluded from the current resource estimation (Table 9-2).

Downhole deviation survey records were available for 28 holes. Of the 21 PMC holes requested, only eight appeared to have been surveyed, and none of the records were available. Of the 13 Atna holes requested, two did not have downhole surveys and 11 had been surveyed, of which four had records available for review. All of the requested Barrick holes had been surveyed, with one lacking archived records for review. Downhole survey records were available for all of the selected i-80 holes. All of the available downhole survey records match the values in the database. All hole traces were viewed in Vulcan and six with excessive deviation were excluded from the mineral resource estimation.

Geology logs were available for 41 of the requested holes. Logs match the database quite well. i-80 logs geology data directly into acQuire which eliminates the possibility of data entry error. Barrick paper logs matched the data in the database. Atna geology logs appear to have been simplified when they were digitized into the database, particularly in the alteration fields. The lithology and formation fields match fairly closely. PMC geology logs generally matched the database with three exceptions: one hole had a geology log that had not been entered in the database, one hole had a 5-foot discrepancy in the TD, and one hole had a discrepancy in the depth of a unit contact. Practical Mining viewed all drillhole traces coded by lithology in Vulcan and observed that the drill data coincides very well with i-80's lithological and structural models.

Assay certificates were available for 41 of the requested holes. Certificate assay values were compared with the values in the database and only one mismatch was identified, a minor error where the preliminary value was exported instead of the final value. Practical Mining viewed all drillhole traces coded by assay grades in Vulcan and noted that grade and thickness correlate well between adjacent holes and along geological contacts. Table 9-4 summarizes the number of holes reviewed per data field.

Table 9-4 Drillhole Data Fields Reviewed

	Collar Surveys	Downhole Surveys	Geology Logs	Assay Certificates
Holes Reviewed	38	28	41	41
Percent of Population	4.5%	3.3%	4.8%	4.8%

Practical Mining concludes the database is suitable for use in the mineral resource estimation.

10 Mineral Processing and Metallurgical Testing

10.1 Introduction

Multiple historical metallurgical programs were completed from 1999 to 2014. Dawson Metallurgical Labs and McClelland Labs completed these programs. Both Homestake Mining and Atna commissioned the work. The test programs included cyanide solubility testing, pregnant solution robbing testing, bottle roll testing, percolation column testing, carbon-in-leach (CIL) testing and autoclave testing. A more recent program was completed in 2023 by FLS focusing on oxidative pretreatment of the underground sulfide material using pressure oxidation.

The Granite Creek Mine was an operating open pit mine, processing oxide material using heap leaching and conventional milling from 1980 to 1999. Although the majority of the current resource at Pinson is similar to the historically processed material, the deeper material is more difficult to treat than the historic oxide material. Atna mined high grade mineralized material from the Ogee underground deposit between 2012 and 2013. This material was treated at the Twin Creeks autoclave facility under a toll treatment agreement. Newmont Mining previously operated the Twin Creeks facility and is now operated by Nevada Gold Mines, a Newmont / Barrick joint venture of which Barrick is the operating partner.

The Author has reviewed the historical metallurgical testwork programs on Pinson feed material including:

- Report on Heap Leach, Direct and CIL Cyanidation, and “Preg-Robbing” Tests – Various Mag Pit Samples and Composites, and CX Pit Bulk Material, MLI Job No. 2532, Addendum, and Change Orders #1, #2, and #3, March 1999. (McClelland, 1999a) (Homestake)
- Report on Column Heap Leach Testing, Pinson CX Pit Material Bulk Samples, MLI Job No. 2630, June 1999 (McClelland, 1999b) (Homestake)
- Summary Report on Material Variability Testing – Mag Pit Pinson Drill Core Composites, MLI Job. No. 3746, 7 February 2013. (McClelland, 2013) (Atna)
- Summary Report on Heap Leach Cyanidation Testing – Mag Pit Pinson Drill Core Composites, MLI Job No. 3746, 16 January 2014. (McClelland, 2014) (Atna)
- Pinson Underground Autoclave-Cyanide Leach Tests, DML P-2895A,B&C, April 14,2006. (Dawson, 2006a) (Atna)
- Results of Sample Preparation and Head Analysis on Ogee Samples, DML P-2895D April 2006. (Dawson, 2006b) (Atna)

- Wilmot Metallurgical Consulting Met Test Work Results Atna-Pinson Project, (no date) (Wilmot, 2006).
- Dawson autoclave leach report, DML P-2895, final date November 3, 2005. (Dawson, 2005)(Atna)
- Osgood Mining Company – Granite Creek, Pressure Oxidation, Carbon-in-leach Testing, Preg-Rob Testing, FLS, March 2023. (FLS, 2023)

These reports are the basis for estimating recoveries for the various materials at the Pinson site, including the Mag Pit, the CX Pit, and the Ogee underground. Recoveries used in support of the economic evaluation are detailed within the geometallurgical subsections within this Section 13.

Tables are from various reports, and the units have been left as direct quotes.

10.2 Metallurgical Test Work

10.2.1 McClelland Laboratories, Inc. March and June 1999

During March and June of 1999, McClelland Laboratories completed a test work program on samples sourced from the Mag Pit and CX Pit on behalf of Homestake Mining. The program was described as a multiphase program testing various Mag Pit bulk high grade samples, core and cuttings composites, and a CX Pit bulk material sample. The results were reported under two different McClelland labs job numbers: #2630 and #2532 (McClelland, 1999a; McClelland, 1999b).

Bulk material samples from Mag Pit (Mag Pit I to Mag Pit VI) and CX pit were tested for the following items:

- Pregnant solution robbing (preg-robbing) tests to establish preg-robbing characteristics
- pH control tests to determine lime requirements for subsequent agitated cyanidation tests and column percolation leach tests
- Direct and CIL Cyanidation tests on the CX Pit (CX-2) bulk material sample to confirm the non-preg-robbing character of the material
- Column percolation leach tests on the Mag Pit bulk material samples at 4-inch size and the CX bulk material sample at three different feed sizes: run-of-mine (ROM), 3-inch, and ¾-inch.

Samples for the program were as follows:

- Six Mag Pit bulk material samples that were sampled from the pit. These samples were labeled as “Mag Pit I” through “Mag Pit VI”. The specific coordinates of these samples were not included in the document.

- One bulk material sample from the CX Pit labelled "C2." The specific coordinates for this sample were not provided.
- Six Mag Pit composites were made of drilled cuttings material. These were labeled as "Mag Pit Cuttings Composite 1" through "Mag Pit Cuttings Composite 6." Drill hole and drill intervals were documented for all of these samples.
- Five Mag Pit composites were made of drill core material. These were labelled as "Mag Pit Drillcore Composite 1" through "Mag Pit Drillcore Composite 5." Drill hole and drill intervals were documented for all of these samples.
- A Mag Pit master composite was made up of the Mag Pit drill core composites. All intervals and proportions of the composites were documented.

The drillhole IDs and intervals used to make up the Mag Pit cuttings and drill core composites are shown in Table 10-1.

Table 10-1: Mag Pit Composites for 1999 Test Work Program

Sample Description	Drill Hole Identification	Intervals
Mag pit cuttings composite 1	HPC - 129	505 to 555, 600 to 615
Mag pit cuttings composite 2	HPC - 109	155 to 165
Mag pit cuttings composite 2	HPC - 129	190 to 200
Map pit cutting composite 3	HPC 129	455 to 465, 490 to 495, 570 to 580, 585 to 590
Mag pit cutting composite 4	HPC - 109	255 to 275, 280 to 290, 310 to 320
Mag pit cutting composite 5	HPC -129	210 to 220, 255 to 270, 275 to 285, 470 to 485
Mag pit cutting composite 5	HPC 109	200 to 205, 215 to 235
Mag pit cutting composite 6	HPC 109	275 to 280, 290 to 295, 300 to 305, 320 to 340, 345 to 350, 360 to 370

The Mag Pit master composite was created using material from Mag pit cutting composite 1 to Mag pit cutting composite 6. The material was blended using 11.6% of Mag Pit 1 sample, 21.7% of Mag Pit 2, 14.5% of Mag Pit 3, 24.6% of Mag Pit 4, and 27.6% of Mag Pit 5.

Preg-robbing tests were completed on some of the samples to determine the preg-robbing characteristics of the Mag Pit and CX Pit samples. In these tests, barren solutions were "spiked" with a diluted gold solution to create a 1 ppm gold solution. The donor solution was a barren solution from column leaching of oxide materials. A standard cyanide leach bottle roll test with a 1-kilogram (kg) rock charge was completed on the slurry with the spiked solution. The pregnant leach solution was then assayed for gold at 2, 6, 24, 48, 72, and 96 hours. The percentage of gold that was preg-robbed was determined by the following formula:

$$\frac{\text{Original gold solution concentration - Final gold solution concentration}}{\text{Original gold solution concentration (\%)}}$$

Preg-robbing test work completed in this manner does not account for any gold dissolution from the 1-kg rock charge; it only considers gold loss from the spike solution. Typically, tests are conducted first (base case) without spiking to determine how much gold will be dissolved, if any. The test is then repeated with the spiked solution, and the base test gold dissolution is included within the preg-rob calculation.

Table 10-2 shows the percentages of preg-robbed gold. Negative values indicate where the final gold concentration was higher than the original gold concentration. Cyanide is destroyed in the 1 ppm gold solution before conducting the test. Some samples have negative preg-robbed values. These samples leach gold during the test. This is most likely due to the presence of Weak Acid Dissociable cyanide, which was not destroyed and is yet available for continued leaching of the material during the bottle roll test.

Table 10-2: Preg-Robbing Test Results from the 1999 Test Work Program

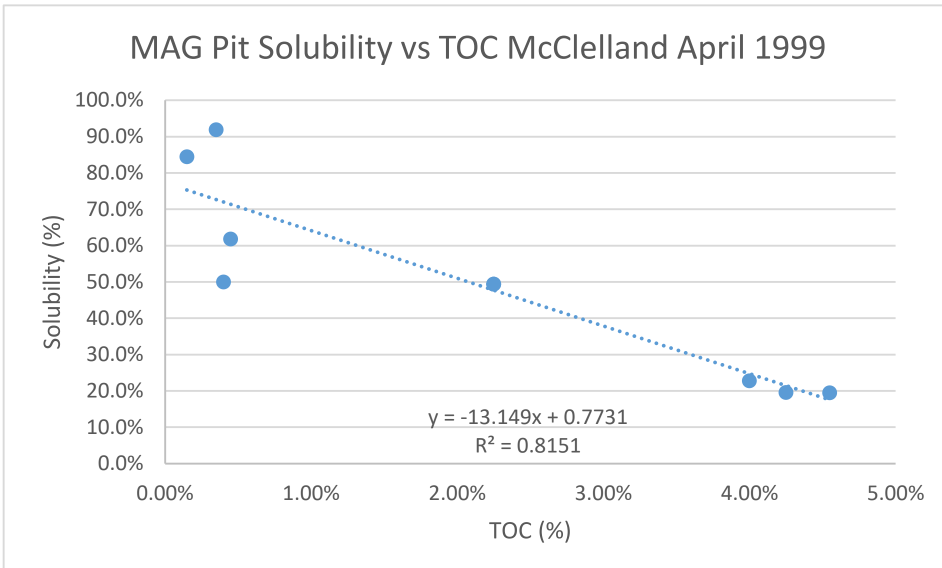
Sample	Sample Type	Feed Size	Preg-robbed Gold (%)
Mag Pit I	Bulk Ore	P_{80} ¾-inch (19 millimeter [mm])	87.9
Mag Pit II	Bulk Ore	P_{80} 3/4-inch (19 mm)	76.6
Mag Pit III	Bulk Ore	P_{80} 3/4-inch (19 mm)	-8.6
Mag Pit IV	Bulk Ore	P_{80} 3/4-inch (19 mm)	76.0
Mag Pit V	Bulk Ore	P_{80} 3/4-inch (19 mm)	-8.6
Mag Pit VI	Bulk Ore	P_{80} 3/4-inch (19 mm)	-5.0
CX-2	Bulk Ore	P_{80} 3/4-inch (19 mm)	-31.0
Mag Pit cuttings composite 1	Drill core	10 Mesh (1.7 mm)	14.1
Mag Pit cuttings composite 2	Drill core	10 Mesh (1.7 mm)	14.9
Mag Pit cuttings composite 3	Drill core	10 Mesh (1.7 mm)	15.5
Mag Pit cuttings composite 4	Drill core	10 Mesh (1.7 mm)	19.1
Mag Pit cuttings composite 5	Drill core	10 Mesh (1.7 mm)	60.5
Mag Pit cuttings composite 6	Drill core	10 Mesh (1.7 mm)	47.4

Many samples had relatively high preg-robbing values (greater than 50%), demonstrating that preg-robbing is a potential significant issue for some Pinson material (Table 10-2).

Plotting of preg-robbed percentage vs. the Total Organic Carbon (TOC) of the feed material shows a trend but the correlation is not statistically significant. However, plotting gold solubility vs. TOC shows a statistically significant trend. Solubility (%) is calculated as follows:

$$\frac{\text{Cyanide Leach Gold Grade}}{\text{Fire Assay Gold Grade}} x100$$

Figure 10-1: Gold Cyanide Solubility and Total Organic Carbon Influence



The TOC appears to have a strong influence on cyanide-soluble gold extraction. The organic carbon is capable of adsorbing gold during cyanide leaching, reducing the final gold recovery. This can have a major implication in the selection of the leaching and recovery process. Processes like CIL have activated carbon present during leaching that helps reduce the impact of preg-robbing carbon, while processes like heap leaching do not.

Predictors of recovery as they relate to Pinson and the currently available metallurgical database is discussed within the geometallurgical section.

Cyanide leach bottle roll tests were completed on the Mag Pit bulk material samples using caustic soda (NaOH) to adjust pH, rather than hydrated lime. The test work report postulated that NaOH passivates the preg-robbing (carbonaceous) surfaces by occupying active carbon sites with hydroxide (OH^-) ions so the gold cyanide complex $Au(CN)_2^-$ ions do not absorb onto the active carbon sites. Reducing the amount of $Au(CN)_2^-$ ions that are absorbed onto the carbon sites would improve gold recovery. For each sample, two tests were conducted: at pH 10.5 and 12 (using NaOH to adjust pH). The results of these tests are shown in Table 10-3.

The test data shows an increase in recovery with higher pH as well as a reduction in cyanide consumption. The reduced cyanide consumption is most likely a direct result of the increase in pH.

Generally, the largest recovery increases between the pH 12.0 tests and the pH 10.5 tests were associated with samples that showed the highest preg-robbed gold. This would indicate the OH^{-} ions are passivating sites on the available pre-robbing carbon and preventing the uptake of gold. No tests were completed using lime. Some gold processing facilities have demonstrated success using a lime-boil for similar issues. However, CIL processing is likely the best alternative for material of this nature.

There were no baseline tests using lime on these samples, so a proper comparison between lime and NaOH cannot be completed.

Table 10-3: NaOH Bottle Roll Tests from 1999 Test Work Program

Sample	Preg-robbing Factor (%)	NaOH Tests				
		pH 10.5 Tests		pH 12.0 Tests		Difference in Gold Recovery between pH 12.0 Tests and pH 10.5 Tests
		Gold Recovery (%)	Cyanide Consumption (pound [lb]/short ton)	Gold Recovery (%)	Cyanide Consumption (lb/short ton)	
Mag Pit I	87.9	8.5	1.1	32.0	1.0	23.5
Mag Pit II	76.6	13.2	3.0	24.7	1.5	11.5
Mag Pit III	-8.6	74.2	1.4	83.6	0.7	9.4
Mag Pit IV	76.0	26.4	1.6	40.8	0.5	14.4
Mag Pit V	-8.6	50.0	1.8	53.8	0.4	3.8
Mag Pit VI	-5.0	62.2	1.1	65.0	0.3	2.8

CIL tests were completed on the CX-2 bulk material and Mag Pit cuttings samples. The objective of these tests was to test the applicability of CIL processes to Pinson open pit material. The CIL process is used to help overcome the impact of the organic pre-robbing carbon that naturally occurs in some of the Pinson material. The conditions of these tests were:

- Tests were conducted in agitated bench-scale beakers
- Samples were ground to a P_{80} of 200 mesh (75 microns [µm])
- 72 hours residence time
- Kinetic samples taken at 6 hours, 12 hours, 24 hours, 36 hours, and 48 hours
- Hydrated lime was added to raise the pH to 10.5
- A sodium cyanide (NaCN) concentration of 1 gram per liter (g/L)
- Pulp density of 40% solids weight for weight (w/w)
- Activated carbon was added to absorb the gold in solution onto the carbon.

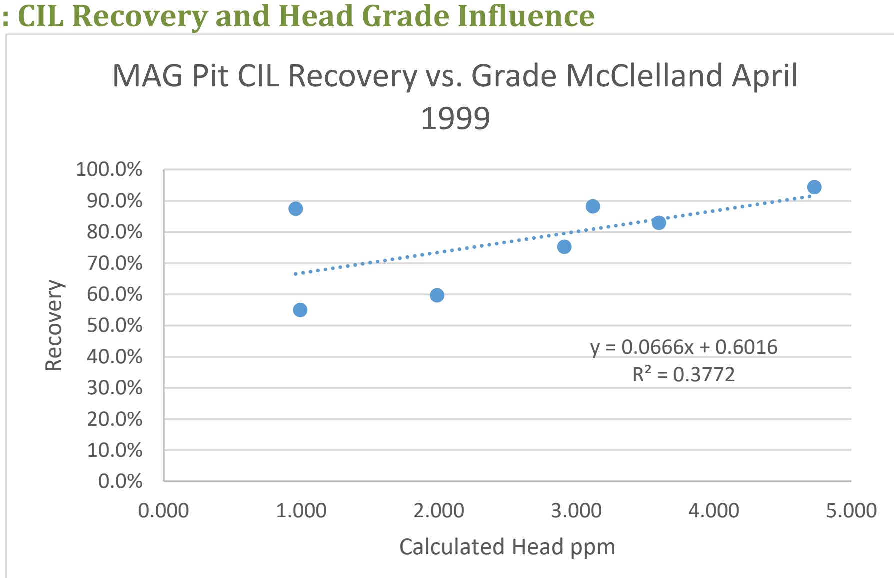
The results of these tests are shown in Table 10-4.

Table 10-4: CIL Tests from 1999 Test Work Program

Sample	CIL Gold Recovery (%)	Cyanide Consumption (lb/short ton)	TOC (%)	S= (%)	Solubility (%)
CX-2 Bulk ore	88.2	2.4	0.35	<0.01	91.9
Mag Pit cuttings I	94.4	3.3	4.55	0.05	19.5
Mag Pit cuttings II	75.3	2.3	4.25	0.12	19.6
Mag Pit cuttings III	59.7	3.0	0.15	<0.01	84.5
Mag Pit cuttings IV	82.9	4.8	4.00	0.83	22.8
Mag Pit cuttings V	55.0	3.0	0.40	<0.01	50.0
Mag Pit cuttings VI	87.5	3.9	0.45	<0.01	61.8

With the exception of Mag Pit 3 and Mag Pit 5 samples, these tests generally achieved high gold recoveries. However, these lower recoveries appear to be an anomaly when compared to column tests on the same material. When comparing the solubility value to the CIL gold recovery, it can be seen there is not a strong correlation between the two values. There is also no relationship between the CIL recovery and the sulfide sulfur grade (S=). It appears that the CIL process can overcome the presence of organic carbon in this material. There is a relationship between gold feed grade and CIL recovery as shown in Figure 10-2. Based on this test work, a CIL process would be applicable to Pinson material.

Figure 10-2: CIL Recovery and Head Grade Influence



Column leach tests were conducted on some of the samples from the 1999 program. Sixteen columns were completed representing 11 different samples. The samples were from the Mag Pit, bulk samples, core, and bulk samples from the CX Pit. The conditions of these tests were:

- Leach time of between 50 and 90 days. If the kinetic leach curve demonstrated that a test was approaching terminal gold recovery, the test was stopped.
- Varying crush sizes.
- Hydrated lime was added to agglomerate the material in the column.
- Lime was added to most tests to raise the pH to 10.5.
- NaOH was added to the Mag Pit I and Mag Pit II samples, given the successful NaOH bottle roll tests on these samples. The pH was initially 10.5 but was increased to 12.0 later in the test to ascertain the impact on leaching.
- NaCN was added at an initial concentration of 1 g/L and was pumped into the columns at a rate of 0.005 gpm per square foot (/ft^2) of cross-sectional area.
- Three tests with varying particle sizes were conducted on the CX Pit sample to ascertain the impact of crush size on gold recovery.

Three tests were conducted on the Mag Pit master composite where pH and alkali were varied:

- Test 1: pH 10.5 (lime)
- Test 2: pH 11.8 (lime.
- Test 3: pH 11.8 (NaOH)

Table 10-5 shows the results from the 1999 column leach tests.

Table 10-5: Column Leach Tests from 1999 Test Work Program

Sample	Sample Type	Feed Size (inches)	Gold Recovery (%)	Cyanide Consumption (lb/short ton)	Lime Consumption (lb/ short ton)
Mag Pit I	Bulk ore	-4	18.8	9.9	5.2
Mag Pit II	Bulk ore	-4	35.3	9.0	10.2
Mag Pit III	Bulk ore	-4	93.1	4.6	5.2
Mag Pit IV	Bulk ore	-4	49.5	5.3	12.0
Mag Pit V	Bulk ore	-4	51.7	3.9	2.5
Mag Pit VI	Bulk ore	-4	60.7	3.7	4.0
Mag Pit 2	Drill core	-1	69.0	4.0	11.0
Mag Pit 3	Drill core	-1	62.0	1.6	9.6
Mag Pit 4	Drill core	-1	47.9	1.5	8.1
Mag Pit 5	Drill core	-1	61.7	2.1	10.0
Mag Pit master (pH 10.5, Lime)	Drill core	-1	65.0	6.3	8.4
Mag Pit master (pH 11.8, Lime)	Drill core	-1	70.7	4.2	19.3

Sample	Sample Type	Feed Size (inches)	Gold Recovery (%)	Cyanide Consumption (lb/short ton)	Lime Consumption (lb/ short ton)
Mag Pit master (pH 11.8, NaOH)	Drill core	-1	69.0	3.5	n/a
CX Pit, CX-2	Bulk ore	-6	77.7	5.1	3.0
CX Pit, CX-2	Bulk ore	P_{80} 3	81.7	4.8	3.0
CX Pit, CX-2	Bulk ore	P_{80} ¾	82.2	5.4	3.0

This test work program had the following findings:

- There was a wide range of gold recoveries, varying from 19% to 93%
- Gold recovery closely followed the gold cyanide solubility percentage
- Sulfide grade was generally too low to show any impact on gold recovery
- The bulk sample tested from the CX Pit showed near 5% improvement in recovery when crushing from 6 inches to ¾ inch. There was very little difference in recovery between the 3 inch and ¾ inch size, less than 1%.

The tests on the Mag Pit master composite sample had the following conclusions:

- Increasing pH demonstrated an increase in gold recovery
- The NaOH and the lime test (pH 11.8) had a slightly less than 2% recovery difference.

Figure 10-3: Column Recovery and Solubility Influence

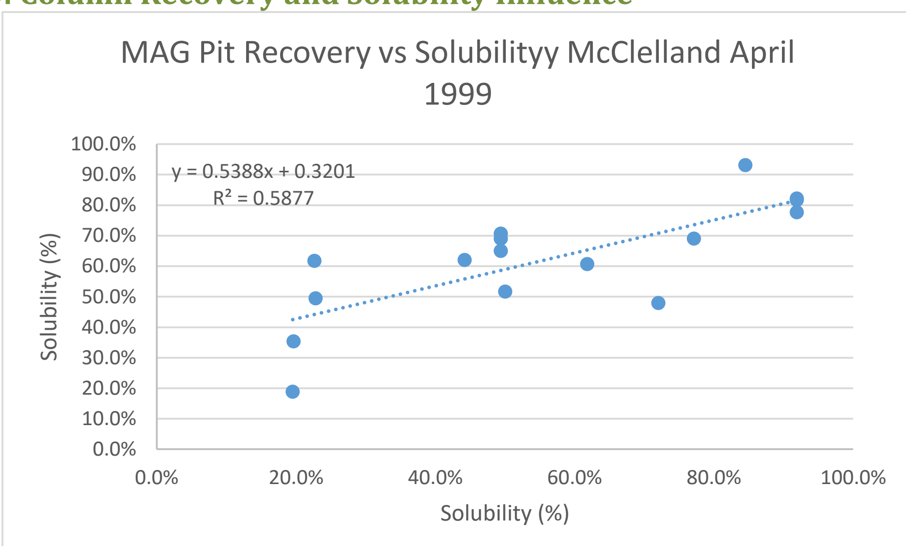


Figure 10-3 shows that the gold cyanide solubility influences the column gold recovery (all particle sizes and parameters shown). This is expected as the degree of gold solubility is highly dependent on the presence of organic carbon as shown in Table 10-4. Unmitigated organic carbon will adsorb gold from solution negatively impacting the solubility percentage and the gold recovery from a column test.

McClelland reported on an additional column leach test program in June 1999. The materials were all from the CX Pit area. Coordinates within the pit were not provided, although a bench elevation within the pit was provided for each sample. Three bulk samples, 4840-Silty, 4840-Clay Ore and 4680–Typical CX-4, were tested at a 100% passing 1-inch crush size in percolation columns. The materials leach very well and rapidly. The majority of the leaching was complete in 30 days. The samples were run from 78 to 95 days, including leaching and washing.

Table 10-6 Column Leach Tests from June 1999 Test Work Program

Sample	Sample Type	Column Feed Size	Gold Recovery (%)	Cyanide Consumption (lb/short ton	Lime Consumption (lb/short ton)	Final pH	Calculated Head (opt)
4840 Silty	Bulk Material	P_{100} 1-inch	91.9	3.16	3	10.1	0.111
4840 Clay Ore	Bulk Material	P_{100} 1-inch	95.6	2.36	3	9.4	0.110
4680 Typical CX-4	Bulk Material	P_{100} 1-inch	94.2	2.73	3	10.9	0.119

The grade of these samples was high, above what the typical heap leach feed would be. McClelland noted in the final report that the Silty and Clay Ore materials demonstrated moderate to severe percolation issues. Agglomeration was recommended for commercial heap leaching operations. During the column testing, the pH levels were lower than desired. Additional lime, above 3 pounds per short ton, will be necessary to keep cyanide consumption to a minimum. For typical column leaching, the cyanide consumption is high. The CX materials are very amenable to heap leaching.

10.2.2 McClelland Laboratories Inc 2013 & 2014

McClelland Laboratories completed a metallurgical test work program on Mag Pit samples in 2013 and 2014 on behalf of Atna Resources Ltd. (Atna).

The 2013 program used 32 drill core composite samples. The samples were well identified by the drill hole and down-hole depth. The 32 samples were then subjected to detailed head analysis, ICP scan, carbon and sulfur speciation analysis, and preg-robbing tests.

Bottle roll tests were completed in pairs, with one being 80% passing ¼-inch and the second as a 150 Mesh sample (P100 100 um). The program intended to complete an evaluation of the impact of feed size, potential for heap leaching, and especially testing for preg-robbing problems associated with the Pinson materials.

A summary of the drillholes and intervals used to make up the samples for this program are shown in Table 10-7. This is sourced from the appendix within the McClelland 2013 report (McClelland, 2013).

Table 10-7 Sample Composite List from 2013 Test Work Program

Drillhole	Sample	Interval	
		To (ft)	From (ft)
Magmet-001	Magmet-001-01	0.0	27.5
	Magmet-001-02	27.5	99.5
	Magmet-001-03	99.5	157.0
	Magmet-001-04	170.5	228.5
	Magmet-001-05	228.5	251.5
	Magmet-001-06	251.5	302.5
	Magmet-001-07	302.5	364.5
	Magmet-001-08	364.5	415.5
Magmet-002	Magmet-002-01	211.5	254.5
	Magmet-002-02	254.5	292.0
	Magmet-002-03	292.0	337.0
	Magmet-002-04	337.0	397.0
	Magmet-002-05	397.0	446.0
	Magmet-002-06	450.0	497.0
	Magmet-002-07	497.0	567.5
	Magmet-002-08	567.5	599.8
Magmet-003	Magmet-003-01	179.0	225.0
	Magmet-003-02	225.0	283.0
	Magmet-003-03	283.0	304.5
	Magmet-003-04	304.5	361.5
	Magmet-003-05	361.5	409.0
	Magmet-003-06	409.0	459.5
	Magmet-003-07	459.5	514.0
Magmet-004	Magmet-004-01	125.0	148.0
	Magmet-004-02	148.0	220.5
	Magmet-004-03	220.5	270.0
	Magmet-004-04	270.0	330.0
	Magmet-004-05	330.0	372.0
	Magmet-004-06	372.0	415.0
	Magmet-004-07	415.0	439.0

Drillhole	Sample	Interval	
		To (ft)	From (ft)
	Magmet-004-08	439.0	496.0
	Magmet-004-09	496.0	551.5

Cyanide leach bottle roll tests were conducted on Mag Pit samples. The objective of these tests was to identify the impact of particle size on gold recovery. The conditions for the bottle roll tests were:

- A pulp density of 40% solids w/w
- pH was maintained between 10.8 and 11.2 using lime
- NaCN was added at a concentration of 1 g/L
- The bottle rolls were sampled at 6, 12, 24, 36, and 48 hours
- The tests were terminated at 48 hours and final analysis was completed

Preg-rob factors were measured for each sample. The preg-robbing factor methodology was not presented within the laboratory report.

A summary of the results from the bottle roll tests is shown in Table 10-8.

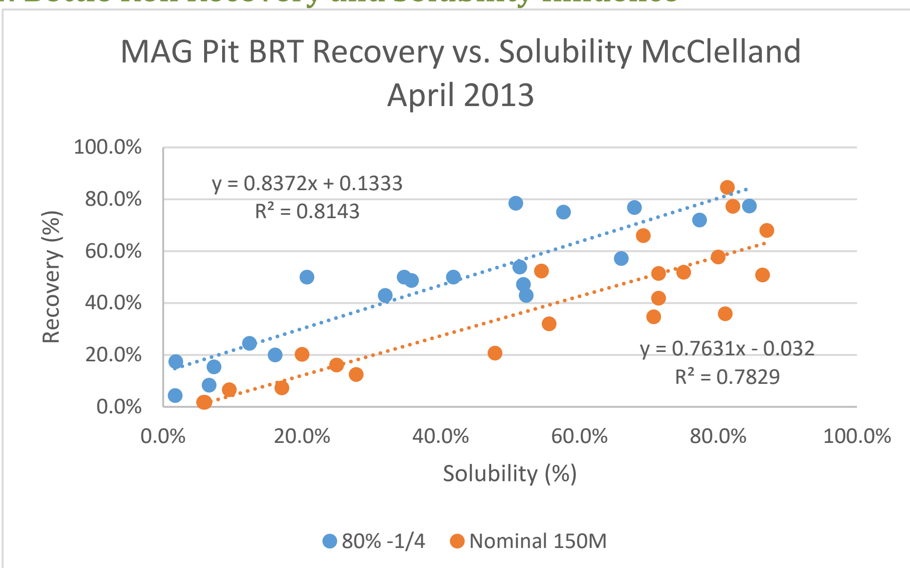
Table 10-8 Bottle Roll Tests Results from 2013 Test Work Program

Sample	Au head grade (oz/ton)	S= (%)	TOC (%)	Solubility (%)	Preg-Rob Factor	BRT Gold recovery (%)		
						P_{80} ¼-inch	150 Mesh	Difference 150# and P_{80} ¼-inch
Magmet-001-01	0.031	0.03	0.24	84.5	0	77.4	81.3	3.9
Magmet-001-02	0.089	0.04	3.49	20.7	93	50.0	47.8	-2.2
Magmet-001-03	0.032	0.01	0.46	77.3	0	72.0	82.1	10.1
Magmet-001-04	0.030	0.02	1.78	66.0	36	57.1	69.2	12.1
Magmet-001-05	0.057	0.20	3.47	1.8	97	17.4	6.0	-11.4
Magmet-001-06	0.050	0.53	3.67	12.4	92	24.4	27.8	3.4
Magmet-001-07	0.058	1.24	4.15	1.7	100	4.3	5.8	1.5
Magmet-001-08	0.018	1.14	3.81	16.1	87	20.0	25.0	5.0
Magmet-002-02	0.030	0.94	2.75	6.6	92	8.3	9.5	1.2
Magmet-002-03	0.005	0.57	1.73	N/A	84	50.0	40.0	-10.0
Magmet-002-04	0.128	0.83	2.48	50.8	76	78.4	86.4	8.0
Magmet-002-05	0.114	0.69	4.57	34.7	78	50.0	70.7	20.7
Magmet-002-06	0.043	1.57	5.07	7.3	95	15.4	17.1	1.7
Magmet-003-04	0.014	1.12	3.39	N/A	0	53.8	71.4	17.6
Magmet-003-05	0.059	2.55	0.40	20.2	10	76.8	87.0	10.2
Magmet-003-06	0.022	1.80	0.18	51.4	11	75.0	80.0	5.0
Magmet-003-07	0.044	1.05	3.34	67.9	48	48.7	81.0	32.3

Sample	Au head grade (oz/ton)	S= (%)	TOC (%)	Solubility (%)	Preg-Rob Factor	BRT Gold recovery (%)		
						P_{80} ¼-inch	150 Mesh	Difference 150# and P_{80} ¼-inch
Magmet-004-04	0.018	0.59	3.58	57.7	42	42.9	55.6	12.7
Magmet-004-05	0.006	1.15	3.66	35.8	15	50.0	71.4	21.4
Magmet-004-06	0.018	1.64	2.94	32.0	19	47.1	75.0	27.9
Magmet-004-08	0.023	0.98	1.90	41.8	10	42.9	54.5	11.6

Some conclusions can be drawn from the results of the tests. The ¼-inch material had a recovery range from 4.3% to 78.4%, with an average of 45.8%. Gold recoveries for the 150 Mesh material ranged from 5.8% to 87.0%, averaging 52.9%. The recovery was not very sensitive to feed size considering the size difference.

Figure 10-4: Bottle Roll Recovery and Solubility Influence



McClelland stated the refractory nature of the Pinson material is poorly understood. Preg-robbing (PR factor) assay results indicate 11 of the 23 samples would be expected to exhibit moderate to severe preg-robbing character, PR factor >50%. There is a strong correlation between the calculated solubility and the gold extraction from the bottle roll tests, and a related correlation between the TOC grade and the solubility, as would be expected when activated carbon is not employed in the leaching (CIL).

The sulfide grade did not show a strong correlation to the BRT gold extraction but in most cases the sulfide grade was low.

Cyanide leach bottle roll tests were completed on a select set of the Mag Pit samples to determine the impact of using NaOH rather than lime to abate the preg-robbing impact. For each sample type, tests were completed at pH 10.5 and 12.0. The conditions of these tests were:

- NaOH was added to raise the pH
- Samples were crushed to a P_{80} of ¼-inch
- Residence time of 48 hours

Table 10-9 shows that increasing the pH using NaOH increased gold recovery for all tests.

Table 10-9 Results from Bottle Roll Tests Using NaOH from 2013 Testwork Program

Sample	Au Head Grade (opt)	Gold Recovery (%)		
		pH 10.5 Test	pH 12.0 Test	Difference Between pH 12.0 Test and pH 10.5 Test
Magmet-001-02	0.089	53.2	60.8	7.6
Magmet-001-05	0.057	14.0	43.1	29.1
Magmet-002-02	0.030	12.5	20.8	8.3
Magmet-002-06	0.043	20.5	35.1	14.6
Magmet-003-05	0.059	75.0	78.2	3.2

Column leach tests were conducted on composites of the Mag Pit samples. The samples used to make up these composites are shown in Table 10-10. The rationale behind the compositing methodology was not clear from the report.

Column leach tests and bottle roll tests were conducted on each composite to determine gold recovery kinetics and reagent addition rates. The conditions of the bottle roll tests were:

- 48 hours residence time
- P_{80} of ¼-inch
- Hydrated lime was added to raise the pH to 12.0
- Residence time for the column leach tests varied between 72 and 76 days

Generally, the tests were conducted on samples that had been crushed to 2 inches. The Mag Column 2 sample had an additional test on a sample crushed to ½-inch to ascertain the impact of size on gold recovery. The ½-inch column recovered essentially the same as the 2-inch column of the same material and grade.

Lime was added to agglomerate the single ½-inch column. The lime was cured in the column for 72 hours prior to applying the leach solution. Agglomeration is required for any material that might exhibit percolation issues. Materials that exhibit mild percolation issues in the lab may exhibit more severe percolation issues during commercial operations. Larger and taller lab column tests can help predict the potential for percolation issues.

Lime additions were based on the bottle roll test lime requirements. This is typical for column tests and one of the intents of the bottle roll tests.

The results of the McClelland column tests are shown in Table 10-11.

Table 10-10 Sample Composition for Column Leach Tests from 2013 Test Work Program

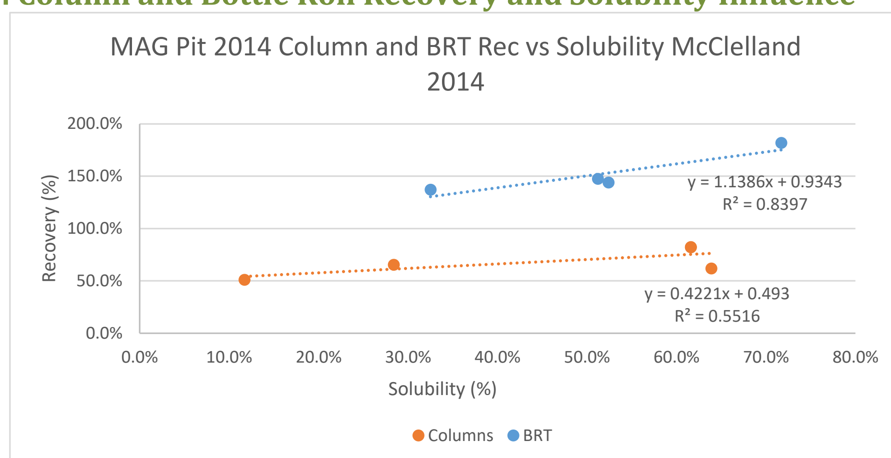
Sample	% of Composite
Mag Column 1	
Magmet-001-01	14.3
Magmet-001-03	15.5
Magmet-001-04	33.2
Magmet-003-05	17.1
Magmet-004-04	19.9
Mag Column 2	
Magmet-002-04	18.5
Magmet-002-05	8.8
Magmet-003-04	28.4
Magmet-003-06	14.5
Magmet-003-07	13.8
Magmet-004-06	13.2
Magmet-004-08	12.8
Mag Column 3	
Magmet-001-02	26.2
Magmet-001-05	6.4
Magmet-001-06	10.1
Magmet-001-07	17.7
Magmet-001-08	13.9
Magmet-002-02	12.2
Magmet-002-06	13.5
Mag Column 4	
Mag Column 1	13.8
Mag Column 2	43.1
Mag Column 3	43.1

Table 10-11 : Bottle Roll and Column Test Results from 2013 Test Work Program

Sample	Calc Au Grade (opt)	Test Type	Feed Size (inches)	Leach Time (days)	Gold Recovery (%)	Cyanide Consumption (lb/short ton)	Lime Consumption (lb/short ton)
Mag Column 1	0.042	Bottle roll	¼	2	52.4	0.6	14.6
	0.034	Column	2	73	61.8	2.5	14.6
Mag Column 2	0.053	Bottle roll	¼	2	71.7	1.1	21.4
	0.051	Column	2	76	82.4	3.3	21.4
	0.050	Column	½	72	82.0	3.8	21.4
Mag Column 3	0.040	Bottle roll	¼	2	32.5	0.7	16.1
	0.057	Column	2	72	50.9	2.6	16.2
Mag Column 4	0.043	Bottle roll	¼	2	51.2	0.8	17.3
	0.049	Column	2	76	65.3	3.0	17.4

Table 10-11 shows that the gold recoveries in the column tests varied from 51% to 82%. The Mag Column 2 tests showed no benefit to gold recovery by crushing finer. The ½ inch and 2-inch columns had similar recovery for the materials tested. The gold solubility percentage correlated to the column and bottle roll gold extractions.

Figure 10-5: Column and Bottle Roll Recovery and Solubility Influence



10.2.3 Dawson Metallurgical Program 2005 and 2006

Dawson Metallurgical Laboratories completed autoclave metallurgical test work programs on samples from underground (Ogee samples) on behalf of Atna (Dawson, 2005; Dawson, 2006a; Dawson, 2006b). The 2005 report reported on samples LR&RR and 33941, 33942, 34259. These samples were treated to determine applicability of the material for autoclave treatment. This initial program was followed by the 2006 test work and report, which appears to be the final report.

This test work program was completed on the following samples:

- A composite from the Ogee underground deposit labelled “Right Rib and Left Rib”

- Composites from the RFZ labelled as:
  - LR&RR
  - RF_Met-1 (33941)
  - RF_Met-2 (33942)
  - RF_Met-4 (34259)
- Composites from the CX Zone labelled as (drill footages identified):
  - APCX-204
  - APCX-211
  - APCX-219
  - APCX-226
- Undefined samples:
  - AMW-002
  - Met1 and Met 2 from the 2005 program (re-tested)
  - Met1 and Met 2 from the 2006 program (re-tested)

The objective of these programs was to determine whether the underground materials identified as refractory could be treated using autoclave pre-treatment. Atna was considering contracting with a third party for autoclave treatment and downstream processing of the underground material.

The scope of these test work programs included:

- Head assays including gold, sulfur speciation, and carbon speciation
- Baseline cyanide leach shake-out tests on ground feed samples
- Pressure oxidation test work: grinding of samples to either a P_{80} of 75 µm or 45 µm.
- Acidulation, where sulfuric acid was added to achieve a pH of 1.8 to 2.0 and processed for one hour. The purpose of this stage was to digest carbonate minerals ahead of the autoclave stage, which is a standard methodology for whole material autoclave treatment in Nevada.

The acid leach residue was then processed in an autoclave with the following conditions:

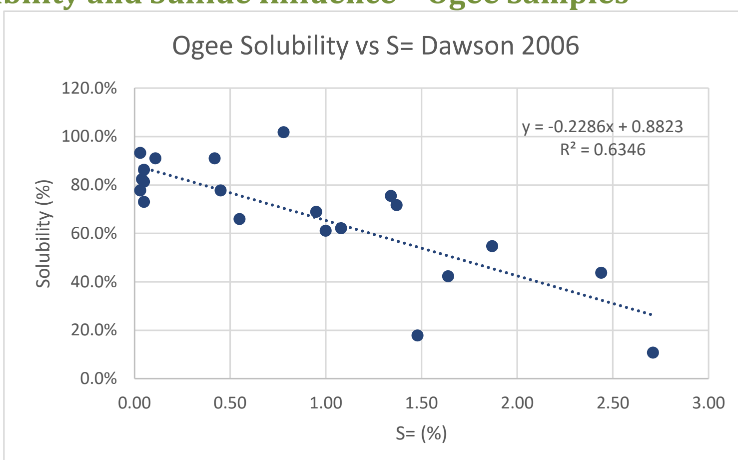
- Temperature of 225 °C
- Residence time of one hour
- Pulp density of 35% solids w/w
- Oxygen overpressure of 460 pounds per square inch (psi)

Lime was added to the autoclave residue to raise the pH to a range of 10.0 to 10.5 prior to leaching with cyanide. The autoclave residue underwent a cyanide leaching test simulating CIL (addition of carbon) processing to determine gold recovery.

The cyanide leach gold recoveries from the baseline tests and the autoclave tests are shown inTable 10-12. The refractory material responded well to the autoclave pre-treatment, and, for those tests with a baseline cyanide leach (no pre-treatment) compared to the autoclave pre-treatment, there was an increase from an average recovery of 52% to 92%.

An attempt was made to fit an equation to demonstrate the relationship between sulfide sulfur analysis and baseline gold recovery. The Coefficient of Determination (R^2) was very low for the equation and for this reason it is not presented. There was an inverse relationship between sulfide sulfur content and cyanide solubility (no autoclave treatment). The solubility in this case showed a reasonable correlation to the sulfide grade.

Figure 10-6: Solubility and Sulfide Influence – Ogee Samples



There is a range of recoveries from 11% to 86% for those samples with measured sulfide sulfur. The previous test work completed by McClelland labs noted that the sulfide sulfur content did not correlate well with cyanide solubility. Other potential issues impact cyanide solubility, such as the presence of organic carbon, and not all of the gold is directly associated with pyrite, as demonstrated by the wide range of baseline cyanide solubility tests. Table 10-12 Samples MET1 and MET2 did not undergo baseline cyanide leach tests. These samples were from prior autoclave tests and were submitted for re-testing.

Table 10-12: Autoclave Pre-treatment Tests from Dawson Test Work Program

Sample	Year of Test work Program	Grind P_{80} (µm)	Gold Head Assay (opt)	Total Sulfide Sulfur (%)	Total Carbon (CO_2) Head Assay (%)	Cyanide Leach Gold Recovery (%)	
						Baseline Tests	Tests on Autoclave Residue
Ogee Samples							
Ogee (Right Rib + Left Rib)	2005	75	0.40	0.0	0.82	86	93
RF Zone Samples							
RF_Met-1 (33941)	2005	75	0.24	1.21	2.27	52	93
RF_Met-2 (33942)	2005	75	0.43	2.61	1.76	61	95
RF_Met-4 (34259)	2005	75	0.43	2.32	2.43	11	89
CX Zone Samples							
APCX-204	2006	75	0.27	0.00	5.34	94	N/A
APCX-211	2006	75	0.33	0.00	4.29	85	N/A
APCX-219	2006	75	0.33	0.84	0.77	60	91
APCX-226	2006	45	0.56	1.53	2.70	42	94
Undefined Samples							
AMW-002	2006	75	0.33	0.03	0.35	77	N/A
MET 1	2005	75	0.51	1.21	2.27	N/A	93
MET 2	2005	75	0.32	2.61	1.76	N/A	95

10.2.4 FLS Metallurgical Program 2023

In early 2023 FLS was contracted to undertake a series of tests related to the potential underground material. The objective of this report was to provide the following: ore characterization, mineralogical testing, comminution testing, acid-alkaline batch pressure oxidation (POX), followed by batch cyanidations, and preg robbing tests. This program also included a continuous POX test followed by batch neutralization testing and subsequent cyanidation, and cyanide detoxification testing. A batch of POX-CIL tests for 3 different composite blends was also undertaken.

10.2.4.1 Samples Characterizations

The samples received were assayed with the results shown in Table 10-13.

Table 10-13: Underground Samples Head Assays from FLS Program

Sample Number	Description	ID	Au opt	Ag opt	S= %	Corg %
1	OG Zone Upper (Sulfide)	OGU	0.424	0.135	1.96	0.44

Sample Number	Description	ID	Au opt	Ag opt	S= %	Corg %
2	OG Zone Lower (Sulfide)	OGL	0.830	0.274	2.98	0.12
	OG Zone Oxide	OGOX	0.364	0.337	0.43	0.34
3	OG Zone High Grade (Sulphide) Variability Sample	OGHG	1.10	0.216	3.89	0.10
4	OG Zone Low Grade (Sulphide) Variability Sample	OGLG	0.246	0.65	2.73	0.36
	OG Zone Comminution Sample No. 1	OGCOM1	0.372	4.47	2.33	0.33
	OG Zone Comminution Sample No. 2	OGCOM2	1.15	0.285	0.04	0.03
	OG Zone Comminution Sample No. 3	OGCOM3	0.303	0.207	0.15	0.02
	OG Zone Comminution Sample No. 4	OGCOM4	0.290	0.123	0.00	0.19
	OG Zone Comminution Sample No. 5	OGCOM5	0.765	0.108	0.00	0.03
	OG Zone Comminution Sample No. 6	OGCOM6	0.376	0.056	1.41	1.12
5	Otto Zone Upper (Sulfide)	OTU	0.487	0.117	1.14	0.18
6	Otto Zone Lower (Sulfide)	OTL	0.437	0.319	1.67	0.38
7	Otto Zone High Grade (Sulfide) Variability Sample	OTHG	0.595	0.376	0.79	0.19
8	Otto Zone Low Grade (Sulfide) Variability Sample	OTLG	0.182	0.032	1.48	0.22
	Otto Zone Comminution Sample No. 1	OTCOM1	0.245	0.00	3.01	0.11
	Otto Zone Comminution Sample No. 2	OTCOM2	0.406	0.029	1.81	0.47
	Otto Zone Comminution Sample No. 3	OTCOM3	0.277	0.031	2.84	0.27
9	Adams Peak Zone Upper	APU	0.223	0.027	5.84	0.33
10	Adams Peak Zone Lower	APL	0.133	0.143	4.29	0.25
11	Adams Peak Zone High Grade Variability Sample	APHG	0.596	0.029	2.86	0.20
12	Adams Peak Zone Low Grade Variability Sample	APLG	0.188	0.240	3.20	0.13
	Adams Peak Zone Comminution Sample No. 1	APCOM1	0.322	0.091	3.12	0.37
	Adams Peak Zone Comminution Sample No. 2	APCOM2	0.343	0.439	3.85	0.17
13	Otto/Adams Peak Zone Composite	OAPC	0.331	0.107	2.25	0.24
14	Deep Range Front Zone Variability Sample	DRFV	0.184	0.081	1.73	0.26
15	Range Front Zone Variability Sample	RFV	0.213	0.083	2.33	0.17
16	South Pacific Zone Variability Sample	SPZV	0.588	0.036	2.79	0.75

Mineralogy (XRD) and swelling clay analysis was conducted on the samples. The gangue mineralogy consists of quartz, k-feldspar, muscovite, clays (kaolinite and swelling clay) and calcite. Pyrite and marcasite were present in all samples as the primary sulfides. Although most samples had minor amounts of swelling clay, two samples showed higher percentages that may require attention during POX treatment (APL and APLG).

10.2.4.2 Comminution Testing

Comminution testing (Ball Mill Bond Work Index, BBWi) was conducted on selected samples.

Comminution testing was limited to Bond ball mill work index testing in consideration of the fact that no comminution design is required for this study. Production will be initially toll milled at a Nevada Gold Mines process facility or eventually through the refurbished Lone Tree process facility. Bond ball mill work index tests were conducted using a 140 mesh (106) µm closing screen size, aiming to achieve a grind size k_{80} of 200 mesh (75 µm).
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- The overall average is 16.4 (imperial), classified as hard while the 75th percentile value (normal design value) is 19.0 (imperial), classified as very hard.
- For the OG Zone samples the average value is 18.9 (imperial), classified as very hard while the 75th percentile value is 20.3 (imperial), classified as very hard. The OG Zone oxide samples have similar hardness values compared to the sulphide samples.
- ·For the Otto Zone samples the average value is 16.2 (imperial), classified as hard while the 75th percentile value is 17.4 (imperial), classified as hard.
- ·For the Adams Peak Zone samples the average value is 12.8 (imperial), classified as medium while the 75th percentile value is 13.2 (imperial), also classified as medium.
- The Deep Range Front and Range Front Zone samples are classified as medium hardness while the South Pacific Zone sample is classified as very hard.

10.2.4.3 Cyanide Shake Tests and Preg-Robbing Testing

Analytical Direct Cyanide Leach Shake Tests were conducted to provide a baseline recovery on representative samples from each of the 17 selected samples. Tests were done on pulverized samples and conducted in centrifuge tubes. The tubes were placed on a shaker for 60 minutes before centrifuging and before the collection of the pregnant solution for gold analysis by AA. Sample pH was adjusted using lime slurry to a pH of 10.5.

Preg-robbing leach test samples were pulverized and placed on shakers for 60 minutes after being spiked with a stock solution containing a known amount of gold. This test was performed to measure the preg-rob index (PRI) of each sample by comparing the amount of gold adsorbed onto the solids to the leached gold.

Cyanide shake gold extractions ranged from 4.8% to 54.7%, with the OG oxide sample having the highest extraction. The overall average gold extraction was 26.2%, reflecting the refractory sulphide nature of the majority of the samples. The average baseline CIL gold recovery was 31%. All samples demonstrated some degree of preg robbing. Historical testing showed that Mag Pit samples also demonstrated preg robbing, so the current results are not unexpected. The average preg-robbing index was 17.9%, ranging from 4.4% to 54.1% (OAPC sample). If the shake tests are used as a cyanide leach test proxy, CIL improved recoveries by an average of 4.1% (Otto Zone upper and lower samples were well above this average). This is not a significant increase showing CIL does counteract the preg robbing to a small extent but doesn't overcome the refractory nature of the samples. The only oxide sample (OGOX from the OG Zone) showed the highest extraction at 54.7% Au but only 31% Au CIL recovery. This is well below expected recovery for an oxide sample. OGOX showed low preg robbing at 7.23%, suggesting the sample has some refractory qualities.

In some deposits, organic carbon can be corelated to pre-robbing index No relationship is apparent between organic carbon content and preg-robbing or CIL gold recoveries.

10.2.4.4 Initial BTAC Tests and CIL Testing

All metallurgical samples (except for OGOX) were run through a series of BTAC tests under a series of tests with six different conditions shown in Table 10-14 . Continuous POX testing was employed to confirm the batch testing results. Test conditions B, C, D and F replicate Lone Tree autoclave operating conditions. Trona addition can counteract the problems caused by swelling clays in POx and was evaluated in test conditions C and D. Test conditions A and E include acid conditions with pre-acidulation at two temperatures. Conditions A and E were acidulated for an hour using 98% concentrated sulfuric acid. For Condition A, sulphuric acid was added in a stoichiometric ratio to enable carbonate destruction. Condition E included sulphuric acid addition to target a CO_3/S^{2-} weight ratio of 1.

Six different Batch Autoclave Tests (BTAC) conditions were run on each of the selected zones as shown. As a result of the sulfide oxidation and gold recovery determined during the batch testing for the material sample OAPC, the three conditions selected for the continuous POX run were conditions A, B, and E. Condition A had the highest average gold recovery of 89% with 96% sulfide oxidation, condition B with an average of 69% gold recovery and 60% sulfide oxidation—expected to improve at a larger scale and thus chosen over condition F, and finally condition E with the second best average gold recovery of 79% and 72% sulfide oxidation.

Table 10-14: Underground Samples Batch Autoclave Conditions from FLS Program

POX Condition	A	B	C	D	E	F
Acidulation	Yes	No	No	No	Yes	No
POX	Acid	Alkaline	Alkaline	Alkaline	Acid	Alkaline
Trona Dosage (lb./ton)	None	None	20	10	None	None
Temperature (°F)	437	390	390	390	390	390
O_2 Overpressure (psig)	100	100	100	100	100	100
Target Gauge Pressure (psig)	455	305	305	305	305	305
Pulp Density (% solids)	30	30	30	30	30	30

POX Condition	A	B	C	D	E	F
Particle Size, k_{80} (mesh)	200	200	200	200	200	270
Retention Time (min)	60	45	45	45	45	45

Carbon-in-Leach Bottle Roll Tests were conducted each sample and on the discharge from each BTAC test. These tests aimed to evaluate the effects of the different batch POX conditions on gold and silver recoveries. The conditions for these CIL tests were the same as the baseline CIL tests.

CIL Operating Conditions

- Temperature (ºF): 70
- Pulp Density (% solids): 35
- Carbon (lb./gal.): 0.17 (20 g/L)
- pH Control: 10.5-11
- Initial Cyanide Addition (lb./ton NaCN): 5.0
- Residence Time (hours): 24

Averaged results by zone are shown in Table 10-15.

Table 10-15: Underground Samples Baseline and Batch Pressure Oxidation CIL Results from FLS Program

BTAC Conditions	Parameter	OG Zone	Otto Zone	Adam's Peak	Otto/ Adam's Peak	Deep Range Front	Range Front	South Pacific
N/A	Baseline Recovery (% Au)	41.50	46.75	9.25	24.00	10.00	34.00	43.00
A	S Oxidation (%)	86.26	99.49	98.87	99.33	95.87	96.62	99.48
	Recovery (% Au)	84.12	88.77	90.65	83.18	89.05	94.96	95.65
	NaCN Consumption (lb./ton)	1.32	1.5	1.32	1.00	1.22	0.62	1.60
B	S Oxidation (%)	71.11	56.10	57.71	44.00	32.95	42.06	98.68
	Recovery (% Au)	85.17	67.12	61.33	56.88	42.79	59.32	94.19
	NaCN Consumption (lb./ton)	2.82	2.14	1.46	3.62	3.24	2.62	3.64
C	S Oxidation (%)	66.56	38.66	32.35	38.22	35.26	38.63	43.01
	Recovery (% Au)	75.75	66.71	45.43	52.17	43.33	57.64	79.09
	NaCN Consumption (lb./ton)	2.22	2.14	3.20	3.58	3.42	3.82	2.34
D	S Oxidation (%)	60.60	37.91	42.81	35.56	31.21	34.76	80.47
	Recovery (% Au)	80.40	66.93	52.71	50.99	40.10	57.79	84.98
	NaCN Consumption (lb./ton)	1.50	2.22	3.30	1.84	3.30	3.80	3.30

BTAC Conditions	Parameter	OG Zone	Otto Zone	Adam's Peak	Otto/ Adam's Peak	Deep Range Front	Range Front	South Pacific
E	S Oxidation (%)	65.20	92.98	67.32	68.67	36.42	42.92	98.84
	Recovery (% Au)	86.59	89.31	70.02	81.70	41.47	63.04	93.95
	NaCN Consumption (lb./ton)	2.62	1.98	2.84	0.76	0.94	3.60	3.70
F	S Oxidation (%)	67.68	60.17	58.13	44.89	32.95	45.06	98.80
	Recovery (% Au)	86.40	72.76	65.74	61.13	45.18	60.99	94.83
	NaCN Consumption (lb./ton)	1.52	2.66	1.42	2.14	3.02	2.78	4.16

Observations on the results:

- Baseline CIL tests confirmed the refractory nature of the samples with an average recovery of 31.3% Au.
- Acidic BTAC conditions produced the highest gold recoveries:
    - BTAC A conditions produced the highest $S^{=}$ oxidation, averaging 95.6% and highest average recovery averaging 88.6% Au. Gold recoveries ranged from 70.2% (99.5% $S^{=}$ oxidation) with sample OGLG to 95.7% (99.6% $S^{=}$ oxidation) with sample SPZV.
    - BTAC E conditions produced the next highest $S^{=}$ oxidation, averaging 71.8% and next highest average recovery averaging 79.0% Au. Gold recoveries ranged from 42.2% (31.8% $S^{=}$ oxidation) with sample APHG to 94.5% (61.7% $S^{=}$ oxidation) with sample OGL.
- Alkaline BTAC conditions produced lower $S^{=}$ oxidations and lower gold recoveries.
    - BTAC F conditions produced the best alkaline results, likely from the finer grind compared to the baseline alkaline conditions (B). The average $S^{=}$ oxidation was 60.3% with an average gold recovery of 72.6%.
    - BTAC B conditions had an average $S^{=}$ oxidation of 59.8%, with an average gold recovery of 69.2%.
    - The alkaline condition tests with trona addition produced the lowest $S^{=}$ oxidations and the lowest gold recoveries. BTAC C conditions had an average $S^{=}$ oxidation of 44.1%, with an average gold recovery of 61.5%. BTAC D conditions had an average $S^{=}$ oxidation of 46.7%, with an average gold recovery of 64.6%.

In terms of the how samples from the various zones responded:

- Adam's Peak and Deep Range Front are the most refractory zones based on their baseline CIL recoveries. The South Pacific Zone variability sample responded well to all BTAC conditions.
- OG Zone samples demonstrate a trend between increasing gold recovery with increasing gold head grade. In terms of the primary zones, OG Zone samples had the highest recoveries, regardless of the POx conditions. The highest average OG Zone gold

recoveries was with conditions F, (alkaline with finer grind), followed by BTAC A conditions.

- Otto Zone and Adam's Peak Zone samples showed no clear relationship between gold head grade and gold recovery.
- The highest average Otto Zone gold recoveries with acidic BTAC conditions (E followed by A).
- The highest average Adams Peak Zone gold recoveries was with BTAC A conditions. The average gold recoveries for the other three conditions were significantly lower.
- Composite OAPC best responded to acidic BTAC conditions A and E and responded poorly to alkaline conditions with respect to gold recovery.
- SPZV responded well to all BTAC conditions, with an average gold recovery of 90.5%. Sulphide oxidations ranged from 43.0% with BTAC C conditions (79.0% Au recovery) to 99.5% with BTAC A conditions (95.7% Au recovery).

RFV and DRFV only responded well to BTAC A conditions, with gold recoveries of 95.0% and 89.1% respectively. Gold recoveries for the other conditions were all below 60%.

Overall, there is a positive trend between $S^{=}$ oxidation and gold recovery as shown in Figure 10-7.

Figure 10-7: CIL Gold Recovery as a Function of Sulfide Sulfur Oxidation – Underground Samples

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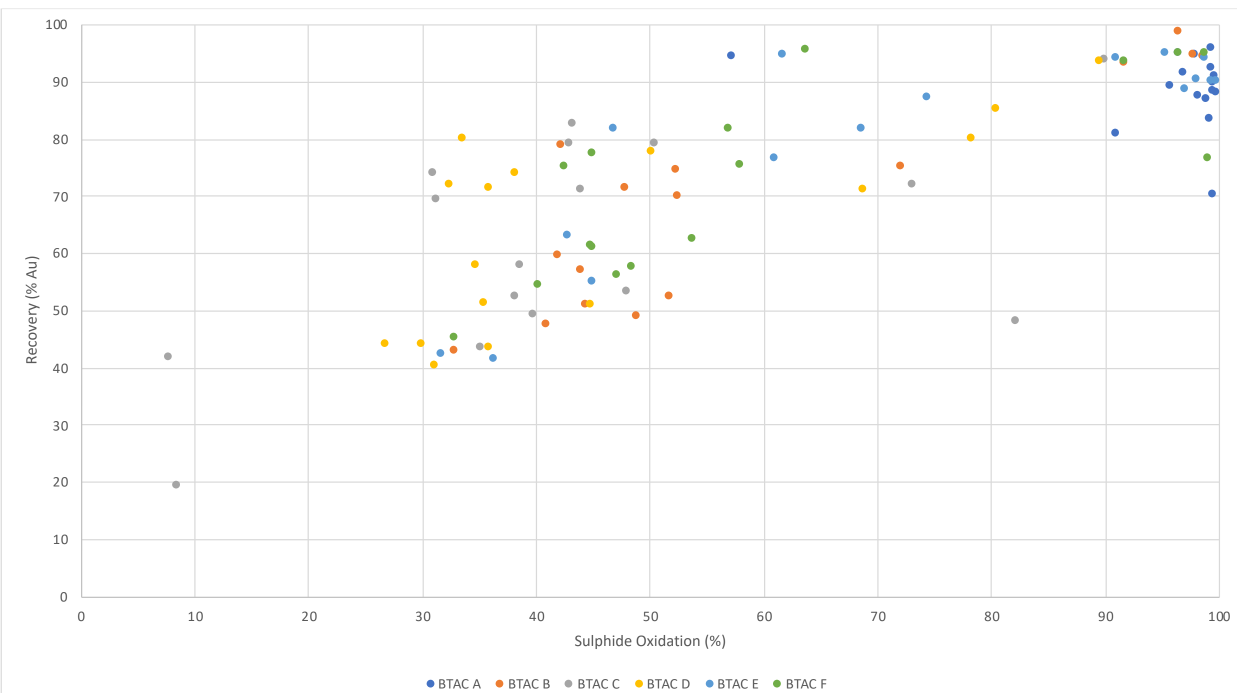

10.2.4.5 Follow Up BTAC Tests and Roasting Testing

Follow tests aimed at optimizing alkaline POx conditions and to compare against bench top roasting. The blends are shown in Table 10-16. The objective was primarily to evaluate longer retention time under alkaline conditions (Lone Tree autoclave) and to confirm the results from adding trona to POx. Predicted sulphide oxidations and up tests were completed on three sample blends. The test conditions are shown in Table 10-17.

Table 10-16: Granite Creek Underground Metallurgical Sample Blends for Additional Testing

Sample	Composition
Blend 1	65% OTHG, 34% APHG
Blend 2	50% OTLG, 34% OUT, 16% APL
Blend 3	60% OUT, 27% APHG, 13% OTLG

Table 10-17: Granite Creek Underground Metallurgical Testing Program Follow Up BTAC Test Conditions

Test Condition	I	II	III	IV	V
Acidulation	No	No	No	No	No
POx Condition	Alkaline	Alkaline	Alkaline	Alkaline	Alkaline
Trona Dosage (lb./ton)	None	None	40	20	None
Temperature (^oF)	390	390	390	390	390
O_2 Overpressure (psi_g)	100	100	100	100	100
Pulp Density (% solids)	30	30	30	30	30
Particle Size, k_{80} (mesh)	200	200	200	200	200
Retention Time (min)	60	45	45	45	75

Benchtop roasting (BTR) was performed in a Carbolite HTR rotary reactor tube furnace. A dry ground sample was weighed into a tared baffles borosilicate glass reactor and subjected to a two-stage roast with oxygen (@99.9% O2 purity) gas applied across the roaster bed. BTR test conditions included:

- Temperature 1 (°C): 986
- Residence 1 Time (min): 30
- Temperature 2 (°C): 1,058
- Residence 2 Time (min): 15
- Ramp Rate (°F/min): 9
- Oxygen Rate (cc/min): 2
- Sample Feed Weight (lb.): 1.1-1.2 (500-550 g)
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Results for sulfide oxidation are shown in and results for gold recovery are shown in Table 10-18 and Table 10-19 respectively. Overall, the actual sulphide oxidations exceeded predicted values and gold recoveries slightly exceeded predicted values. Extended retention times did not make a significant impact on gold recoveries. Trona additions again did not prove to be beneficial. BTR provided superior sulfur oxidation and gold recoveries for all blend samples.

Table 10-18: Granite Creek Underground Metallurgical Testing Program Follow Up BTAC Sulfide Oxidation Results Compared to Predicted Results

Test Condition	Actual Results				Predicted Results			
	Blend 1	Blend 2	Blend 3	Average	Blend 1	Blend 2	Blend 3	Average
I	48	57	49	51	-	-	-	-
II	43	52	42	46	43	60	50	51
III	39	49	39	42	39	26	33	33
IV	37	35	41	38	36	33	37	35
V	55	52	45	51	-	-	-	-
BTR	99.0	93.9	98.2	97.0				

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Table 10-19: Granite Creek Underground Metallurgical Testing Program Follow Up BTAC Gold Recovery Results Compared to Predicted Results

Test Condition	Actual Results				Predicted Results			
	Blend 1	Blend 2	Blend 3	Average	Blend 1	Blend 2	Blend 3	Average
I	46	75	71	64	-	-	-	-
II	65	74	70	70	63	69	66	66
III	67	73	68	69	63	64	64	64
IV	52	73	69	65	62	68	65	65
V	69	77	66	71	-	-	-	-
BTR	84.0	85.0	80.0	83.0				

10.2.4.6 Continuous Pressure Oxidation Testing

10.2.4.6.1. Continuous Pressure Oxidation Operation

A continuous POx test was completed using sample OAPC for three conditions shown in Table 10-20. These conditions were based on the batch BTAC results for the OAPC sample with highest sulfide oxidation and gold recovery. Run 1-Condition B was an alkaline test with no reagents added with six 45-minute turnover intervals, Run 2-Condition E was a partially acidulated test (only partial carbonate destruction) with six 45-minute turnover intervals, and Run 3–Condition A was fully acidulated (total carbonate destruction) with six 60-minute turnover intervals and 437^{0}F.

The three test runs operated back-to-back for approximately 16 hours excluding autoclave heating and cooling time. The first set of POx profile samples were collected once steady state was achieved and another set was collected after three volume changeovers. In addition to sampling, 4.22 gal. (16 L) of POx discharge were collected and weighed at each changeover for downstream testing. Continuous autoclave feed rate for Runs 1 and 2 was approximately 29.3 lb./hour (13.31 kg/hour) of solids to provide 45-minute residence time runs, and for Run 3 the feed rate was approximately 22.0 lb./hour (9.98 kg/hour) to provide the 60-minute residence time.

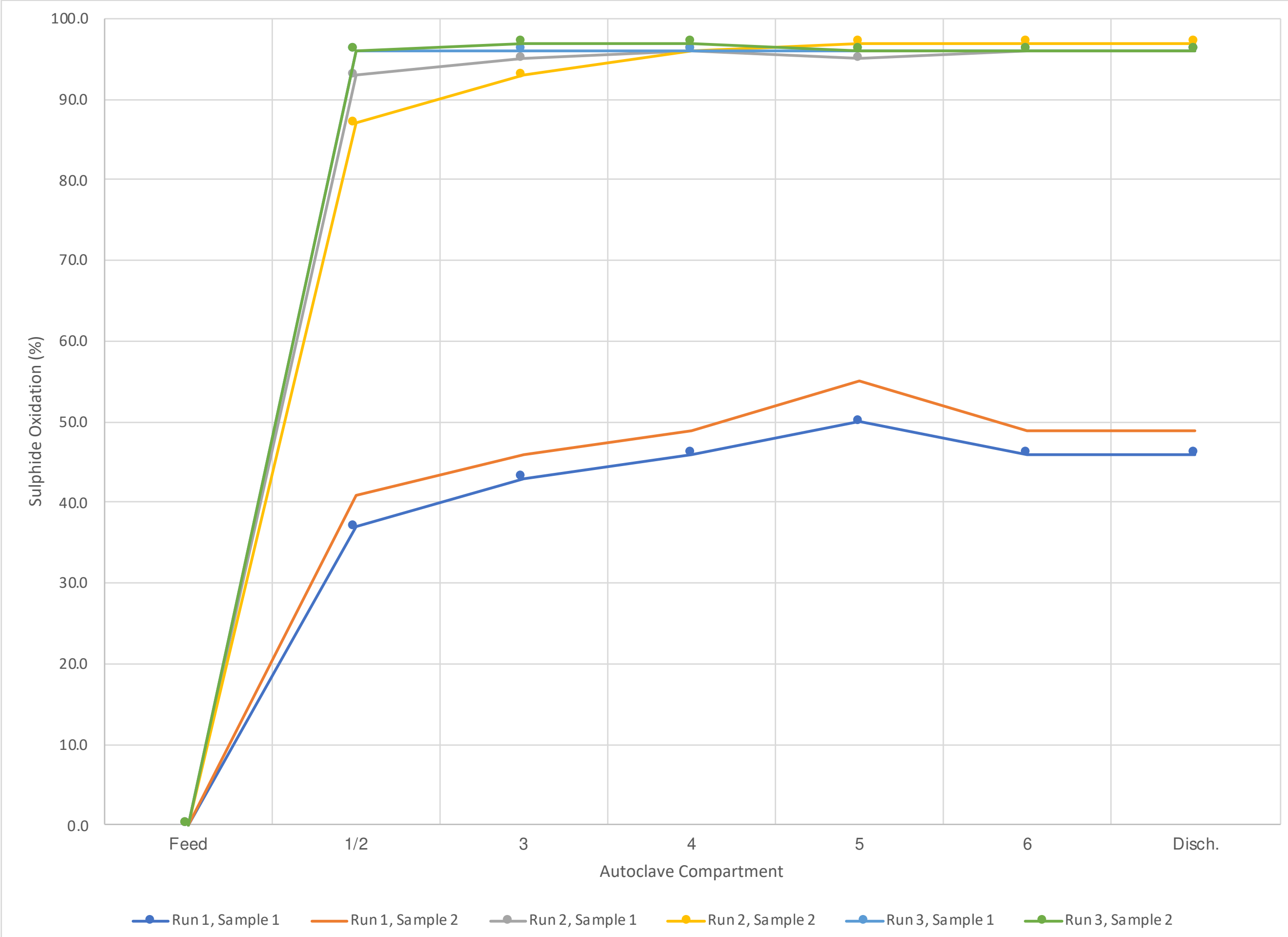
Table 10-20: Granite Creek Underground Metallurgical Testing Program Continuous POx Run Test Conditions

Test Condition	B (Run 1)	E (Run 2)	A (Run 3)
POx Condition	Alkaline	Acid	Acid
Acidulation	No	Partial	Complete
Trona Dosage (lb./ton)	0	0	0
Temperature (^oF)	390	390	437
O_2 Overpressure (psi_g)	100	100	100
Total Pressure (psi_g)	304	304	454
Pulp Density (% solids)	30	30	30
Particle Size, k_{80} (mesh)	200	200	200
Retention Time (min)	45	45	60

10.2.4.6.2. Continuous Pressure Oxidation Results

The primary objective of the POX process is to oxidize sulfide sulfur to liberate contained refractory gold. Figure 10-8 shows sulfide oxidation by autoclave compartment, starting with the autoclave feed and ending with autoclave discharge. Sulfide oxidation starts quickly in compartments ½ and reached near full oxidation by compartment 3 for Run 2 and Run 3. As expected, Run 1 did not reach the same level of oxidation as the next two runs under alkaline conditions.

Figure 10-8 Granite Creek POx Pilot Plant Sulfide Oxidation Profile



Source: TR Raponi Consulting Ltd. (2023)

Continuous POX discharge leached in a hot stir tank showed no significant gold recovery difference compared to POX discharge leached in a bottle roll at ambient temperature. The effect temperature had on the discharge was increased lime and cyanide consumption.

Table 10-21 compares the continuous conditions. Conditions A and E had much higher gold recovery and sulfide oxidation when compared to Condition B but also consumed significantly more lime and recovered much less silver.

Table 10-21: Underground Samples (OAPC) Continuous Autoclave Tests from FLS Program

Parameter	Baseline Results	Run 1 (B) – Alkaline			Run 2 (E) – Partial Acidulation			Run 3 (A) – Full Acidulation		
Bottle Roll Conditions		Hot Stir	Bottle Roll	BTAC	Hot Stir	Bottle Roll	BTAC	Hot Stir	Bottle Roll	BTAC
Sulphide Oxidation (%)		49		44	97		69	96		99
CIL Recovery (% Au)	24	63	63	57	89	91	81.7	91	91	83
CIL Recovery (% Ag)	16	50	42	-	3	3	-	2	1	-
Lime Consumption (lb./ton)	2.60	10.5	6.0	-	86.6	78.38	-	111.76	105.50	-
Cyanide Consumption (lb./ton)	4.90	5.90	3.58	3.62	5.12	3.66	0.76	6.60	3.28	1.00

The acid POX conditions (A and E) had much higher sulfide oxidation when compared to alkaline POX (B). This resulted in higher gold recoveries but also required significantly more lime for neutralization. As a result, the increased lime consumption likely caused much of the silver to be locked within jarosite, which did not appear to form in alkaline POX.

10.2.4.7 Cyanide Destruction Testing

Cyanide destruction tests were performed using the SO_2/air process. The SO_2/air process was originally developed and patented by Inco Ltd. (now Vale). SO_2 plus air oxidizes cyanide into cyanate, catalyzed by the addition of copper ions. Typical retention times to achieve <5 ppm weak acid dissociable cyanide (CN_{WAD}) are 1 to 2 hours, at pH levels of 7.5 to 9.5.

SO_2 is now usually provided in the form of sodium metabisulphite solution dissolved on site or elemental Sulphur combusted to generate SO_2 on site for larger users.

This process is capable of achieving discharge concentrations of < 1 mg/L CN_{WAD}. The addition rate of SO_2 to cyanide is optimized for consumption and desired discharge cyanide concentration. The process is not suited to directly reducing total cyanide (CN_T).

A total of 8.45 gal. (32) L of transitional continuous POX material was collected as discharge from the beginning of Run 2. This material was used to conduct a bulk Carbon-in-Leach test for 24 hours using the autoclave feed reactor and the standard CIL conditions.

After the 24-hour leach, the sample underwent cyanide detox testing with continuous sodium metabisulfate (SMBS) and copper sulfate addition. Conditions of the tests are shown Table 10-22.

Table 10-22: Underground Blend Samples Cyanide Detox Conditions from FLS Program

Test Number	1	2
Feed Slurry	Bulk CIL	
Oxidizing Agent	SMBS	
Feed CN_{WAD} (ppm)	177	182
SO_2:CN_{WAD} (mass basis)	8	5
Retention Time (h)	1	1
Copper (ppm)	25	25
pH range	8.0 - 9.0	8.0 - 9.0
Number of Turnovers	6	6
Total Continuous Time (h)	4.5	4.5
Temp ºC	77ºF	77ºF

The CIL slurry was screened for carbon and prepared for the cyanide detox set up. Two of the 32 L of carbon-free slurry were then used to prime the cyanide detox set up, and detox testing was started. Samples were taken from the feed every two hours, and from reactors 1, 2, and the discharge every hour. Cyanide speciation assays for each of these solutions were determined.

Reagents consumptions in both cyanide detox tests can be found in Table 10-23. Lime was used during bulk CIL testing to create the feed for both detox tests.

Table 10-23: Underground Cyanide Detox Reagent Consumption from FLS Program

Reagent	Detox 1	Detox 2
Lime Consumption (lb./ton)	2.88	2.88
SO_2 Consumption (lb./ton)	6.90	6.84
Lime:SO_2 (wt/wt)	0.42	0.60
$CuSO_4$ Maintained (ppm)	29.6	32.1
$CuSO_4$ Consumption (lb./ton)	0.14	0.18

Table 10-24 shows the final cyanide results from both cyanide detox tests. The additional SMBS added in Test 1 is shown to result in significantly less CN_{WAD} when compared to Test 2.

Table 10-24: Underground Cyanide Detox WAD from FLS Program

Time (hours)	Detox 1	Detox 2
	CN_{WAD} (ppm)	
1	22.5	51.1
2	25.9	42.8
3	23.2	46.1
4	33.2	50.5

10.2.4.8 Solids Liquids Separation Testing

10.2.4.8.1. Thickener Testing

POX discharge slurry samples were retained from the POX Runs 1, 2, and 3 plus product slurry from Detox 1 (POx Run 2 sample) for solids-liquids separation testing for optimum sizing and selection of thickener operating parameters. Testing was for pre-leach thickener duty for the POx discharge samples and tailings thickener on the cyanide detox sample.

The Run 1 sample was used for the flocculant screening testing. The flocculants tested were 905VHM, 910, VHM, 913VHM, 923VHM and 934VHM. Flocculant 913VHM was found to provide the best overflow clarity and settling velocities when compared to four other flocculants. The findings were consistent for all samples and 913 VHM flocculant was selected for subsequent testing on all samples. This flocculant can be substituted with any comparable anionic polyacrylamide flocculant with a medium molecular weight and low charge density.

Flux testing showed the optimum feedwell suspended solids concentration for flocculation to be approximately 4% by weight for all samples except for the Run 3 sample which is approximately 2% by weight. These diluted densities are much lower than seen for most applications that are in the 10% to 15% solids by weight. Testing shows all samples produced acceptable overflow clarity.

The Run 1 sample (alkaline POX) and the Detox Tailings were the only samples that produced acceptable underflow densities at 51% and 48% solids by weight. Run 2 and Run 3 samples produced thickener underflow densities that were no better than POx discharge densities. Underflow slurry yield stresses are above what is considered the maximum yield stress for centrifugal pumping of 25 Pa.

Thickening results indicate that use of a thickener downstream of POx is not recommended.

10.2.4.8.2. Filtration Testing

FLSmidth conducted pressure filtration tests using a bench-scale filtration testing unit. The bench-scale testing unit can simulate FLSmidth's recessed chamber and membrane squeeze chamber configurations allowing for various feed solids concentrations, pressure profiles, and cake thicknesses. Filter testing was completed on the Detox Tailings sample to assess the potential for filtered tailings disposal. The pressure filtration test was fed at 48% solids by weight to simulate feeding from a thickener underflow.

While the final filter cake moisture and density are suitable for disposal in a typical disposal area, the filtration rate is about an order of magnitude below what is typical for this duty. The filtration equipment requirements to implement filtered tailings would be inordinately high but would provide suitable tailings for dry disposal.

10.3 Sample Representativity

Many of the samples used for the metallurgical test work were bulk samples collected either underground or on the benches of the open pits. The precise location of these samples was not fully documented, although the samples were identified as to which pit they originated from. Mag pit and some C and CX samples did include some core drill samples that were well documented as to the precise drill hole location and interval.

10.3.1 Overview

Samples used for the metallurgical test work have been sourced from the open pits (Mag Pit, CX Pit, and underground). Within each zone, drilling has been localized to relatively small portions of the deposit. The metallurgical response of the samples is likely to represent the general behavior of the zone, but sampling different areas of each zone to confirm the metallurgical response will reduce uncertainty. Additional targeted drilling in different zones is recommended to mitigate the risk. Figure 10-9 and Figure 10-10 show the recommended targeted drill hole locations.

Figure 10-9 Plan View Showing Metallurgical Sample Locations

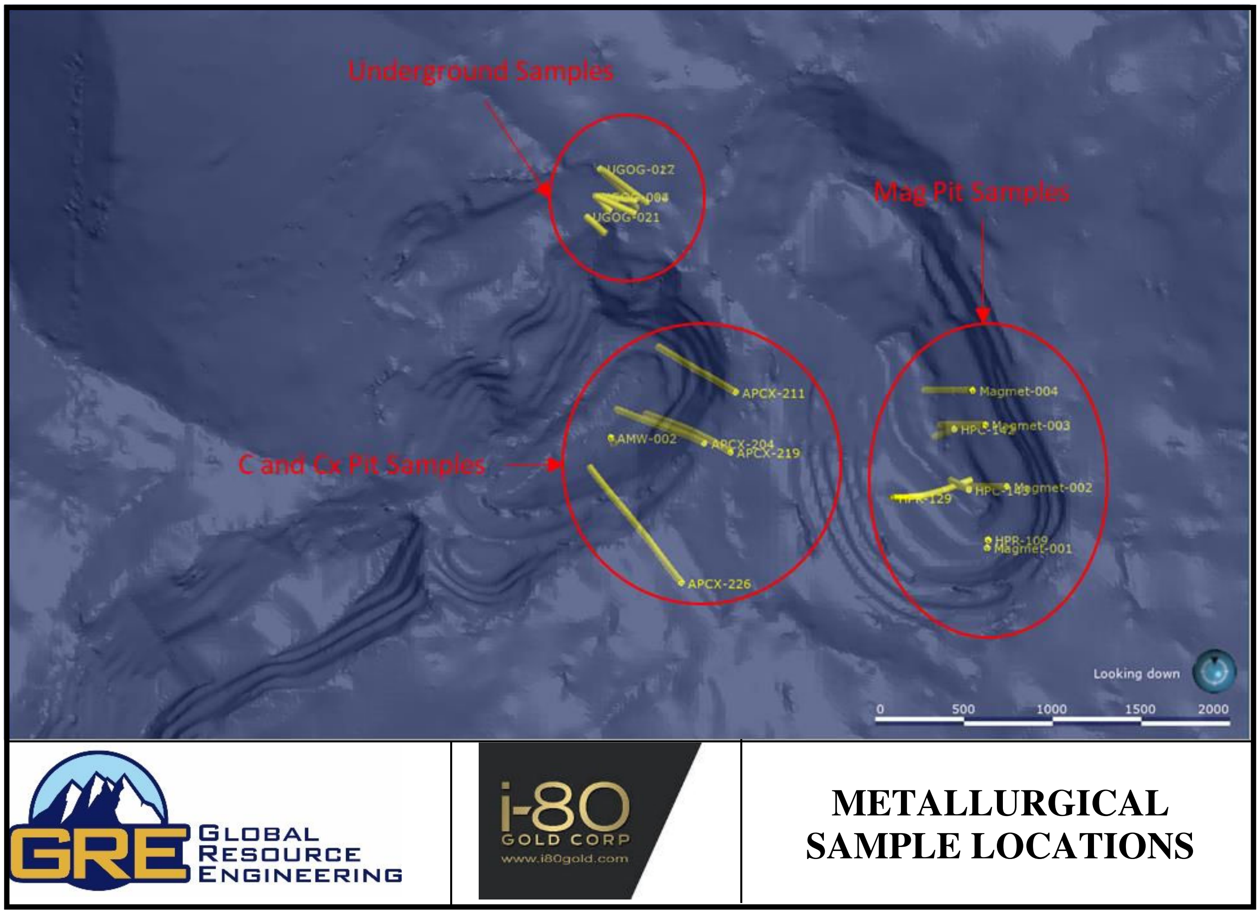
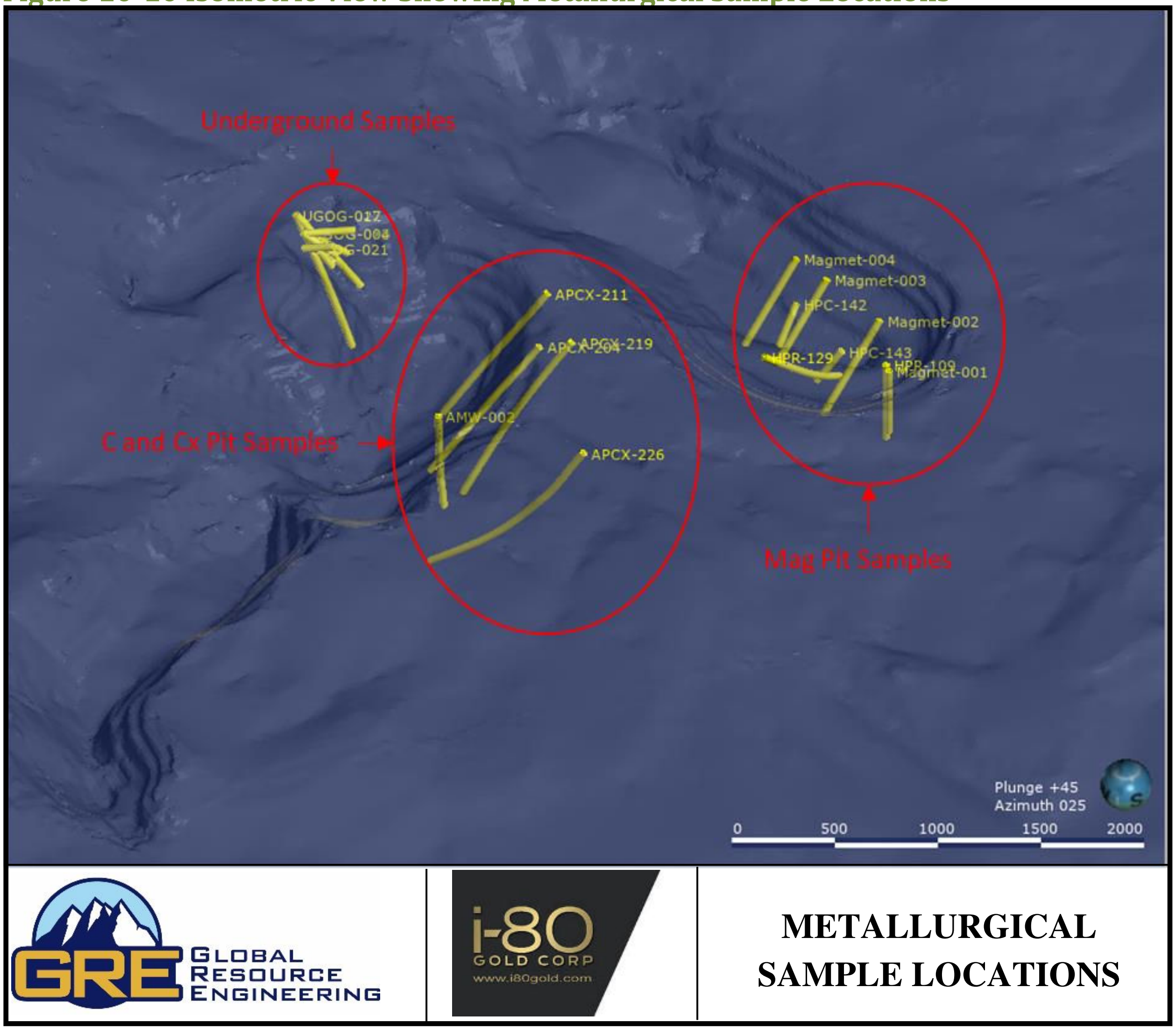


Figure 10-10 Isometric View Showing Metallurgical Sample Locations



10.3.2 Bulk Samples

Bulk samples were sourced from the Mag Pit and CX Pit. Six bulk samples of approximately 1,000 lbs each were sourced from the Mag Pit and designated Mag Pit I through Mag Pit VI. One bulk sample of approximately 4,300 lbs was sourced from the CX Pit. The locations of the samples were not reported, so it is not possible to assess whether they are representative of the eventual Mineral Resource volume.

10.3.3 Drillhole Samples

The samples selected from drilling on the Project over its life are listed in Table 10-25.

Table 10-25: Drillhole Sample Selection and Testing Matrix

Sample ID	Location	Testing
HPR109	Mag Pit	Preg-robbing, bottle roll, column percolation leach tests
HPR129	Mag Pit	
HPC142	Mag Pit	
HPC143	Mag Pit	
Magmet-001	Mag Pit	Bottle roll, column percolation leach tests
Magmet-002	Mag Pit	
Magmet-003	Mag Pit	
Magmet-004	Mag Pit	
APCX-204	CX Zone	Bottle roll, column percolation leach tests
APCX-211	CX Zone	
APCX-219	CX Zone	
APCX-226	CX Zone	
AMW-002	CX Zone	Bottle roll, column percolation leach tests
UGOG-004	Underground resource	Head assays and CN soluble Au only.
UGOG-010	Underground	
UGOG-013	Underground	
UGOG-015	Underground	
UGOG-017	Underground	
UGOG-018	Underground	
UGOG-019	Underground	
UGOG-021	Underground	
UGOG-022	Underground	

Mag Pit drillholes intersect only one end of the mineralized domain. The sampling of the CX Pit is heavily clustered, and much of the mineralized domain has not been assessed metallurgically. Ogee (from the old underground developments) metallurgical test drilling intersects a restricted portion of the mineralized domain. The lens parallel to the existing workings is not intersected by any drilling. No metallurgical test work is available for the A Pit and B Pit.

Generally, drilling intersects only limited areas of the mineralized domains, and testing of additional areas is recommended. Selection of further drilling and sampling for metallurgical testing should be guided by a future mine plan. Metallurgical testing that spatially represents all zones of the project is recommended.

10.3.4 Metallurgical Composite Assembly

For all test work conducted, composites for metallurgical test work were prepared by combining drillhole intervals. From the information available, it is not apparent if the composites were prepared in a way that represents the grade and mineralogical variability within the deposits.

The samples provided by Homestake in 1999 were composited in a manner that tended to reduce the variability in the provided samples. Table 10-26 displays the composite assay and the highest and lowest assays of the intervals in the composite.

Table 10-26: Composite Assays

Composite	Composite Assay (opt)	High/Low Interval (opt)
Mag Pit Cuttings composite 1	0.094	0.027 – 0.266
Mag Pit Cuttings composite 2	0.070	0.028 – 0.103
Mag Pit Cuttings composite 3	0.068	0.036 – 0.125
Mag Pit Cuttings composite 4	0.074	0.035 – 0.102
Mag Pit Cuttings composite 5	0.059	0.014 - 0.142
Mag Pit Cuttings composite 6	0.076	0.041 – 0.163
Mag Pit Drill Core Composite 1	0.032	0.014 – 0.058
Mag Pit Drill Core Composite 2	0.077	0.003 – 0.162
Mag Pit Drill Core Composite 3	0.098	0.021 – 0.191
Mag Pit Drill Core Composite 4	0.058	0.016 – 0.093
Mag Pit Drill Core Composite 5	0.146	0.015 – 0.272

The samples do not represent the full variability of the mineralization, and test work should be undertaken on samples that represent different grade variations of the mineralization.

10.4 Deleterious Elements

Both arsenic (As) and mercury (Hg) are present in the mineralization, which is very common in Nevada gold deposits. The arsenic is not cyanide-soluble. However, the mercury is cyanide-soluble and must be collected using appropriate technology at any thermal device processing, stripping, or regenerating carbon. Much like precious metals, mercury will report to the carbon.

Naturally occurring pregnant solution robbing organic carbon is also present within some of the materials at Granite Creek. This limits the applicable processing methods for these materials. High preg-robbing materials are unsuitable for heap leaching and should be treated by CIL methods.

Any autoclave or roasting treatment for the underground refractory material will mobilize the arsenic. If adequate iron is present within the autoclave discharge, the arsenic can be fixed as ferric-arsenate. Any material treated by third-party toll treatment will potentially be subject to additional charges for mercury, arsenic, sulfides, and organic carbon.

10.4.1 Homestake Mining 1999

Test work conducted for Homestake did not report the presence or deportment of arsenic.

Mercury was assayed in Mag Pit and CX Pit bulk material samples. Assays ranged from 2.53 ppm to 43.12 ppm Hg; levels high enough to require consideration of mercury capture during refining.

TOC was present at greater than 4% in Mag Pit samples I, II, and IV, all of which displayed significant preg-robbing characteristics. TOC was less than 0.4% in the other samples, and gold recoveries were high. Based on the currently available metallurgical test work, there is not a well-defined relationship between TOC and the level of preg-robbing, but there does appear to be a relationship with the cyanide-soluble gold. A relationship between TOC, sulfide sulfur, and recovery is most likely.

10.4.2 Atna Resources 2005

Drillhole samples from the underground extension of the CX Pit were received for autoclave and cyanide leach testing. These were assayed for arsenic, and measured values between 0.054% and 1.65% were reported and are tabulated with gold by fire assay in Table 10-27.

Table 10-27: Gold and Arsenic Assays CX Pit

Sample	Au (ppm) by Fire Assay	As (%) by AA
APCX-204	8.16	0.066
APCX-211	8.33	0.130
APCX-219	10.25	0.180
APCX-226	17.5	1.650
AMW-002	10.29	0.054

A further four samples designated R Rib & L Rib, 33941, 33942, and 34259 were also received for autoclave and cyanide leach testing. These tests were completed on three composite samples from the RFZ and one composite from the Ogee Zone mineralization. These samples are understood to be samples collected from the previously mined underground mineralized body. Assays for As and Hg were not reported and assumed not measured.

Approximately 200 samples from nine drillholes in the Ogee Underground resource area were submitted for sample preparation and assaying in March 2006 (Dawson, 2006a). The individual samples were composited into 21 samples for further work. Gold assays ranged from 7.1 ppm to 54.7 ppm and arsenic from 0.09% to 0.46%.

10.4.3 Atna Resources 2013

Thirty-two (32) drill core composites from the Mag Pit area were submitted for heap leach amenability testing. A full elemental analysis was done on each sample, including As, Hg, copper (Cu), and organic carbon. The range of assays is shown in Table 10-28.

Table 10-28: Mag Pit Drill Core Composite Assays

Element	Assay Range
As	77 ppm to 671 ppm
Hg	2.1 ppm to 30 ppm
Cu	8.7 ppm to 154 ppm
C	0.14% to 5.1%

Twenty-three (23) of these composites were tested for heap leach amenability.

10.4.4 Osgood 2023

All the material for the testing campaign was received. Twenty-eight (28) samples were obtained for the various tests outlined in the campaign. A total of 621.6 kg of material was received. Drill hole identifiers were supplied with the samples and included intervals from 57 holes. Mineralogy detected arsenopyrite in all of the sample composites. Realgar was identified in low quantities in several of the samples. It does not appear that As or Hg was assayed for directly in the samples.

10.5 Geometallurgical Modeling

The test programs included cyanide solubility testing, pregnant solution robbing testing, bottle roll testing, percolation column testing, CIL testing, and autoclave testing. The objective of the study was to determine the factors that impact cyanide solubility. Since the cyanide solubility information is unavailable for all the drill holes, a geometallurgical model was created to predict the cyanide solubility based on the available information in the drill hole data. A trend between the cyanide solubility and the column test was determined to predict the heap leach recovery.

The study is divided into 5 zones: Mag pit area, C and CX pit area, A pit area, and B pit area (associated with the open pit zones), and a single underground zone. In all the zones, the gold grade, alterations, and cyanide solubility in some intervals are available. A model for cyanide solubility was created in all zones based on the available cyanide solubility information, see Table 10-29 .

Table 10-29: Available Cyanide Solubility Data in Different Zones

Zone	Number of Fire Assays Available	Number of CN Solubility Data Available	Percent of Data with CN Solubility Information
Mag Pit	26,589	20,722	77.9%
C and CX Pit	42,732	23,849	55.8%
A Pit	4,234	1,211	28.6%
B Pit	5,109	1,994	39.0%
Underground	139,191	56,887	40.9%

The deposit is mainly associated with silicification, iron oxide, argillization, pyrite, decarbonatization, carbonate, carbon, hydrogen chloride, bleaching, propylitic, and realgar orpiment alterations. All the alterations were logged as shown in Table 10-30.

Table 10-30: Numerical Equivalent Alteration Codes.

Alteration Code	Description
0	None
1	Trace (Incipient)
2	Weak (Patchy, Poorly developed)
3	Moderate (Occurs through most rock)
4	Strong (Occurs throughout, textures remain)
5	Complete (Destruction of lithologic texture)

To perform the geometallurgical modeling, different analyses were performed in all the zones:

1. **Principal Component Analysis:** A principal component analysis was performed to better understand the relationship between the elements in the ICP geochemical data and its variability. Principal component analysis reduces the variables in a data set to components that attempt to describe the greatest amount of variance in the data set. The first component describes the greatest amount of variance in the data set, the second component is orthogonal to the first and describes the second greatest amount of variance, and so on.
   a. **Scree Plots:** A scree plot shows the variance of different principal components (variables). On a scree plot, dimensions indicate the amount of variance in the data set described by each component.
   b. **Biplots:** A biplot allows information on the variables of a data matrix to be displayed graphically, where variables are displayed as vectors. The further away these vectors are from a Principal Component origin, the more influence they have on that Principal Component. Biplots also hint at how variables correlate with the principal components and one another: a small angle implies positive correlation, a large one suggests a negative correlation, and a 90° angle indicates no correlation between two variables.

2. **Regression Tree:** Regression techniques contain a single output (cyanide solubility) variable and one or more input variables (alteration mineral species, elevation, and assay grades). The output variable is numerical, and the general regression tree building methodology allows input variables to be a mixture of continuous and categorical variables. A regression tree is generated when each decision node in the tree contains a test on some input variable's value. The terminal nodes of the tree contain the predicted output variable values. A regression tree is built through a process known as binary recursive partitioning, which is an iterative process that splits the data into partitions or branches and then continues splitting each partition into smaller groups as the method moves up each branch. The prediction provided by the regression tree model is the mean cyanide solubility for the groups created.
3. **Multivariate Adaptive Regression Splines (MARS):** MARS are a form of a non-parametric regression technique and can be seen as an extension of linear models that automatically model nonlinearities and interactions between variables. MARS can handle both continuous and categorical data. Building MARS models often requires little or no data preparation. The hinge functions automatically partition the input data, so the effect of outliers is contained. In this respect, MARS is similar to recursive partitioning which also partitions the data into disjointed regions, although using a different method. The result of the GRE's non-parametric geometallurgical modeling effort was a formula (the MARS model formula) to determine the cyanide solubility for gold.

10.5.1 Cyanide Solubility for Different Zones

For open pits material only the samples with Au >0.1 ppm (0.003 oz/ton) were considered in all zones. Assays less than this value were considered below cut-off grade and were removed from the study. The material below 0.1 ppm was removed to reduce the impact of waste material on the model.

10.5.1.1 Mag Pit Zone

There were 11,812 samples with cyanide solubility information and Au grade above 0.1 ppm. Figure 10-11 shows the Scree plot, where the first component comprises of the maximum variance. Figure 10-12 shows the biplot, where gold, pyrite, and iron oxides have strong positive correlation and strong negative correlation to elevation. Figure 10-13 shows the regression tree model, which shows that gold cyanide solubility is highly dependent on gold grade and elevation.

The MARS model for the gold cyanide solubility in the Mag pit area is shown by two different models, depending on the gold grade. The first model was created where the gold grade is less than 15 ppm, and the second model was created when the gold grade is above 15 ppm.

1. Au >=0.1 ppm and Au <15 ppm

The gold cyanide solubility when gold grade is greater than 0.1 ppm and less than 15 ppm is given by:

$$AuCN\ ppm = 0.5888 - 0.6124 * max(0, 1.33 - Au\ ppm) + 0.7781 * max(0, Au\ ppm - 1.33) + 0.26 * max(0, 2 - Pyrite) - 0.4271 * max(0, Pyrite - 2) - 0.001047 * max(0, 4920 - Elevation(ft)) - 0.0009474 * max(0, Elevation(ft) - 4920)$$

The cyanide solubility equation uses gold grade, pyrite alteration, and elevation in the model. The graph showing the observed and predicted gold cyanide solubility is given in Figure 10-14. The model has an R^2 of 0.82, implying that 82% of the variations are explained by the model.

Figure 10-11 PCA- Scree Plot for Mag Pit

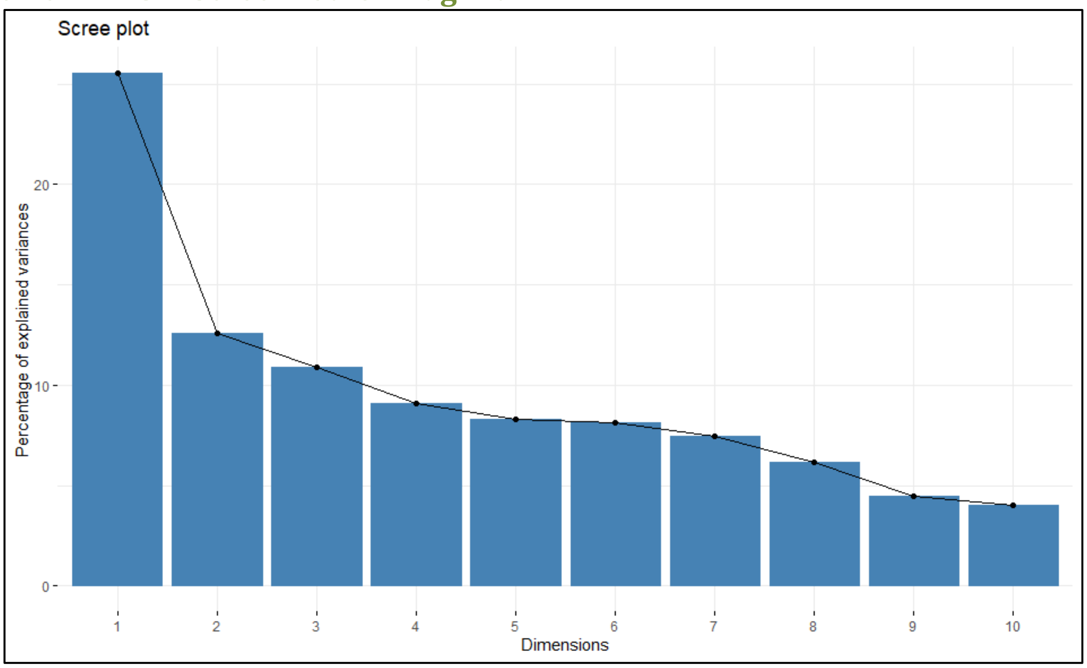


Figure 10-12 PCA – Biplot for Mag Pit

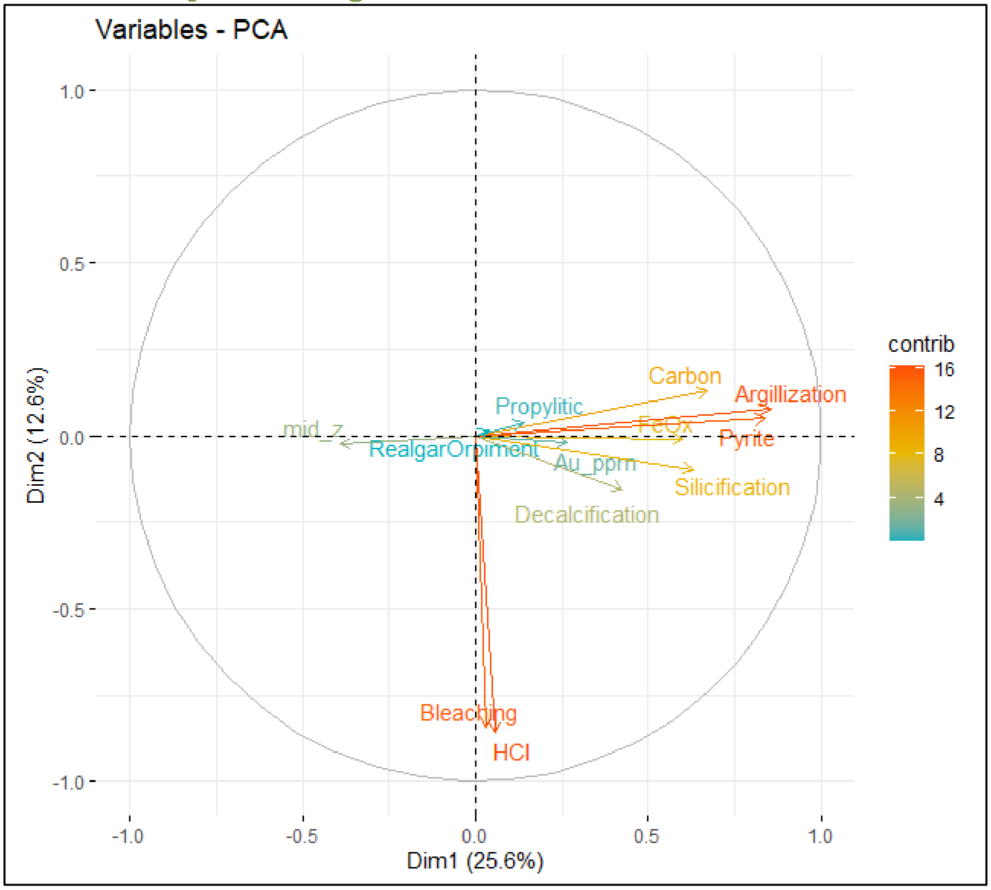


Figure 10-13 Regression Tree Model for Mag Pit

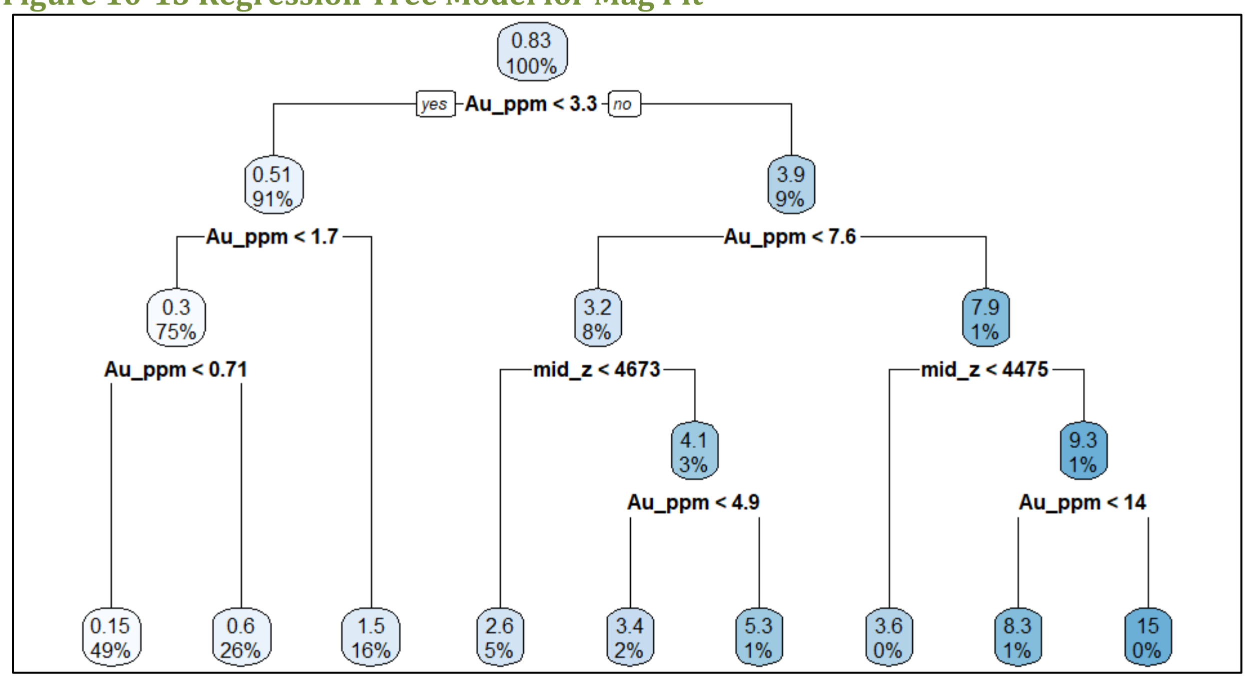
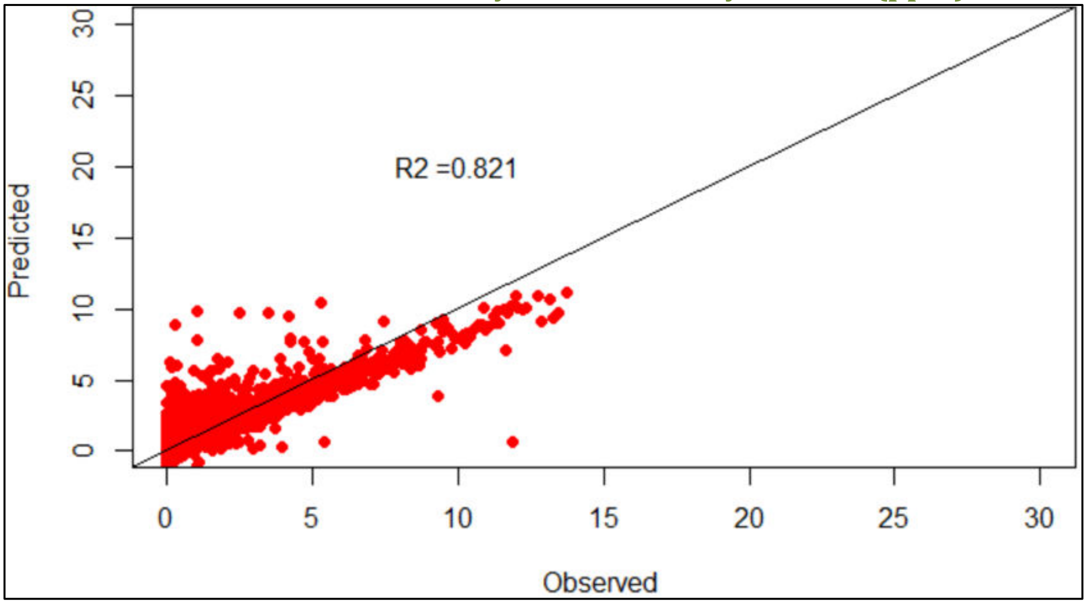


Figure 10-14 Observed and Predicted Cyanide Solubility for Gold (ppm)



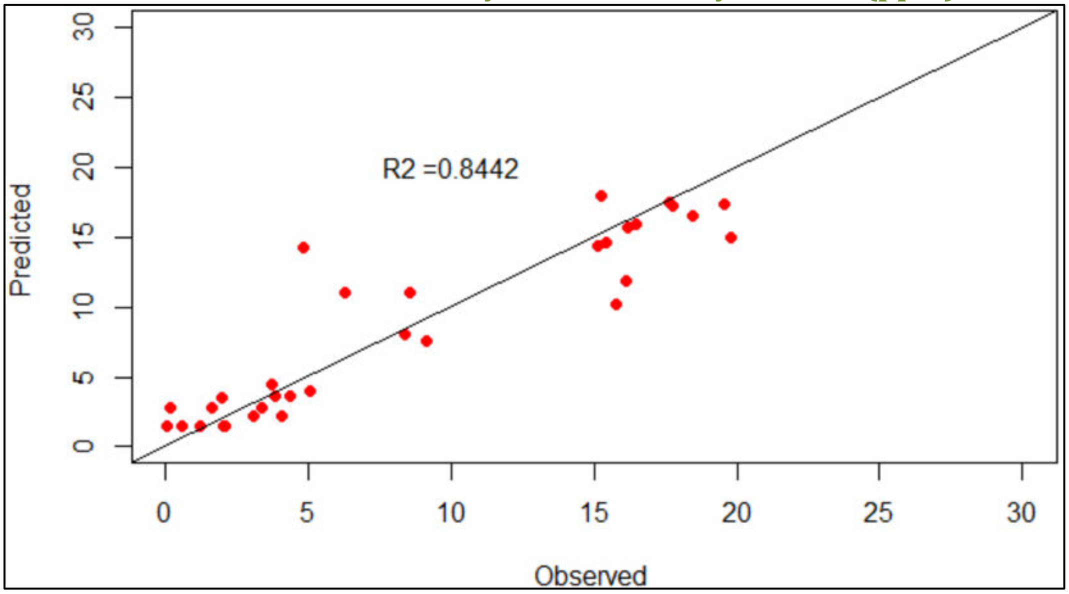
2. Au >=15 ppm

The gold cyanide solubility when gold grade is greater than 15 ppm is given by:

$$AuCN\,ppm = 1.463 + 0.1723 * max(0, Au\,ppm - 20.74) + 0.01517 * max(0, Elevation(ft) - 3934)$$

The cyanide solubility equation uses gold grade and elevation in the model. The graph showing the observed and predicted gold cyanide solubility is given in Figure 10-15. The model has an R^2 of 0.84, implying that 84% of the variations are explained by the model.

Figure 10-15 Observed and Predicted Cyanide Solubility for Gold (ppm)



10.5.1.2 C and CX Pit Zone

There were 5,656 samples with cyanide solubility information and Au grade above 0.1 ppm. Figure 10-16 shows the scree plot, where the first component comprises of the maximum variance and second component has relatively higher variance. Figure 10-17 shows the biplot, where gold, pyrite, and iron oxides have a weak positive correlation and weak negative correlation to elevation. Figure shows the regression tree model, which shows that gold cyanide solubility is highly dependent on only gold grade.

Figure 10-16 PCA- Scree Plot for C and CX Pit

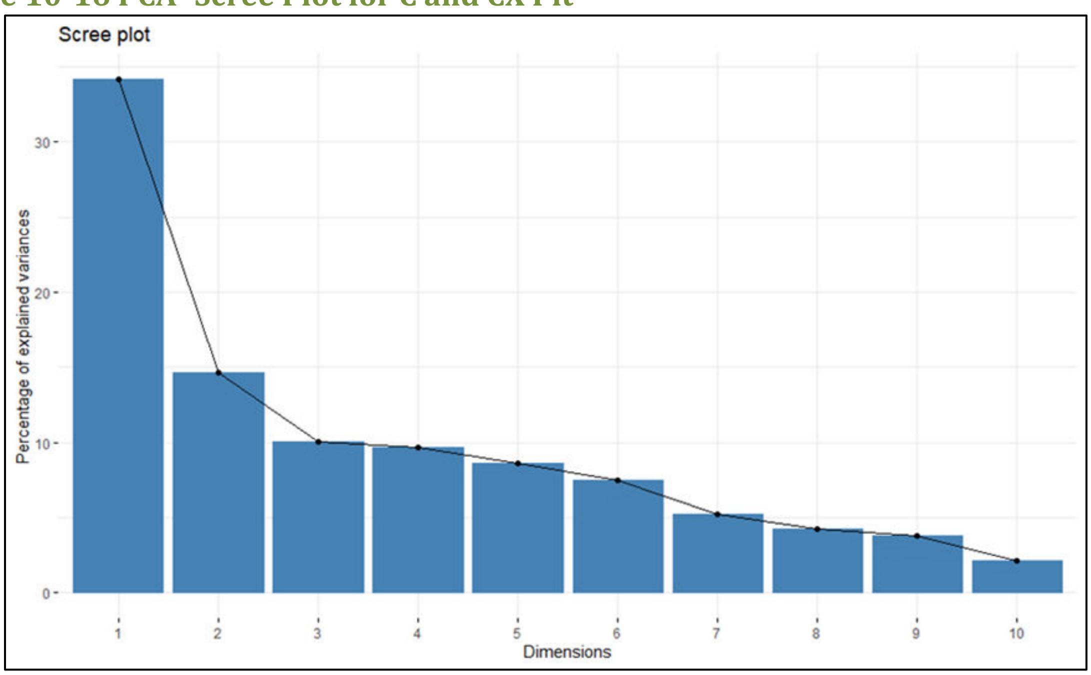


Figure 10-17: PCA – Biplot for C and CX Pit

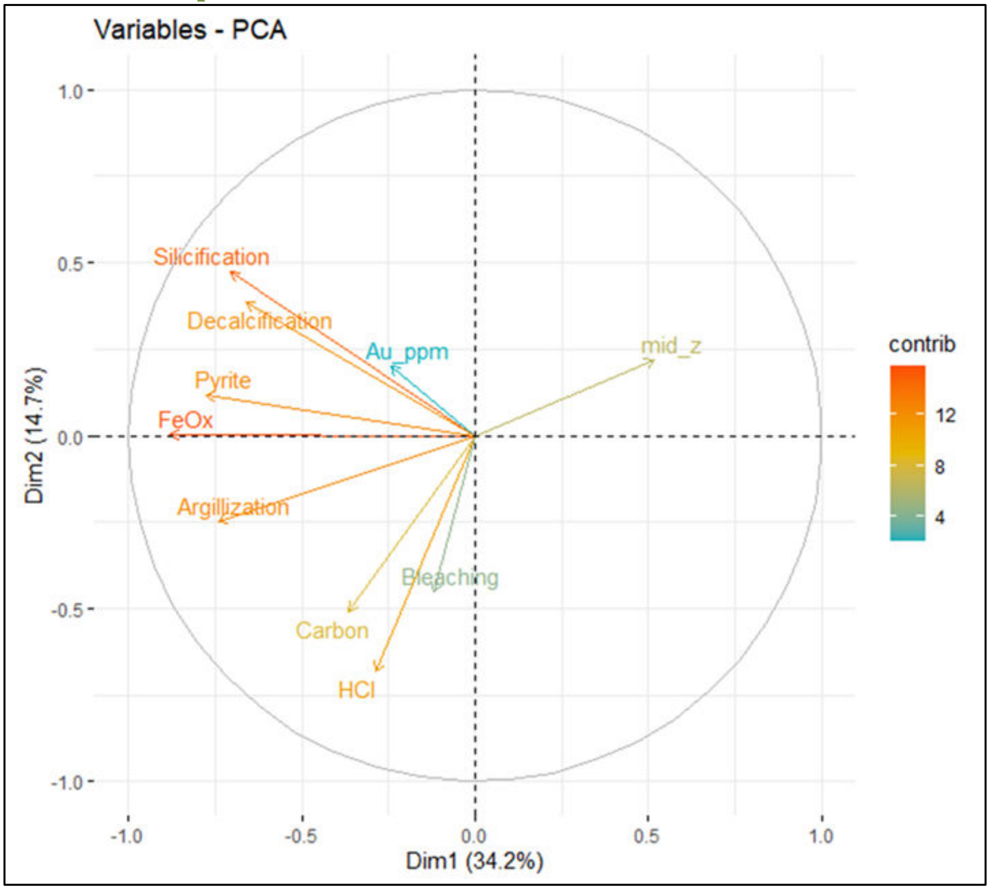
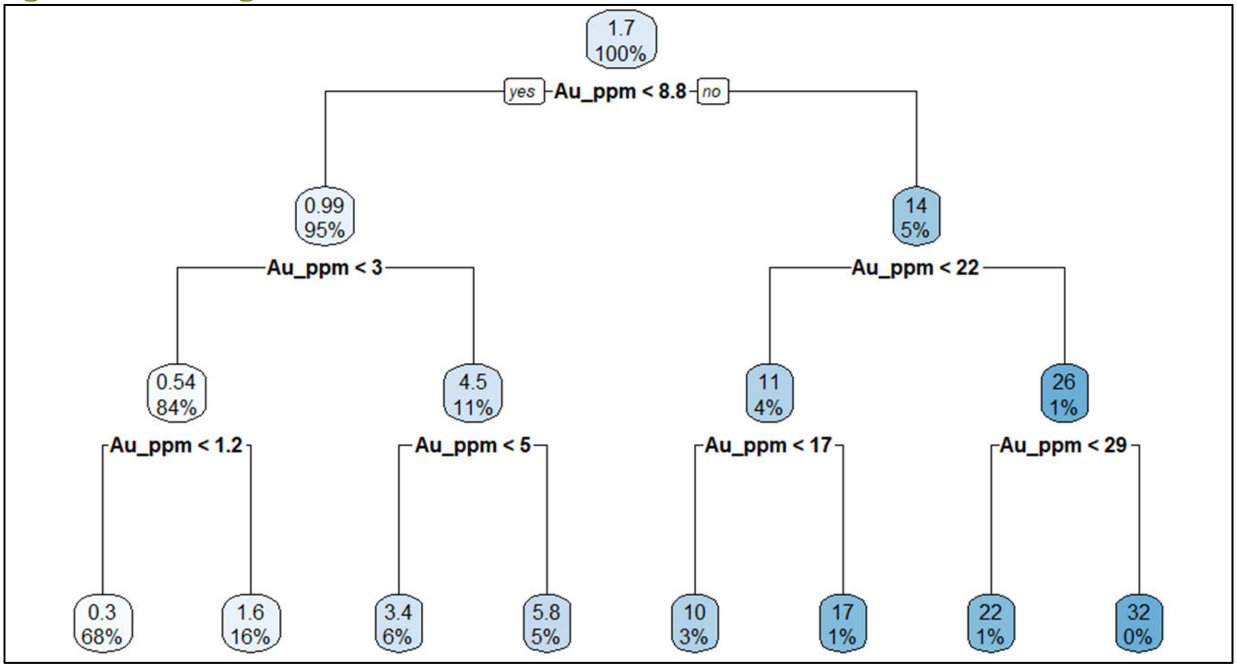


Figure 10-18: Regression Tree Model for C and CX Pit

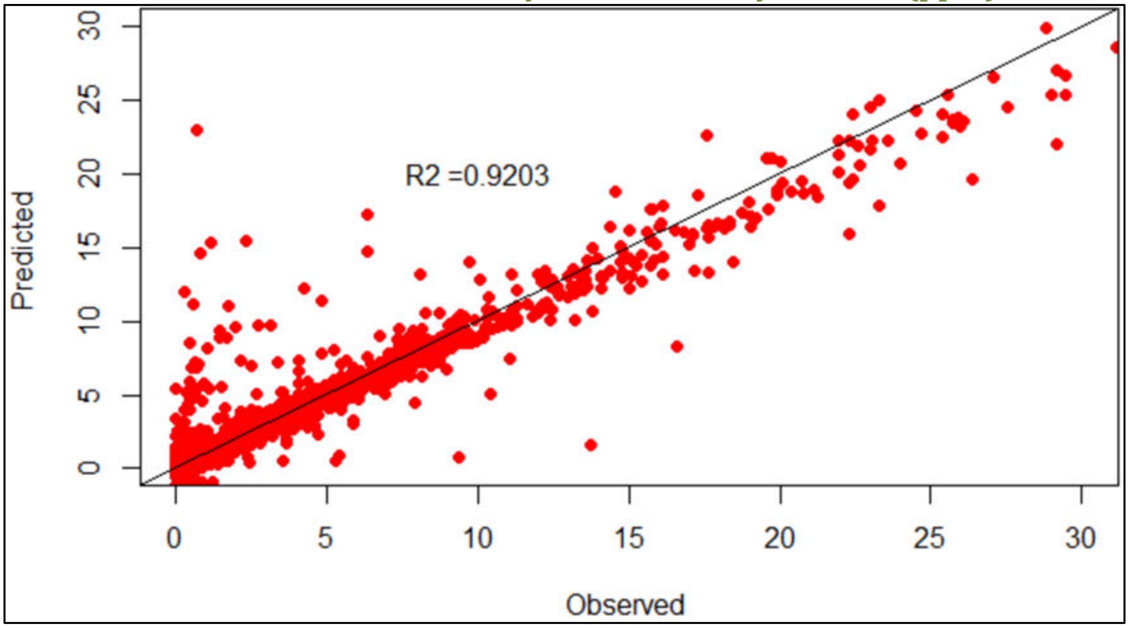


The MARS model for the gold cyanide solubility in C and Cx pit area is shown as:

$$AuCN\,ppm = 23.87 - 0.8948 * max(0, 26.91 - Au\,ppm) + 0.6038 * max(0, Au\,ppm - 26.91) + 0.2289 * max(0, 1 - Pyrite) - 1.748 * max(0, Pyrite - 1) - 0.00116 * max(0, 4308 - Elevation(ft))$$

The cyanide solubility equation uses gold grade, pyrite alteration, and elevation in the model. The graph showing the observed and predicted gold cyanide solubility is given in Figure . The model has an R^2 of 0.92, implying that the 92% of the variations are explained by the model.

Figure 10-19: Observed and Predicted Cyanide Solubility for Gold (ppm)



10.5.1.3 A Pit Zone

There were 487 samples with cyanide solubility information and Au grade above 0.1 ppm. Figure 10-20 shows the scree plot, where the first component comprises the maximum variance. Figure 10-21: PCA – Biplot for A Pit shows the biplot, where gold and elevation have a positive correlation. Figure the regression tree model shows that gold cyanide solubility is highly dependent on only gold grade.

The MARS model for the gold cyanide solubility in A pit area is shown as:

$$AuCN\,ppm = 1.365 - 0.9794 * max(0, 1.44 - Au\,ppm) + 1.005 * max(0, Au\,ppm - 1.44)$$

The cyanide solubility equation uses only gold grade in the model. The graph showing the observed and predicted gold cyanide solubility is given in Figure 10-23 The model has an R^2 of 0.999, implying that the 99.9% of the variations are explained by the model.

Figure 10-20: PCA- Scree Plot for A Pit

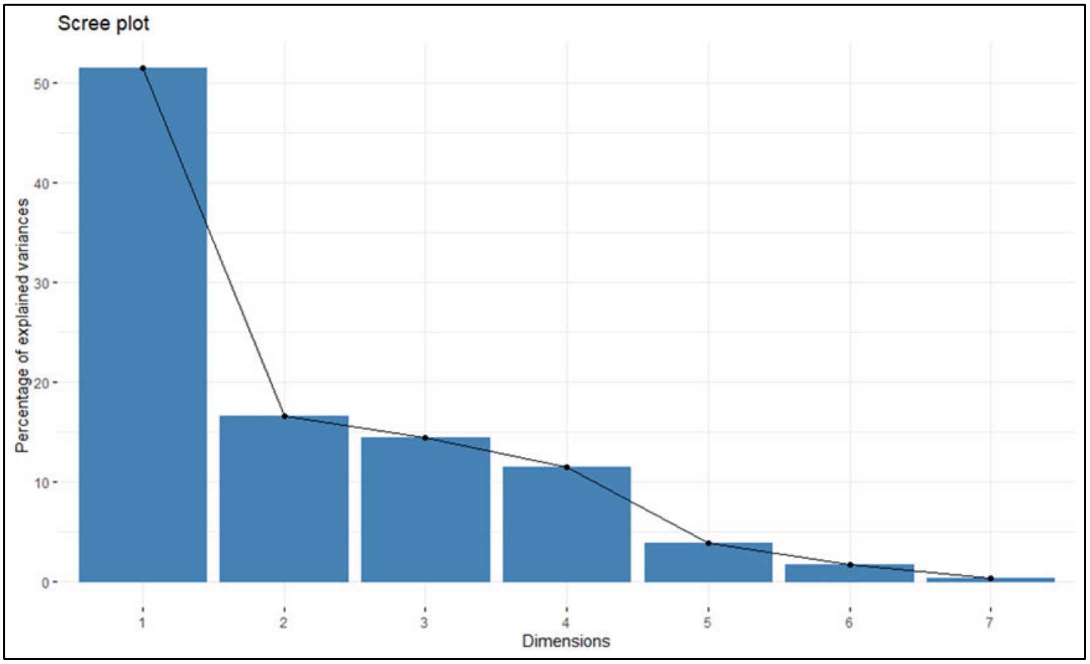


Figure 10-21: PCA – Biplot for A Pit

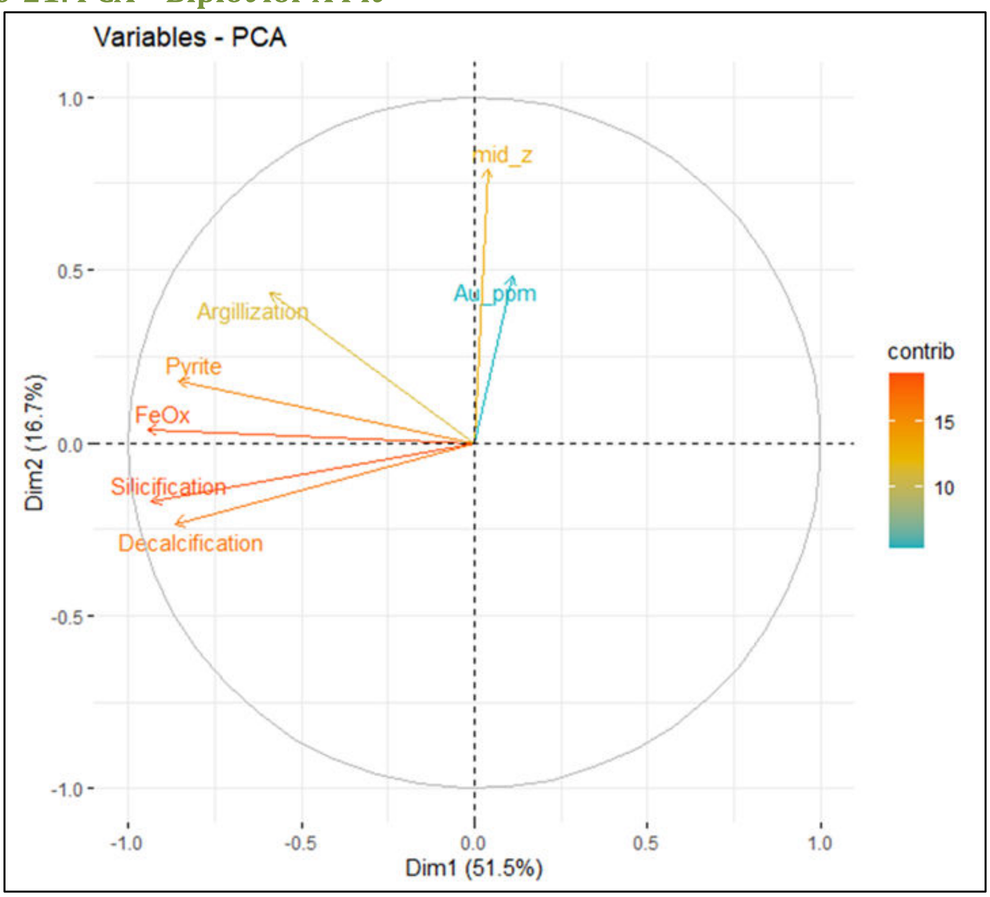


Figure 10-22: Regression Tree Model for A Pit

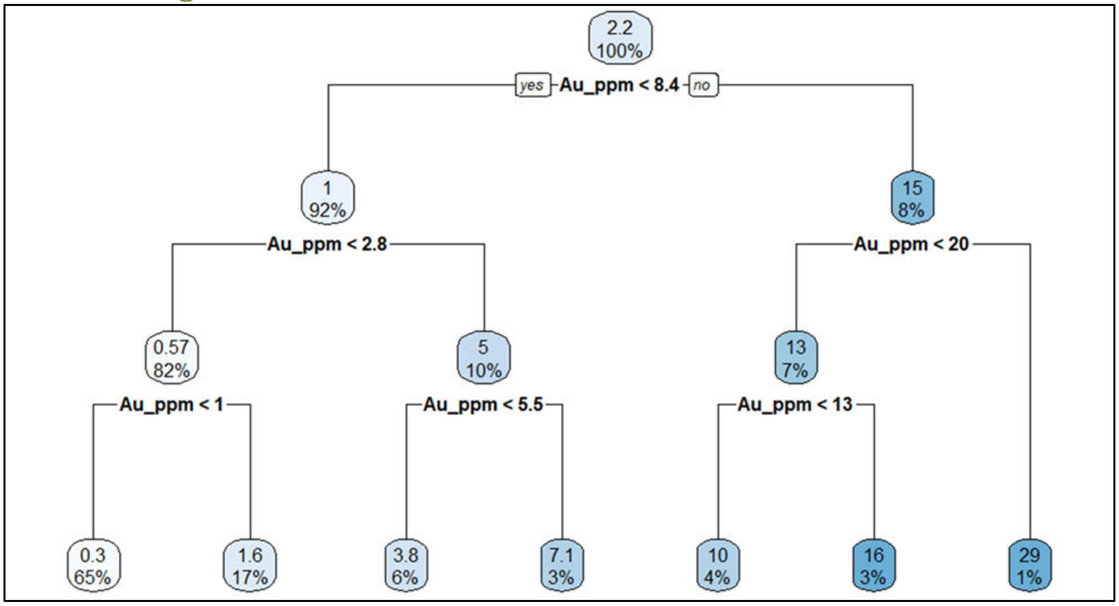
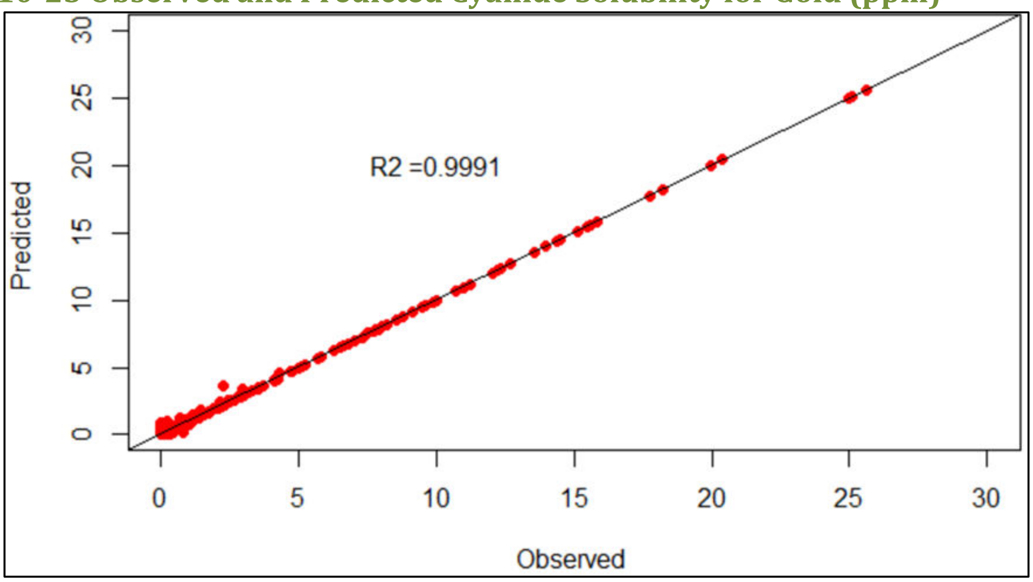


Figure 10-23 Observed and Predicted Cyanide Solubility for Gold (ppm)



10.5.1.4 B Pit Zone

There were 617 samples with cyanide solubility information and Au grade above 0.1 ppm Figure 10-24 shows the scree plot, where the first component comprises of the maximum variance. Figure 10-25 shows the biplot, where gold and elevation have a positive correlation. Figure 10-26 shows the regression tree model, which shows that gold cyanide solubility is highly dependent on gold grade and pyrite alteration.

The MARS model for the gold cyanide solubility in B pit area is shown as:

$$AuCN\ ppm = 3.002 - 0.6082 * max(0, 4.121 - Au\ ppm) + 1.071 * max(0, Au\ ppm - 4.121) - 0.4479 * max(0, 1 - FeOx) - 0.4769 * max(0, Pyrite - 1)$$

Figure 10-24 PCA- Scree Plot for B Pit

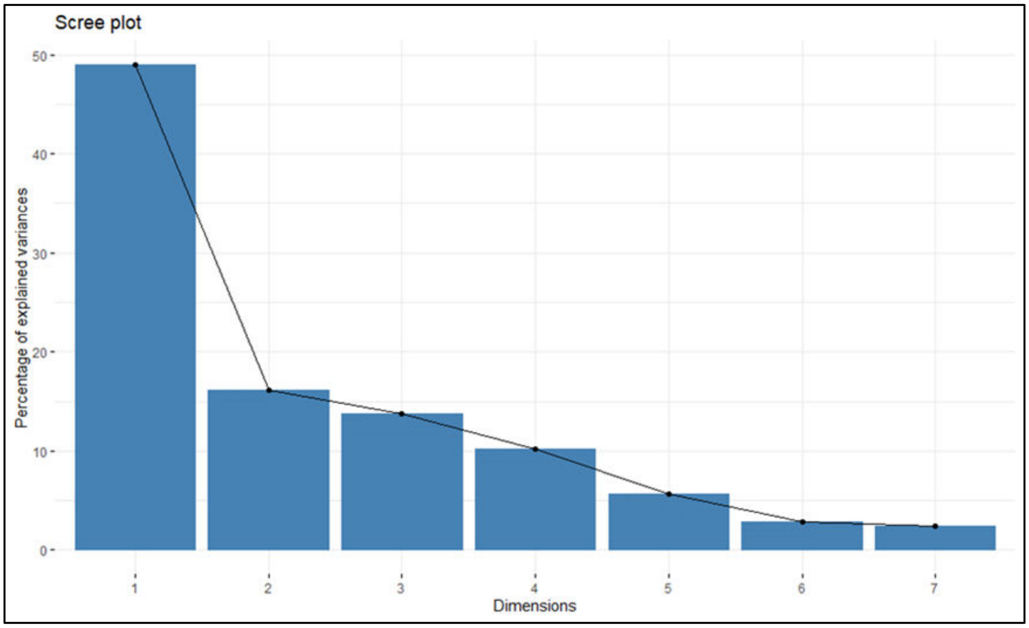


Figure 10-25: PCA – Biplot for B Pit

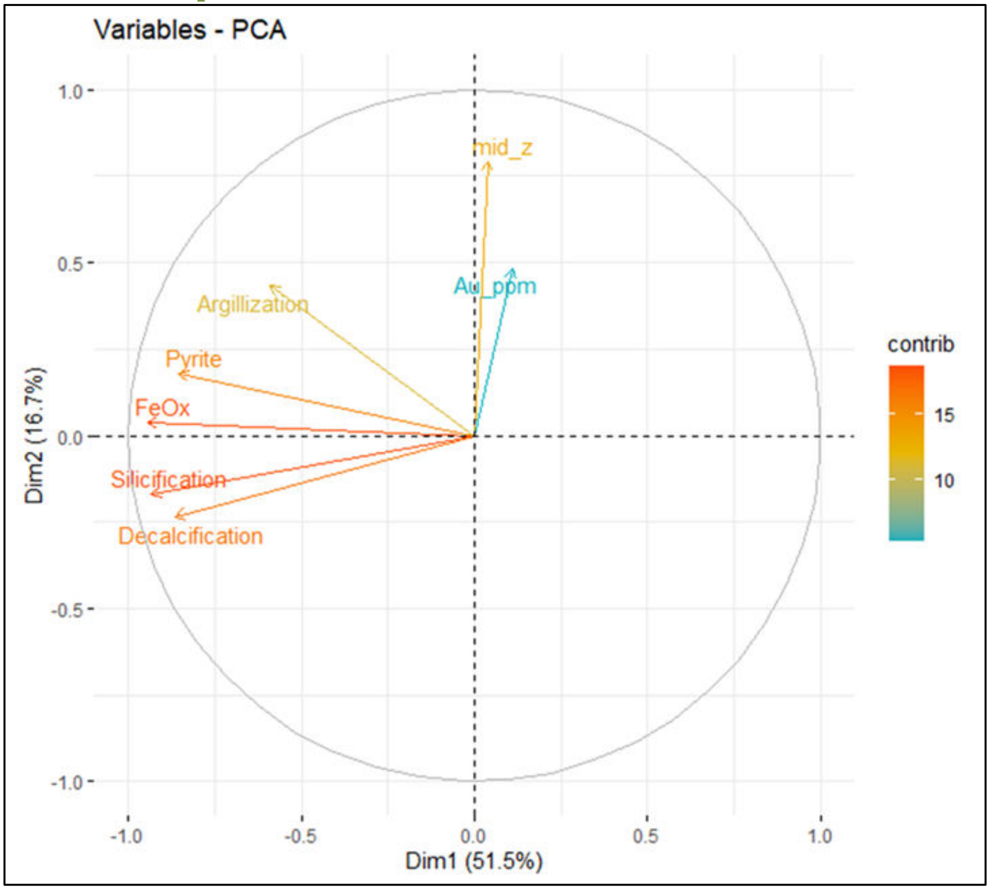
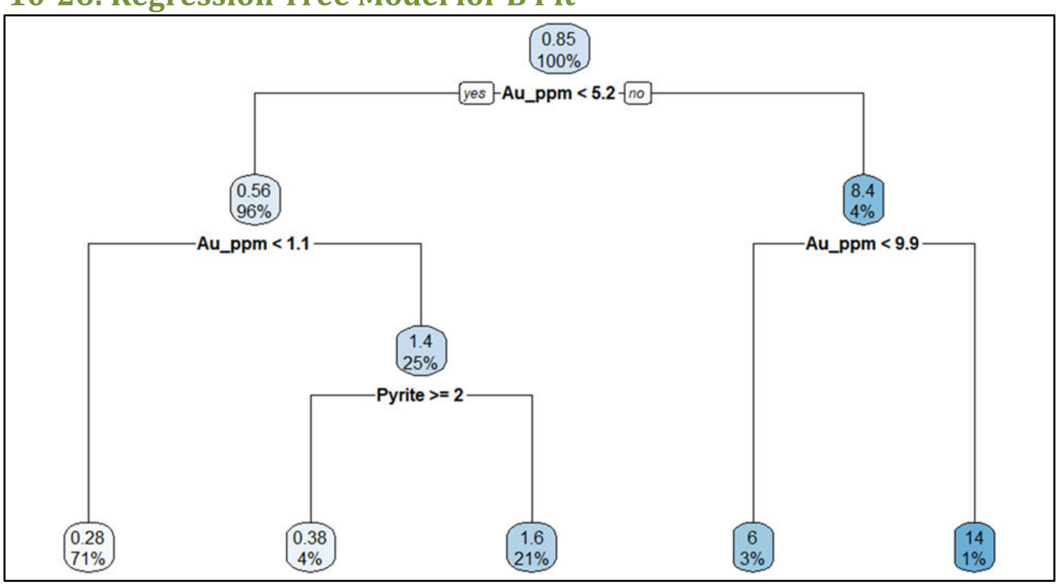
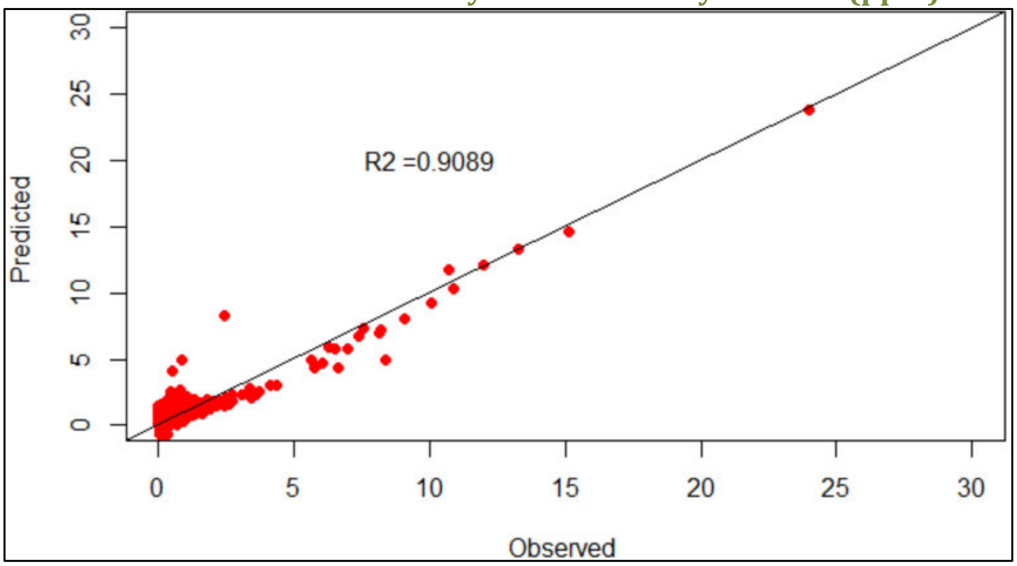


Figure 10-26: Regression Tree Model for B Pit



The cyanide solubility equation uses gold grade, pyrite alteration, and iron oxide alteration in the model. The graph showing the observed and predicted gold cyanide solubility is given in Figure 10-27. The model has an R^2 of 0.91, implying that 91% of the variations are explained by the model.

Figure 10-27: Observed and Predicted Cyanide Solubility for Gold (ppm)



10.5.2 Cyanide Solubility Estimation in the Block Model

Using the drill hole information in each pit and the underground zone, the gold grades were estimated, and alteration information was coded for all the blocks. All the blocks are associated with the elevation, and the elevation for the block center is used for cyanide solubility estimation. Using the MARS model for each domain, the cyanide solubility information was calculated. This information was later used to calculate the recovery for each block.

10.5.3 Metallurgical Test and Recovery

Column test and the CIL test information are available in the McClelland April 1999 report (McClelland, 1999a). The test samples are primarily located in the Mag pit area and a single sample in the C and CX pit area. Autoclave results are available in the DML Wilmot 2005 -2006 Memo (Wilmot, 2006).

There are 16 Column leach tests available and seven CIL tests. The number of samples and the locations of the samples are not spatially representative of the deposit.

Table 10-31: Column Test and CIL Test (McClelland April 1999 Report)

Sample Composite	Sample Type	Column Feed Size	Column Au Recovery Percent	NaCN lb/ton	Lime lb/ton	NaOH	C organic Percent	CIL Au Percent	Calc Head opt	CN Solubility %
Mag Pit I	Bulk High-Grade Material	-4 inch	18.8%	9.94	5.2	2.8	4.55%	94.4%	0.138	19.5%
Mag Pit II		-4 inch	35.3%	8.96	10.2	2.34	4.25%	75.3%	0.085	19.6%
Mag Pit III		-4 inch	93.1%	4.55	5.2		0.15%	59.7%	0.058	84.5%
Mag Pit IV		-4 inch	49.5%	5.30	12.0		4.00%	82.9%	0.105	22.8%
Mag Pit V		-4 inch	51.7%	3.85	2.5		0.40%	55.0%	0.029	50.0%
Mag Pit VI		-4 inch	60.7%	3.66	4.0		0.45%	87.5%	0.028	61.8%
CX Pit, CX-2		-6 inch	77.7%	5.11	3.0		0.35%	88.2%	0.091	91.9%
CX Pit, CX-2		P80 3 inch	81.7%	4.80	3.0				0.089	91.9%
CX Pit, CX-2		P80 3/4 inch	82.2%	5.39	3.0				0.090	91.9%
Mag Pit 2	Core Comp	Nom 1 inch	69.0%	3.98	11.0				0.058	77.1%
Mag Pit 3	Core Comp	Nom 1 inch	62.0%	1.60	9.6				0.079	44.2%
Mag Pit 4	Core Comp	Nom 1 inch	47.9%	1.51	8.1				0.048	72.0%
Mag Pit 5	Core Comp	Nom 1 inch	61.7%	2.08	10.0				0.141	22.6%
Mag Pit Master (pH 10.5)	Core Comp	Nom 1 inch	65.0%	6.26	8.4		2.25%		0.080	49.4%

Mag Pit Master (pH 11.8, Lime)	Core Comp	Nom 1 inch	70.7%	4.15	19.3		2.25%		0.082	49.4%
Mag Pit Master (pH 12.0, NaOH)	Core Comp	Nom 1 inch	69.0%	3.48	N/A	10.7	2.25%		0.084	49.4%

Assumption: The recovery properties observed in the Mag pit are similar to the entire area (Mag Pit, C and Cx pit, A Pit, B Pit, and Underground Zone).

10.5.3.1 Recovery Model

Recovery models are based on Table 10-31. The recovery models are based on the actual metallurgical tests. The Heap Leach Recovery model is used for both the open pit areas (Mag pit, C and Cx pit, A pit, and B pit) and the Underground area. CIL recovery model is used only in the open-pit area. The autoclave is used only for the underground area, except for any oxide material encountered above the 4670-foot level. The material above the 4670-foot level in the underground area can be treated in the heap leach or autoclave depending on the revenue generated by the blocks in the different processing options.

10.5.3.1.1. Heap Leach Recovery

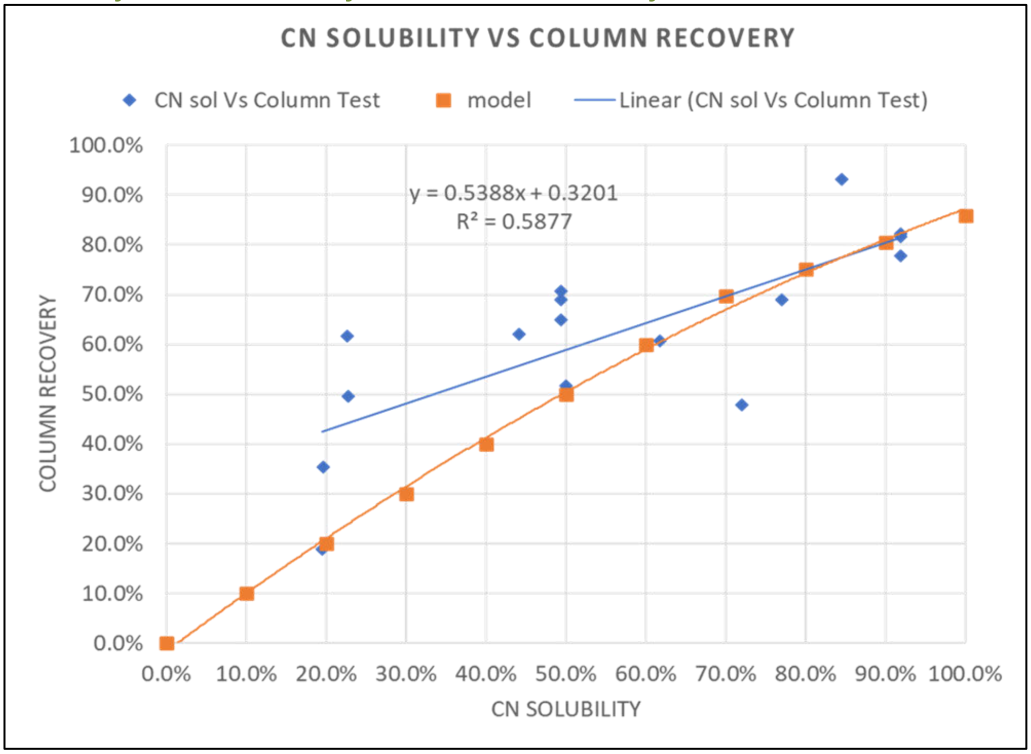
Heap Leach recovery is determined by plotting the cyanide solubility with the column recovery. Figure 10-28 shows that a few samples have lower cyanide solubility but have variable column recovery, ranging from 20% to 70%.

A few samples show that the model has 20% column recovery at 20% cyanide solubility, 50% Column Recovery at 50% cyanide solubility, and 60% Column Recovery at 60% cyanide solubility. Looking at the trend, a conservative model is created, where up to 60% cyanide solubility, the heap leach recovery is 60%, and above 60% cyanide solubility, the model follows the trend on the regression line. $Heap\ Leach\ Recovery\ =$

$$\begin{cases} CN\ Solubility\ , CN\ Solubility < 0.6 \\ (0.5388 * CN\ Solubility) + 0.3201, CN\ Solubility \geq 0.6 \end{cases}$$

Where $CN\ Solubility = \ AuCN\ ppm\ (from\ MARS\ model) /\ Au\ ppm$

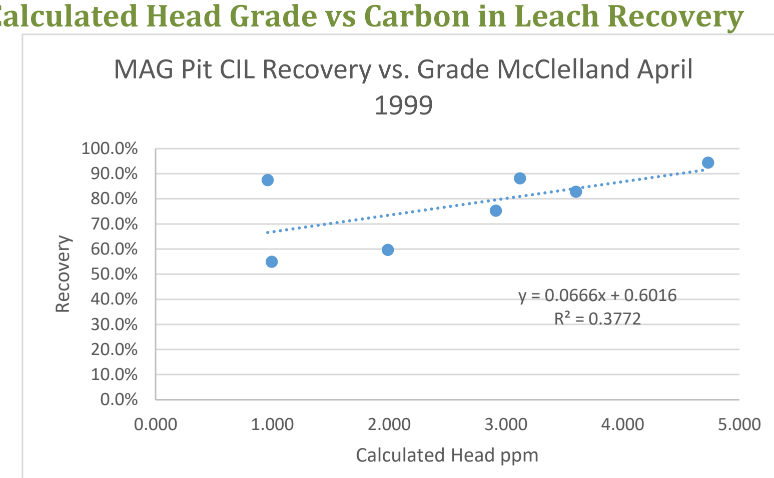
Figure 10-28: Cyanide Solubility vs Column Recovery



10.5.3.1.2. Carbon in Leach Recovery

Within the Mag Pit, the samples were looked at, and the trend between the calculated head grade and CIL Recovery was plotted (Figure 10-29).

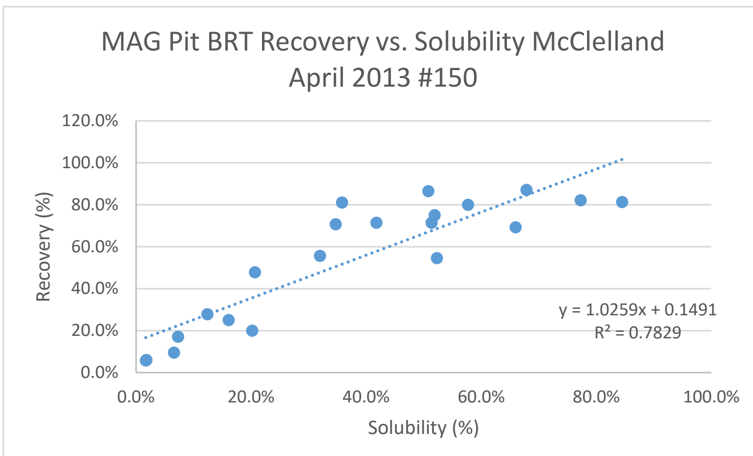
Figure 10-29: Calculated Head Grade vs Carbon in Leach Recovery



Comparing the CIL results to column leach tests of the same samples revealed that several outliers needed to be addressed. In one case, a column test showed a gold extraction of 93% while the CIL test provided only 55% gold recovery, two additional samples were removed for similar reasons. To create a more realistic model, the outliers were removed as shown in Figure 10-30. Multiple regression analysis did not find a statistical relationship between CIL recovery and organic carbon or sulfide sulfur. In most cases, the sulfide sulfur grade of the samples was low.

The number of CIL tests available was low, but significant direct leach bottle roll tests (BRT) informed the selection of the CIL results. Gold extraction from BRT showed a strong relationship to the organic carbon concentrations and a weaker relationship to sulfide sulfur. The sulfide sulfur relationship was likely impacted by the low sulfide grade variability examined. CIL testing indicated that the preg-robbing impact of the TOC was largely overcome in most cases.

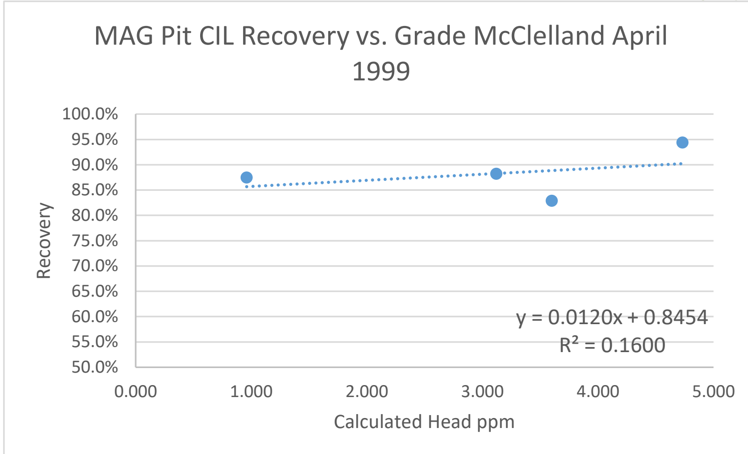
Figure 10-30: Solubility vs BRT Recovery



By parsing the data Figure 10-31 a reasonable trend between the head grade and recovery can be observed. This trend is used to predict CIL recovery, which is capped at 95%.

$$CIL\ Recovery\ = (0.012\ *\ Au\ grade\ (ppm))\ + 0.8454$$

Figure 10-31 Calculated Head Grade vs Carbon in Leach Recovery (Outlier Removed)



It is important to note that no limit on sulfide sulfur has been included in this recovery formula but it likely would play a role. Additional test work is required to better define this relationship and careful ore control measures will need to be adopted to ensure that refractory material is not directed to the CIL.

10.5.3.1.3. Autoclave Recovery

For the autoclave recovery tests, 25 samples were available (Table 10-32, through Table 10-34. The majority of the tests had a high recovery, ranging from 81% to 97%. One sample had a low recovery (OGLG) and the reason for this was not determined.

Table 10-32: DML Wilmot 2005 -2006, Memo Autoclave Test Results (Samples from 2005)

October 2005	One hour 225 C, 460 psi, Acidulate to 1.8 to 2.0 pH Autoclave Discharge, 2 gm/l NaCN 24 hr										
Description	Test #	Sample ID	Grind P80 (microns)	Head Assay (ppm)	Head Calc (ppm)	Leach Residue (ppm)	Gold Extracted (ppm)	Gold Recovery (%)	Consumption CaO (Kg/Mt)	Consumption NaCN (Kg/Mt)	CN Shake Test
Ogee channel sample, oxide	1	LL&RR	75	13.71	12.22	0.81	11.4	93.2%	4.2	1.32	86%
Range Front Sample, surface drilling	4	34259	75	14.74	14.8	1.49	13.1	88.5%	32.1	4.72	11%
Range Front Sample, surface drilling	2	33941	75	8.23	9.01	0.58	8.4	93.0%	19.6	1.56	53%
Range Front Sample, surface drilling	3	33942	75	14.74	15.63	0.67	14.9	95.2%	47.2	2.37	61%
Range Front Sample, surface drilling	5	33942	45	14.74	15.57	0.41	15.1	97.0%	47.7	2.44	61%

Table 10-33: DML Wilmot 2005 -2006, Memo Autoclave Test Results (Samples from 2006)

April 14, 2006	One hour 225 C, 460 psi, Acidulate to 1.8 to 2.0 pH Autoclave Discharge, 2 gm/l NaCN 24 hr										
Description	Test #	Sample ID	Grind P80 (microns)	Head Assay (ppm)	Head Calc (ppm)	Leach Residue (ppm)	Gold Extracted (ppm)	Gold Recovery (%)	Consumption CaO (Kg/Mt)	Consumption NaCN (Kg/Mt)	CN Shake Test
CX Sample composites	8	APCX-219	75	10.25	8.35	0.72	7.6	91.0%	17.5	1.2	60%

CX Sample composites	9	APCX-226	75	17.5	15.81	0.89	14.84	93.8%	21.2	2.38	42%
From Barrick pre-ground	6	MET 1	88	n/a	15.96	0.96	14.8	92.7%	71.4	1.36	n/a
From Barrick pre-ground	7	MET 2	88	n/a	10.06	2.06	7.81	77.6%	23.4	0.88	n/a
Coarse sample -3/8	2	MET 1	75	8.23	9.01	0.58	8.38	93.0%	19.6	1.56	n/a
Coarse sample -3/8	3	MET 2	75	14.74	15.63	0.67	14.88	95.2%	47.2	2.37	n/a

Table 10-34: Autoclave Test Results (Samples from 2023)

	Condition A - Acid			Condition E - Partial Acid		
Sample ID	S= Oxidation (%)	CIL Recovery		S= Oxidation (%)	CIL Recovery	
		Au (%)	Ag (%)		Au (%)	Ag (%)
OGU	97	91	13	47	82	84
OGL	57	94	0	62	94	0
OGHG	91	81	15	91	94	1
OGLG	100	70	0	61	76	53
OTU	100	88	42	99	90	0
OTL	99	87	27	98	90	77
OTHG	99	92	45	74	87	62
OTLG	100	88	87	100	90	0
APU	98	95	1	95	95	0
APL	98	87	3	97	88	20
APHG	100	91	0	32	42	0
APLG	100	90	2	45	55	0
OAPC	99	83	19	69	82	16
DRFV	96	89	0	36	41	30
RFV	97	95	0	43	63	0
SPZV	99	96	0	99	94	0

10.5.4 Recovery in the Block Model

The recovery in the block model uses the estimated cyanide solubility information (Section 10.5.2). For each block, the recovery equations from Section 10.5.3 are used to calculate the recovery in different processing destinations.

10.6 Conclusions

10.6.1 Sample Representativity

Within each zone, drilling has been localized to relatively small portions of the mineralized domains, as seen in Figure 10-9 and Figure 10-10. The samples' metallurgical response is likely to represent the zone's general behavior, but additional sampling of each zone to confirm the metallurgical response will reduce uncertainty. The lack of this metallurgical drilling remains a risk to the project.

10.6.2 Test Work on Open Pit Samples

Cyanide leach bottle roll tests and column leach tests were completed on samples from both the Mag and CX open pits. Both Homestead and Atna commissioned these tests.

The test work demonstrated that many of the Mag Pit samples had high preg-robbing factors due to carbonaceous material in the feed. Due to the variable preg-robbing characteristics of the feed material, a higher degree of representativity of the Mag Pit should be evaluated.

Bottle roll tests were conducted on Mag Pit samples using NaOH as an alternative to hydrated lime, as a method of treating material with preg-robbing characteristics. These tests demonstrated that raising the pH improved gold recovery and decreased cyanide consumption.

A column leach test on a Mag Pit sample showed that there was no gold recovery benefit in using NaOH rather than lime (at the equivalent pH).

Test work on ground materials showed that Mag Pit materials were amenable to CIL methods. CIL treatment showed low impact from the TOC. Gold recoveries ranged from 83% to 94%.

Column leach tests on the Mag Pit samples achieved gold recoveries in the range of 19% to 82%.

Column leach tests on the CX Pit samples achieved gold recoveries of 82%.

10.7 Recommendations

The following recommendations have been put forward:

10.7.1 Test Work Recommendations

A metallurgical drilling program should be undertaken to collect samples within the various zones representing the spatial, mineralogical, and grade differences. The collected samples should be tested for the following:

- Paired fire assays and cyanide soluble assays to define cyanide solubility.
- Bottle roll tests with and without carbon to predict reagent consumption as well as amenability to CIL treatment and to evaluate the impact of sulfide sulfur on the CIL performance.
- Column leach tests at various sizes to predict field recovery for material to be heap leached. This should be performed on those materials with a cyanide solubility of greater than 50%. Recovery by size fraction should be completed as part of the testing program.
- Conduct SAG and ball mill testing to determine the work index.
- Additional autoclave pretreatment of underground materials should be completed, especially for those materials that showed lower gold extraction.
- Infill the drill hole database with TOC and S= assays.
- Conduct arsenic and mercury assays on all samples employed for metallurgical testing.

10.7.2 Geometallurgy Recommendations

The geometallurgical work completed as part of this technical report should be expanded using the planned metallurgical test program results. The intent will be to confidently define those materials that can be treated by heap leaching or CIL methods and those that require autoclave treatment.

11 Mineral Resource Estimates

11.1 Introduction

Securities and Exchange Commission (SEC) S-K regulations (Title 17, Part 229, Items 601 and 1300 through 1305 provides the following definitions for mineral resources:

Mineral resource is a concentration or occurrence of material of economic interest in or on the Earth's crust in such form, grade or quality, and quantity that there are reasonable prospects for economic extraction. A mineral resource is a reasonable estimate of mineralization, taking into account relevant factors such as cut-off grade, likely mining dimensions, location or continuity, that, with the assumed and justifiable technical and economic conditions, is likely to, in whole or in part, become economically extractable. It is not merely an inventory of all mineralization drilled or sampled.

Inferred mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. The level of geological uncertainty associated with an inferred mineral resource is too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability. Because an inferred mineral resource has the lowest level of geological confidence of all mineral resources, which prevents the application of the modifying factors in a manner useful for evaluation of economic viability, an inferred mineral resource may not be considered when assessing the economic viability of a mining project, and may not be converted to a mineral reserve.

Indicated mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of adequate geological evidence and sampling. The level of geological certainty associated with an indicated mineral resource is sufficient to allow a qualified person to apply modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Because an indicated mineral resource has a lower level of confidence than the level of confidence of a measured mineral resource, an indicated mineral resource may only be converted to a probable mineral reserve.

Measured mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of conclusive geological evidence and sampling. The level of geological certainty associated with a measured mineral resource is sufficient to allow a qualified person to apply modifying factors, as defined in this section, in sufficient detail to support detailed mine planning and final evaluation of the economic viability of the deposit. Because a measured mineral resource has a higher level of confidence than the level of confidence of either an indicated mineral resource or an inferred mineral resource, a measured mineral resource may be converted to a proven mineral reserve or to a probable mineral reserve. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into Mineral Reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

The Mineral Resource Statement presented herein represents the updated Mineral Resource Estimate for the Granite Creek deposit located in Humboldt County, Nevada, USA. This Mineral Resource Estimate was prepared by GRE for i-80 to complete a S-K 1300 Technical Report Summary. The most recent previous Mineral Resource Estimates were contained in the reports titled:

- "Preliminary Economic Assessment NI 43-101 Technical Report, Granite Creek Mine Project, Humbolt County, Nevada, USA" (GRE, 2021)
- "Technical Report, Osgood Pinson Deposit NI 43-101 Technical Report, Osgood Mining Company, LLC, Humboldt County, Nevada, USA" produced by Osgood Mining Company, LLC (AMC, 2019).

The mineral resources were estimated in conformity with and are reported in accordance with the S-K 1300 requirements. This mineral resource estimate includes inferred mineral resources, which are defined as "that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. The level of geological uncertainty associated with an inferred mineral resource is too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability…" (CFR, 2018). There is no certainty that the inferred mineral resources will be converted to the measured or indicated categories through further drilling or into mineral reserves, once economic considerations are applied. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The project presently has no mineral reserves. Whittle Pit optimization was applied to the open pit mineral resource estimate to assess the reasonable prospects for economic extraction for the resource.

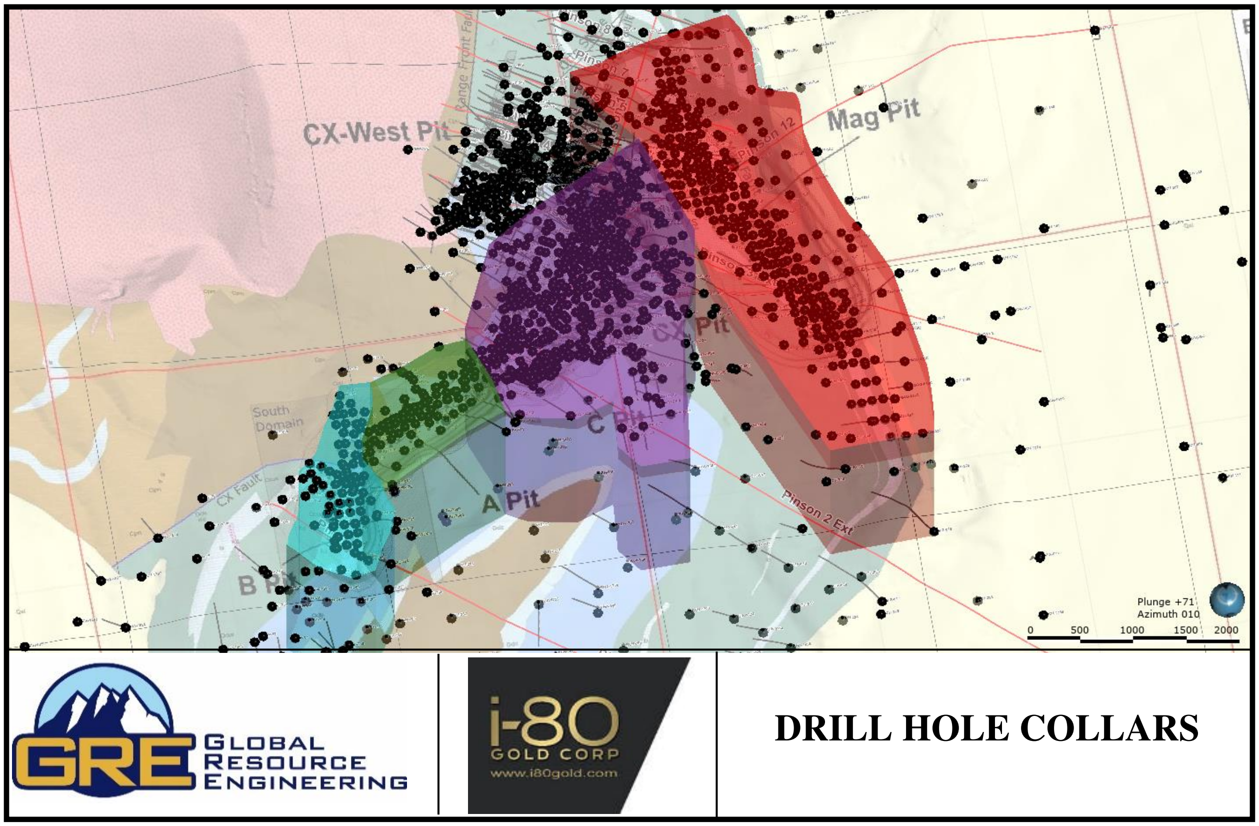
The open pit Mineral Resource Estimate for the Granite Creek Mine Project was completed by GRE. The effective date of the resource statement is December 31, 2024. In the opinion of GRE, the Mineral Resource Estimate reported here is a reasonable representation of the mineral resources found in the open pit portion of the Granite Creek Mine Project at the current level of sampling.

11.2 Drill Hole Database

GRE performed a data validation of the drill hole database prepared by i-80 for the Granite Creek deposit and determined it to be of suitable accuracy to perform a mineral resource estimate for the property. More detail regarding the validation of the drill hole database can be found in Section 9. The drill hole data for the Granite Creek Mine Project was delivered as a separate .csv file that contained exploration and production collar locations, drill hole survey orientations, sample intervals with gold assays in ppm, geologic intervals with lithology, alteration type, and alteration strength. The collar locations are projected in a local grid system, with planar and elevation units in feet. All downhole intervals are captured in feet.

The complete data set contained assays, collar, and survey data for a total of 2,855 exploration holes (surface, underground, and trench samples) and 695 production holes (surface and underground). Drilling is a mix of RC drilling, diamond drilling, and RC pre-collar with diamond drilling to final depth. The exploration assay file contains 212,839 gold assays. The production data assay file contains 1,477 gold assays. The drill hole collar locations are shown in Figure 11-1.

Figure 11-1: Drill Holes Used Plan View on Topography



Note: This figure is intended to show the relative distribution of surface drill hole collars on topography around the areas of interest. This figure does not show all collars that have been drilled.

A number of negative, missing, and blank assay values exist in the drill hole data files provided to GRE by i-80. Missing intervals and values were assumed to be non-mineralized and therefore assigned a value of half of the most common detection limit used to assay the samples. Negative assay values were replaced according to Table 11-1.

Table 11-1: Negative Values in Drill Hole Database

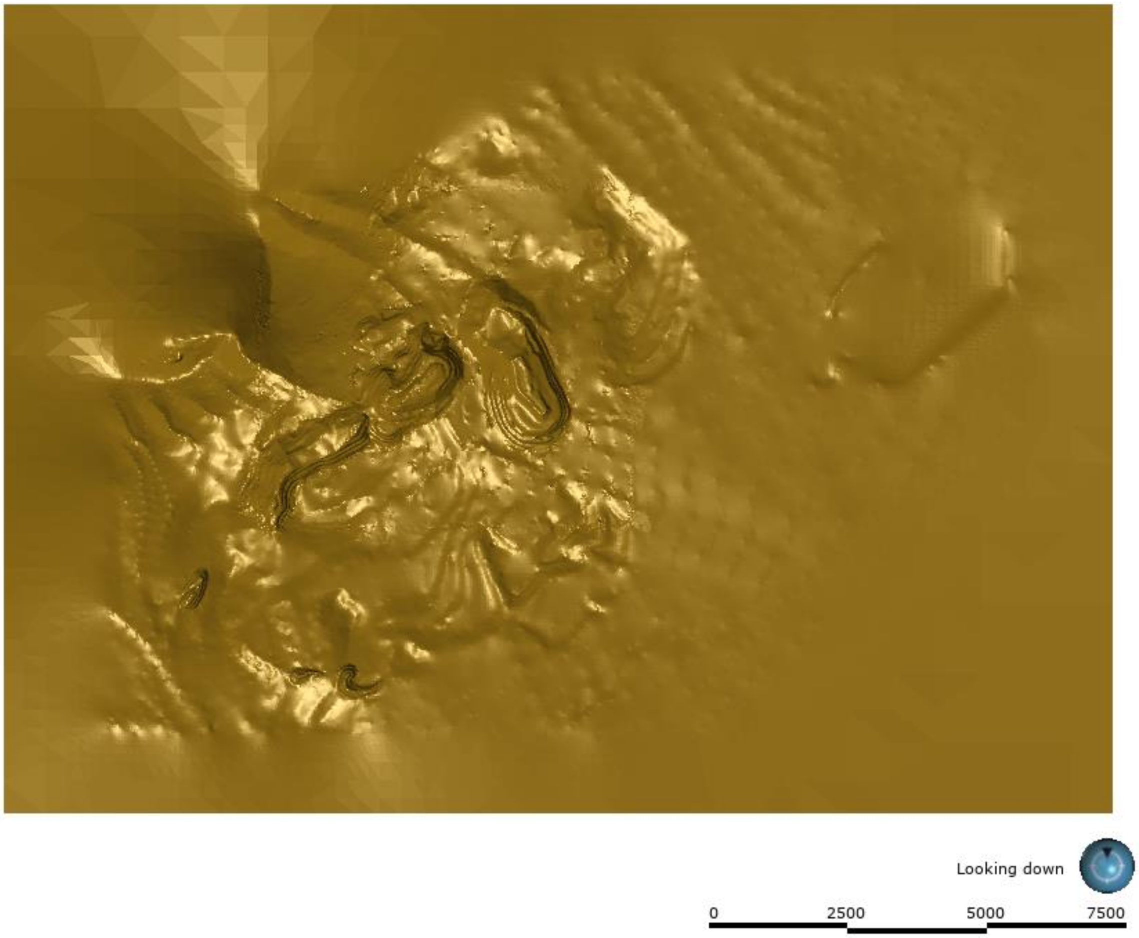
Non-Positive Value	Interpretation	Count In Data Set	Action Taken
-0.005	below detection limit of 0.005 opt	6417	replaced with 0.0857 ppm
-0.003	below detection limit of 0.003 ppm	2723	replaced with 0.0015 ppm
-0.9943	below detection limit of 0.029 opt	429	replaced with 0.0857 ppm
-5557	Sample Not Received	3	Omit
-5556	Sample Not Received	80	Omit
-0.0343	Half the detection limit of 0.002opt	926	replaced with 0.0343 ppm

Non-Positive Value	Interpretation	Count In Data Set	Action Taken
-0.1714	below detection limit for 0.005 opt	52	replaced with 0.0857 ppm
-3394.2842	conversion of -99 opt to ppm	13	replaced with 0.0857 ppm

11.3 Topography

Topography was provided by i-80 as dxf files with triangulated surfaces. The files included both as built surfaces showing dimensions of previously mined pits at their maximum depths, and present topography which includes backfill, pits, dumps, and surrounding topography. The current topographic data was loaded into Leapfrog Geo and used to constrain the block model. The topographic data provided by i-80 was not rectangular, which is required within Leapfrog to generate models; therefore, GRE extrapolated topographic data around the edges to form a rectangular surface (Figure 11-2). The extrapolated area, however, is not part of the resource estimate.

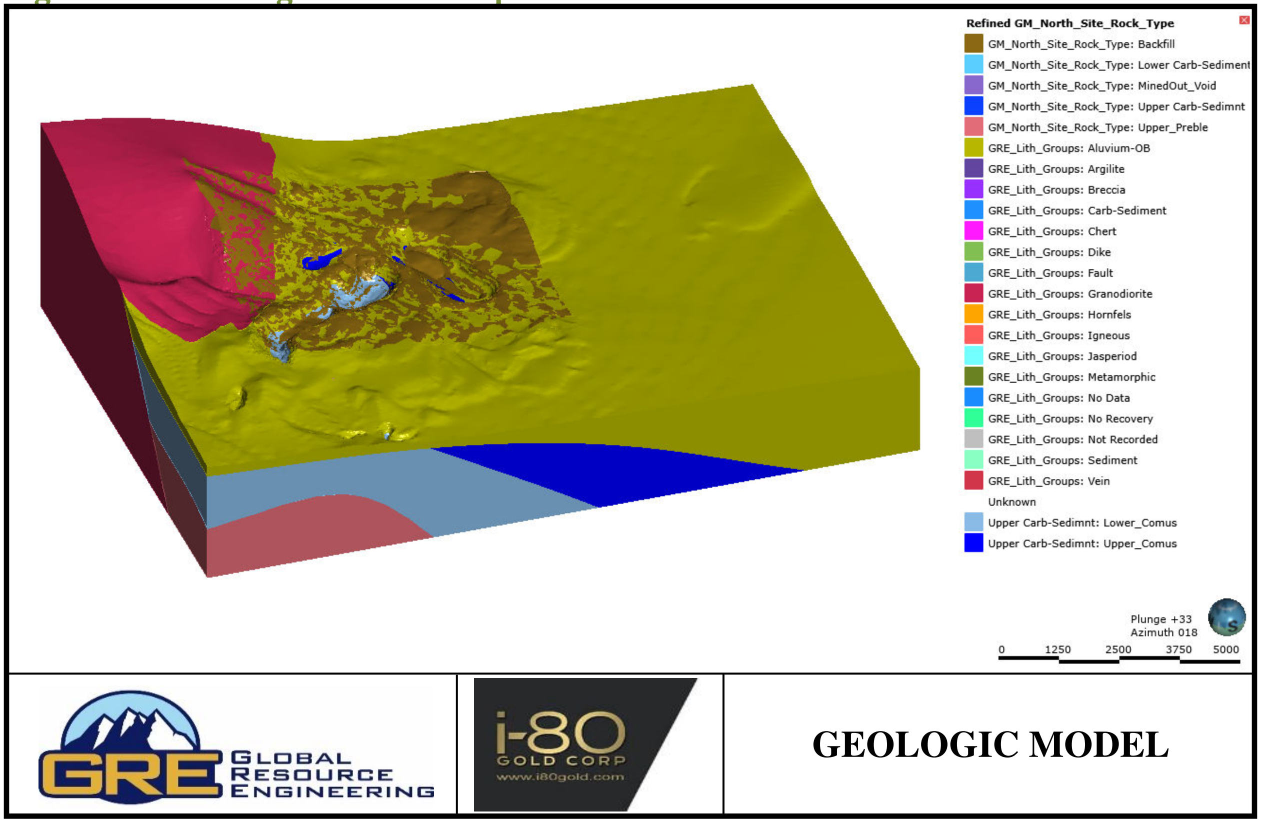
Figure 11-2: Current Topography Used for Resource Estimation



11.4 Geologic Model

The geologic model used to complete the mineral resource estimate was developed by GRE using grouped majority composites for lithology based on data provided to GRE as part of the drill hole database. Material below the current topography and above the as-built surface was classified as backfill and assigned an Au ppm grade of zero. See Figure 11-3 for illustration of the geologic model used in the resource estimation. The model was validated for geologic accuracy and found to be suitable for the purpose of Mineral Resource Estimation by GRE.

Figure 11-3: Geologic Model Oblique View



11.5 Open Pit Estimation

11.5.1 Estimation Domains

Estimation zones were recreated by bounding the assays that surround the pit areas (Figure 11-4). The underground zone to the North of Zone 3/CX Pit was not considered since this area was estimated in the underground resource section of this report. Vertical extents of the estimation zones range from 5500 feet amsl to 2500 feet amsl.

Figure 11-4: Open Pit Estimation Zones

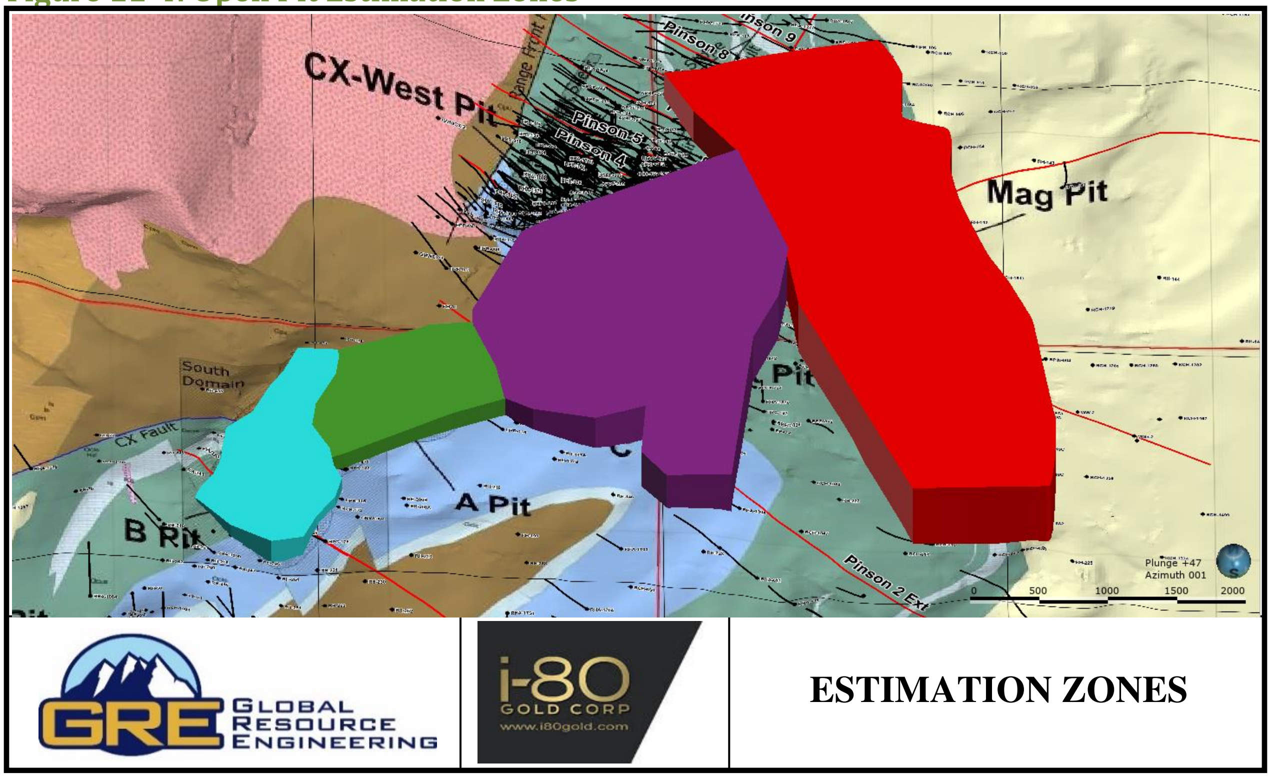


Table 11-2 summarizes the estimation zone numbers along with the corresponding pits.

Table 11-2: Open Pit Estimation Zone and Pit Name

Estimation Zone	Pit Name
Zone 1	A Pit
Zone 2	MAG Pit
Zone 3	CX Pit
Zone 4	B Pit

Numeric indicator models were constructed to better define the high-grade mineralized domains contained within the generalized estimation domains fit around the existing pits. The parameters used to define the indicator models are shown in Table 11-3.

Table 11-3: Open Pit Numeric Indicator Model Parameters

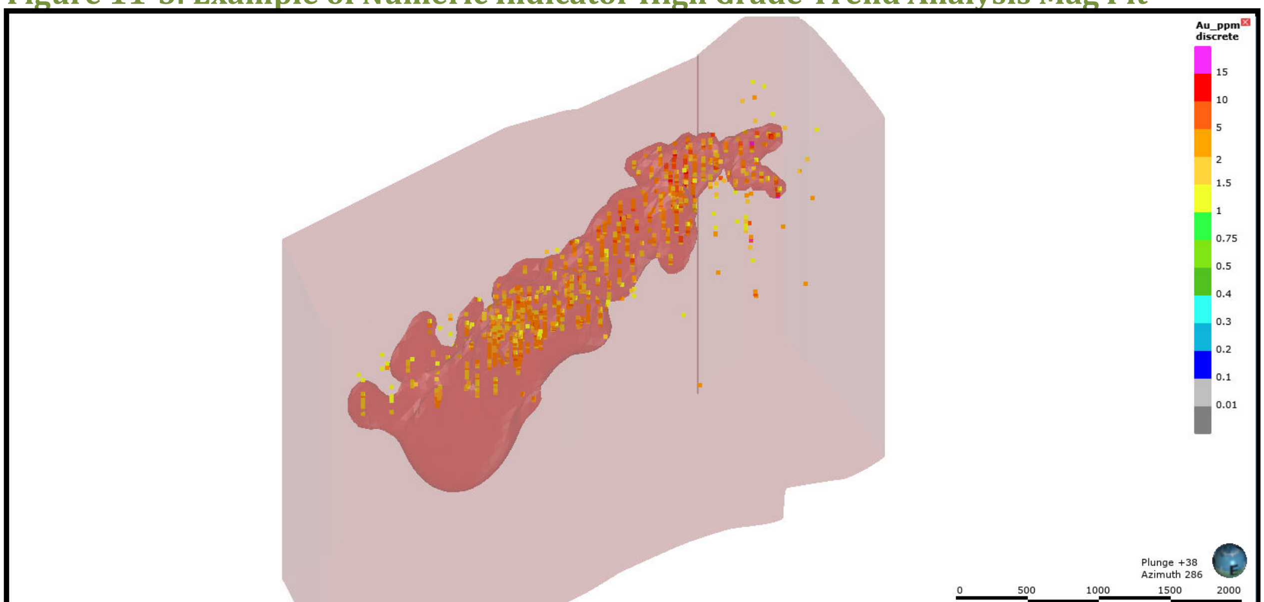
Estimation Zone	Indicator Model Cutoff (ppm)	ISO Value	Search Distance (feet)	Dynamic Anisotropy
Zone 1	1.0	0.4	200	CX Fault
Zone 2	1.0	0.3	250	Mag Fault
Zone 3 CX	0.1	0.4	180	CX Fault
Zone 3 SOS DIKE	0.1	0.4	100	SOS Dike

Zone 3 SOS XSECTION	0.1	0.3	150	SOS X Section Fault
Zone 4	1.0	0.3	200	NA*

*Because Zone 4 did not use dynamic anisotropy, a global trend set to the following parameters was used: dip 90, dip Azimuth 100, pitch 75, and ellipse ratios max. 200, int. 200, min. 100.

Initial search ellipse orientations were set from examining the spatial orientation of the composites greater than 1 g/t (see Figure 11-5). After initial construction of the high grade solids using indicator models, it was noted that some of the high-grade numeric models had voids or otherwise poor geometry. When examining the fault structures, it was noted that the high grade domain corresponded well with the location and orientation of several fault structures. GRE then attempted to add a structural trend using dynamic anisotropy to the numeric model by constructing a structural trend from the fault meshes. This improved the continuity of the indicator models and helped eliminate voids that were previously present in the indicator models. ISO factors, which are defined as the probability that the enclosing volume encloses the values above the cutoff, for interpolants were based on visually examining the mineralized body and using an iterative process to select a value that produced a reasonable geologic shape. If small islands of volume were created off the major trend, they were clipped out.

Figure 11-5: Example of Numeric Indicator High Grade Trend Analysis Mag Pit



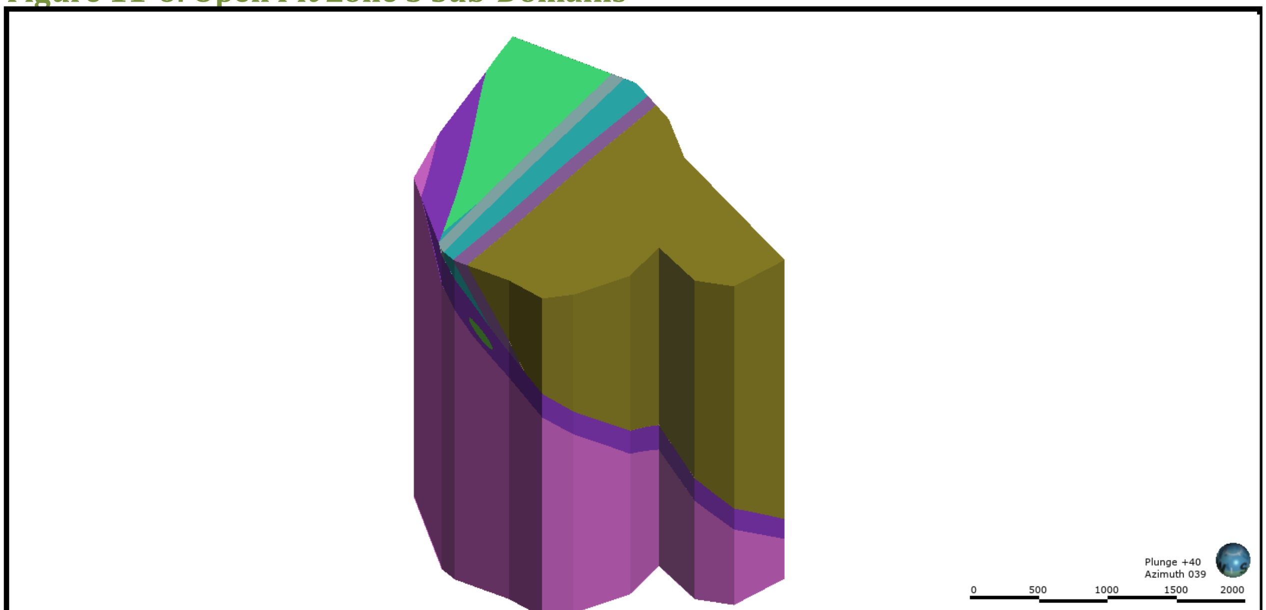
EXAMPLE OF NUMERIC INDICATOR HIGH GRADE TREND ANALYSIS MAG PIT

Note: The opaque red solid represents the high grade domain. The transparent red solid is the low grade domain. Composite grades are shown at a cutoff of 1ppm.

One resource estimation zone, Zone 3, was broken into several sub domains after failing to produce a reasonable high grade numeric indicator model using a single global trend within the Zone 3 domain. It was noted that during the initial attempt, branching solids formed displaying three distinct trends. Upon further investigation it was found that these trends corresponded to fault structures that crosscut the Zone 3 domain. The identified structural trends are the CX fault, the SOS Dike, and the SOS X-Section.

To construct estimation sub-domains, GRE offset the fault meshes both forwards and backwards to a thickness that contained the majority of the high-grade intercept (see Figure 11-6). Separate numeric estimators were constructed within these domains, and high grade and low-grade zones were defined within the zones that contained the mineralized trends (see Figure 11-7). To later avoid estimation boundary issues during resource estimation, the volumes on either side of the mineralized domains were separated into sub domains. These include the HW 1, HW 2, HW 3, and FW zones.

Figure 11-6: Open Pit Zone 3 Sub-Domains



GLOBAL RESOURCE ENGINEERING

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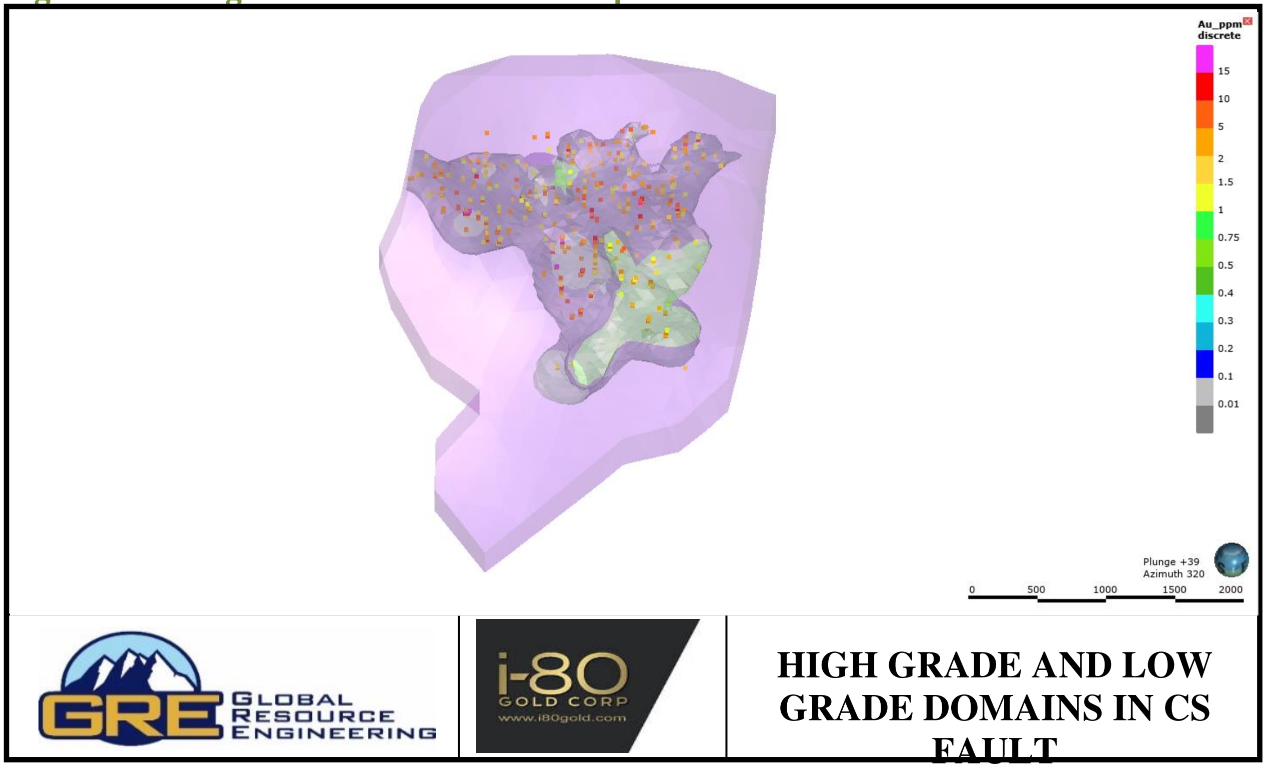
ZONE 3 SUB-DOMAINS

Figure 11-7: High Grade and Low Grade Open Pit Domains in CX Fault



Note: The high grade zone is shown in green, the low grade domain is shown in purple.

11.5.1.1 Domain Analysis

To check the validity of the high grade and low-grade estimation domains, box and whisker plots were constructed, as shown in Figure 11-8. Generally, a good correlation was observed between the high grade and low-grade solids and the distribution of the grades contained within them.

Figure 11-8: Box and Whisker Plot of Open Pit Estimation Domains

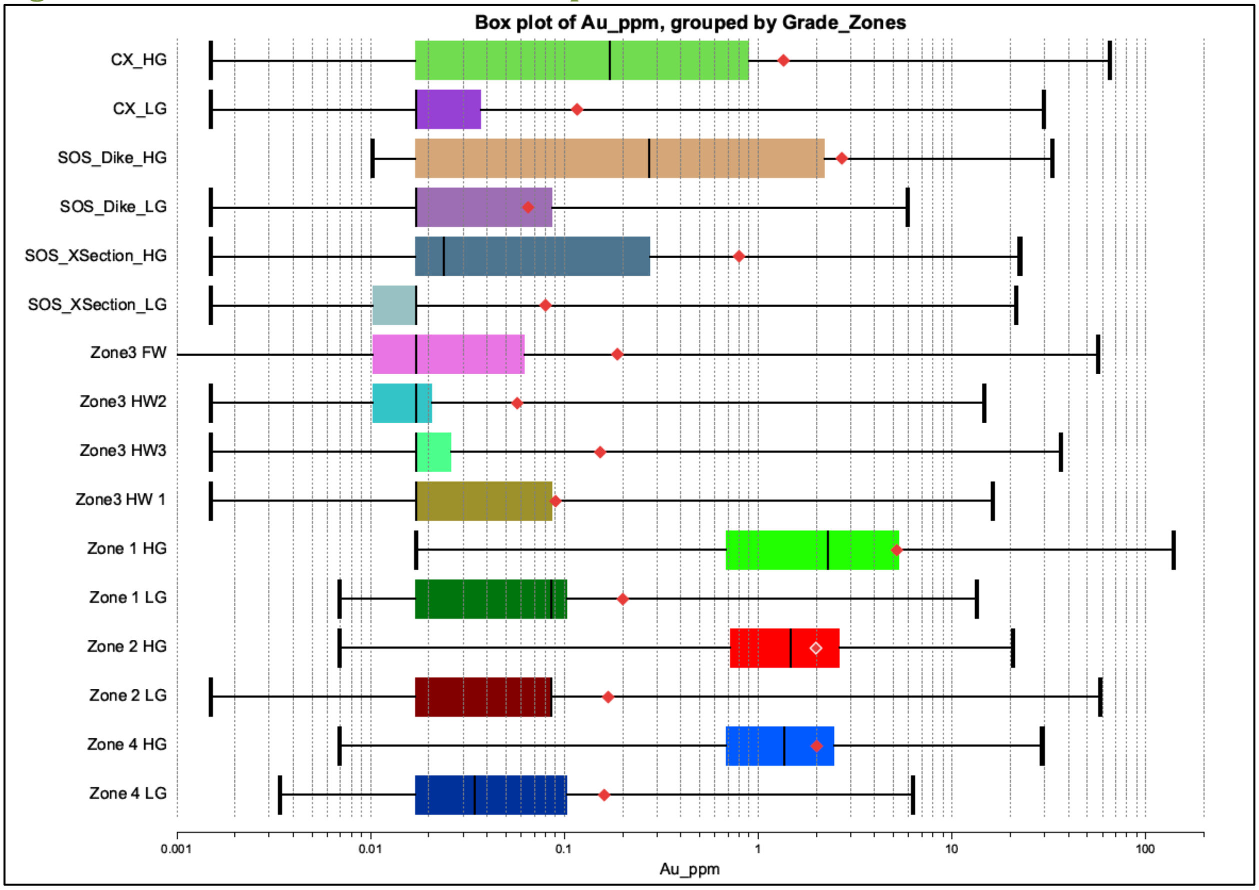


Figure 11-9: HG and LG Distributions in Zones 1 and 2

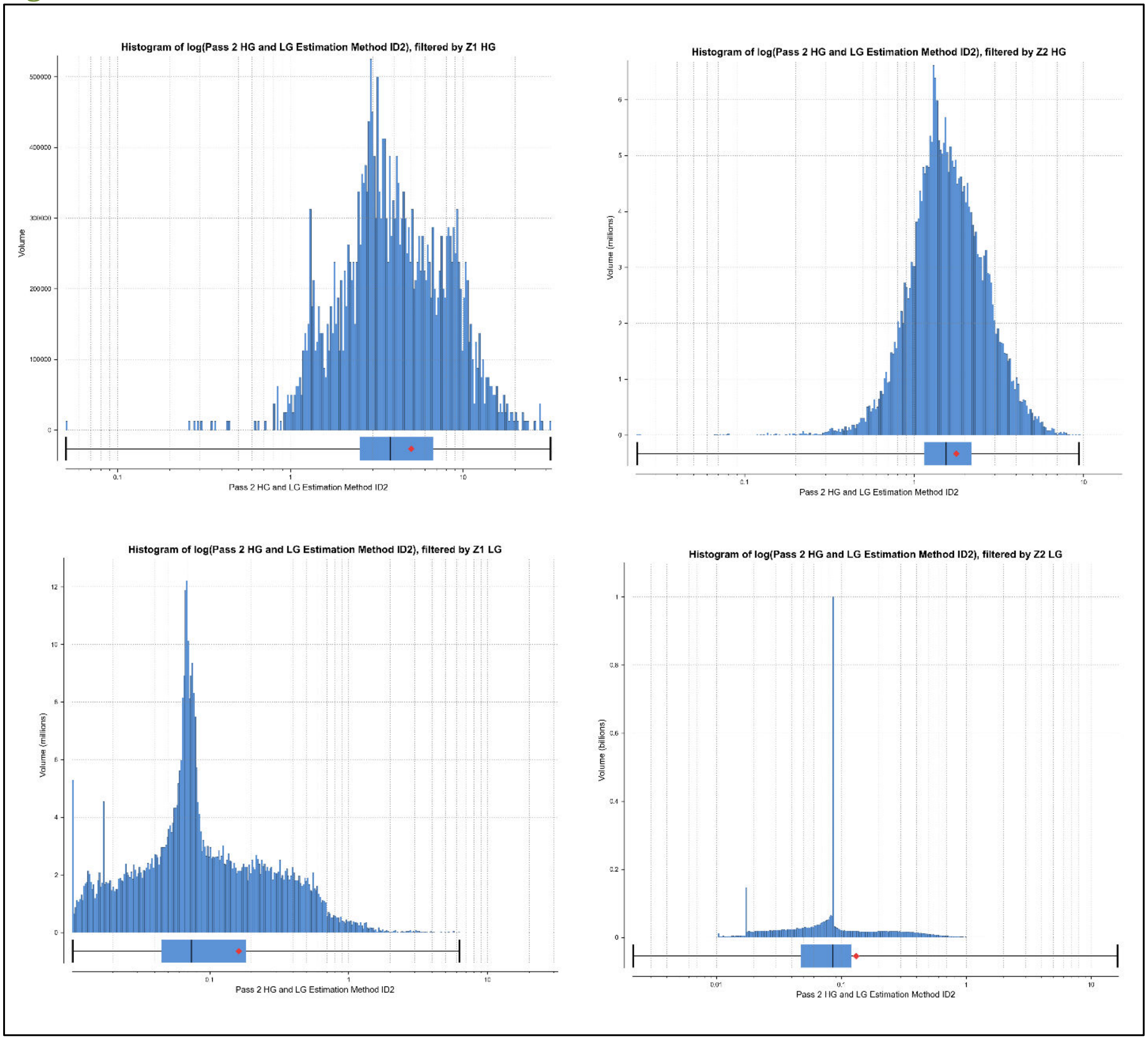
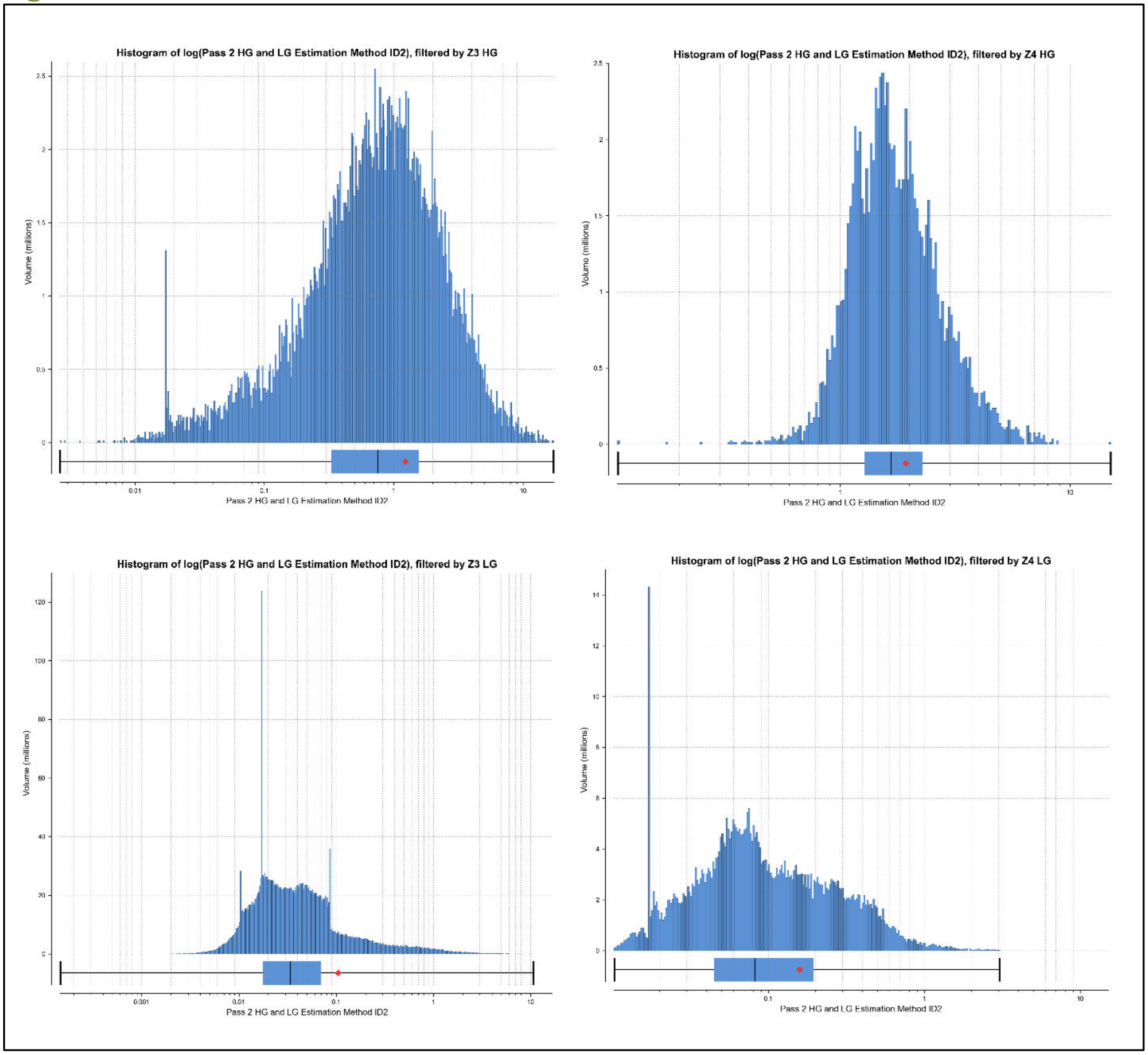


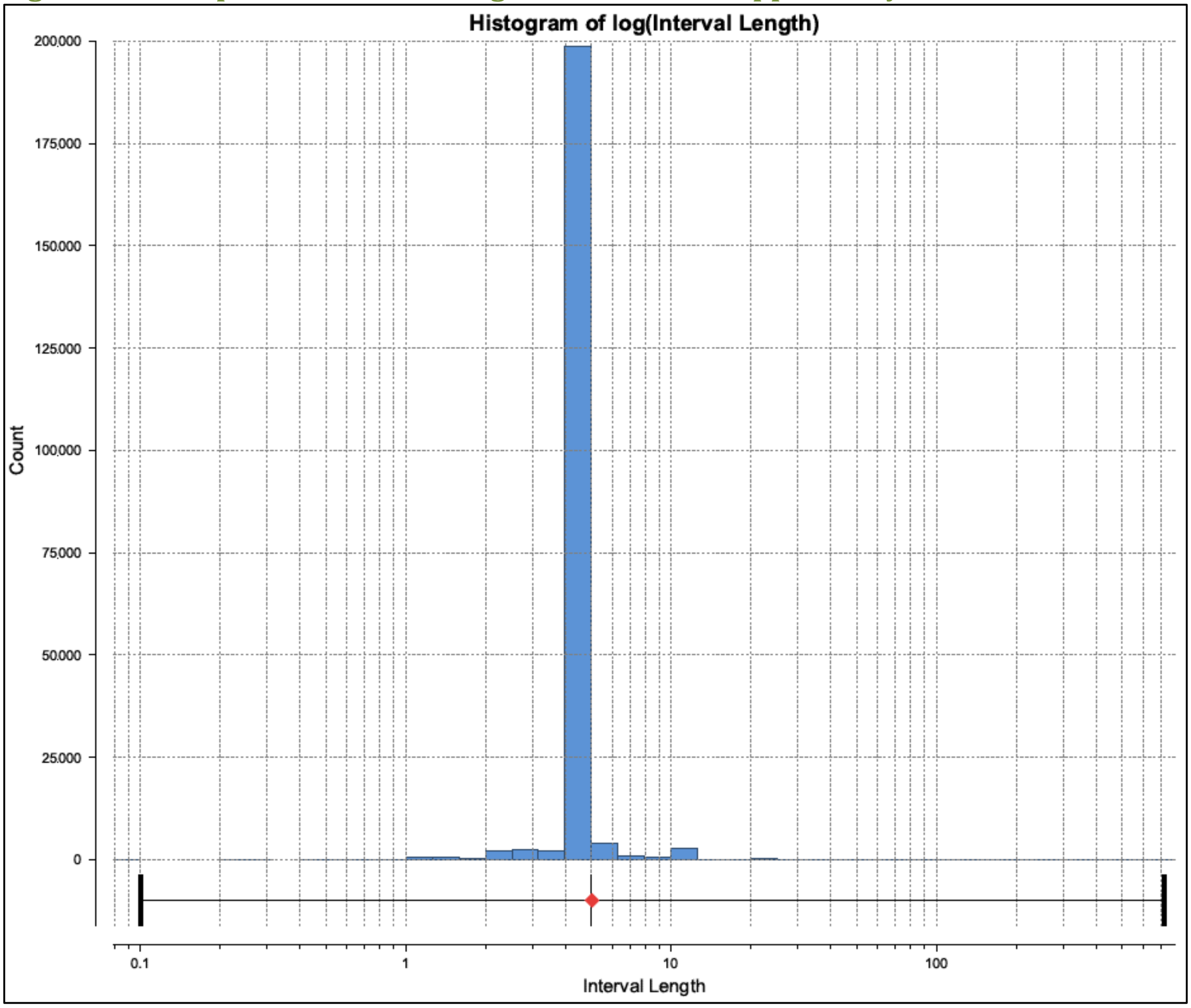
Figure 11-10: HG and LG Distributions in Zones 3 and 4



11.5.2 Assay Compositing

Sample data was composited to intervals of equal length to ensure that the samples used in statistical analysis and estimations were equally weighted. To accomplish compositing, GRE first examined the interval histogram to determine the most common assay length (see Figure 11-11).

Figure 11-11: Open Pit Interval Length Statistics of Au ppm Assays



Once it was determined that five feet was the primary assay length, GRE evaluated various compositing lengths using 5-foot intervals to avoid splitting assays. It was decided that compositing on a 20-foot interval represented a significant decrease in the variance of the data while not adversely decreasing the mean of the data set, as shown in Table 11-4. Therefore, GRE selected a 20-foot composite interval.

Table 11-4: Open Pit Compositing Interval Statistics

Statistic	AU_ppm_Assays	Composite Interval				
		5-foot	10-foot	15-foot	20-foot	25-foot
Count	221,062	264,712	133,121	88,217	66,103	52,752
Length	1,323,435.3	1,323,396.9	1,323,367.4	1,322,037.1	1,321,859.4	1,321,327.9

Statistic	AU_ppm_Assays	Composite Interval				
		5-foot	10-foot	15-foot	20-foot	25-foot
Mean	0.362	0.362	0.361	0.358	0.357	0.354
SD	2.58	2.49	2.31	2.08	1.95	1.87
CV	7.1	6.9	6.4	5.8	5.5	5.3
Variance	6.6	6.2	5.4	4.3	3.8	3.5
Minimum	0	0	0	0	0	0
Q1	0.0170	0.0171	0.0171	0.0171	0.0171	0.0171
Q2	0.0170	0.0171	0.0274	0.0343	0.0343	0.0343
Q3	0.0860	0.0857	0.0857	0.0857	0.0857	0.0857
Maximum	290.06	290.06	290.06	119.89	94.33	85.08

A box plot comparison of the 20-foot composited and the uncomposited assays is shown in Figure 11-12. This comparison shows that compositing, while not changing the mean or quartiles, does drastically reduce the maximum value of grades.

Figure 11-12: Open Pit Compositing Comparison 20 Foot Intervals

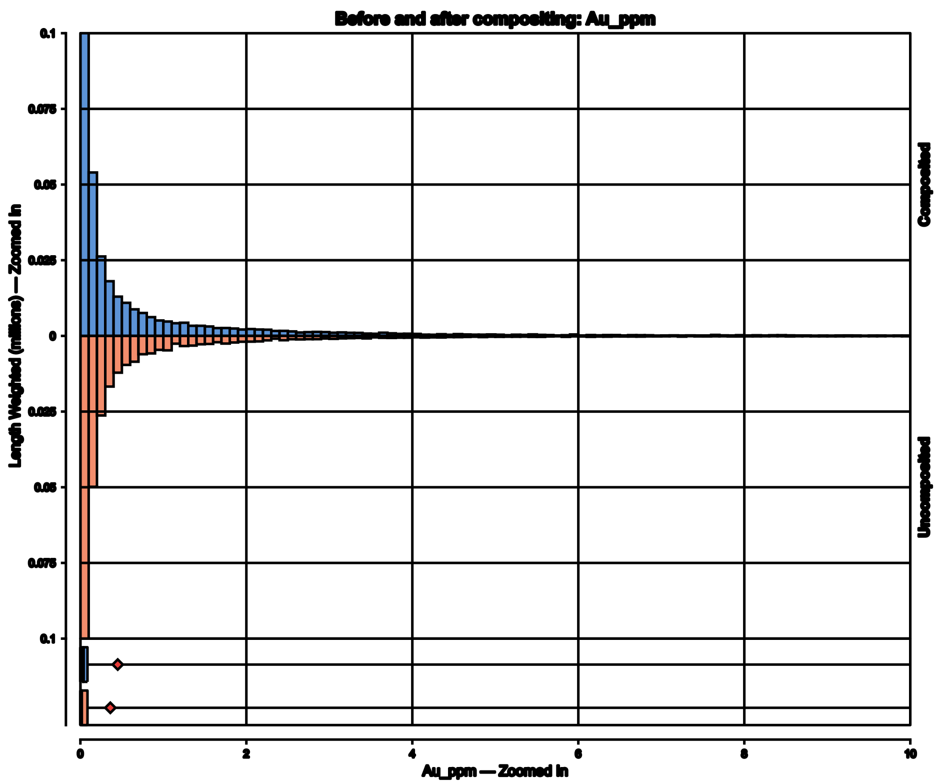


Table 11-5: Open Pit Compositing Comparison 20 Foot Intervals

Statistic	Composited	Uncomposited
Count	50,633	221,675
Length	1,011,524	1,327,165

Statistic	Composited	Uncomposited
Mean	0.45	0.36
SD	2.22	2.57
CV	4.93	7.13
Variance	4.93	6.63
Minimum	0	0
Q1	0.02	0.02
Q2	0.04	0.02
Q3	0.09	0.09
Maximum	94.33	290.06

11.5.3 Evaluation of Outliers

Cumulative probability plots for gold were completed for the composites within each estimation domain (see Figure 11-13 for an example). A break in the population was identified and marked with the clipping line. Based on this analysis, GRE applied a maximum allowable value for the gold grade within each separate domain, as shown in Table 11-6.

Figure 11-13: Example of Open Pit Cumulative Log Probability Plot Zone 1 HG

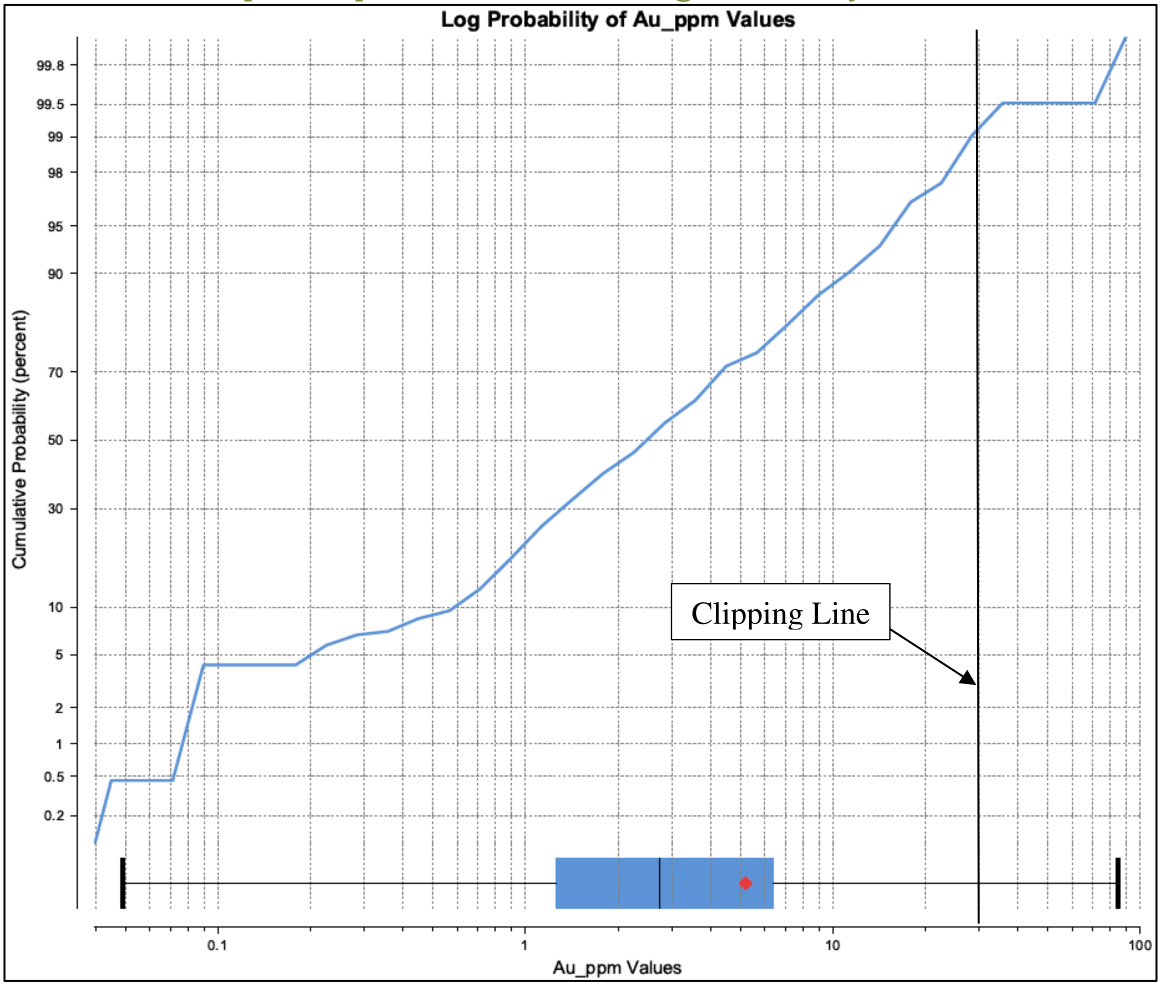


Table 11-6: Open Pit Upper Clipping Au ppm Values by Domain

Zone	Sub-Domain	Clipping Value
Zone 1	HG	35
	LG	NA
Zone 2	HG	10
	LG	5
Zone 3	CX HG	20
	CX LG	10
	SOS DIKE HG	NA
	SOS DIKE LG	NA
	SOS XSECTION HG	NA
	SOS XSECTION LG	NA
	HW 1	7
	HW 2	3
	HW 3	7

Zone	Sub-Domain	Clipping Value
	FW	12
Zone 4	HG	NA
	LG	3

11.5.4 Density

Density was assigned to each domain in the block model based on a combination of rock type and grade, as shown in Table 11-7. The bulk densities are the same as those used in the 2020 Getchell Project Technical Report (AMC, 2020) and were originally supplied by OMC. The results for each domain fit well with GRE's experience with similar rock types.

Table 11-7: Open Pit Domain Density Summary

Unit	Au>=0.008 opt (tonne/m3)	Au<0.008 opt (tonne/m3)
Backfill	1.85	1.85
Alluvium	1.85	1.85
Granodiorite	2.7	2.7
Upper Comus	2.5	2.7
Lower Comus	2.51	2.64
Preble	2.42	2.6

11.5.5 Variography

After iterative analysis, a good fit for the gold grade variography was found using pairwise relative variograms. The pairwise relative variogram helps to smooth the variogram by scaling $\gamma(h)$ using the square of the mean of each sample pair of the data from calculating $\gamma(h)$. This makes the interpretation of the variogram model easier, and all variances calculated this way are relative to the mean of the sample pairs within the distribution.

Variogram analysis was completed on the samples within each of the high grade and low grade estimation domains to establish the direction of maximum continuity between sample pairs. The range for each variogram was found using a global variogram. The nugget was determined by examining the downhole variograms and determining where the short-range trend crossed the y-axis. Variograms were orientated along the strike and dip of the visually observed high grade trend of the composites, with the major axis oriented along the direction of maximum continuity.

Table 11-8: Open Pit Variogram Parameters

Zone	Sub-Domain	Dip	Dip Azimuth	Pitch	Major Axis	Semi-Major Axis	Minor Axis
Zone 1	Overall	38	134	75	200	200	100
	HG	38	134	160	160	160	75
	LG	38	134	75	175	175	60
Zone 2	Overall	50	70	105	250	250	100
	HG	50	70	75	180	125	75
	LG	50	70	105	300	200	200
Zone 3	Overall*	56	135	75	160	160	125
	CX	56	135	75	180	160	70
	CX HG	56	135	75	80	80	50
	CX LG	56	135	75	50	50	50
	SOS DIKE	65	170	45	100	80	50
	SOS DIKE HG	65	170	45	100	80	75
	SOS DIKE LG	65	170	45	100	50	25
	SOS XSECTION	80	170	80	150	125	50
	SOS XSECTION HG	80	170	80	80	80	80
	SOS XSECTION LG	80	170	105	150	125	50
	HW 1	56	135	150	160	100	125
	HW 2	56	135	100	160	160	125
	HW 3	56	135	75	160	160	125
	FW*	56	135	70	140	125	125
Zone 4	Overall	90	100	75	200	200	100
	HG	90	100	90	60	60	25
	LG	90	100	75	300	225	75

11.5.6 Block Model Parameters

A 3D block model was developed to represent the deposit using a block size of 25 feet x 25 feet x 20 feet. The block model dimensions and model limits are shown in Table 11-9. The coordinate system used for the 3D modelling was based on the local grid system using imperial units of feet. The block model is un-rotated and contains no sub blocking.

Table 11-9: Block Model Parameters Open Pit

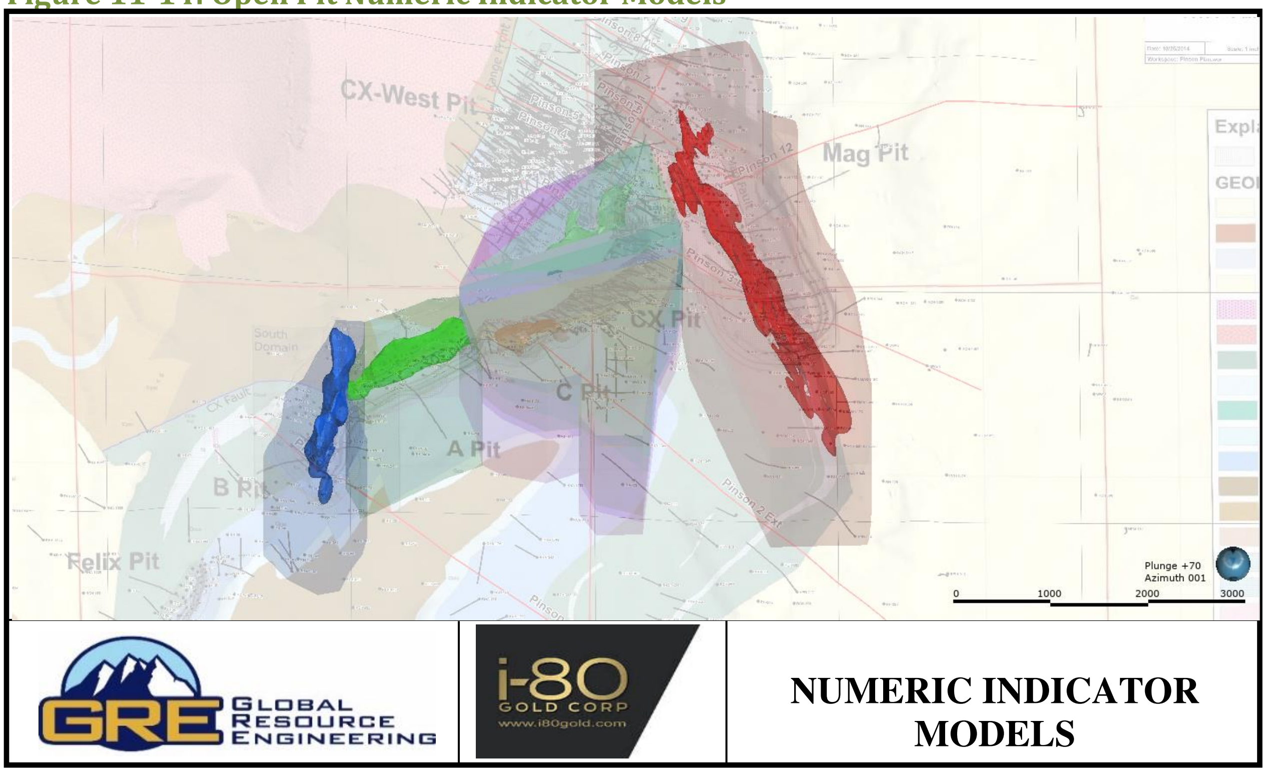
Parameter	Value
Base point	6000,7000,5800 (X,Y,Z)
Parent block size	25x25x20 (X,Y,Z)
Azimuth	0

Boundary size	9000,7000,3000 (X,Y,Z)
Size in Blocks	360x280x150 (X,Y,Z)
Sub-blocking	None

11.5.7 Estimation Domains

The estimation domains used to constrain the mineral resource estimate resulted from the numeric indicator models developed as part of the geologic model as discussed in Section 11.5.1. Figure 11-14 shows an overview of the estimation domains that were used to constrain the mineral resource estimation for the open pit mineral resource estimate.

Figure 11-14: Open Pit Numeric Indicator Models



11.5.8 Estimation Parameters

Estimation within mineralized domain boundaries was performed using an inverse distance squared method with a minimum of 4 samples, a maximum of 20 samples, and a drill hole limit of 2. Declustering objects were applied to all high-grade estimation domains. Dynamic anisotropy was applied where it was applicable based on faults that structurally control mineralization. The exception to this was Zone 4, which has no apparent structural control that has yet been mapped. In this case, search ellipse orientation was determined from examining the spatial orientation of the composites greater than 1 g/t.

Search distances of the domained estimators were based on the variography for each sub-domain, as discussed in Section 11.5.4. Search distances for domains that showed poor variography were replaced by the overall sub-domain or overall domain search distances. This method was used for the Zone 4 HG and LG estimations and the Zone 3 sub-domains. All estimations used hard boundaries with the exception of the boundaries between Zone 1 HG and Zone 3 HG and between Zone 1 LG and Zone 4 LG. Soft 40-foot boundaries were set up with filters between these boundaries since they are immediately adjacent to each other and could potentially have continuity in grade estimation across these boundaries. The inverse distance estimation parameters for each domain are given in Table 11-10.

Table 11-10: Open Pit ID2 Estimation Parameters

Zone	Sub-Domain	Dynamic Anisotropy	Trend	Major	Semi-Major	Minor
Zone 1	HG	Yes	CX Fault	160	160	75
	LG	Yes	CX Fault	175	175	60
Zone 2	HG	Yes	MAG Fault	180	125	75
	LG	Yes	MAG Fault	300	200	200
Zone 3	CX HG	Yes	CX Fault	180	160	70
	CX LG	Yes	CX Fault	180	160	70
	SOS DIKE HG	Yes	SOS Dike	100	80	50
	SOS DIKE LG	Yes	SOS Dike	100	80	50
	SOS XSECTION HG	Yes	SOS Xsection	150	125	50
	SOS XSECTION LG	Yes	SOS Xsection	150	125	50
	HW 1	Yes	CX Fault	160	100	125
	HW 2	Yes	CX Fault	160	160	125
	HW 3	Yes	CX Fault	160	160	125
	FW	Yes	CX Fault	160	160	125
Zone 4	HG	No	90,100,90	200	200	100
	LG	No	90,100,75	200	200	100

After each domained estimator was constructed, a combined estimator was used to assign a hierarchical value to each domained estimation to produce a single gold grade value. The combined estimator hierarchy is shown in Table 11-11.

Table 11-11: Open Pit Combined Estimator Hierarchy

Priority	Domained Estimation	Priority	Domained Estimation
1	Zone 1 HG	11	Zone 3 HW 3
2	Zone 1 LG	12	Zone 3 HW 1
3	Zone 2 HG	13	Zone 3 HW2
4	Zone 2 LG	14	Zone 3 FW

Priority	Domained Estimation	Priority	Domained Estimation
5	Zone 3 CX HG	15	Zone 4 HG
6	Zone 3 CX LG	16	Zone 4 LG
7	SOS Dike HG	17	Zone 1
8	SOS Dike LG	18	Zone 2
9	SOS Xsection HG	19	Zone 3
10	SOS Xsection LG	20	Zone 4

The overall domain grade estimations were assigned to priority 17 to 20 so that they would fill in areas of the numeric estimator domains that lacked the required number of samples to be able to estimate grade due to the small volume being estimated that excluded drill holes.

11.5.9 Geometallurgical Modeling

Section 10.510.5 goes into detail of how the gold recovery model was estimated and implemented in the block model.

Cyanide solubility was compared to all available interval information from the drilling data: gold assay, alteration, lithology, depth, etc. From this available data, a principal component analysis, regression tree, and multivariate adaptive regression spline analysis were performed. Using multivariate adaptive regression spline analysis model was created to predict cyanide solubility in different zones using the available drilling data. Heap Leach (HLCH) recovery is determined by plotting the cyanide solubility with the column recovery. The carbon in leach (CIL) recovery equation was determined by plotting the trend with the calculated head grade and CIL recovery.

The input fields required in the recovery equations were added to the block estimations. Then, the recovery equations were applied to the block model for HLCH and CIL recoveries. These recoveries, along with the Whittle inputs from Table 11-15, were used to determine which of the two processes would be applied to each block.

11.6 Open Pit Resource

11.6.1 Block Model Validation

Validation of the estimated block grades for the Granite Creek deposit was completed for each of the estimation domains. The resource block model estimate was validated by:

- Completing a series of visual inspections by comparisons of gold assay and composite grades to estimated block values across the deposit in both horizontal and vertical sections.

- Statistical comparison of parameters such as means, quantiles, and variance between 20-foot composites, Nearest Neighbor (NN), Inverse Distance squared (ID2), and Ordinary Kriged (OK) estimators to ensure that the grade estimations are representative of the composites they are based on.
- Comparing average composite sample values with average estimated block grades along east, north, and elevation orientations using swath grade trend plots.

11.6.1.1 Visual Inspection

The model was examined in plan and section views to compare to drill hole locations and grades. Plan views and section views for each of the estimation areas are shown in Figure 11-15 though Figure 11-22. Comparison of the model grade from the assays did not reveal any major discrepancies.

Figure 11-15: Open Pit Zone 1 Visual Comparison Composite to Block Model Grade Plan View

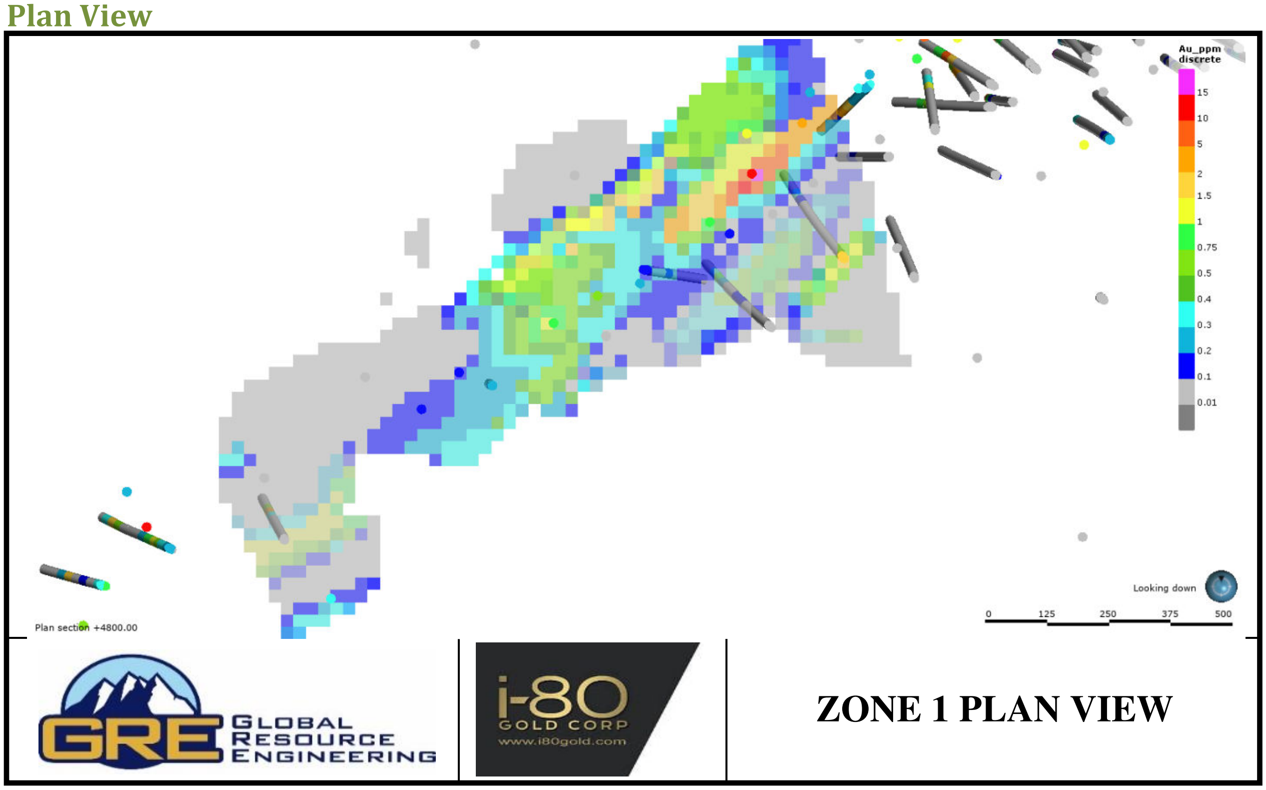


Figure 11-16: Open Pit Zone 2 Visual Comparison Composite to Block Model Grade Plan View

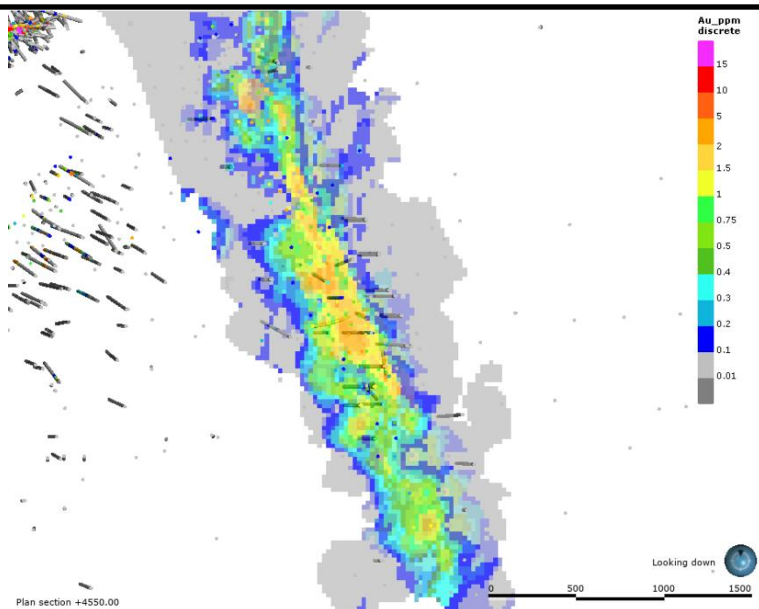


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Figure 11-17: Open Pit Zone 3 Visual Comparison Composite to Block Model Grade Plan View

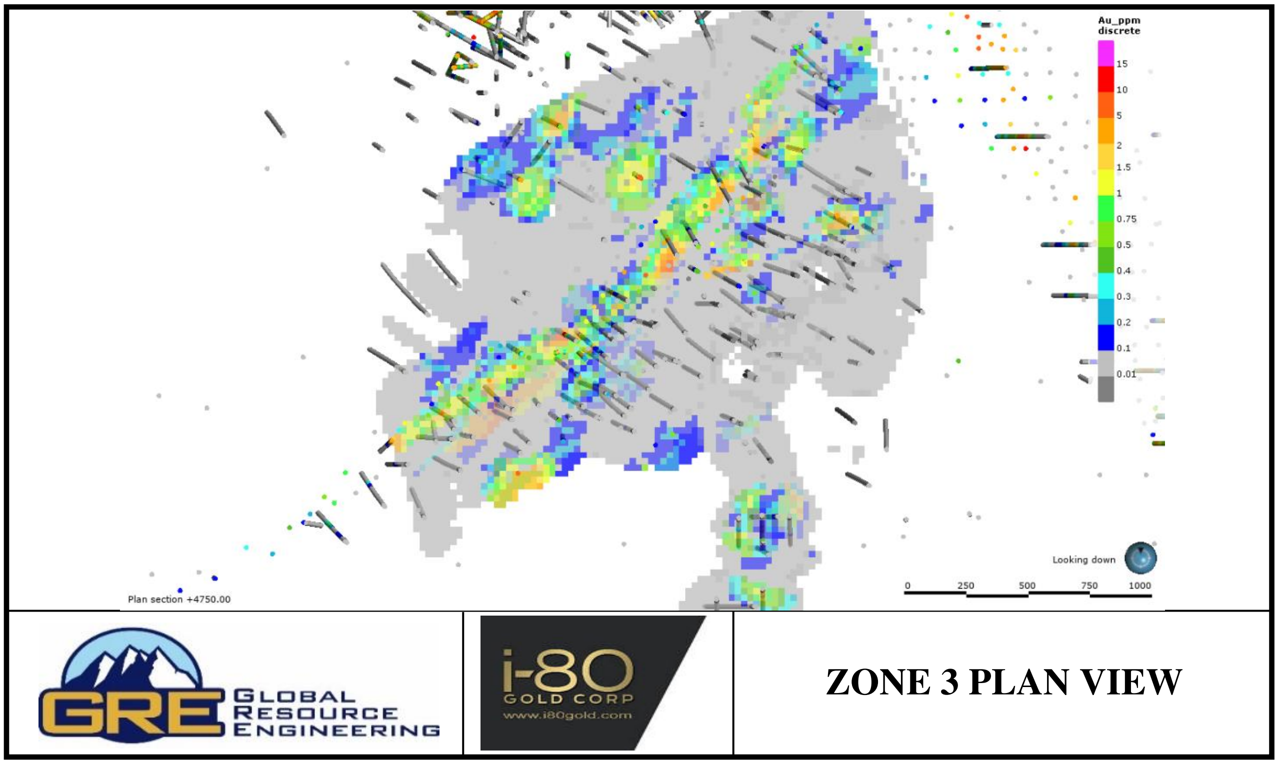


Figure 11-18: Open Pit Zone 4 Visual Comparison Composite to Block Model Grade Plan View

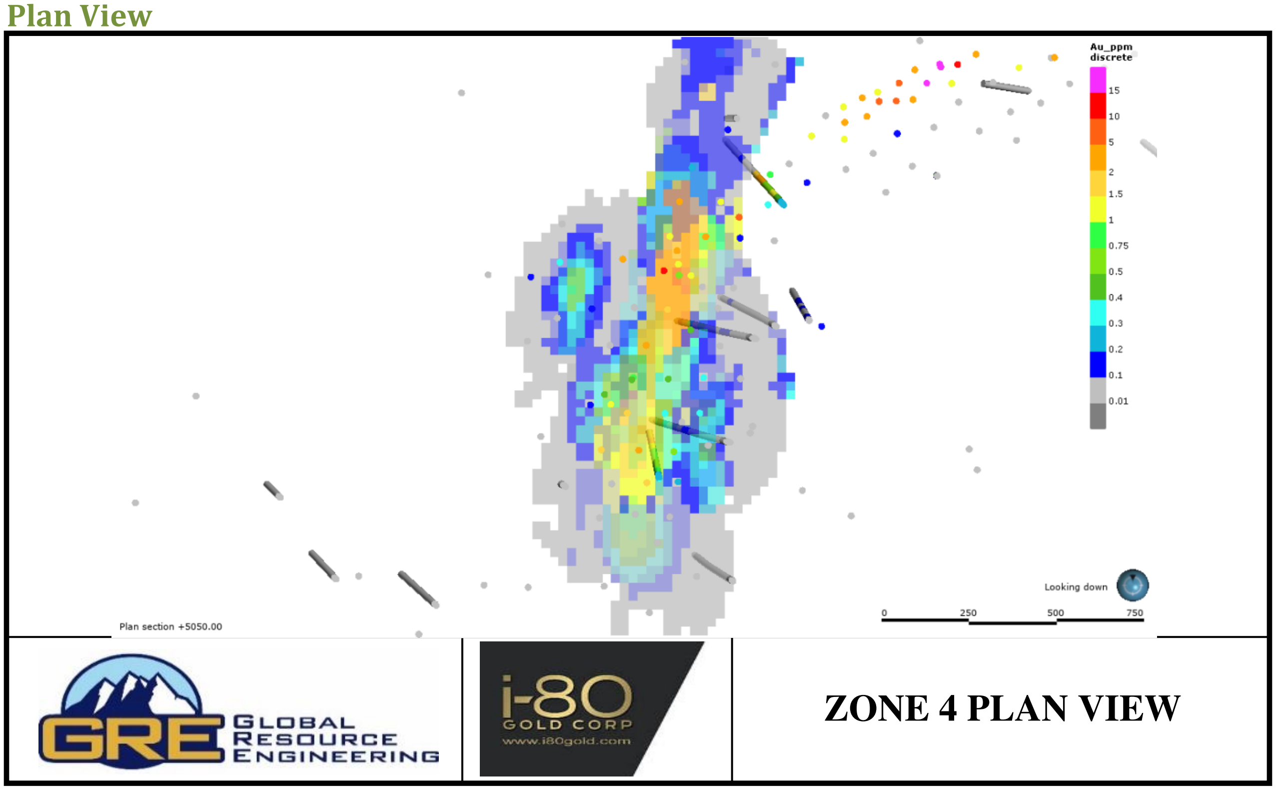


Figure 11-19: Open Pit Zone 1 Section Composites and Block Model Cross Section

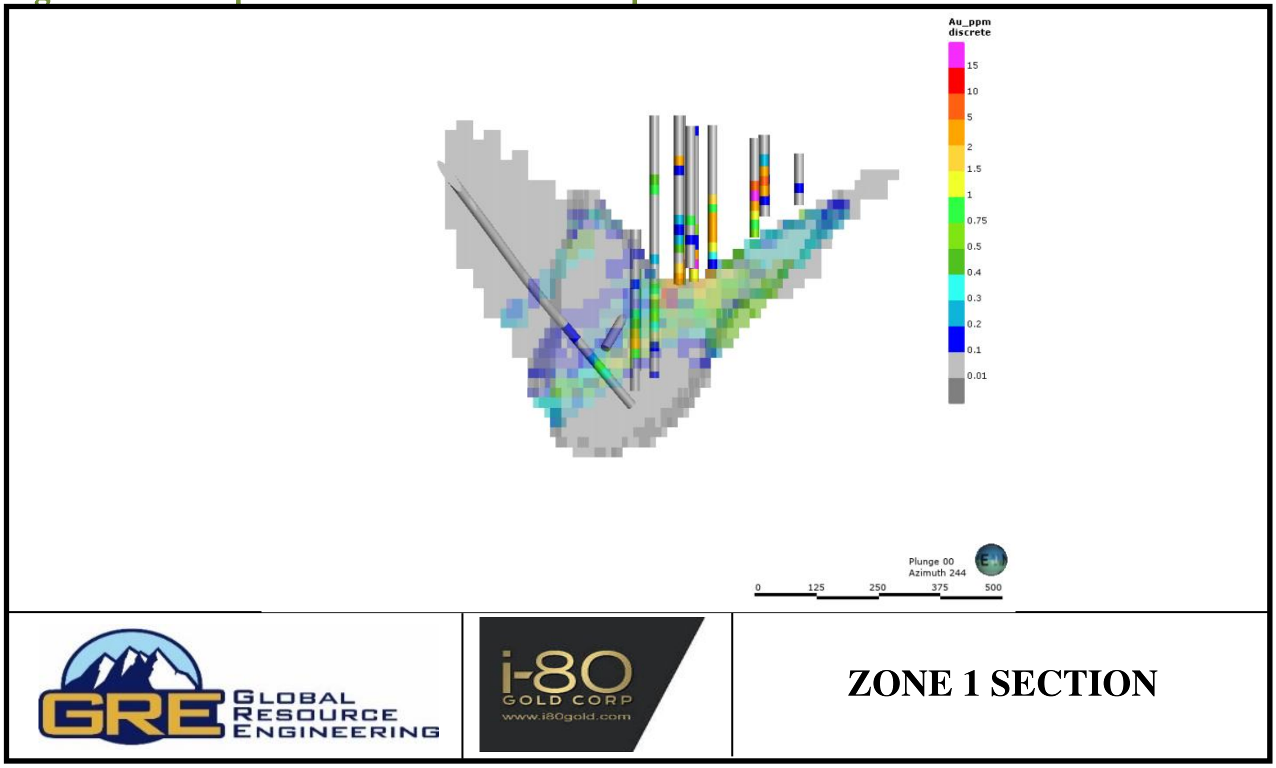


Figure 11-20: Open Pit Zone 2 Section Composites and Block Model Cross Section

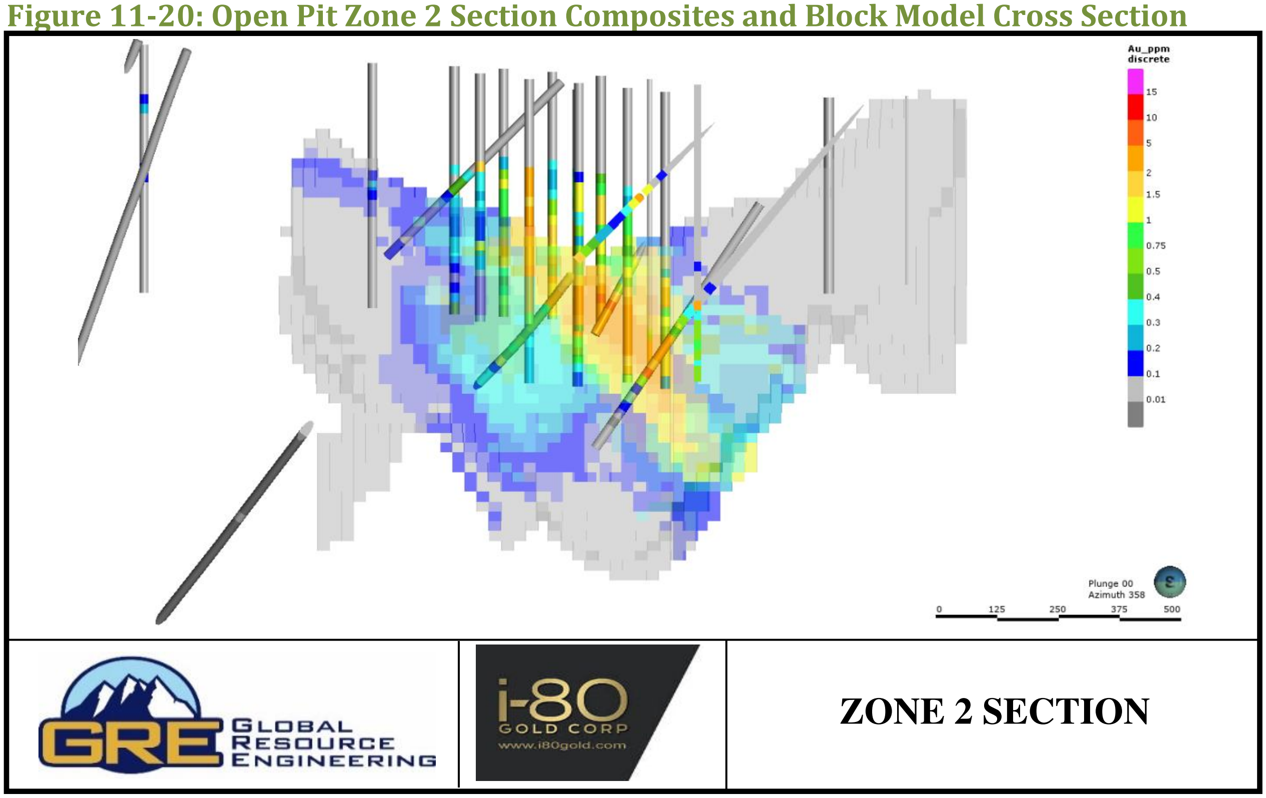


Figure 11-21: Open Pit Zone 3 Section Composites and Block Model Cross Section

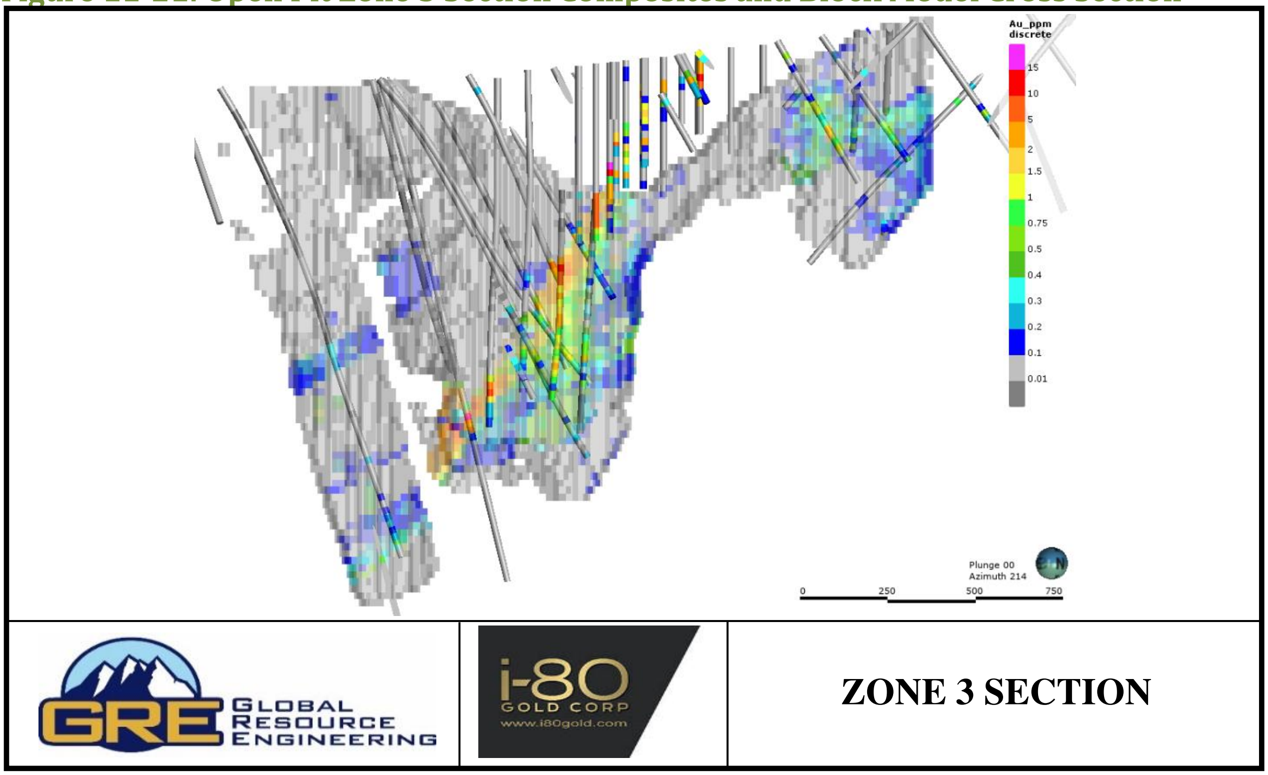
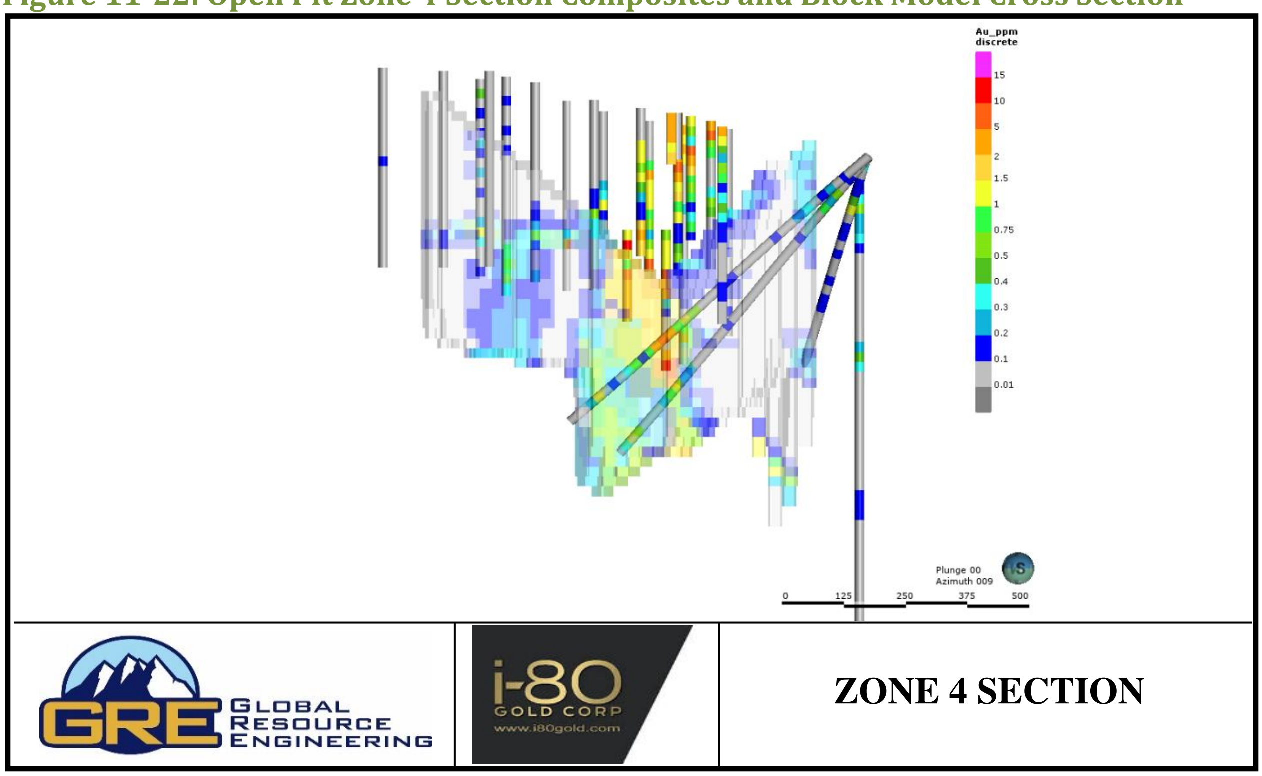


Figure 11-22: Open Pit Zone 4 Section Composites and Block Model Cross Section



11.6.1.2 Statistical Comparison

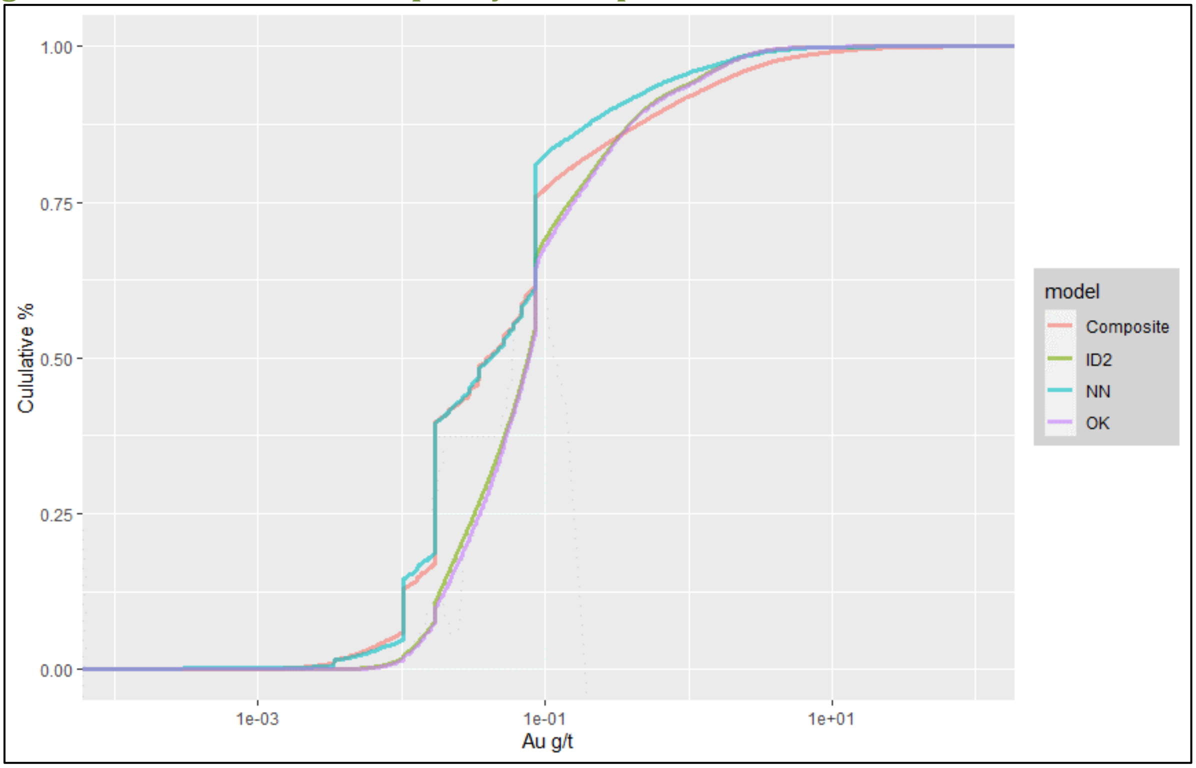
To ensure that the grade estimations are representative of the composites they are based on and validate the resource estimation results, the block model grade estimation statistics were analyzed. GRE compared the means, quantiles, and variance between 20-foot composites, Nearest Neighbor (NN), Inverse Distance squared (ID2), and Ordinary Kriged (OK) estimators, as shown in Table 11-12. Blocks are confined to the 2000 $/tr oz Whittle pit.

Table 11-12: Open Pit Comparison of Composite Values to Grade Estimation Methods

Parameter	Composites	Parameter	NN	ID2	OK
Count	66,300	Block Count	216,544	253,284	216,544
Mean	0.36	Mean	0.35	0.31	0.36
SD	1.95	SD	0.81	1.07	0.78
CV	5.47	CV	2.31	3.49	2.19
Variance	3.80	Variance	0.66	1.14	0.62
Minimum	0.00	Minimum	0.00	0.00	-0.35
Q1	0.02	Q1	0.03	0.02	0.03
Q2	0.03	Q2	0.08	0.05	0.08
Q3	0.09	Q3	0.27	0.09	0.28
Maximum	94.33	Maximum	32.11	35.00	27.39

As expected, the NN estimator generates more blocks due to the lack of restrictions of having to use multiple samples and multiple drillholes to estimate a block grade. Both ID2 and OK produced similar quantiles, means, variance, etc. The one marked difference seen between ID2 and OK is that it is possible to have a negative value in Kriging as seen in the minimum value of the OK estimator.

Figure 11-23: Cumulative Frequency of Composite and Block Data



11.6.1.3 Swath Plots

Swath plots of the various estimation methods (NN, ID2, and OK) were used to compare the results from each estimation method to the composite values and examine which method smoothed the estimated grades. As an example, swath plots are provided below for the Zone 1 high grade and low -grade domains. The swatch plots show a general trend that the ID2 estimator smoothed out drastic swings in grade while not over smoothing local variability.

Figure 11-24: Open Pit Swath Plot X axis, Zone 1 High Grade Domain

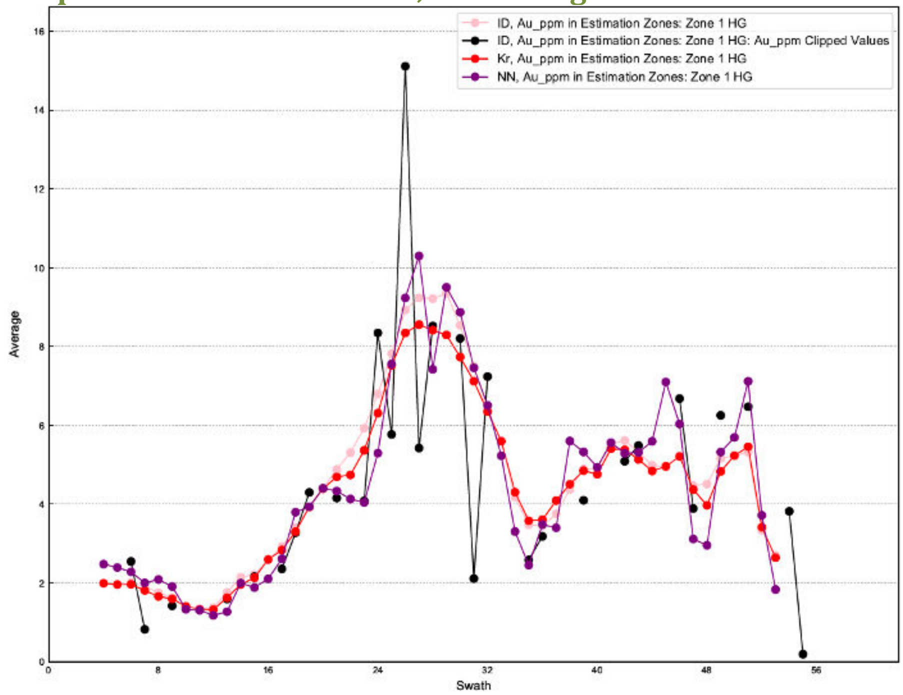


Figure 11-25: Open Pit Swath Plot Y axis, Zone 1 High Grade Domain

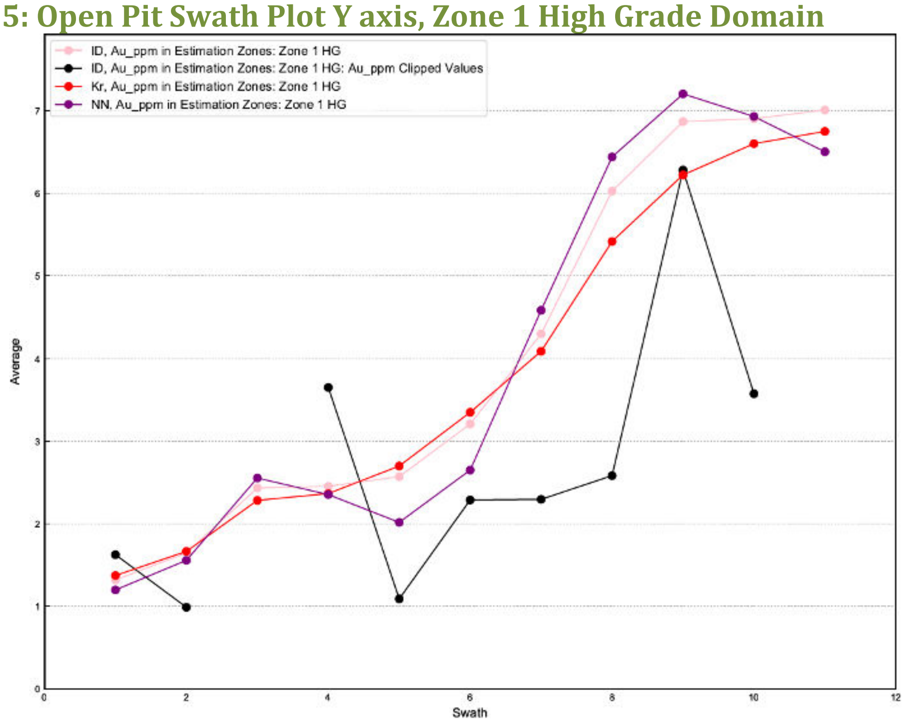
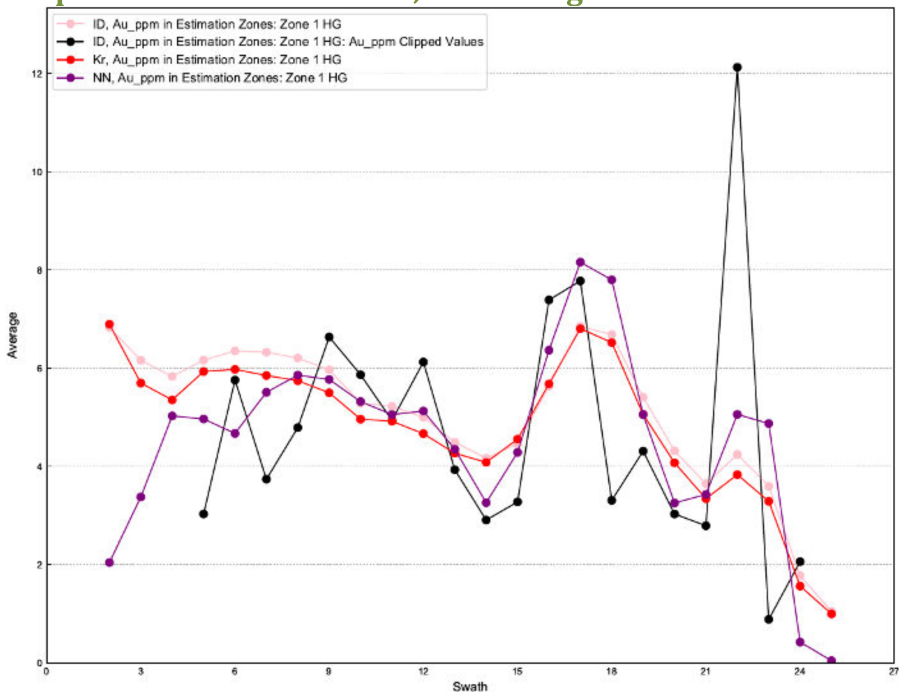


Figure 11-26: Open Pit Swath Plot Z axis, Zone 1 High Grade Domain



11.6.2 Mineral Resource Classification

Block model quantities and grade estimates for the Granite Creek deposit were classified according to the S-K 1300 definitions for Mineral Resources and Mineral Reserves (CFR, 2018).

Mineral resource classification involved a two-step process using minimum distances and minimum numbers of samples to define resource classification initially before applying numeric indicator model to define a more continuous and reasonable resource classification. The criteria used in the first step of the resource classification are listed in Table 11-13. Parameters used for the numeric indicator models for the second step of the resource classifications are listed Table 11-14.

Table 11-13: Open Pit Mineral Resource Classification Parameters

Resource Class	Minimum Distance	Minimum Number of samples
Measured	50	7
Indicated	100	5
Inferred	150	NA

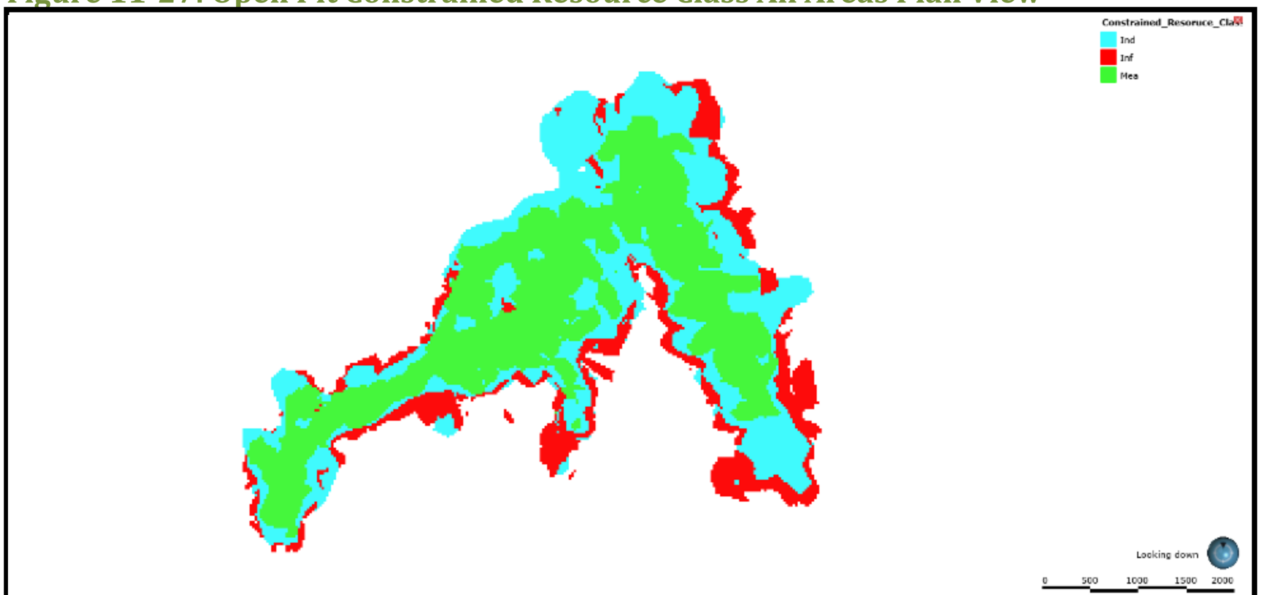
Table 11-14: Open Pit Parameters for Resource Class Numeric Indicator Model

Resource Class	Shape	Interpolant Distance	ISO

Measured	Isotropic	250	0.4
Indicated	Isotropic	250	0.4
Inferred	NA	NA	NA

Because the classification was performed across all resource estimation domains, an isotropic search was used along with an interpolant distance of 250, which was based on the average continuity of grade seen in the deposit. No numeric indicator model was constructed for the inferred resource class, rather it was defined as any block with a calculated gold grade that did not fall within the measured or indicated numeric indicator domains that had a calculated gold grade. A plan view of the estimated resource classes is shown in Figure 11-27.

Figure 11-27: Open Pit Constrained Resource Class All Areas Plan View



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CONSTRAINED RESOURCE CLASS

11.6.3 Mineral Resource Statement

S-K 1300 (CFR, 2018) defines a mineral resource as: "a concentration or occurrence of material of economic interest in or on the Earth's crust in such form, grade or quality, and quantity that there are reasonable prospects for economic extraction. A mineral resource is a reasonable estimate of mineralization, taking into account relevant factors such as cut-off grade, likely mining dimensions, location or continuity, that, with the assumed and justifiable technical and economic conditions, is like to, in whole or in part, become economically extractable…". The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic, and other factors. Mineral resources are not mineral reserves and do not have demonstrated economic viability. As a result, no mineral reserves have been estimated as part of this study. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

The requirement, "reasonable prospects for economic extraction," generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at a cutoff grade considering appropriate extraction scenarios and processing recoveries. To meet this requirement, GRE considered that major portions of the Granite Creek deposit are amenable for open pit extraction.

To determine the quantities of material offering "reasonable prospects for economic extraction" by an open pit, GRE constructed open pit scenarios developed from the resource block model estimate using Whittle's Lerchs-Grossman miner "Pit Optimizer" software. Reasonable mining assumptions were applied to evaluate the portions of the block model (Measured, Indicated, and Inferred blocks) that could be "reasonably expected" to be mined from an open pit. The optimization parameters presented in Table 11-15 were selected based on experience and benchmarking against similar projects. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cutoff grade. GRE considers that the blocks located within the resulting conceptual pit envelope show "reasonable prospects for economic extraction" and can be reported as a mineral resource.

Table 11-15: Granite Creek Resource Parameters for Open Pit Optimization

Parameter	Items	Unit	Value
Costs	Mining Cost (waste/mineralized material)	$/tonne mined	2.46
	Heap Leach[1]	$/tonne mineralized material treated	9.04
	Carbon in Leach[2]	$/tonne mineralized material treated	17.22
Recovery	Heap Leach (HLCH) Recovery with CN Solubility <60	%	CN Solubility*100
	HLCH Recovery with CN Solubility >= 60	%	((0.1225 * [Au_ppm]) + 0.4164)*100

Parameter	Items	Unit	Value
	CIL Recovery	%	((0.5388 * CN Solubility) + 0.3201)*100
Net revenue gold	Gold price[3]	$/oz	2,040
	Selling costs and penalties[4]	$/oz	114
Royalty	Total royalty (simplified)	%	6.00%
Slope angles	Slope Angle	degrees	41
Limits	HLCH	tonnes per year	2,975,000
	CIL	tonnes per year	1,050,000

1 HLCH and CIL costs include $1.56/tonne milled for admin costs.
2 Various royalties are applicable at various points throughout the mine life, however for the scope of this iA, GRE has used a single 6% royalty for the open pit mineral resource.
3 The gold price used for this analysis is the 36-month trailing average gold price as of December 31, 2024.
4 This selling cost is used to apply the 6% royalty

Due to the large ratio of deposit size to block size and method of grade estimation, the grade model is fully diluted, and the resource is 100% recoverable as estimated.

The Granite Creek open pit mineral resource constrained by a Whittle pit shell that corresponds to a gold price of $2,040 per troy ounce is shown in Table 11-16. The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are presently no mineral reserves for the project.

Table 11-16: Granite Creek Open Pit Mineral Resource

Class	Zone	Total Process Material (1000s Tonnes)	Total Process Material (1000s Tons)	Au Grade (g/t)	Au Grade (opt)	Total Contained Au (10000s t. oz)
Measured	Pit B	2,910	3,207	1.32	0.042	123.41
	Pit A	563	620	1.07	0.034	19.30
	CX	10,889	12,003	1.30	0.042	455.27
	MAG	12,000	13,228	1.21	0.039	467.97
	Total	**26,362**	29,059	**1.26**	0.040	**1,065.95**
Indicated	Pit B	360	397	1.10	0.035	12.73
	Pit A	689	760	0.80	0.026	17.78
	CX	2,973	3,277	1.25	0.040	119.62

Class	Zone	Total Process Material (1000s Tonnes)	Total Process Material (1000s Tons)	Au Grade (g/t)	Au Grade (opt)	Total Contained Au (10000s t. oz)
	MAG	7,317	8,066	0.93	0.030	219.16
	Total	**11,339**	12,499	**1.01**	0.033	**369.29**
Measured + Indicated	Pit B	3,270	3,604	1.29	0.042	136.14
	Pit A	1,252	1,380	0.92	0.030	37.08
	CX	13,862	15,280	1.29	0.041	574.89
	MAG	19,317	21,293	1.11	0.036	687.13
	Total	**37,701**	41,558	**1.18**	0.038	**1,435.24**
Inferred	Pit B	32	36	0.64	0.021	0.67
	Pit A	205	226	0.59	0.019	3.88
	CX	1,347	1,485	1.16	0.037	50.24
	MAG	563	620	1.11	0.036	20.17
	Total	**2,148**	2,367	**1.09**	0.035	**74.95**

1) The effective date of the Mineral Resources Estimate is December 31, 2024.

2) The Qualified Person for the estimate is GRE.

3) Mineral resources are not ore reserves and are not demonstrably economically recoverable.

4) Mineral resources are reported at a 0.30 g/t cutoff, an assumed gold price of 2,040 $/tr. oz, using variable recovery, a slope angle of 41 degrees, 6% royalty, heap leach processing cost $9.04 per tonne (includes admin), CIL processing cost of $17.22 per tonne (includes admin).

11.6.4 Calculation of Cutoff Grade

The cutoff grade of 0.30 ppm used for the Mineral Resource Statement was calculated as the maximum of the following:

Heap leach cutoff grade:

Process & G&A Cost:	$9.04
Cost @ 70% Recovery (average heap leach recovery):	$12.91/tonne
Au Price:	$2,040/Au oz
Economic Cutoff grade	$12.91/$2040 = $0.00633 oz/tonne

= 0.20 g/tonne or ppm

CIL cutoff grade:

Process & G&A Cost:	\$17.22
Cost @ 85% Recovery (average CIL recovery):	\$20.85/tonne
Au Price:	\$2,040/Au oz
Economic Cutoff grade	\$20.85/\$2040 = \$0.010 oz/tonne
	= 0.30 g/tonne or ppm

These calculated cutoff grades represent marginal cutoff grades. Mining costs are not included because they are applied during pit optimization.

11.6.5 Sources of Uncertainty

Sources of uncertainty in the mineral resource estimate are described below.

a) Data: As noted in Section 11.2, a number of negative, missing, and blank assay values exist in the drill hole data files. These values were replaced as shown in Table 11-1. GRE followed standard practice in this regard; however, these intervals represent some uncertainty but are believed to not have a significant impact on the resource estimate.
b) Geologic Model: The geologic model was prepared by a QP geologist and represents GRE's understanding of the project geology as of the effective date of this report; however, the geologic model may evolve as more data becomes available. This may have an impact on the resource estimation in the future.
c) Classification Criteria: GRE followed standard practice in determining Measured, Indicated, and Inferred resources; however, the interpretations may evolve as more data becomes available.
d) Grade Interpolation: GRE interpolated grades following completion of a variography analysis of the data. The variogram ellipsoid directions and ranges could evolve as more data becomes available, which could impact the resource estimation in the future.
e) Parameters for Open Pit Optimization: GRE used mining, processing, G&A, royalty, and selling costs from the 2021 Technical Report (GRE, 2021) and the 36-month trailing average gold price as of December 31, 2024. Changes in any of the values, for example from potential tariffs and trade wars increasing operating costs, would alter the resulting Lerchs-Grosman pit shell and the resource reporting within that pit shell.
f) Cutoff Grade Calculation: GRE used processing and G&A costs, average royalty, and average recoveries from the 2021 Technical Report (GRE, 2021) to calculate economic cutoff grades. Changes in any of the values would alter the cutoff grade.

g) Metal Price: Use of the 36-month trailing average gold price is deemed reasonable. It is likely that acceptable gold prices will change over time and could be influenced by shifts in the economy.

11.6.6 Mineral Resource Sensitivity

Table 11-17 shows the sensitivity of the mineral resource to cutoff grade in each domain.

Table 11-17: Granite Creek Mineral Resource Sensitivity to Cutoff Grade

Deposit	Cutoff Grade (ppm)	Mass (1000sn tonnes)	Mass (1000s tons)	Au Grade (g/t)	Au Grade (opt)	Au Contained (million tr oz)
Pit B	0.1	4,655	5,132	0.89	0.029	0.134
	0.15	3,994	4,402	1.02	0.033	0.131
	0.2	3,543	3,905	1.13	0.036	0.128
	0.25	3,201	3,529	1.22	0.039	0.126
	0.3	**2,910**	**3,207**	**1.32**	**0.042**	**0.123**
	0.35	2,662	2,935	1.41	0.045	0.121
	0.4	2,476	2,729	1.49	0.048	0.119
	0.45	2,283	2,517	1.58	0.051	0.116
	0.5	2,098	2,312	1.68	0.054	0.113
Pit A	0.1	1,052	1,160	0.66	0.021	0.022
	0.15	890	981	0.75	0.024	0.022
	0.2	762	839	0.85	0.027	0.021
	0.25	646	712	0.96	0.031	0.020
	0.3	**563**	**620**	**1.07**	**0.034**	**0.019**
	0.35	486	536	1.18	0.038	0.019
	0.4	424	467	1.30	0.042	0.018
	0.45	370	407	1.43	0.046	0.017
	0.5	320	353	1.58	0.051	0.016
CX	0.1	17,873	19,702	0.86	0.028	0.495
	0.15	14,942	16,470	1.01	0.032	0.483
	0.2	13,162	14,508	1.12	0.036	0.473
	0.25	11,837	13,048	1.22	0.039	0.464
	0.3	**10,889**	**12,003**	**1.30**	**0.042**	**0.455**

Deposit	Cutoff Grade (ppm)	Mass (1000sn tonnes)	Mass (1000s tons)	Au Grade (g/t)	Au Grade (opt)	Au Contained (million tr oz)
	0.35	10,083	11,114	1.38	0.044	0.447
	0.4	9,401	10,363	1.45	0.047	0.439
	0.45	8,787	9,686	1.52	0.049	0.430
	0.5	8,208	9,048	1.60	0.051	0.421
Mag	0.1	16,755	18,469	0.92	0.030	0.497
	0.15	15,088	16,631	1.01	0.032	0.490
	0.2	14,006	15,439	1.07	0.035	0.484
	0.25	12,970	14,297	1.14	0.037	0.477
	0.3	**12,000**	**13,228**	**1.21**	**0.039**	**0.468**
	0.35	11,133	12,272	1.28	0.041	0.459
	0.4	10,263	11,313	1.36	0.044	0.448
	0.45	9,472	10,442	1.44	0.046	0.438
	0.5	8,807	9,708	1.51	0.049	0.427
Pit B	0.1	1,291	1,423	0.44	0.014	0.018
	0.15	931	1,027	0.56	0.018	0.017
	0.2	700	772	0.68	0.022	0.015
	0.25	499	550	0.87	0.028	0.014
	0.3	**360**	**397**	**1.10**	**0.035**	**0.013**
	0.35	284	313	1.31	0.042	0.012
	0.4	229	252	1.53	0.049	0.011
	0.45	208	229	1.64	0.053	0.011
	0.5	183	201	1.81	0.058	0.011
Pit A	0.1	1,076	1,186	0.58	0.019	0.020
	0.15	936	1,032	0.65	0.021	0.019
	0.2	821	905	0.71	0.023	0.019
	0.25	754	831	0.76	0.024	0.018
	0.3	**689**	**760**	**0.80**	**0.026**	**0.018**
	0.35	618	681	0.86	0.028	0.017
	0.4	535	590	0.93	0.030	0.016
	0.45	467	514	1.01	0.032	0.015

Deposit	Cutoff Grade (ppm)	Mass (1000sn tonnes)	Mass (1000s tons)	Au Grade (g/t)	Au Grade (opt)	Au Contained (million tr oz)
	0.5	411	453	1.08	0.035	0.014
CX	0.1	6,222	6,858	0.69	0.022	0.137
	0.15	4,800	5,291	0.85	0.027	0.132
	0.2	3,898	4,297	1.01	0.033	0.127
	0.25	3,301	3,639	1.15	0.037	0.123
	0.3	**2,973**	**3,277**	**1.25**	**0.040**	**0.120**
	0.35	2,740	3,020	1.33	0.043	0.117
	0.4	2,532	2,791	1.41	0.045	0.115
	0.45	2,329	2,567	1.49	0.048	0.112
	0.5	2,188	2,412	1.56	0.050	0.110
Mag	0.1	11,982	13,208	0.64	0.021	0.247
	0.15	10,312	11,367	0.72	0.023	0.240
	0.2	9,142	10,078	0.79	0.026	0.234
	0.25	8,161	8,996	0.86	0.028	0.227
	0.3	**7,317**	**8,066**	**0.93**	**0.030**	**0.219**
	0.35	6,584	7,257	1.00	0.032	0.212
	0.4	5,939	6,546	1.07	0.034	0.204
	0.45	5,260	5,799	1.15	0.037	0.194
	0.5	4,616	5,089	1.24	0.040	0.185
Pit B	0.1	61	68	0.41	0.013	0.001
	0.15	44	48	0.54	0.017	0.001
	0.2	42	46	0.55	0.018	0.001
	0.25	38	42	0.58	0.019	0.001
	0.3	**32**	**36**	**0.64**	**0.021**	**0.001**
	0.35	25	27	0.74	0.024	0.001
	0.4	23	25	0.77	0.025	0.001
	0.45	23	25	0.77	0.025	0.001
	0.5	21	23	0.80	0.026	0.001
Pit A	0.1	457	503	0.37	0.012	0.005
	0.15	376	414	0.42	0.014	0.005
	0.2	327	360	0.46	0.015	0.005
	0.25	263	290	0.52	0.017	0.004
	0.3	**205**	**226**	**0.59**	**0.019**	**0.004**

Deposit	Cutoff Grade (ppm)	Mass (1000sn tonnes)	Mass (1000s tons)	Au Grade (g/t)	Au Grade (opt)	Au Contained (million tr oz)
	0.35	175	193	0.63	0.020	0.004
	0.4	145	160	0.69	0.022	0.003
	0.45	123	135	0.74	0.024	0.003
	0.5	105	115	0.78	0.025	0.003
CX	0.1	2,694	2,969	0.66	0.021	0.058
	0.15	2,093	2,307	0.82	0.026	0.055
	0.2	1,701	1,875	0.97	0.031	0.053
	0.25	1,491	1,643	1.07	0.035	0.051
	0.3	**1,347**	**1,485**	**1.16**	**0.037**	**0.050**
	0.35	1,221	1,346	1.25	0.040	0.049
	0.4	1,127	1,243	1.32	0.042	0.048
	0.45	1,055	1,163	1.38	0.044	0.047
	0.5	984	1,085	1.44	0.046	0.046
Mag	0.1	1,486	1,638	0.54	0.018	0.026
	0.15	1,243	1,370	0.63	0.020	0.025
	0.2	1,031	1,137	0.72	0.023	0.024
	0.25	781	861	0.88	0.028	0.022
	0.3	**563**	**620**	**1.11**	**0.036**	**0.020**
	0.35	488	538	1.24	0.040	0.019
	0.4	444	489	1.32	0.042	0.019
	0.45	430	474	1.35	0.043	0.019
	0.5	425	469	1.36	0.044	0.019

11.7 Underground Mineral Resources

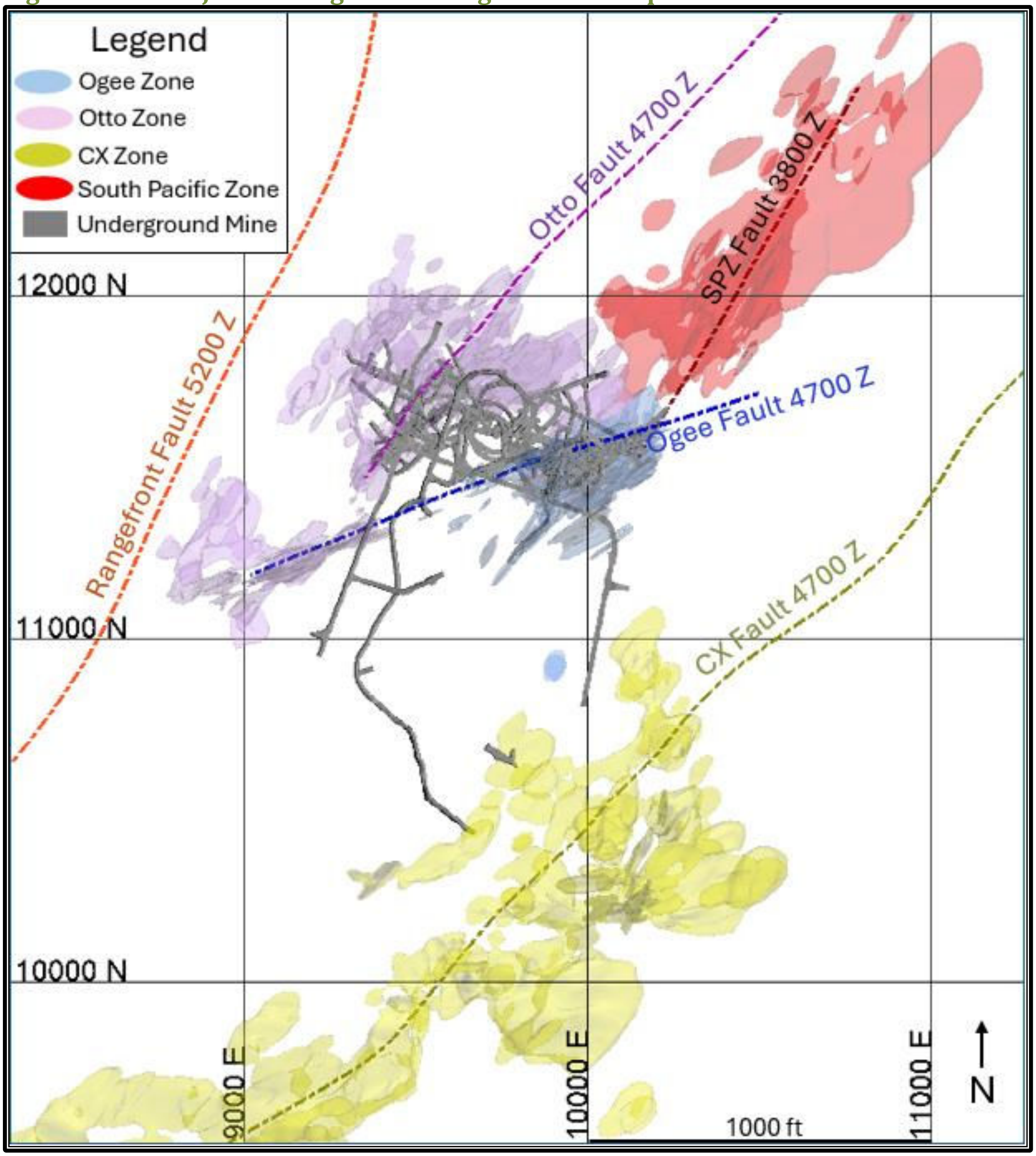
Practical Mining LLC estimated the Granite Creek underground Mineral Resource using all drilling and geological data available through March 29, 2023.

11.7.1 Structural and Mineralized Grade Shell Modelling

The Granite Creek structural model includes 11 major and 46 minor faults. Underground mineralization is controlled by seven of the major faults. The range front fault separates the cretaceous granodiorite to the northwest from the Ordovician Upper and Lower Comus formation. The underground mineralization is hosted almost entirely within the Lower Comus.

Underground mineralization is contained within the fault zones and strikes north easterly with sub vertical dip to the southeast. It is subdivided into the CX, Otto, Ogee and South Pacific Zones. The zones are defined by 0.10 opt grade shells trending parallel to the fault orientations (Figure 11-28).

Figure 11-28 Major Faulting and Underground 0.10 opt Grade Shells



11.7.1.1 Drill Data and Compositing

1.1.6.1.1. Drill Data Set

The drilling data set within the bock model boundary consists of 2,346 drillholes. Of these 93 were excluded due to collar location discrepancies, downhole survey errors or suspected downhole contamination. Table 11-18 summarizes the drilling within the block model extents by operator and hole type.

Table 11-18 Summary of Drilling Within Block Model Extents

COMPANY	Count	Hole Type	Length (ft)
Atna	55	Core	19,730
Atna	197	RC	47,890
Atna	51	RC/Core	44,886
Barrick	1	Rotary	1,000
Barrick	1	RC/Rotary	1,940
Barrick	44	RC/Core	49,532
Barrick	71	Core	30,843
Barrick	34	RC	24,455
Barrick	1	Monitor	1,340
PMC	316	Rotary	99,315
PMC	15	Monitor	8,635
PMC	1,151	RC	505,759
PMC	28	RC/Core	45,748
PMC	4	Unknown	1,797
PMC	5	Core	5,205
i-80	50	RC/Core	70,028
i-80	184	Core	87,159
i-80	16	RC	2,595
Unknown	15	Monitor	7,415
Unknown	2	RC	465
Unknown	4	Pump	3,740
Unknown	5	Core	0
Unknown	3	Met	1,535
Total	2,253		1,061,012

11.7.1.2 Compositing

Drill holes were composited into ten-foot lengths starting and ending with the 0.004 or 0.10 opt grade shells. The grade shells act as hard boundaries and only composites within the shell are used to estimate grades within a particular shell.

11.7.1.3 Statistics

Univariate statistics were calculated for the 0.004 and 0.20 Au opt grade shell. The results are shown in Table 11-19 and Figure 11-29 and Figure 11-30.

Table 11-19 Composite Statistics

Shell	# Comps	Min	Max	Mean	Std Dev	CV
au004	16452	0.0001	19.83	0.030	0.208	6.861
au1	2631	0.0001	3.780	0.317	0.334	1.054

Figure 11-29 Histogram of 0.004 Au opt Composites

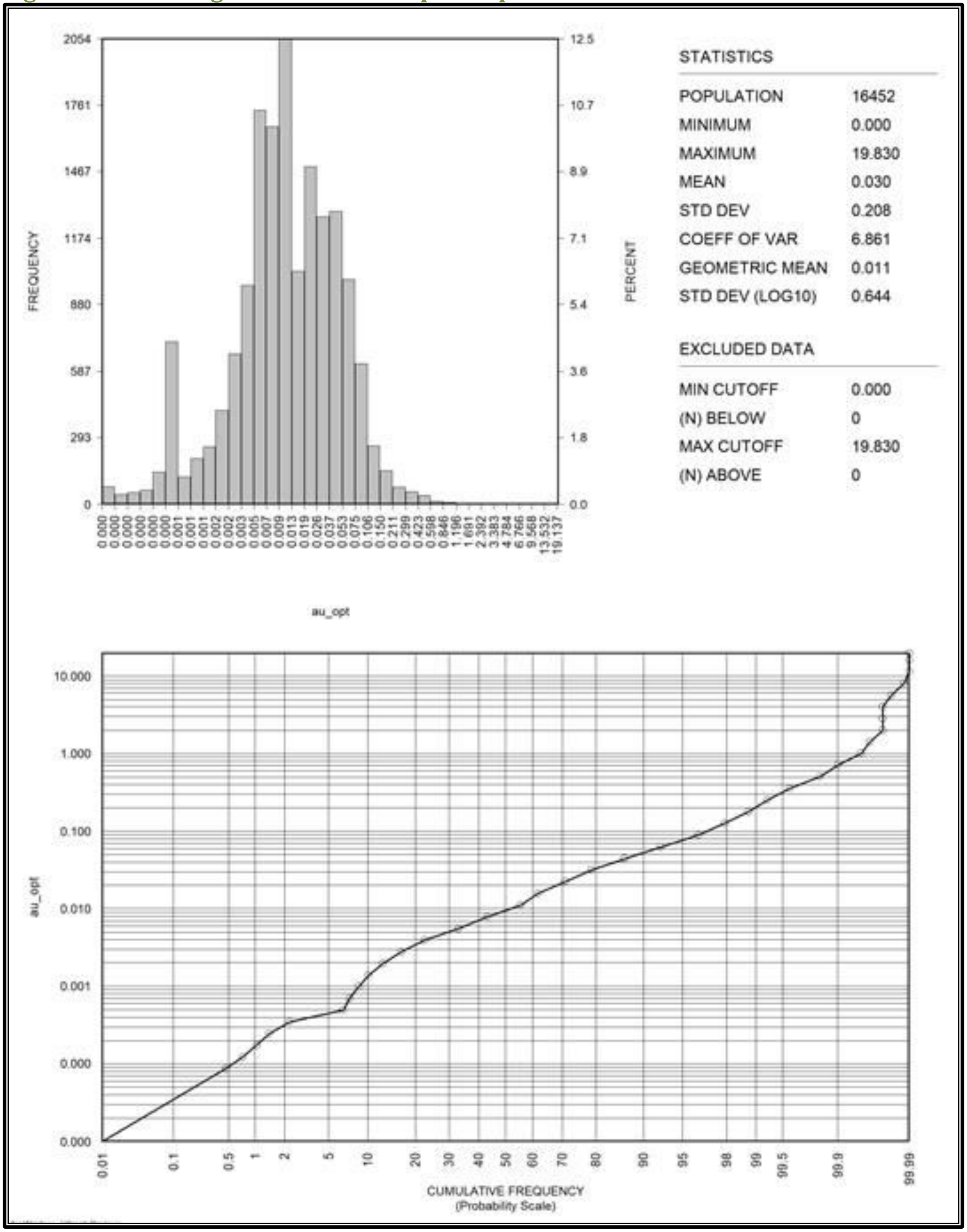
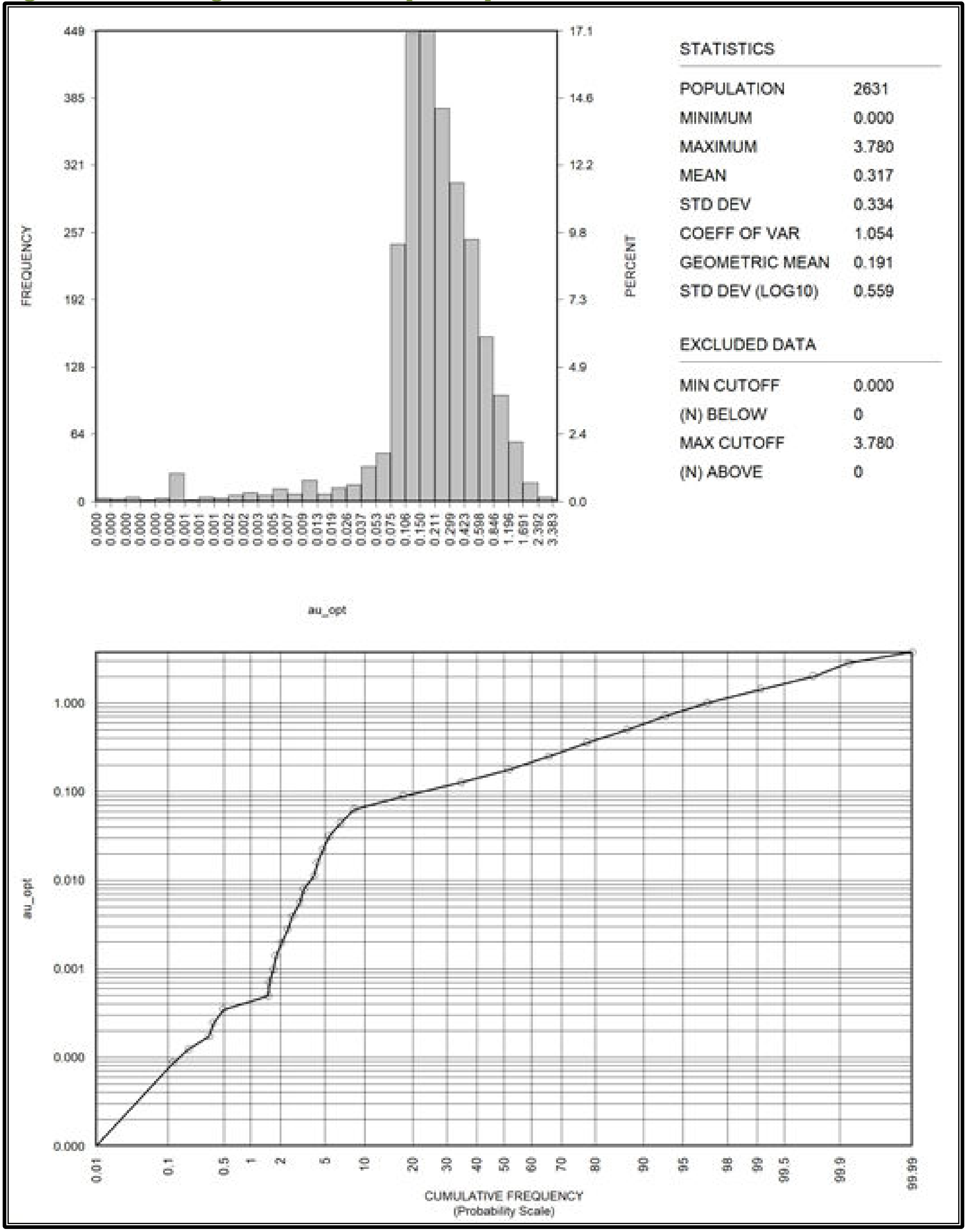


Figure 11-30 Histogram of 0.10 Au opt Composites



11.7.1.4 Density

The drill database contains 419 density measurements collected by past operators of the Granite Creek Project. These values are summarized by lithology in Table 11-20.

Table 11-20 Density Values Used in the Underground Model

Lithologic Unit	Density (tons/ft^3)
Qal	0.0578
Cp	0.0826
Ocl	0.0820
Ocu	0.0814
Kgd	0.0819

11.7.1.5 Block Model

Block model blocks are 20x20x20 feet with sub-blocking to 5x5x5 feet at the 0.004 and 0.10 Au opt grade shells. The model extends across the Ogee, Otto, South Pacific and CX zones.

11.7.1.6 Grade Capping

Grade capping as applied to the 0.10 and 0.004 Au opt composites. Composite grades exceeding the cap value within the 0.004 Au opt grade shells are restricted for use only within the 20x20x20 foot block that contains the capped composite and it is not used to estimate any grades beyond the containing block. For the 0,10 opt grade shells the capped composite is constrained for use within the 10x10x10 foot block. Cap Grades are listed in Table 11-21.

Table 11-21 Underground Grade Capping Values

Grade Shell	Cap Grade Au opt	# Comps Affected
0.004	0.1	248
0.10	1.32	48

11.7.1.7 Grade Estimation and Resource Classification

Block grades were estimated using the Inverse Distance Cubed (ID^3) method. Anisotropic search parameters were set to the general orientation of the grade shells as shown in Table 11-22.

The required number of composites to classify a block as mineral resource must come from at least two drill holes. A block cannot be classified on the basis of only one drill hole. The mineral resource classification parameters are shown in Table 11-23.

Table 11-22 Ellipsoid Search Parameters

Shell	Bearing		Plunge	Dip
ogee004	67		0	-90
otto004	40		0	-63
spz004	33		0	-56
cx004_01	52		0	-40
cx004_02	0		0	0
cx004_03	0		0	0
cx004_04	30		0	0
cx004_05	90		0	0
cx004_06	90		0	-25
cx004_07	90		0	-28
cx004_08	90		0	-15
cx004_09	90		0	-50
cx004_10	90		0	-55
au1_02	0		0	0
cx1_01	52		0	-42
cx1_02	58		-22	-21
cx1_03	90		-52	-35
cx1_04	90		0	-52
cx1_05	90		0	-70
cx1_06	90		0	-53
cx1_07	90		0	0
cx1_08	0		0	0
cx1_09	90		0	-55
cx1_10	0		0	-13
cx1_11	90		0	-57
cx1_12	90		0	-68
cx1_13	90		0	0
cx1_14	90		0	-15
cx1_15	90		0	-27
cx1_16	90		0	-37
ogee1	67		0	90
otto1	40		0	-63
spz1	33		0	-56

Table 11-23 Resource Classification Parameters

Class	Major (ft)	Semi (ft)	Minor (ft)	Min Samp	Max Samp	Min. DH
meas	75	75	37.5	8	16	2

ind	150	150	75	6	12	2
inf	300	300	150	4	12	2

11.7.1.8 Mined Depletion and Sterilization

The fraction of Blocks that intersect the mined volume survey or surface topography is calculated and the block tonnage adjusted accordingly.

11.7.2 Model Validation

11.7.2.1 Visual Comparison

Cross sections showing modelled block grades and drill hole composites provide a visual comparison in a localized area. Comparative cross sections showing blocks greater than 0.10 Au opt are shown in Figure 11-31 through Figure 11-33.

Figure 11-31 Comparative Cross Section Through Otto and Ogee Zones

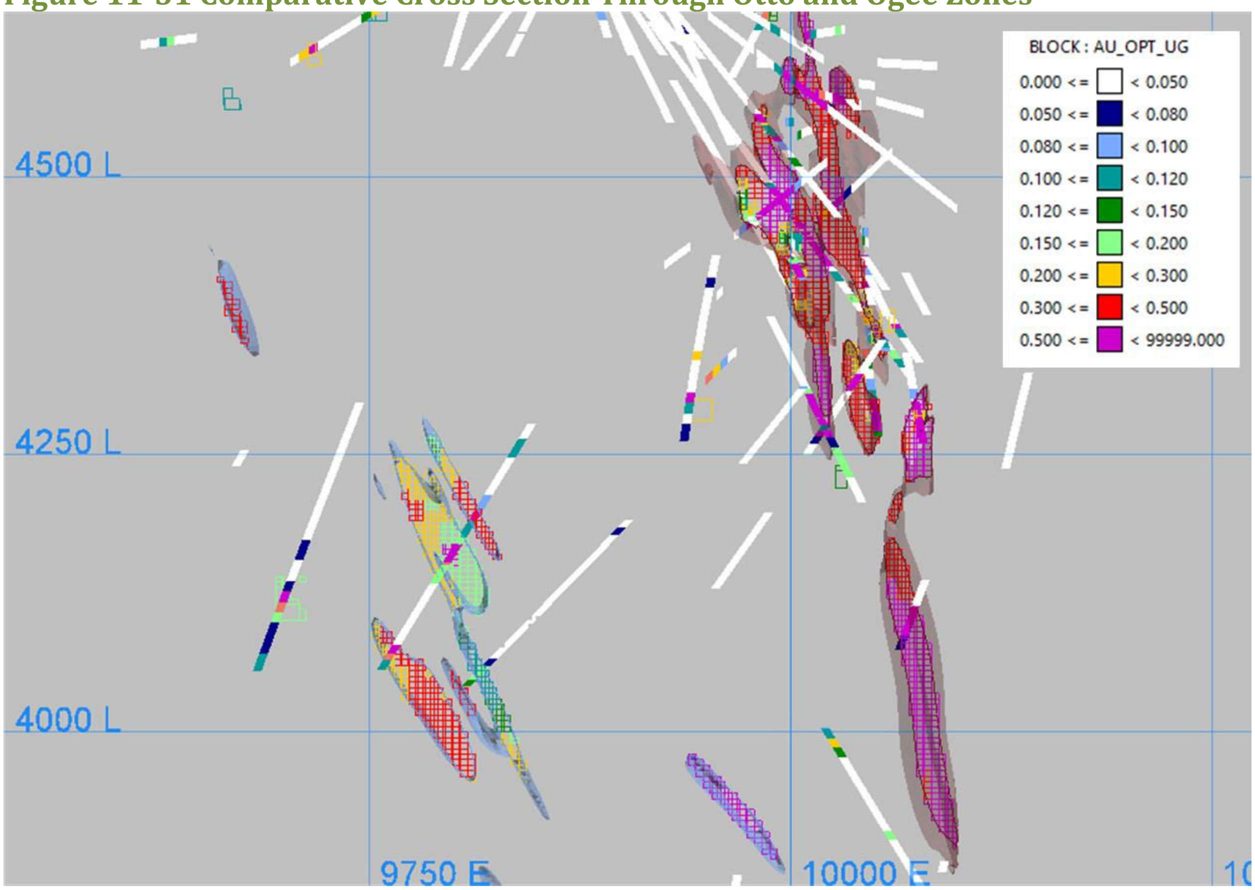


Figure 11-32 Comparative Cross Section Through the CX Zone

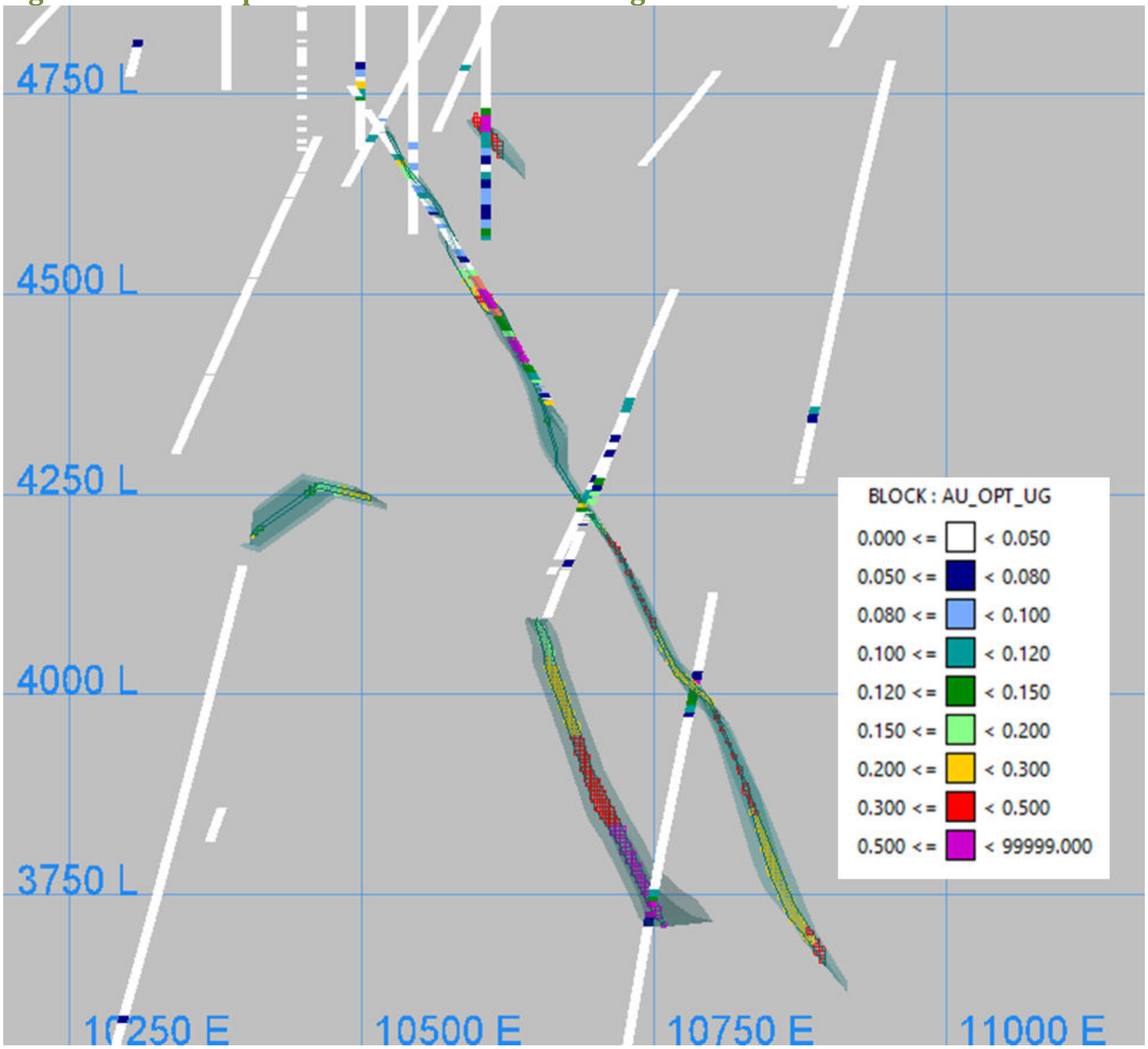
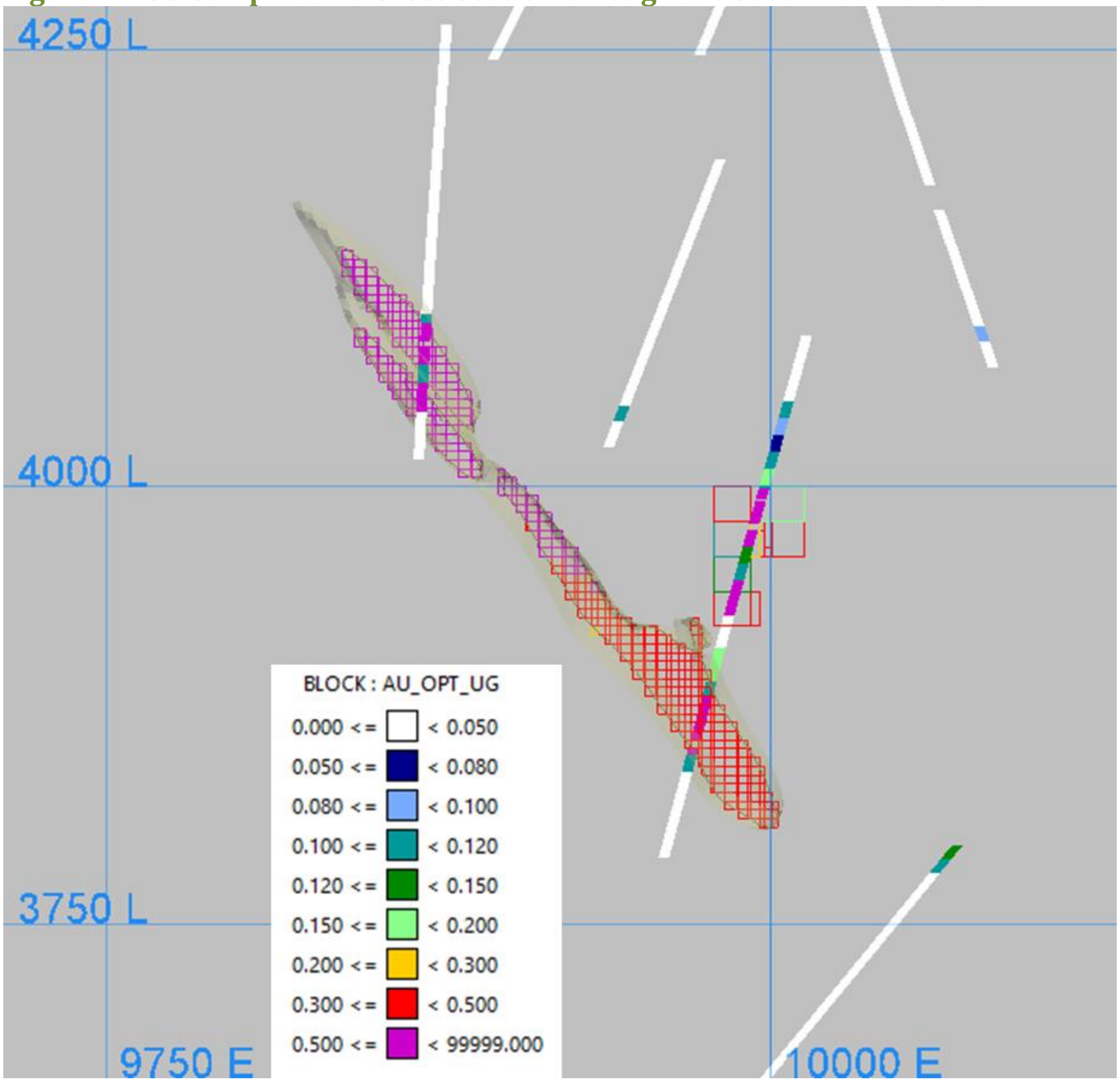


Figure 11-33 Comparative Cross Section through the South Pacific Zone



11.7.2.2 Drift Analysis

Drift analysis or swath plots graphically compare drill hole composite grades to model grades in a specified slice direction and thickness across the modelled extents. Modelled grades should closely follow drilling composite grades. Drift analysis for 100-foot slices are presented in Figure 11-34 and Figure 11-35.

Figure 11-34 Easterly Drift Analysis

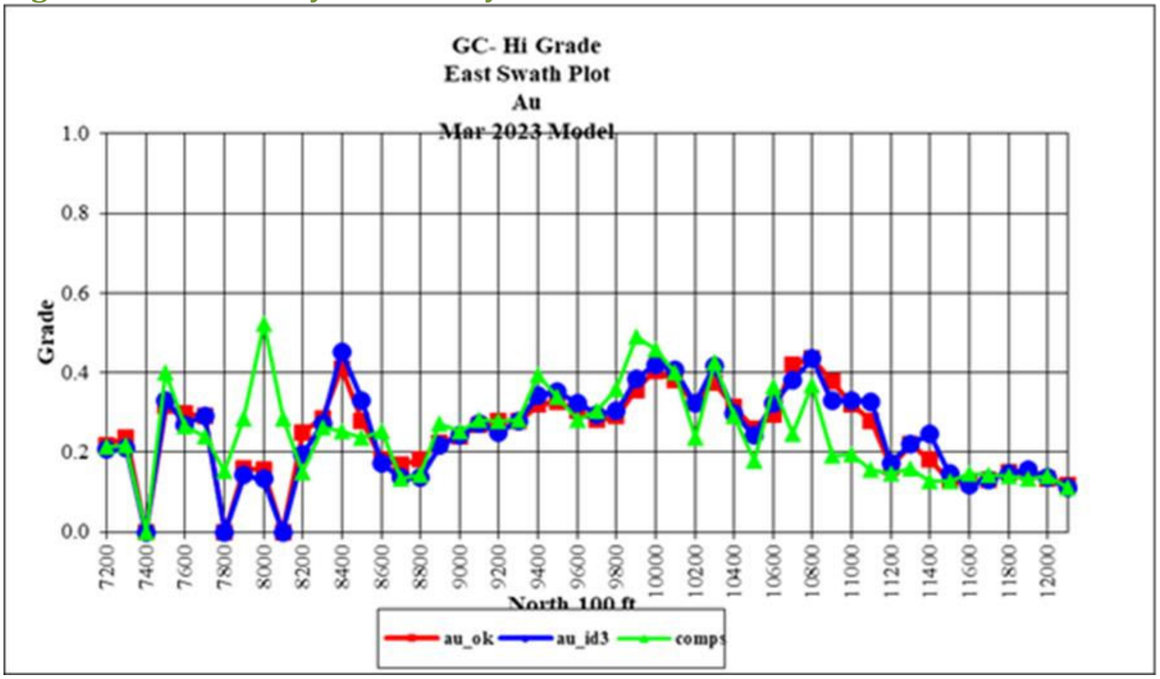
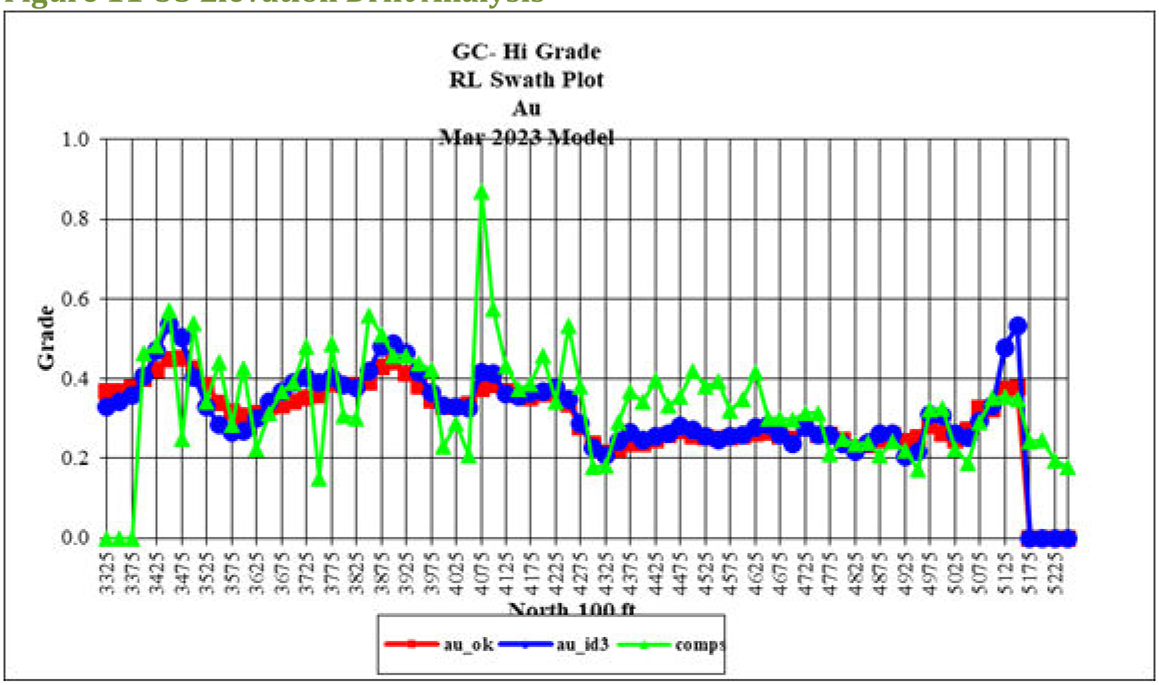


Figure 11-35 Elevation Drift Analysis



11.7.2.3 Reconciliation

i-80 splits the heading survey at the recorded face distance for each round mined. These then correspond to muck assays and grade control tracking. All mineralization determined to be refractory with muck samples above 0.058 opt and oxide material above 0.075 opt is sent to NGM for processing. Low grade oxide material between 0.020 and 0.075 is sent to the Lone Tree heap leach facility. (See Section 13.2.4).

Table 11-24 show the material sent to each process by month in 2024 and Table 11-25 shows the corresponding modeled high grade mineralization contained within the mined volumes and the monthly variance. The monthly ounce variance is shown in Figure 11-36 and the cumulative percentage variance is shown in Figure 11-37.

High grade mill to high grade model reconciliation for all of 2024 shows a 55% increase in tons with a 46% decrease in grade and 16% decrease in ounces. The underperformance of the process with respect to grade and ounces is due to both planned dilution and overbreak. Some high grade mineralization was mixed with low grade mineralization.

Table 11-24 Mineralization Processed in 2024

	Refractory Autoclave			Oxide Carbon in Leach			High Grade Mill			Lone Tree Heap Leach		
	Tons	Au opt	Au oz	Tons	Au opt	Au oz	Tons	Au opt	Au oz	Tons	Au opt	Au oz
Jan	6,812	0.298	2,027	3,778	0.255	963	10,590	0.282	2,990	9,804	0.109	1,069
Feb	11,135	0.238	2,655	3,294	0.195	642	14,429	0.228	3,297	5,663	0.110	623
Mar	4,633	0.185	856	3,062	0.310	948	7,695	0.234	1,804	3,783	0.097	365
Apr	2,291	0.154	352	2,040	0.278	567	4,331	0.212	919	7,592	0.100	757
May	726	0.130	94	2,067	0.226	467	2,793	0.201	561	11,190	0.102	1,140
Jun	939	0.235	221	5,019	0.251	1,260	5,958	0.249	1,481	18,201	0.115	2,084
Jul	2,641	0.183	483	5,978	0.243	1,453	8,619	0.225	1,935	24,069	0.084	2,027
Aug	1,928	0.126	244	3,803	0.251	955	5,731	0.209	1,198	25,791	0.109	2,810
Sep	3,815	0.261	994	5,025	0.211	1,060	8,840	0.232	2,054			
Oct	4,338	0.214	928	5,036	0.230	1,158	9,374	0.223	2,086			
Nov				3,342	0.276	922	3,342	0.276	922			
Dec				4,853	0.243	1,179	4,853	0.243	1,179			
Total	**39,257**	**0.225**	**8,852**	**47,297**	**0.245**	**11,575**	**86,553**	0.236	20,427	**106,093**	**0.103**	**10,875**

Table 11-25 2024 High Grade Block Model Predicted and High Grade Mill - Model Variance

	Block Model			High Grade Mill – Model Variance			Percentage Variance		
	Tons	Au opt	Au oz	Tons	Au opt	Au oz	Tons	Au opt	Au oz
Jan	6,512	0.500	3,253	4,078	(0.217)	(263)	63%	-43%	-8%
Feb	2,676	0.312	834	11,753	(0.083)	2,462	439%	-27%	295%
Mar	6,214	0.346	2,148	1,480	(0.111)	(343)	24%	-32%	-16%
Apr	1,848	0.329	607	2,483	(0.116)	312	134%	-35%	51%
May	3,220	0.369	1,188	(427)	(0.168)	(626)	-13%	-46%	-53%
Jun	5,525	0.509	2,810	432	(0.260)	(1,329)	8%	-51%	-47%
Jul	3,717	0.668	2,482	4,901	(0.443)	(547)	132%	-66%	-22%
Aug	5,002	0.458	2,291	729	(0.249)	(1,092)	15%	-54%	-48%
Sep	5,686	0.330	1,878	3,153	(0.098)	176	55%	-30%	9%
Oct	3,177	0.526	1,672	6,198	(0.304)	415	195%	-58%	25%
Nov	5,658	0.410	2,319	(2,316)	(0.134)	(1,397)	-41%	-33%	-60%
Dec	6,554	0.430	2,815	(1,701)	(0.187)	(1,636)	-26%	-43%	-58%
Total	55,789	0.435	24,296	30,764	(0.200)	(3,870)	55%	-46%	-16%

Figure 11-36 Monthly High Grade Mill to Model Au Ounce Variance in 2024

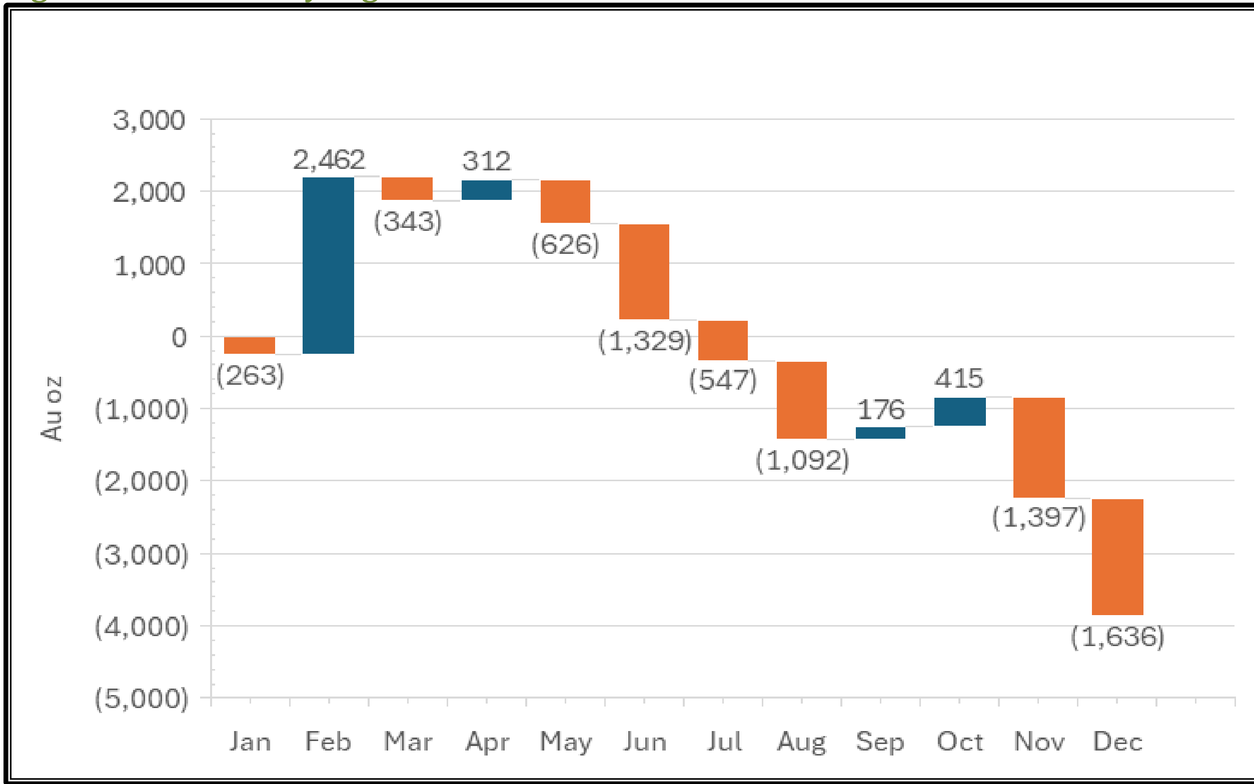
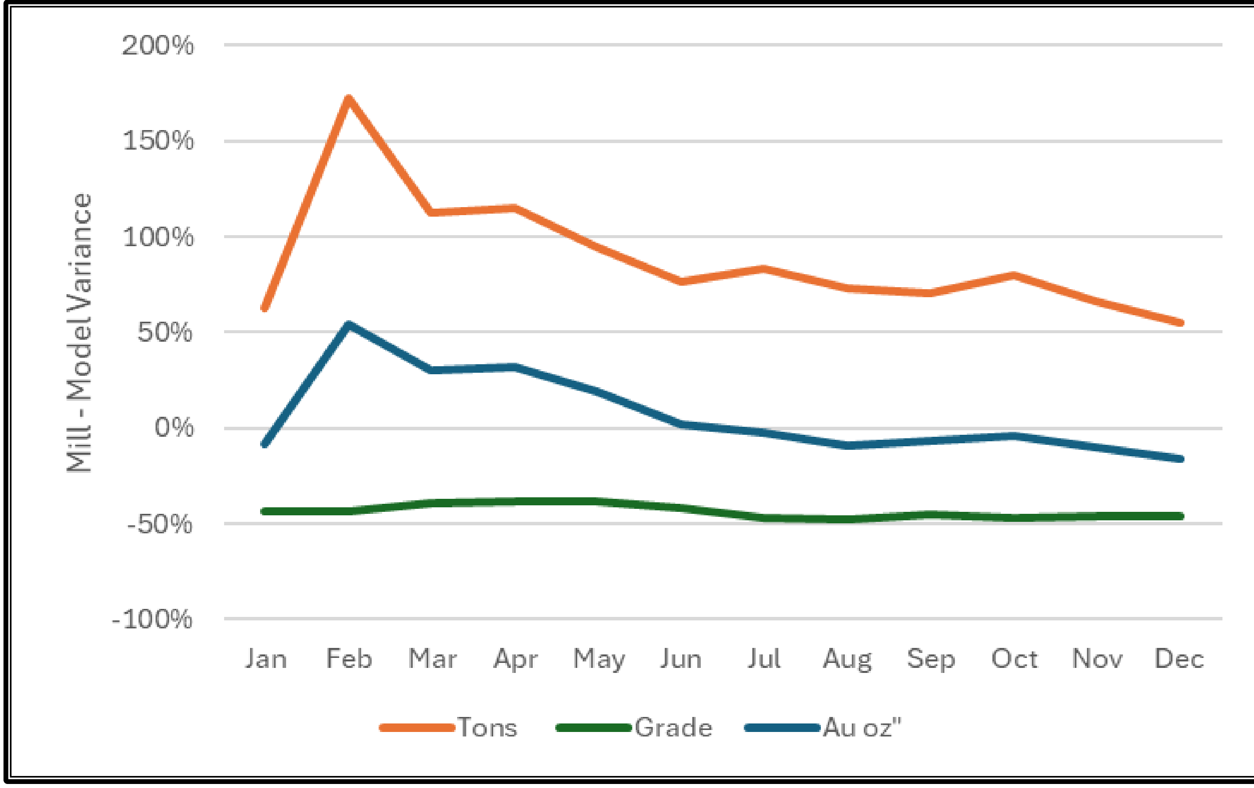


Figure 11-37: 2024 Cumulative High Grade Mill to Model Variance



Reconciliation of all processed material to the model is shown in Table 11-26 and graphically in Figure 11-38 and Figure 11-39. Compared to the high grade reconciliation the model performed much better, processed tons were 5% higher than modeled and processed grade was 15% higher. The model underestimated ounces by 19%.

These results indicate that:

- High grade mineralization is mixed with low grade mineralization, and;
- A significant amount of low grade mineralization is being mined beyond the model limits.

Table 11-26 All Block Model Predicted and Mill - Model Variance in 2024

	Block Model			All Mill – Model Variance			Percentage Variance		
	Tons	Au opt	Au oz	Tons	Au opt	Au oz	Tons	Au opt	Au oz
Jan	19,687	0.174	3,417	707	0.025	641	4%	15%	19%
Feb	9,791	0.097	949	10,302	0.098	2,970	105%	101%	313%
Mar	13,733	0.166	2,278	(2,256)	0.023	(109)	-16%	14%	-5%
Apr	10,304	0.071	728	1,619	0.070	948	16%	99%	130%
May	11,772	0.133	1,562	2,210	(0.011)	139	19%	-8%	9%
Jun	17,233	0.179	3,091	6,925	(0.032)	474	40%	-18%	15%

	Block Model			All Mill – Model Variance			Percentage Variance		
	Tons	Au opt	Au oz	Tons	Au opt	Au oz	Tons	Au opt	Au oz
Jul	15,558	0.174	2,705	17,130	(0.053)	1,257	110%	-30%	46%
Aug	16,503	0.154	2,538	15,019	(0.027)	1,470	91%	-17%	58%
Sep	14,063	0.154	2,172	(5,223)	0.078	(118)	-37%	50%	-5%
Oct	9,958	0.178	1,772	(583)	0.045	314	-6%	25%	18%
Nov	15,094	0.168	2,533	(11,752)	0.108	(1,610)	-78%	64%	-64%
Dec	15,339	0.195	2,989	(10,486)	0.048	(1,810)	-68%	25%	-61%
Total	169,035	0.158	26,734	23,611	0.004	4,568	4%	15%	19%

Figure 11-38 2024 Monthly All Processed to Model Au Ounce Variance

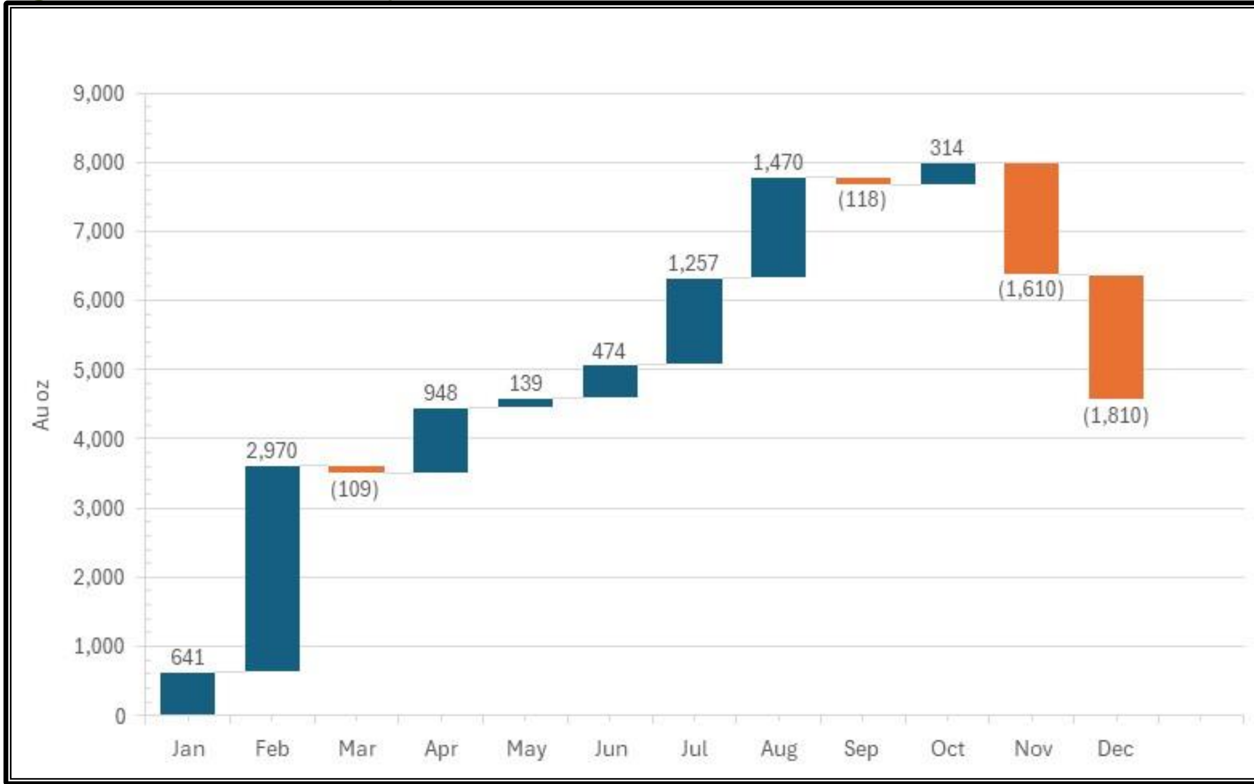
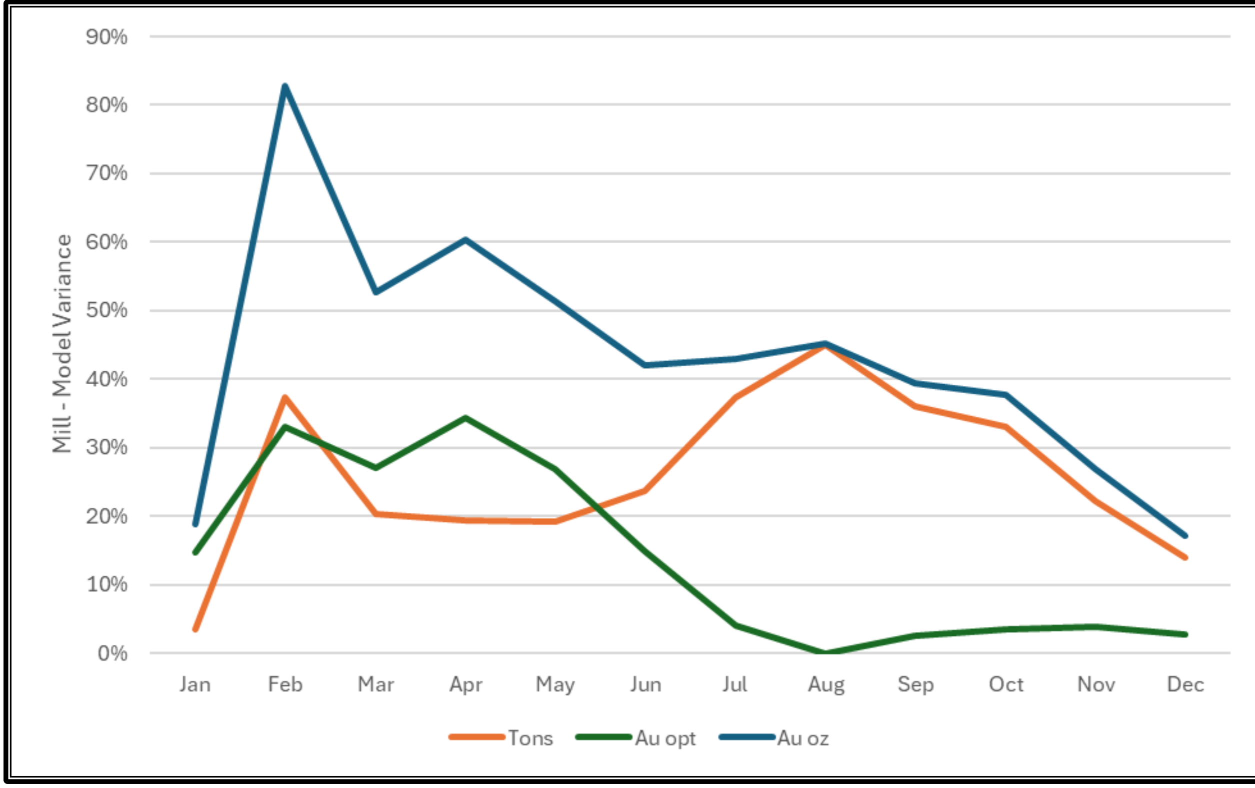


Figure 11-39 2024 Cumulative All Processed to Model Variance



11.7.2.4 Factors that May Affect Mineral Resource

Areas of uncertainty that may materially impact the Mineral Resource Estimates include:

- Changes to long term metal price assumptions.
- Changes to the input values for mining, processing, and G&A costs to constrain the estimate.

- Changes to local interpretations of mineralization geometry and continuity of mineralized domains.

- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Variations in geotechnical, hydrogeological and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social license assumptions.
- Logistics of securing and moving adequate services, labor, and supplies could be affected by epidemics, pandemics and other public health crises.

11.7.3 Reasonable Prospects for Economic Extraction

SK 1300 requires mineral resources demonstrate “Reasonable Prospects for Economic Extraction” (RPEE). Stope optimizer software is well suited to meet this requirement. The software will produce stope designs that meet minimum minable geometric shapes that exceed the cutoff grade. These shapes will include necessary low grade or waste dilution required to produce a minable geometry.

Granite Creek mineral resources are defined by a mining geometry consistent with the drift and fill mining method. The dimensions of a minimum minable stope cross section are 15 feet wide x 15 feet high. Individual stope lengths vary from a minimum of 20 feet to a maximum of 100 feet.

11.7.4 QP Opinion

Practical Mining is not aware of any environmental, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report Summary.

Practical Mining is of the opinion that the Mineral Resources for the Project, which were estimated using industry accepted practices, have been prepared and reported using S-K 1300 definitions.

Technical and economic parameters and assumptions applied to the Mineral Resource Estimate are based on parameters received from i-80 and reviewed by Practical Mining to determine if their appropriateness.

The QP considers that all issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

11.7.5 Underground Mineral Resources

Mineral Resources for the Granite Creek underground mine are summarized in Table 11-27.

Table 11-27 Summary of Mineral Resources at the End of the Fiscal Year Ended December 31, 2024

Zone	ktons	ktonnes	Au opt	Au g/t	Au koz
Measured					
Ogee	88	80	0.244	8.4	22
Otto	59	53	0.256	8.8	15
Meas Total	147	133	0.249	8.5	37
Indicated					
CX	8	7	0.391	13.4	3
Ogee	181	164	0.352	12.1	64
Otto	295	268	0.316	10.8	93

Zone	ktons	ktonnes	Au opt	Au g/t	Au koz
South Pacific	223	203	0.286	9.8	64
Ind Total	707	641	0.317	10.9	224
Measured and Indicated					
CX	8	7	0.391	13.4	3
Ogee	269	244	0.317	10.9	85
Otto	354	321	0.306	10.5	108
South Pacific	223	203	0.286	9.8	64
M&I Total	854	775	0.305	10.5	261
Inferred					
CX	97	88	0.351	12.0	34
Ogee	42	38	0.563	19.3	24
Otto	187	170	0.401	13.7	75
South Pacific	536	486	0.361	12.4	194
Inf Total	862	782	0.378	13.0	326

Notes Pertaining to Underground Mineral Resources:

1. *Mineral Resources have been estimated at a gold price of $2,175 per troy ounce and a silver price of $27.25 per ounce. Refer to Section 16.1 for price selection details.*
2. *Mineral Resources have been estimated using gold metallurgical recoveries of 85.2% to 94.2% for pressure oxidation. Payment for refractory mineralization sold to a third party is 58%. Oxide CIL mineralization payments vary from 40% to 70% based upon the grade of the mineralization.*
3. *The cutoff grade for refractory Mineral Resources varies from 0.151 to 0.184 opt. for acidic conditions. The cutoff grade for oxide mineral resources is 0.075 opt.*
4. *The contained gold estimates in the Mineral Resource table have not been adjusted for metallurgical recoveries.*
5. *Numbers have been rounded as required by reporting guidelines and may result in apparent summation differences.*
6. *A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*
7. *An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
8. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant factors.*
9. *Mineral Resources have an effective date of December 31, 2024, and;*
10. *The reference point for mineral resources is in situ.*

12 Mineral Reserve Estimates

The Granite Creek Project does not have any mineral reserves.

13 Mining Methods

13.1 Open Pit

13.1.1 Introduction

The Granite Creek Mine Project will employ conventional open pit mining techniques using front end loaders and rear dump rigid frame haul trucks. As discussed in Section 17, open pit material will be treated using CIL circuit. The mine plan is designed to deliver an average of 10,000 tonnes of potentially economically viable material per day from the open pit to the crusher which will then be run through the CIL mill. The average daily waste production rate over the life of the mine is 84,750 tonnes per day. Waste material would be either placed on waste rock storage facilities (WRSF) or as backfill in previously mined open pits.

There are three distinct open pit production areas on the project: B pit, CX-A pit, and Mag pit. The CX and Mag pits were each designed with three phases, for a total of seven mining phases for the project.

13.1.2 Whittle Pit Shell Analysis

Whittle pit shell analysis was used to provide a basis for creating the pit designs. The objective of the Whittle pit optimization was to maximize the economic extraction of the mineral resources contained in the block model. The inputs used to develop the Whittle pit shell analysis are listed in Table 13-1.

Table 13-1: Granite Creek Open Pit Mine Project Whittle Pitshell Analysis Parameters

Parameter	Items	Unit	Value
Costs	Mining Cost (waste/mineralized material)	$/tonne mined	2.46
	Heap Leach*	$/tonne mineralized material treated	9.04
	Carbon in Leach**	$/tonne mineralized material treated	17.22
Recovery	HLCH Recovery CN Solubility <60	%	CN Solubility*100
	HLCH Recovery CN Solubility >= 60	%	((0.1225 * [Au_ppm]) + 0.4164)*100

Parameter	Items	Unit	Value
	CIL Recovery	%	((0.5388 * CN Solubility) + 0.3201)*100
Net Revenue Gold	Gold price	$/oz	2,040
	Selling costs and penalties***	$/oz	114
Royalty	Total royalty (simplified)	%	6.00%
Slope angles	Slope Angle	degrees	41
Limits	HLCH	tonnes per year	2,975,000
	CIL	tonnes per year	1,050,000

* CIL costs include $1.1/tonne milled for admin costs

** Various royalties are applicable at various points throughout the mine life, however for the scope of this IA GRE has used a single 6% royalty for the open pit mineral resource.

*** This selling cost is used to apply the 6% royalty

Due to the large ratio of deposit size to block size and method of grade estimation, the grade model is fully diluted, and the resource is 100% recoverable as estimated.

Revenue factors from 0.245 to 1.47 in 0.049 increments were applied to the base gold price of $2,040 per troy ounce to examine gold prices from $500 to $3,000 in $100 increments. After Whittle pit shells were run, GRE analyzed each resource area by examining the marginal impact on undiscounted cashflow. This analysis examines the impact that each incremental increase in the pit shell has on the undiscounted cashflow divided by the number of tonnes that are processed. GRE examined each pit area and selected a case that gave a local spike in the marginal impact on undiscounted cashflow at a revenue factor equal to or less than the base price of $2,040 as shown in Figure 13-1 through Figure 13-4. Pit shells were not adjusted for overlap of backslopes. Visually the overlap only occurs at the top of the pits. However, the effect of this overlap is relatively small and the results of this analysis can still guide the Whittle pit shell selection.

Figure 13-1 Marginal Impact Undiscounted Cashflow Mag Pit

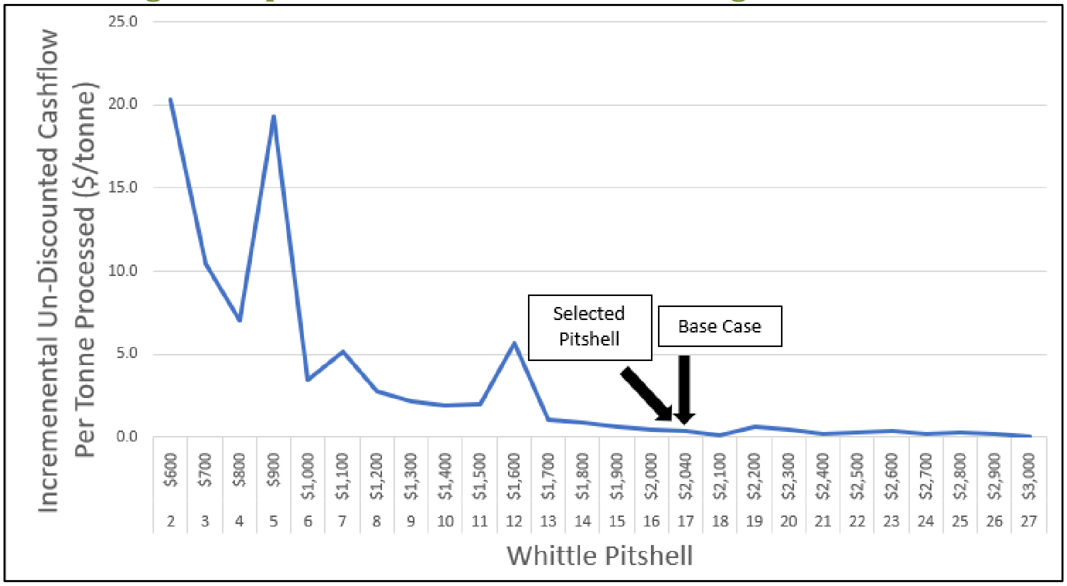


Figure 13-2 Marginal Impact Undiscounted Cashflow CX Pit

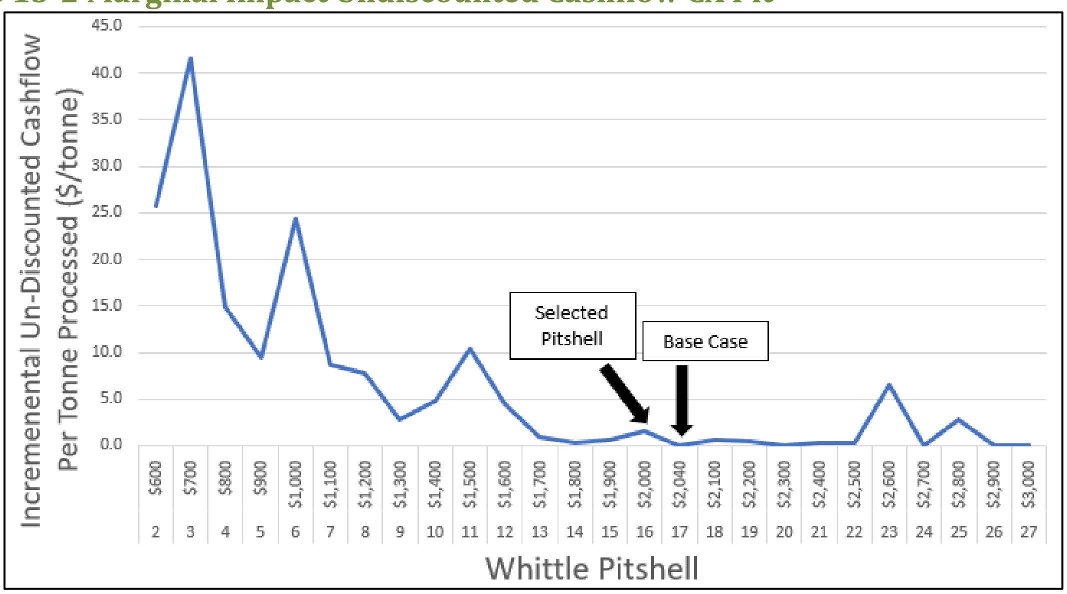


Figure 13-3 Marginal Impact Undiscounted Cashflow Pit B

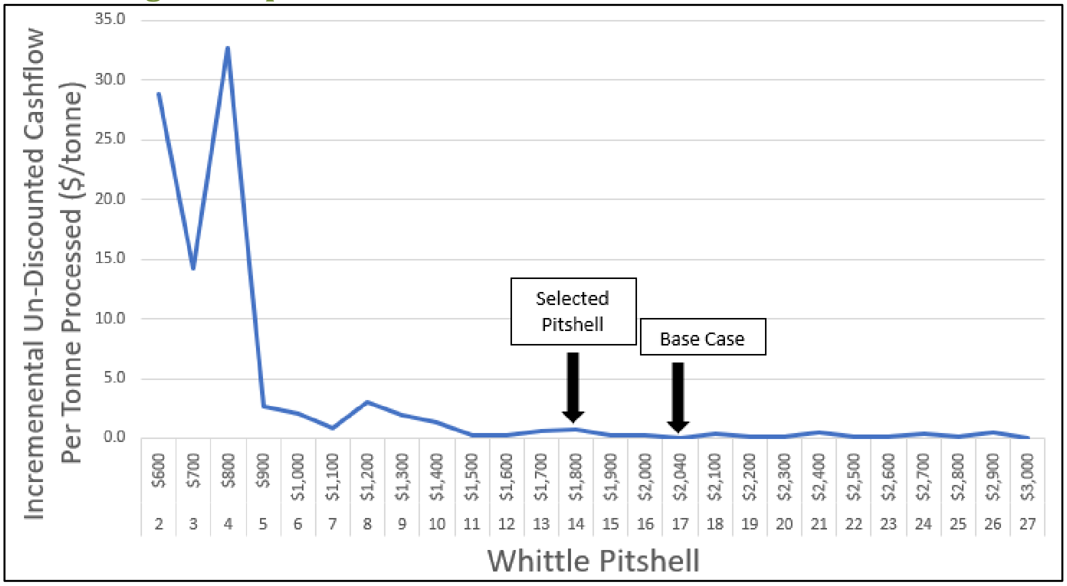
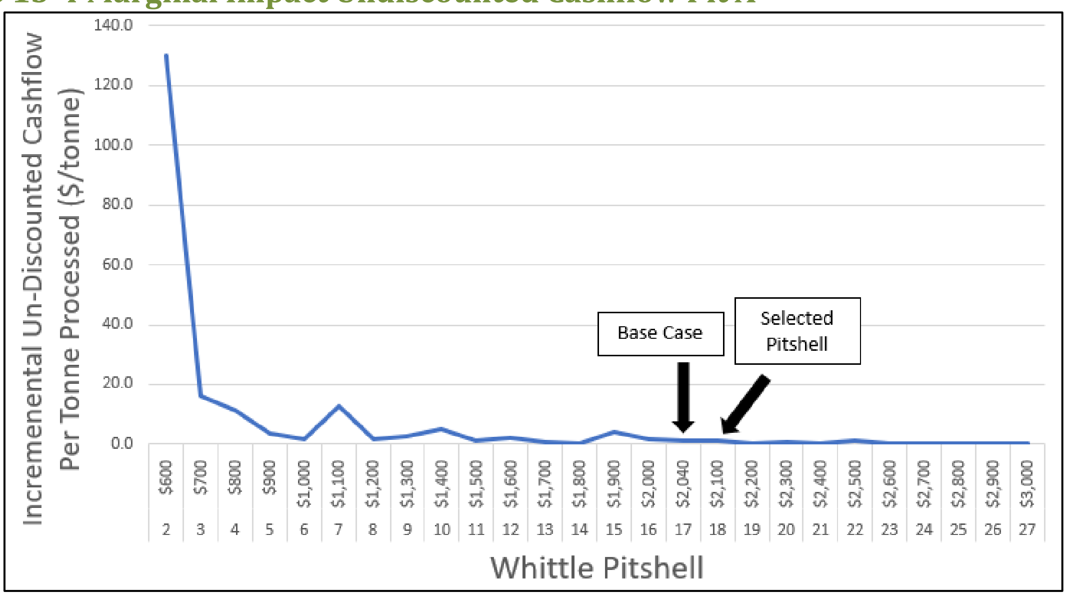


Figure 13-4 Marginal Impact Undiscounted Cashflow Pit A



A summary of the selected Whittle pit shells used to constrain the resource is listed in Table 13-2: Selected Whittle Pit Shells for Resource Areas.

Table 13-2: Selected Whittle Pit Shells for Resource Areas

Area	Whittle Pit Shell
Mag	17
CX	16

Pit A	18
Pit B	14

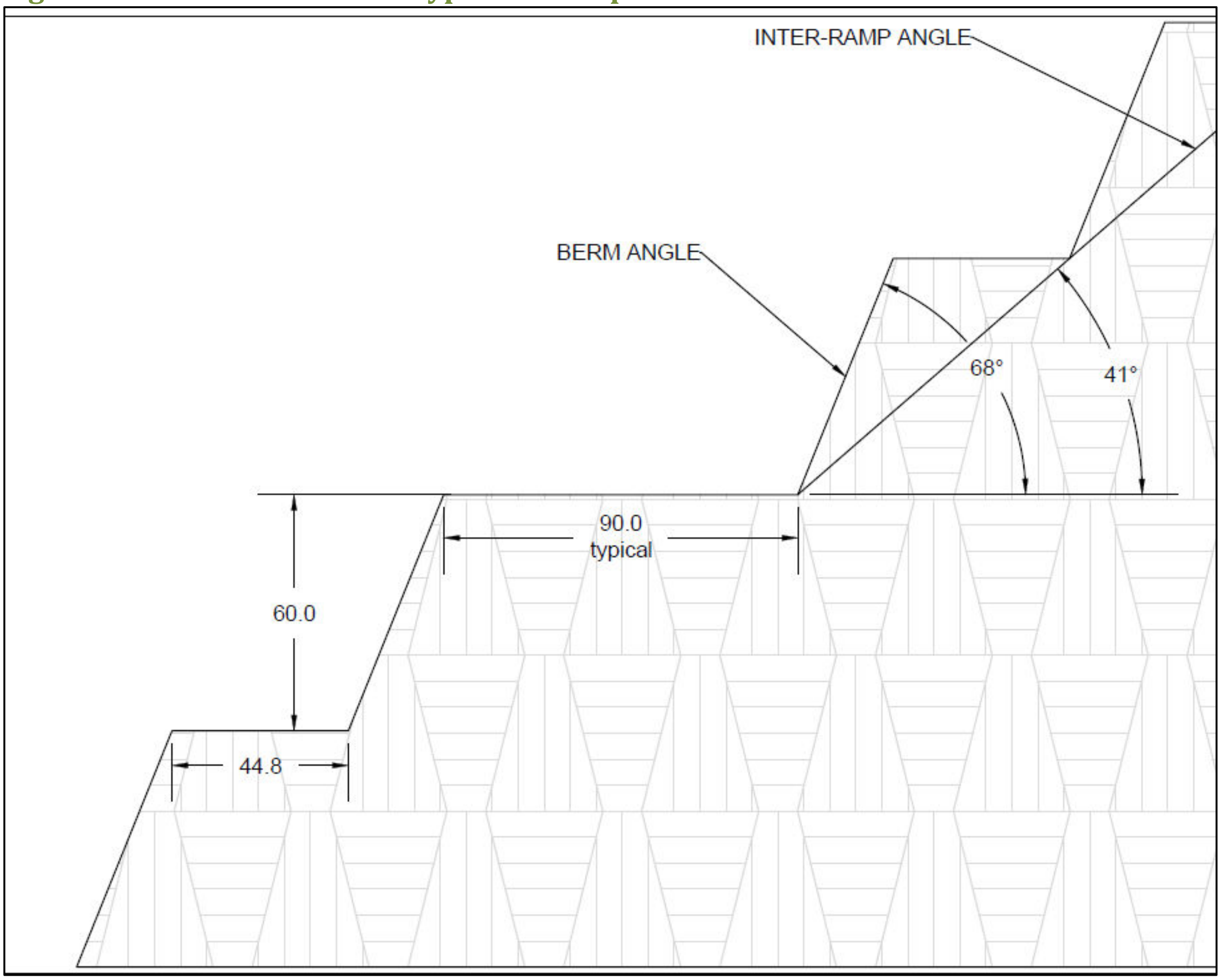
13.1.3 Pit Design

Based on previous engineering analysis performed by Golder (Golder Associates, 2014), GRE used a triple bench format consisting of triple 20-foot vertical benches with a horizontal 44-foot catch bench every three vertical benches. The resulting open pit parameters are listed in Table 13-3. In less competent zones of safety, benches will be wider or placed at more frequent intervals to reduce the slope angle.

Table 13-3: Pit Parameters

Pit Design Parameters	Value (degrees)
Max Inter-ramp Angle Hard Rock	41
Max Bench Face Angle	68

Figure 13-5 Cross-Section of Typical Pit Slope



13.1.4 Block Model Coding

13.1.4.1 Processing Method

GRE coded each block in the block model with a processing method applicable for the block, either heap leach or CIL, or coded the block as waste based on the economic value of each block. The economic value of the block is calculated as:

$$Block\ economic\ value = \begin{cases} -[(Mining\ cost) * tonnage], & Gold\ grade < 0 \\ [(Gold\ grade * Recovery) * (Selling\ price - Royalty - selling\ cost) * tonnage] \\ \quad -[(Mining\ cost + Processing\ Cost + Admin\ Cost) * tonnage], & Gold\ grade \geq 0 \end{cases}$$

The selling price, royalty, selling cost, mining cost, processing cost, and admin cost used in the equation were the preliminary values input into the Whittle analysis, as summarized in Table 13-1. For each block, the economic value for the heap leach and CIL were evaluated using the above equation. Blocks were coded to be processed through CIL operation if the economic value for the block was positive and higher than the heap leach operation. Blocks were coded to be processed through heap leach operation if the economic value for the block was positive and higher than the CIL operation. Remaining blocks were coded as waste material.

13.1.4.2 Recovery

GRE coded each block in the block model with three potential recoveries:

- Multi-process recovery: each block coded with a CIL process method was coded with its calculated CIL recovery and each block coded with a heap leach process method was coded with its calculated heap leach recovery
- Heap Leach only recovery: each block coded with a CIL processing method that was also heap leachable was coded with its calculated heap leach recovery and each block coded with a heap leach processing method was coded with its calculated heap leach recovery
- CIL only recovery: each block coded with a CIL processing method was coded with its calculated CIL recovery and each block coded with a heap leach processing method was coded with its calculated CIL recovery.

13.1.5 Mining Sequence

The proposed mining sequence is based on known engineering information, economic factors, and environmental considerations. The production pits would be sequentially mined with minor overlap of simultaneous production dependent on short term scheduling needs. The proposed mining sequence begins with Pit B and is shown in Table 13-4.

Table 13-4: Summary of Pit Phases

Pit	Start Day	End Day
Pit B	-129	208
CX 1	118	618
CX 2	423	1,077
CX 3	997	1,969
MAG A	1,969	2,137
MAG B	1,918	2,656
MAG C	2,388	3,103

13.1.6 Base Case

GRE selected the CIL processing at a cutoff grade of 0.85 g/t for high grade material for the base case with a low grade to high grade cutover grade of 0.25 g/t. The resources within the base case pits and phases are shown in Table 13-5.

Table 13-5: Base Case Pit Resource

Pit/Phase	High Grade CIL Tonnes (1000s)	Low Grade CIL Tonnes (1000s)	Total Tonnes (1000s)	Au Troy Ounces Contained (1000s)	Au Grade (g/t)	Stripping Ratio
B	1,740.7	1,883.6	26,846.3	132.0	1.13	6.41
CX Phase 1	1,981.1	2,315.5	47,821.5	172.1	1.25	10.13
CX Phase 2	2,137.3	2,707.6	60,396.5	169.3	1.09	11.47
CX Phase 3	3,465.8	4,037.3	90,929.1	313.1	1.30	11.12
Mag Phase 1	489.4	1,498.3	18,423.9	48.3	0.76	8.27
Mag Phase 2	4,546.1	1,695.5	51,912.1	291.7	1.45	7.32
Mag Phase 3	4,549.7	1,806.7	47,847.8	270.8	1.33	6.53
Total	**18,910.1**	**15,944.4**	**344,177.3**	**1,397.2**	**1.25**	**8.87**

13.1.7 Mine Scheduling

A preliminary mining schedule was generated from the base case pit resource estimate. GRE used the following assumptions to generate the schedule:

- Mining Production Rate: 10,000 tonnes per day
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 12

Pre-stripping of waste was included if waste occurred on a bench that had no corresponding processable material or if the tonnage of waste on a bench exceeded ten times the tonnage of processable material on that bench. The production rate for pre-strip benches was set to 20 times the leach material production rate, or 100,000 tpd. Processable material mined along with pre-stripped waste was placed into stockpiles for later processing.

For all other benches, all waste on a bench was scheduled to be mined over the same duration as the processable material on that bench. This scheduling method resulted in some years with high waste quantities relative to the processable material quantity mined. GRE used pre-stripping and phasing, as described above, as much as possible to smooth out the production, but the limitations of the scheduling program resulted in some inefficiencies.

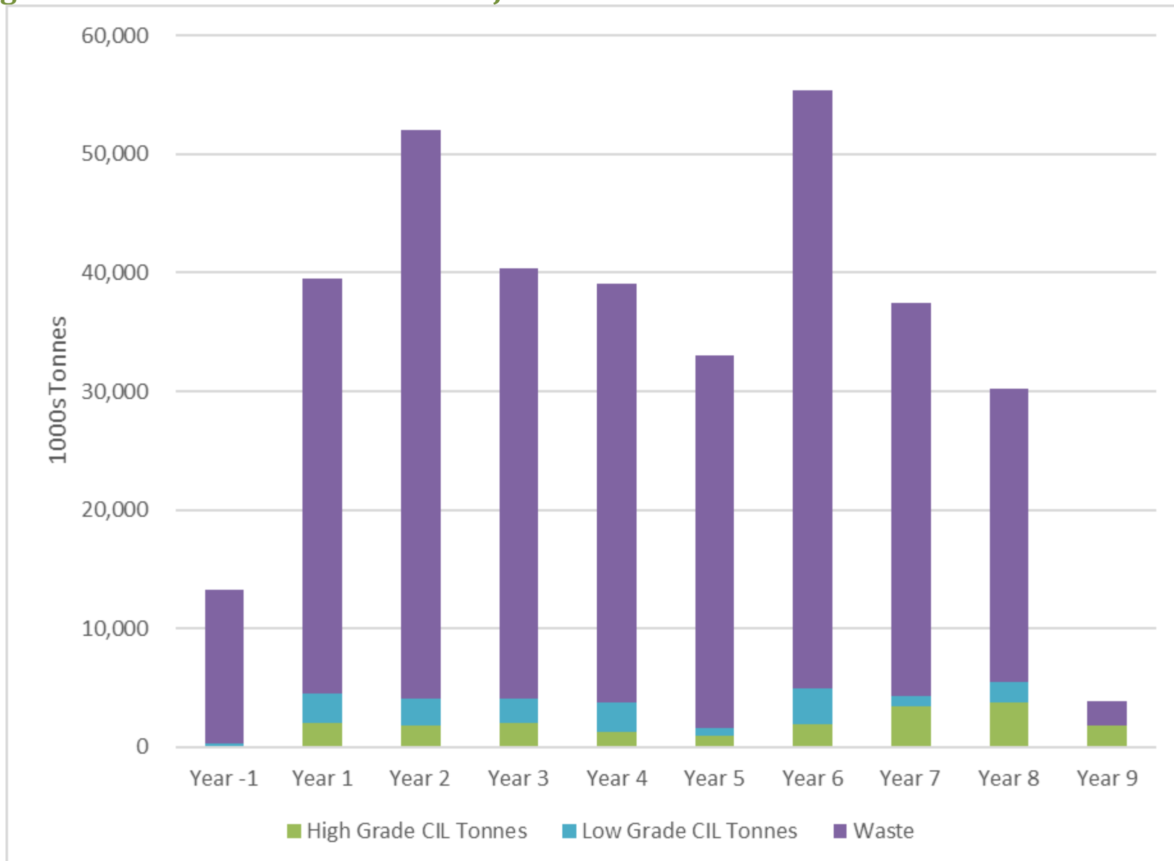
For the economic model, the project was scheduled by quarter for any pre-production years and for the first two production years, then by year for the remainder of the mine life. The mining schedule is summarized in Table 13-6 and illustrated in Figure 13-6.

Table 13-6: Granite Creek Mine Project Open Pit Base Case Mine Schedule Summary

Pit/Phase	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
High Grade CIL Tonnes (1000s)											
B	103.7	1,637.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**1,740.7**
CX Phase 1	0.0	370.1	1,611.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**1,981.1**
CX Phase 2	0.0	0.0	188.1	1,949.1	0.0	0.0	0.0	0.0	0.0	0.0	**2,137.3**
CX Phase 3	0.0	0.0	0.0	96.0	1,228.1	884.1	1,257.6	0.0	0.0	0.0	**3,465.8**
Mag Phase A	0.0	0.0	0.0	0.0	0.0	0.0	489.4	0.0	0.0	0.0	**489.4**
Mag Phase B	0.0	0.0	0.0	0.0	0.0	0.0	202.1	3,374.2	969.8	0.0	**4,546.1**
Mag Phase C	0.0	0.0	0.0	0.0	0.0	0.0	0.0	16.9	2,777.1	1,755.7	**4,549.7**
Total	**103.7**	**2,007.0**	**1,799.1**	**2,045.2**	**1,228.1**	**884.1**	**1,949.2**	**3,391.1**	**3,746.9**	**1,755.7**	**18,910.1**
Low Grade CIL Tonnes (1000s)											
B	187.7	1,695.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**1,883.6**
CX Phase 1	0.0	808.4	1,507.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**2,315.5**
CX Phase 2	0.0	0.0	803.4	1,904.2	0.0	0.0	0.0	0.0	0.0	0.0	**2,707.6**
CX Phase 3	0.0	0.0	0.0	141.4	2,510.8	647.1	738.0	0.0	0.0	0.0	**4,037.3**
Mag Phase A	0.0	0.0	0.0	0.0	0.0	0.0	1,498.3	0.0	0.0	0.0	**1,498.3**
Mag Phase B	0.0	0.0	0.0	0.0	0.0	0.0	768.2	894.4	33.0	0.0	**1,695.5**
Mag Phase C	0.0	0.0	0.0	0.0	0.0	0.0	0.0	47.5	1,703.3	55.9	**1,806.7**
Total	**187.7**	**2,504.3**	**2,310.5**	**2,045.5**	**2,510.8**	**647.1**	**3,004.5**	**941.9**	**1,736.2**	**55.9**	**15,944.4**
Waste Tonnes (1000s)											
B	12,936.7	10,285.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**23,222.0**
CX Phase 1	0.0	24,683.9	18,841.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**43,524.9**
CX Phase 2	0.0	0.0	29,115.8	26,435.8	0.0	0.0	0.0	0.0	0.0	0.0	**55,551.6**

Pit/Phase	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
CX Phase 3	0.0	0.0	0.0	9,878.6	35,287.1	31,496.9	6,763.5	0.0	0.0	0.0	**83,426.0**
Mag Phase A	0.0	0.0	0.0	0.0	0.0	0.0	16,436.2	0.0	0.0	0.0	**16,436.2**
Mag Phase B	0.0	0.0	0.0	0.0	0.0	0.0	27,215.2	16,465.9	1,989.5	0.0	**45,670.5**
Mag Phase C	0.0	0.0	0.0	0.0	0.0	0.0	0.0	16,680.8	22,749.2	2,061.4	**41,491.4**
Total	**12,936.7**	**34,969.3**	**47,956.8**	**36,314.4**	**35,287.1**	**31,496.9**	**50,414.9**	**33,146.7**	**24,738.8**	**2,061.4**	**309,322.8**
Au Troy Ounces (1000s)											
B	12.3	119.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**132.0**
CX Phase 1	0.0	37.1	135.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**172.1**
CX Phase 2	0.0	0.0	21.1	148.2	0.0	0.0	0.0	0.0	0.0	0.0	**169.3**
CX Phase 3	0.0	0.0	0.0	5.6	118.9	68.7	119.9	0.0	0.0	0.0	**313.1**
Mag Phase A	0.0	0.0	0.0	0.0	0.0	0.0	48.3	0.0	0.0	0.0	**48.3**
Mag Phase B	0.0	0.0	0.0	0.0	0.0	0.0	20.0	212.0	59.6	0.0	**291.7**
Mag Phase C	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.5	170.4	99.0	**270.8**
Total	**12.3**	**156.8**	**156.1**	**153.8**	**118.9**	**68.7**	**188.2**	**213.5**	**230.0**	**99.0**	**1,397.2**

Figure 13-6 Granite Creek Mine Project Base Case Mine Schedule



13.1.8 Mine Operation and Layout

Limited facilities for administrative offices, warehouse, and other facilities are present at the site. Other facilities, such as crushing, and CIL plants will need to be constructed.

GRE developed conceptual layouts for the project, including waste dump locations and sizes, leach pad location and size, tailings storage facility, and stockpile locations and sizes. Figure 13-7 illustrates the conceptual project layout with pits, pads, and dumps. Phased site layout plans for the duration of open pit mining are shown in Figure 13-8 through Figure 13-14.

Figure 13-7 Conceptual Project Layout

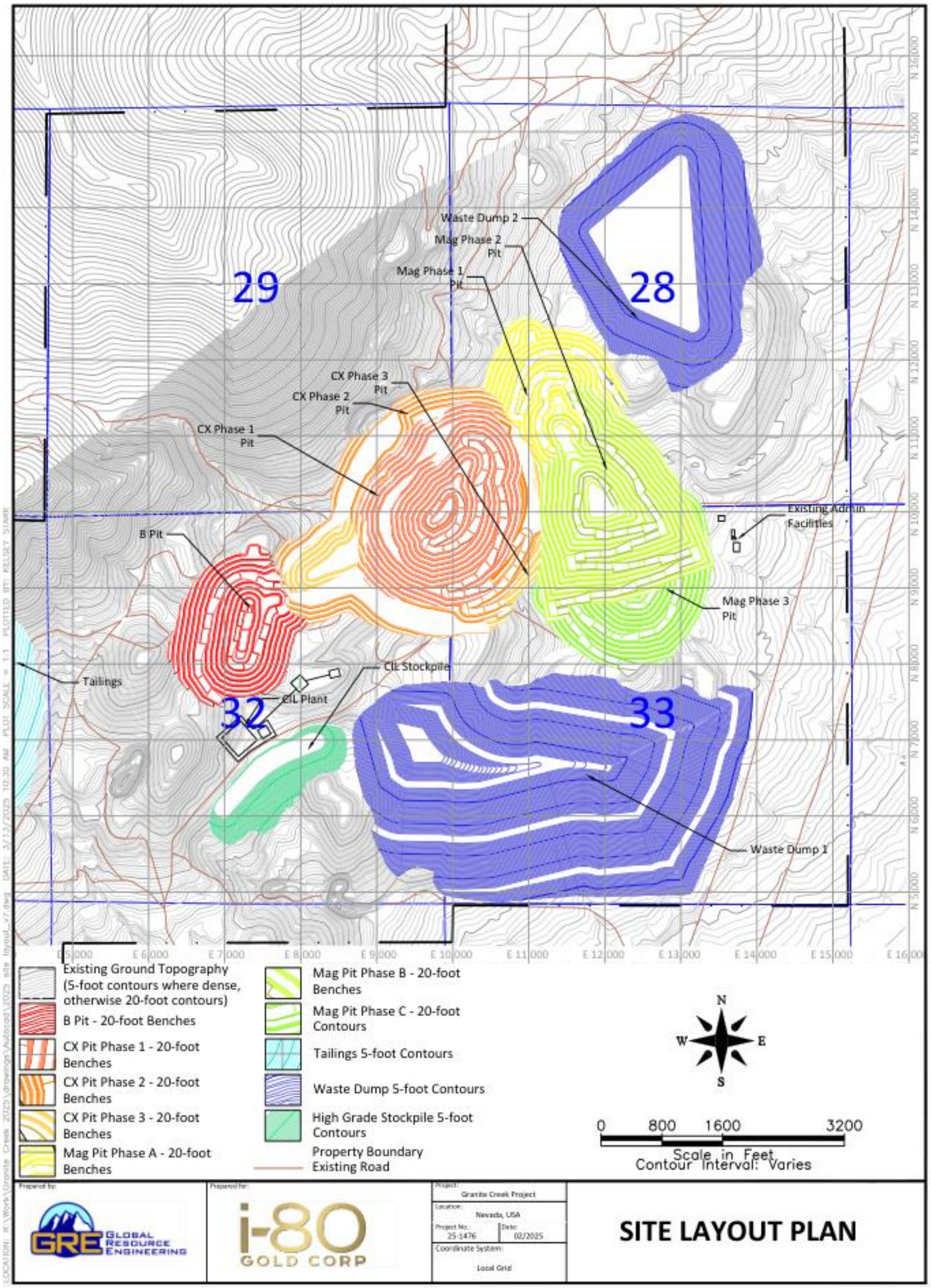


Figure 13-8 Phased Pit and Site Plan Layout B Pit

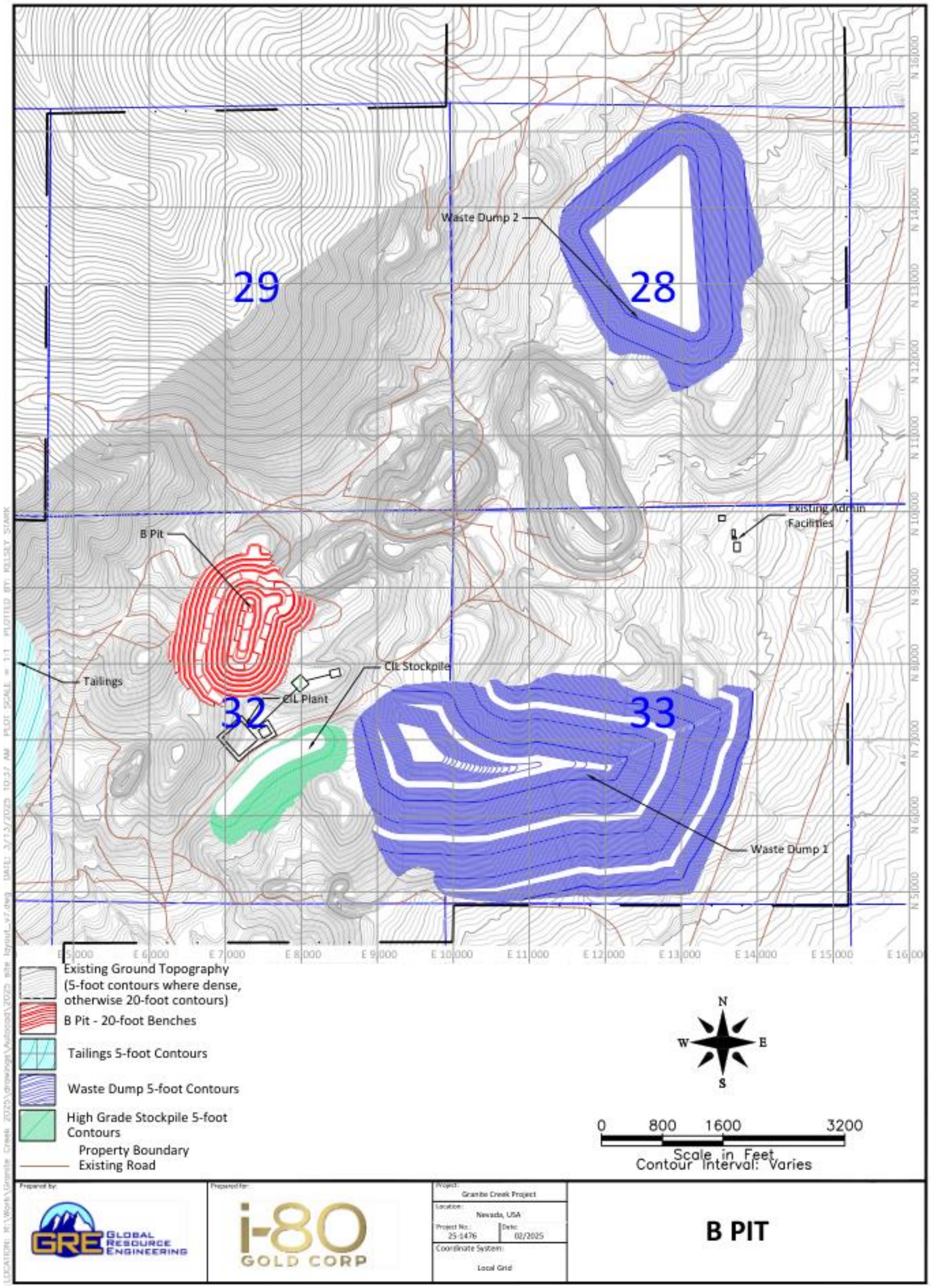


Figure 13-9 Phased Pit and Site Plan Layout B Pit & CX Phase 1

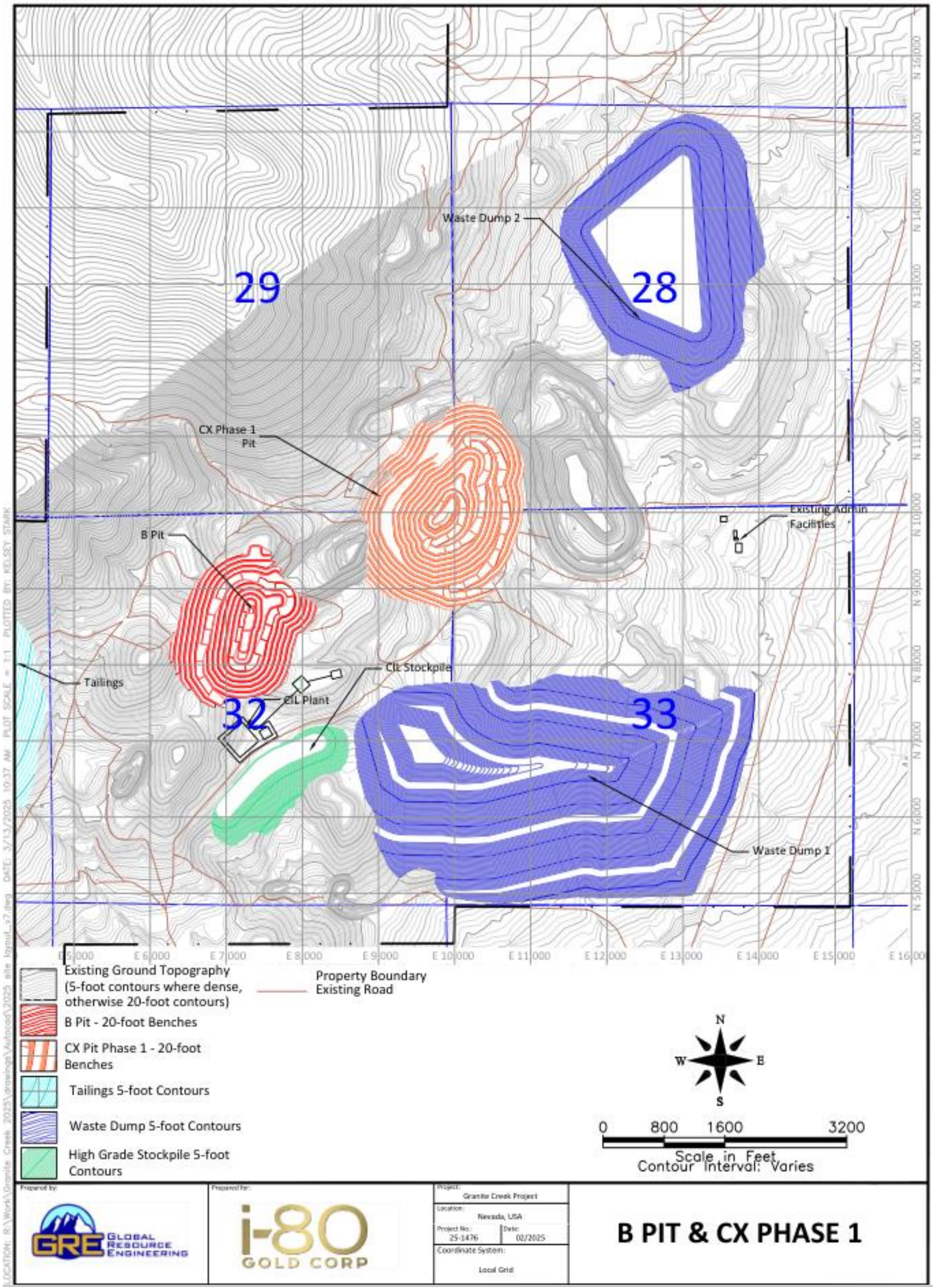


Figure 13-10 Phased Pit and Site Plan Layout B Pit & CX Phases 1 & 2

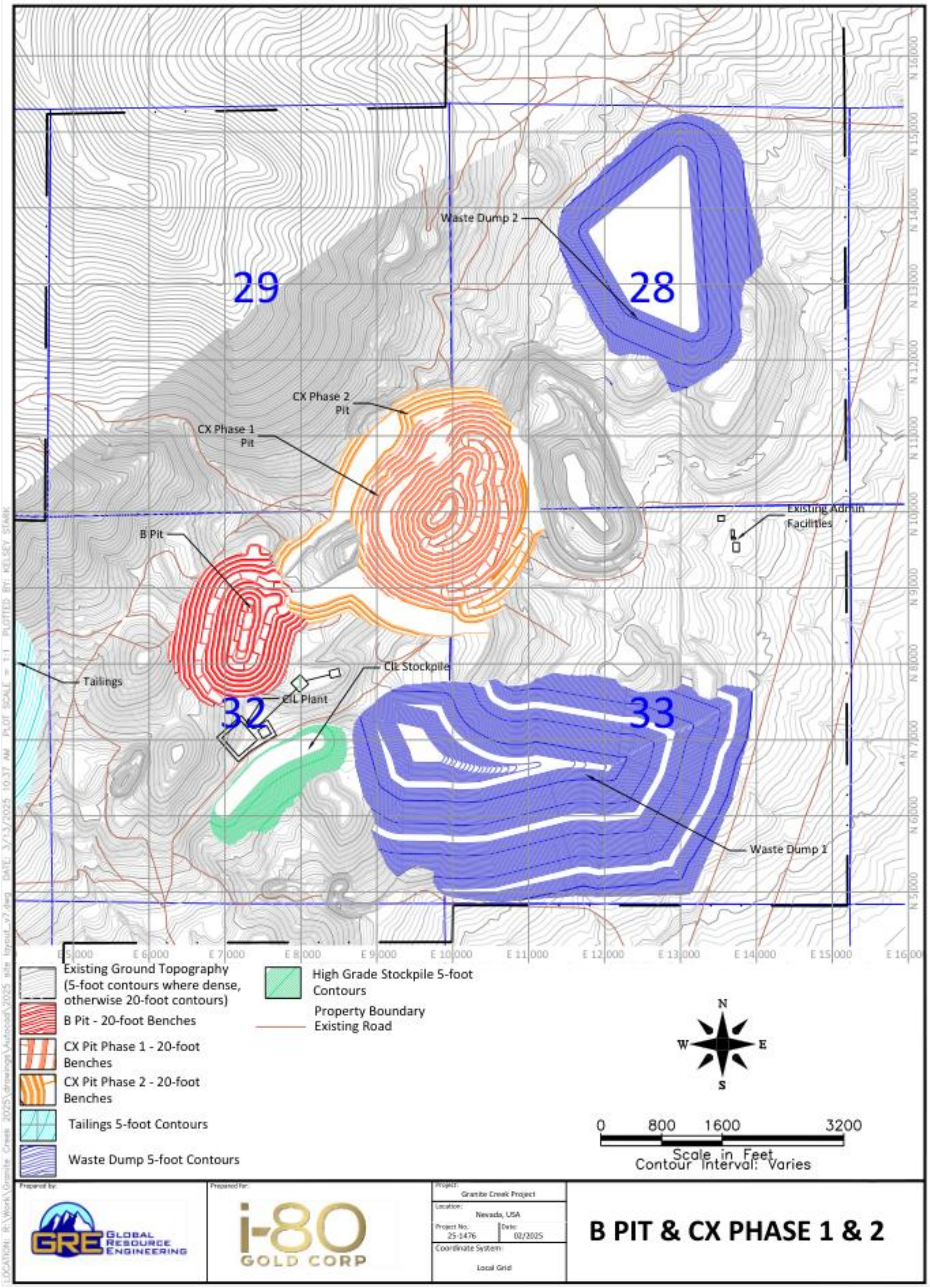


Figure 13-11 Phased Pit and Site Plan Layout B Pit & CX Phases 1, 2, & 3

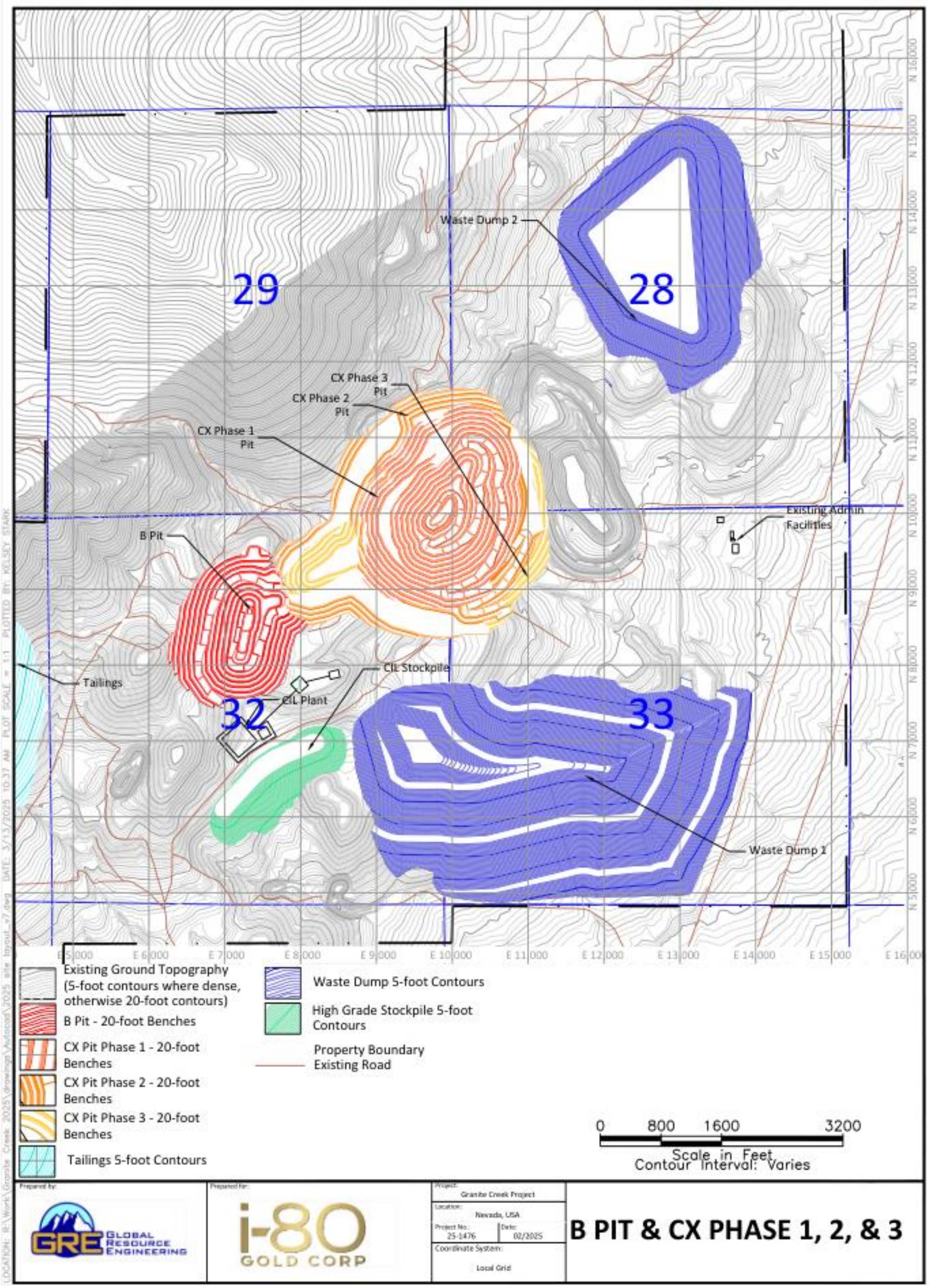


Figure 13-12 Phased Pit and Site Plan Layout B Pit; CX Phases 1, 2, & 3; and Mag Phase1

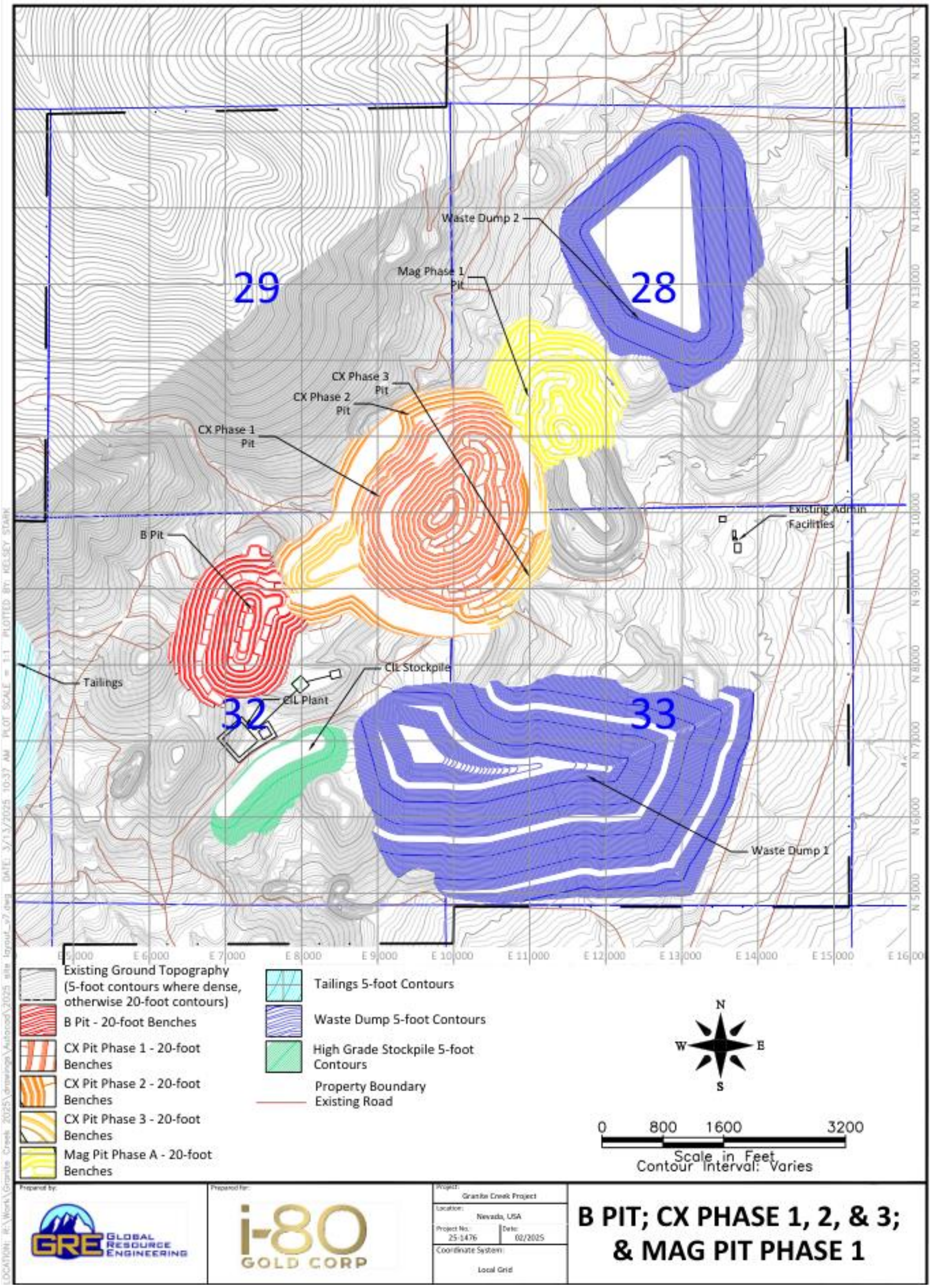


Figure 13-13 Phased Pit and Site Plan Layout B Pit; CX Phases 1, 2, & 3; and Mag Phase 1 & 2

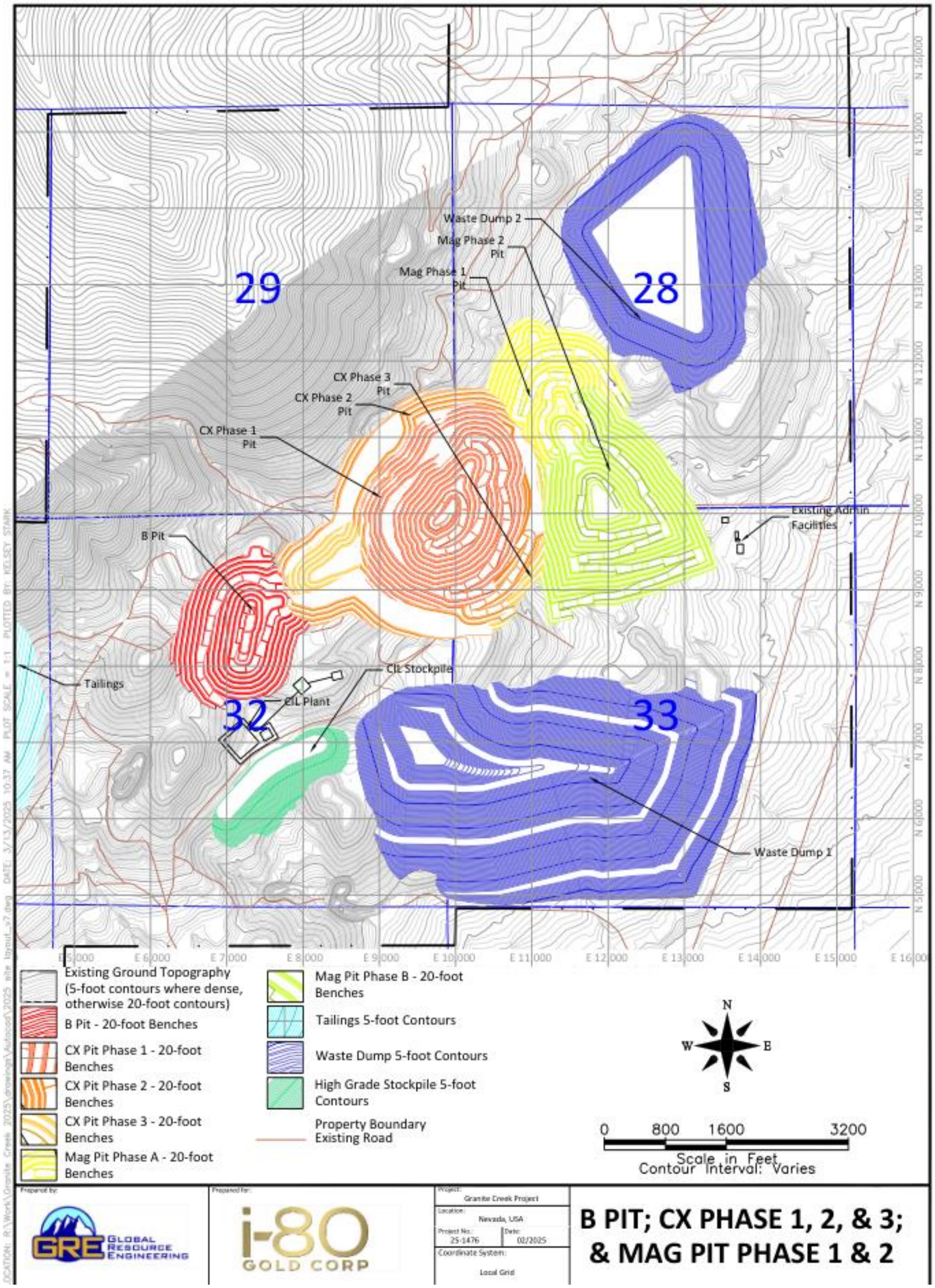
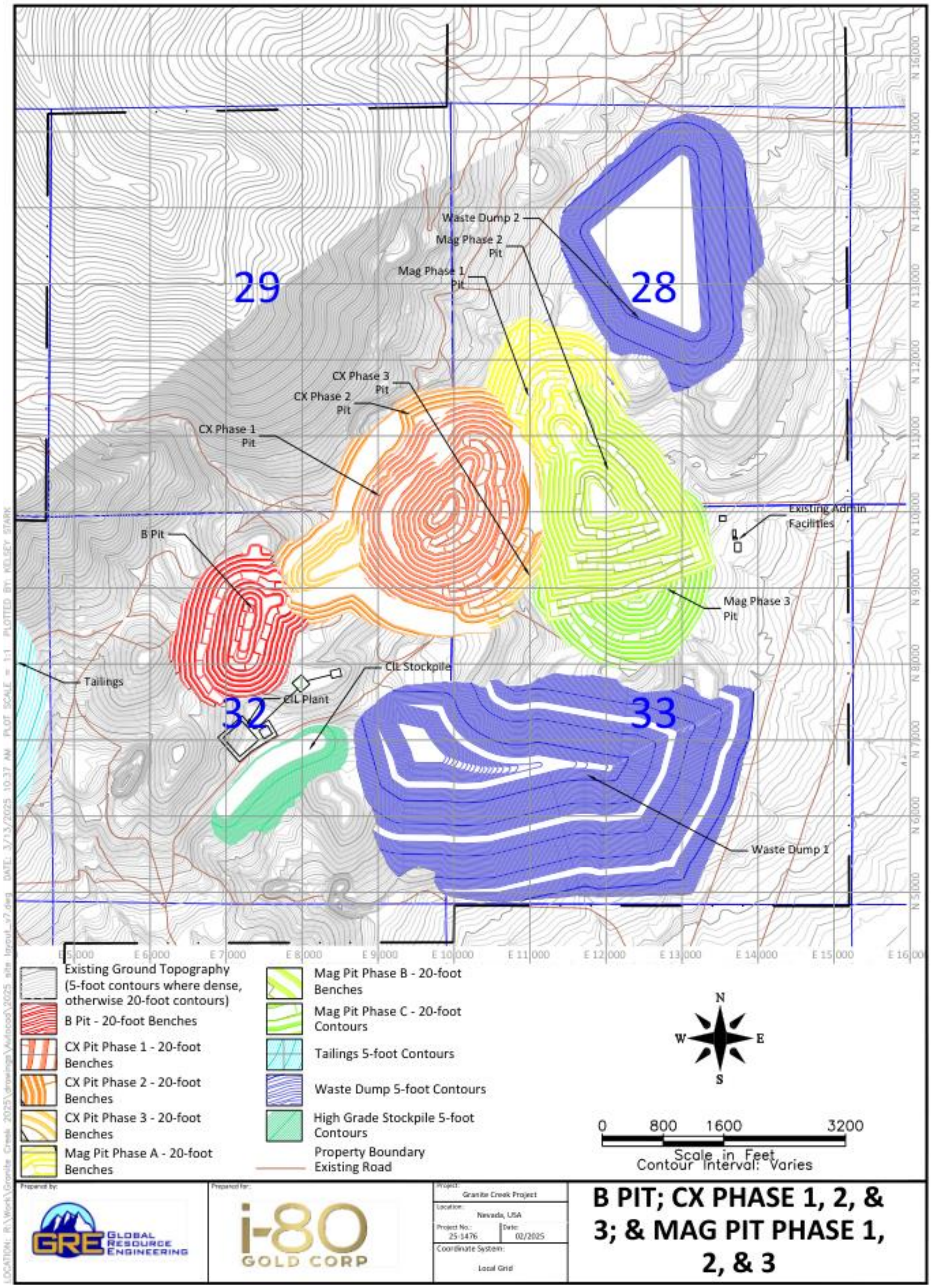


Figure 13-14 Phased Pit and Site Plan Layout B Pit; CX Phases 1, 2, & 3; and Mag Phase 1, 2, & 3



13.1.8.1 Waste Rock Pile

To store the waste material generated during mining activities, two waste rock piles are proposed. The waste piles would be located south of the CX pit and east of the Mag pit. Additionally, as mining progresses, waste rock would be backfilled in portions of the mined-out B and CX pits. These locations are selected to minimize hauling distances and disturbed acreage. Up to approximately 100 million loose cy (approximately 153 million tonnes) of waste rock would be mined and placed into the waste rock piles and approximately 111 million cy (approximately 157 million tonnes) would be backfilled into mined out pits. Waste rock piles would be engineered to have overall final 3H:1V ultimate slopes.

13.1.9 Drilling and Blasting

Fresh mineralized material and waste rock is comprised of a mix of shale, limestone, dolomite, conglomerates, and granodiorite. All of this material would require drilling and blasting prior to excavation. Some areas within the pits to be excavated consist of alluvium or previous backfill; those areas would not require drilling and blasting, except to the extent drill holes are needed for grade control.

Drilling and blasting would employ conventional techniques, which would entail drilling 7-inch diameter blastholes spaced on 18-foot centers. The rock would be blasted with ammonium nitrate fuel oil (ANFO) blasting agent initiated with shock tube, boosters, and nonel blasting caps. Potential noise and dust from blasting is not anticipated to impact the surrounding community due to the project’s remote location far away from residential or commercial structures.

13.1.10 Loading and Hauling

The blasted rock or backfill would be loaded with a 17-cy capacity front end loader into 133-tonne capacity haul trucks.

13.1.11 Haul Roads

Haulage pit ramps were designed with a minimum width of 90 feet and a maximum gradient of 10 percent. Haul ramps and roads have been designed to accommodate two-way traffic using 133-tonne haul trucks, water diversion ditches, and safety berms. Minor sections of temporary ramping for development purposes may be steeper and narrower. Haulage roads outside of the pit areas would typically be 90 feet wide, and in some areas would be up to 150 feet wide to allow for turning lanes, surface drainage, and separate lanes for auxiliary vehicle traffic. A minimum cross slope of 2% on haul roads will accommodate water drainage.

13.1.12 Mining Mobile Equipment

A variety of mobile equipment likely to be used in conducting mining operations is presented in Table 13-7.

Table 13-7: Granite Creek Mine Project Open Pit Mobile Equipment Sizes and Quantities

Major Equipment	Max Quantity
Loader CAT 993K	6
Haul Truck CAT 785D	20
Bulldozer CAT D10	3
Drill	6
Support Equipment	
Wheel Dozer	1
Wheel Loader	1
Water Truck	2
ANFO Truck	1
Lube Truck	2
Mechanics Truck	2
Grader	1
Minor Equipment	
Small Excavator	1
Backhoe	1
Small Crane	1
Light Plant	6
4x4 Pickup	10

Equipment sizes and quantities may vary slightly over the life of the mine in response to changes in stripping ratios, haul distances, or other factors.

13.2 Underground

The Granite Creek Mine is operated by a local contractor. Table 13-8 and Table 13-9 list personnel levels and underground equipment provided. The contractor has operated a number of mines in northern Nevada over the past thirty years.

Table 13-8: Contractors Personnel

Supervision / Overhead	Count
Superintendent	1
Assistant Superintendent	2

Safety Superintendent	2
Master Mechanic	1
Electrician	1
Engineer (Billed on a Day Rate)	1
Total Supervision / Overhead	8
Rotating Crew Manpower (Per Shift)	
Shifter	1
Miners	4
Operators - Mucker / Truck / Jam	6
Shift Mechanic	2
Luber/Nipper	1
Shotcrete	2
Batch Plant	1
Total Crew Manpower 17	17
Rotating Crew Total (4 Crews) 68	68
Day Shift (7 days)	
Mechanics	2
Total Day Shift Crew	4
Total	80

Table 13-9: Contractors Underground Equipment

Granite Creek Equipment List Quantity	
ANFO Truck with boom, Eimco 975	1
Man basket and ANFO pot	1
Single Boom Jumbo, Sandvik D05, backup unit	1
Two Boom Jumbo, Epiroc 282	1
Two Boom Jumbo, Epiroc M2C, Tunnel Manager	1
Mechanized Bolters with Screen Handler, SandvikDS310, Robolt 5, and D05.	3
LHD, 4 cubic yard, Sandvik T6	1
LHD, 6 cubic yard, Cat R1600G/H	3
U/G Articulated Truck, Cat AD30, w/ ejector bodies	4
Shotcrete Spray Truck, YMCO 462, SMD design	1
Shotcrete Remix Truck, Elmac open top and Normet Utimec 1500	1
U/G Fuel/Lube Truck, Getman 644	2

i-80 personnel are responsible for:

- Site security;
- Designation of the proper destination for mined mineralized material;

- Material movement from the portal to appropriate stockpile;
- Screening of mineralized material;
- Operation of the water treatment plant and;
- Contractor oversight.

i-80 fills some of these positions with contract labor as necessary. Table 13-10: i-80 Personnel list i-80 personnel including positions filled with contract labor.

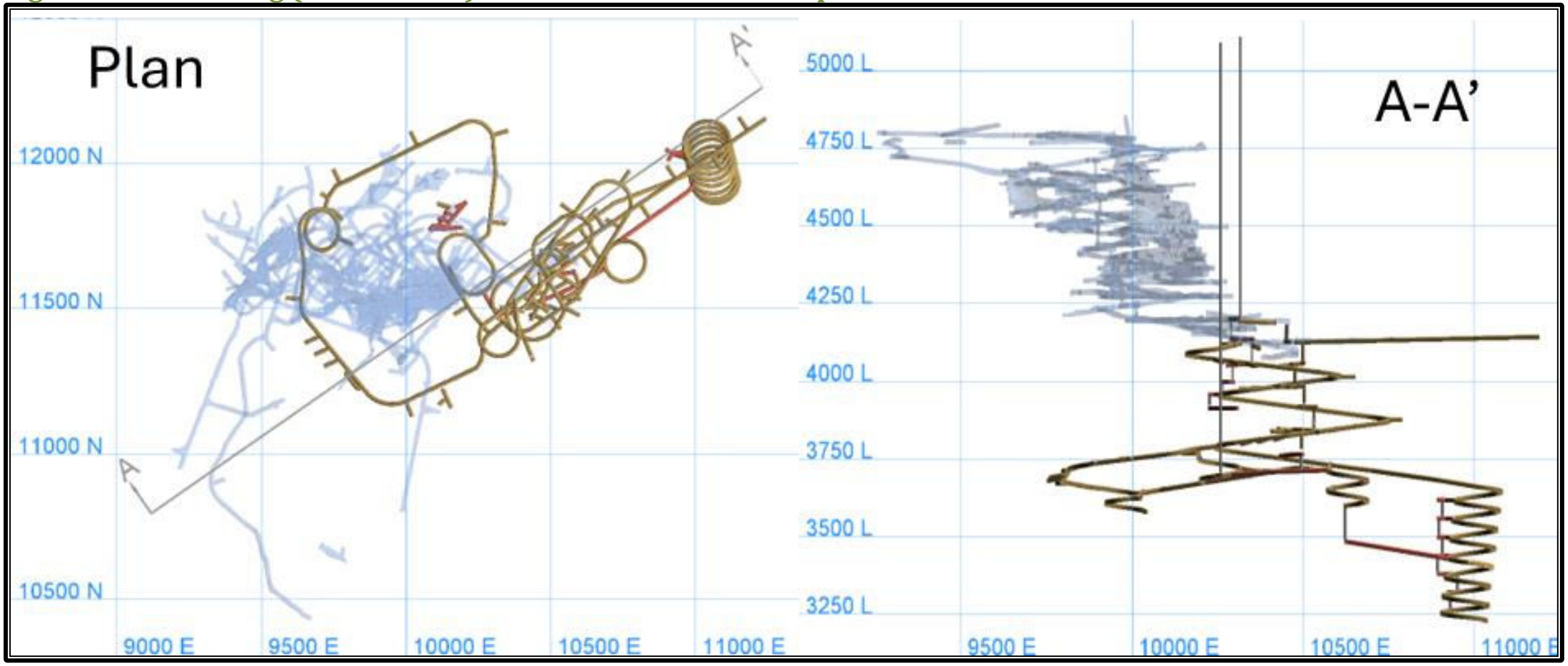
Table 13-10: i-80 Personnel

Description	Count
Manager	1
Safety/Security	6
Geology	2
Ore Control Techs	4
Engineer	1
Surveyor	1
Assay	3
Equipment Operators	4
Labor	2
Clerk	1
Janitor	2
Total	27

13.2.1 Development

Development drifting is excavated 15-feet wide by 17-feet high to allow room for a large diameter ventilation duct and 30-ton truck. Decline gradient cannot exceed +/- 13%. Ventilation is provided through a series of raises and crosscuts located inside the spiral. Two bored raises totaling 909 ft and 1,382 ft are planned to supplement ventilation in the lower levels of the mine. The life-of-mine development plan is shown in Figure 13-15.

Figure 13-15 Existing (Shaded Blue) and Planned Mine Development

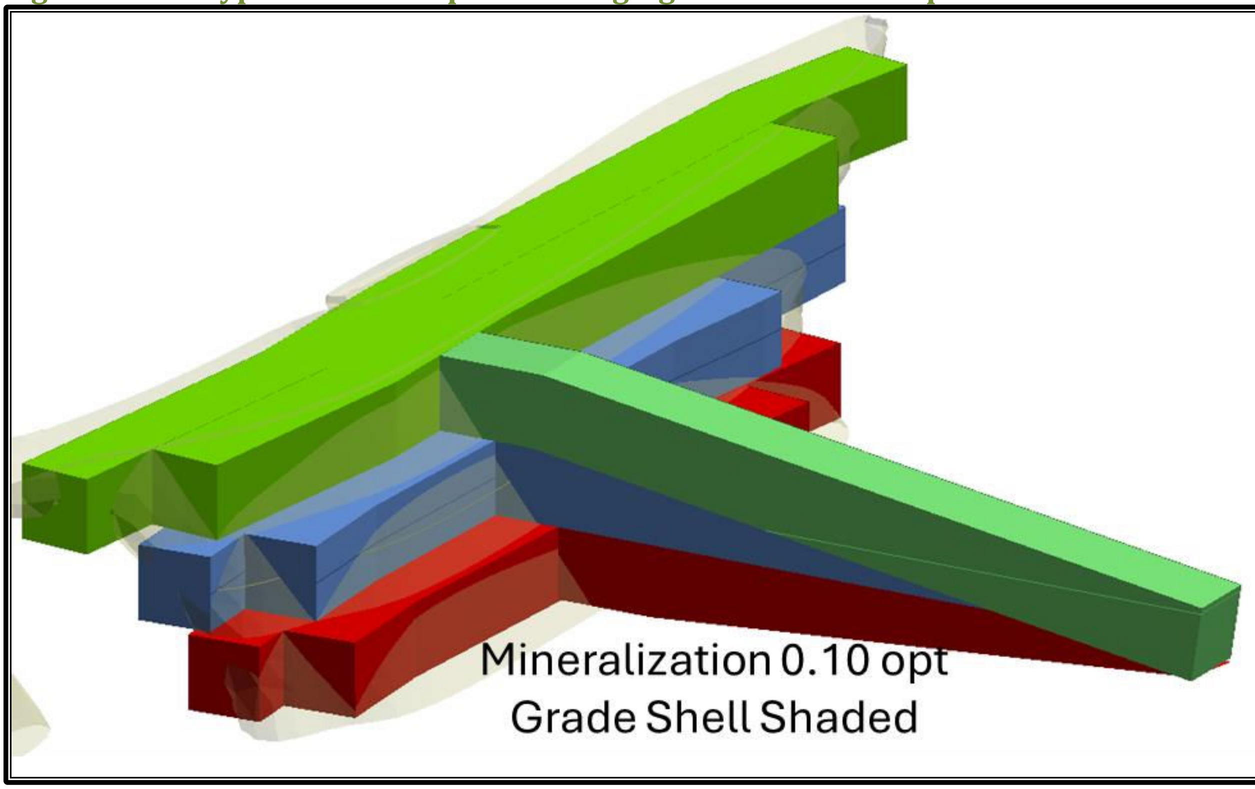


13.2.2 Production

The initial attack ramp to a set of stopes is driven at a +15% grade. Once the initial level is mined and filled the sill of the ramp is excavated to reach the next level. Three levels or 45 vertical feet of mineralization are typically excavated from a single attack ramp. Where sufficient setback distance is available an attack ramp could access four levels.

Underhand drift and fill mining is well suited to the mineralization geometry and ground conditions at Granite Creek. The cross section of individual stope cuts measures 15-feet high by 15-feet wide, large enough for six yd^3 Load Haul Dump (LHD. Mining parallel to strike is preferred. Stope cuts are mined sequentially across the mineralized zone until all mineralization above the cutoff grade is extracted. Once a level is mined and backfilled mining initiates on the next level down.

Figure 13-16 Typical 3-Cut Stope and Hanging Wall Attack Ramp



13.2.3 Ground Support

Primary ground support consists of welded wire mesh and eight-foot Swellex rock bolts with four-foot by four-foot foot spacing. Additional bolts are added to pull the wire tight to the back. Primary support is installed to completely cover the back and to within five feet of the sill. When necessary additional support may consist of two to three inches of shotcrete, 12-foot super Swellex bolts, or grouted cable bolts up to sixteen feet in length.

Figure 13-17 Primary Ground Support Installation

13.2.3.1 Backfill

All stopes where mining is planned alongside or below are backfilled with Cemented Rock Fill (CRF). Backfill aggregate is sourced from waste rock mined in the CX-West open pit. Suitable waste rock is any clean rock free of clay. It is crushed to a nominal three-inch maximum size. The aggregate is mixed with cement slurry to produce a mixture containing six to eight percent cement. Backfill is then loaded in a haul truck where it is transported to the stope. There, a modified LHD with an extended boom and push plate affixed to the end will work the CRF until all the void spaces are filled (Figure 13-18). If no future mining is planned alongside or below the stope cut it can be left open or filled with waste rock from development headings.

Figure 13-18 Cemented Rock Fill in Adjacent Cut

13.2.4 Granite Creek Mineralization Control Procedures

The ore control geologist on shift makes an effort to view each active heading. The geologist takes a photo of the face and makes notes.

Muck samples are collected by haul truck drivers at the windrow. Development headings receive one sample per round and are automatically shipped as waste unless otherwise directed by the geologist. For ore headings, a sample is collected at the rate of roughly one sample per two trucks. The driver uses a hand-held sample scoop to fill a sample bag half full (five to ten pounds), walking along the windrow and taking a scoop every five feet, making sure to collect both coarse and fine material. Sample bags have a tag with duplicate bar codes separated by a perforation. The perforated portion of the tag includes space to hand write sample source information including mine level, heading ID and distance, date and shift. The haul truck driver places completed sample bags in a designated location near the mine office trailer. The ore control geologist collects the samples accumulated from the previous day and night shifts and uses the sample tag information to generate a sample submittal for the laboratory and fill the information to the ore control database. The ore control geologist inserts QAQC samples into the sample stream. A contract driver transports muck samples, Over the Road (OTR) truck samples as well as any drill samples to the Lone Tree laboratory once per day.

The Lone Tree lab analyzes samples for Au grade by fire assay and cyanide absorption, sulfide %, TOC, CO3, and preg-rob potential. The results are used to characterize each round as oxide or autoclave refractory, high grade or low grade, or waste (Table 13-11). Assays must be approved by the database administrator before the ore control geologist can enter assay results in the ore control database and flag windrows for routing. The geologist ties color coded flagging associated with the assessed ore type to a lath at the end of the windrow. The windrow can then be moved to the stockpile corresponding to its ore type. The process typically takes three days from mine face mucking to ore type determination and flagging.

High grade oxide and sulfide ores are screened to 3 inches at the stockpile. The minus 3-inch portion is loaded for shipment to the appropriate ore processing location (autoclave or oxide mill). Screened oversize oxide material is placed in the low grade oxide stockpile, which is shipped to the heap leach facility on a low priority basis. Oversized sulfide material is transferred to long term on-site low grade sulfide stockpiles and is not shipped.

Ore is shipped to processing facilities using contract OTR trucks. The truck driver receives a ticket number at the security gate and gives the ticket number to the loader operator at the stockpile. The loader operator loads the first bucket, then spills a small portion of every other bucket into a small sample pile on the ground during the loading process. Once the loaded truck departs, the loader operator collects a sample from the sample pile and labels the bag with an ID associated with the truck ticket number. Sample bags are waterproof to preserve moisture content. The loader operator places the samples at the designated sample location at the end of the shift, where they are collected by the ore control geologist who prepares a laboratory sample submittal and enters the sample information into acQuire. Trucks are weighed near the security gate when departing the mine, and security personnel email a report of truck tons and ticket numbers at the end of the shift.

Table 13-11 Mineralization Routing Criteria

Criteria	3rd Party Refractory Autoclave	3rd Party Carbon in Leach	Lone Tree Heap Leach
Au opt	0.058	0.075	0.020 <= Au opt <= 0.075
Sulfide	>= 1%	<= 0.6%	<= 0.6%
Total Organic Content (TOC)	<= 0.5%	<= 0.50%	<= 0.50%
Carbonate (CO_3)	<= 15%	N/A	N/A
Preg Rob	<= 40%	<= 40%	<= 40%
Cyanide Solubility	N/A	>= 50%	>= 50%

13.2.5 Mine Production Plan

Individual drift advance rates are estimated to be between six and eight feet per day. The production plan presented in Table 13-12 represents the material mined from the collection of drifts available at any given time and subject to the constraints of people and equipment availability. At least eight drifts mining mineralized material are required to achieve 500 tons per day.

Table 13-12: Annual Production and Development Schedule (Including Inferred Mineral Resources)

Calendar Year	2025	2026	2027	2028	2029	2030	2031	2032	2033	Total
Mineralized Material Mined										
Total Mineralization Mined (000's Tons)	212.4	206.7	221.9	242.2	274.4	205.5	167.7	58.5	-	1,589.4
Gold Grade (Ounce/Ton)	0.328	0.394	0.341	0.346	0.324	0.316	0.316	0.354	-	0.339
Contained Gold (000's Ounces)	69.7	81.4	75.8	83.9	88.9	65.0	53.0	20.7	-	538.4
Production Mining										
Stope Development and Drift and Fill Mining (000's Tons)	212.4	206.7	221.9	242.2	274.4	205.5	167.7	58.5	-	1,589.4
Mineralization Production Rate (tpd)	582	566	608	662	752	563	460	160	-	435
Backfill										
Total CRF Backfill (000's Tons)	212.4	206.7	221.9	242.2	274.4	205.5	167.7	58.5	-	1,589.4
Waste Mining										
Expensed Waste (000's Tons)	115.0	99.9	111.3	110.8	124.3	89.5	75.6	28.0	-	754.4
Primary Capital Drifting (Feet)	6,675	7,913	5,233	1,694	-	-	-	-	-	21,515
Capital Raising (Feet)	1,050	640	180	180	150	150	-	-	-	2,350
Capitalized Mining (000's Tons)	180	201.6	116.4	38.5	1.1	1.1	-	-	-	539
Total Tons Mined (000's Tons)	507.6	508.2	449.6	391.6	399.7	296.1	243.4	86.5	-	2,882.6
Mining Rate (tpd)	1,391	1,392	1,232	1,070	1,095	811	667	236	-	658

Fifty percent of the mineral resource tons are inferred and 56% of the contained gold is inferred. The mine production plan without inferred mineral resources presented in Table 13-13 is a gross factorization of the mine plan containing inferred mineral resources. No adjustments have been made to capital development, productivities, or unit costs.

Table 13-13: Annual Production and Development Schedule (Excluding Inferred Mineral Resources)

Calendar Year	2025	2026	2027	2028	2029	2030	2031	2032	2033	Total
Mineralized Material Mined										
Total Mineralization Mined (000's Tons)	105.7	102.9	110.5	120.5	136.5	102.3	83.5	58.5	-	820.3
Gold Grade (Ounce/Ton)	31.0	36.2	33.7	37.3	39.5	28.9	23.6	9.2	-	0.292
Contained Gold (000's Ounces)	31.0	36.2	33.7	37.3	39.5	28.9	23.6	9.2	-	239.4
Production Mining										
Stope Development and Drift and Fill Mining (000's Tons)	105.7	102.9	110.5	120.5	136.5	102.3	83.5	29.1	-	791.0
Mineralization Production Rate (tpd)	290	282	303	329	374	280	229	160	-	225
Backfill										
Total CRF Backfill (000's Tons)	105.7	102.9	110.5	120.5	136.5	102.3	83.5	29.1	-	791.0
Waste Mining										
Expensed Waste (000's Tons)	57.2	49.7	55.4	55.2	61.8	44.6	37.6	14.0	-	375.5
Primary Capital Drifting (Feet)	6,675	7,913	5,233	1,694	-	-	-	-	-	21,515
Capital Raising (Feet)	1,050	640	180	180	150	150	-	-	-	2,350
Capitalized Mining (000's Tons)	180	201.6	116.4	38.5	1.1	1.1	-	-	-	539
Total Tons Mined (000's Tons)	343.1	354.2	282.2	214.2	199.4	147.9	121.1	43.0	-	1,705.2
Mining Rate (tpd)	940	970	773	585	546	405	332	118	-	389

14 Recovery Methods

14.1 Introduction

The Granite Creek processing facility was selected based on the metallurgical performance of the material. The material generally responds well to cyanide leaching but the presence of organic carbon (TOC) in some of the material hinders the gold recovery. To overcome the impacts of TOC, a carbon-in-leach (CIL) process was selected. The test work has shown that the CIL process largely overcomes the negative impacts of the organic carbon. Further, the CIL test work indicates a substantial gold recovery benefit compared to heap leaching. Heap leaching may still be a future processing option for lower grade materials but the capacity is largely driven by the gold cutover grade. At current gold prices, the use of heap leaching is not justified.

Oxide production from the Granite Creek underground operation will be processed through the planned Granite Creek CIL plant on site.

Refractory production from the Granite Creek underground operation will be initially processed via milling, pressure oxidation followed by carbon in leach (CIL) or roasting followed by CIL at a third party processing facility. The most recent metallurgical testing is described in Section 9, Mineral Processing and Metallurgical Testing, and supports processing parameters at these facilities at NGM.

Granite Creek underground production will be classified based on gold grade, level of oxidation and refractory characteristics (e.g. presence of preg-robbing components in ore, refractory sulfide components) that contribute to recovery at processing facilities and will be routed based on an integrated process production plan that is devised for maximum economic returns. Generally, materials grading >0.080 oz/ton will be routed to a third party processing facility.

Once operational in 2028, production will be processed at i-80's Lone Tree pressure oxidation (POX) operation.

14.2 Process Description

The Granite Creek project would employ open pit mining with a CIL system on a 365 days per year 24 hour per day basis with 91% availability. Run-of mine (ROM) material at a nominal size of approximately 200 mm (8 inches) will be fed into a large jaw crusher. The jaw crusher would be equipped with a dump pocket capable of allowing direct dump from haul trucks or loaders. The dump pocket will have a static grizzly and hydraulic rock breaker to handle oversize material.

The jaw crusher is also equipped with a vibrating grizzly feeder that allows undersize to pass and oversize is fed to the crusher. The crusher will operate to produce a nominal P80 of 100 mm (4 inch). Crushed material will advance to a live stockpile equipped with vibrating feeders to feed the downstream semi-autogenous (SAG) mill. Powdered lime will be added to the SAG mill feed belt via a silo and screw feeder.

The grinding circuit consists of a single SAG and ball mill. The SAG mill grinds the material to a nominal 19 mm (¾ inch) and discharges across a trommel screen to reject oversize scats. The slurry reports to the mill discharge sump that feeds the cyclone pack. The cyclone underflow discharges to a ball mill that discharges back into the common mill sump. The ball mill is also fitted with a trommel to remove oversize scats. The cyclone overflow has a P80 target of 75um at 35% solids.

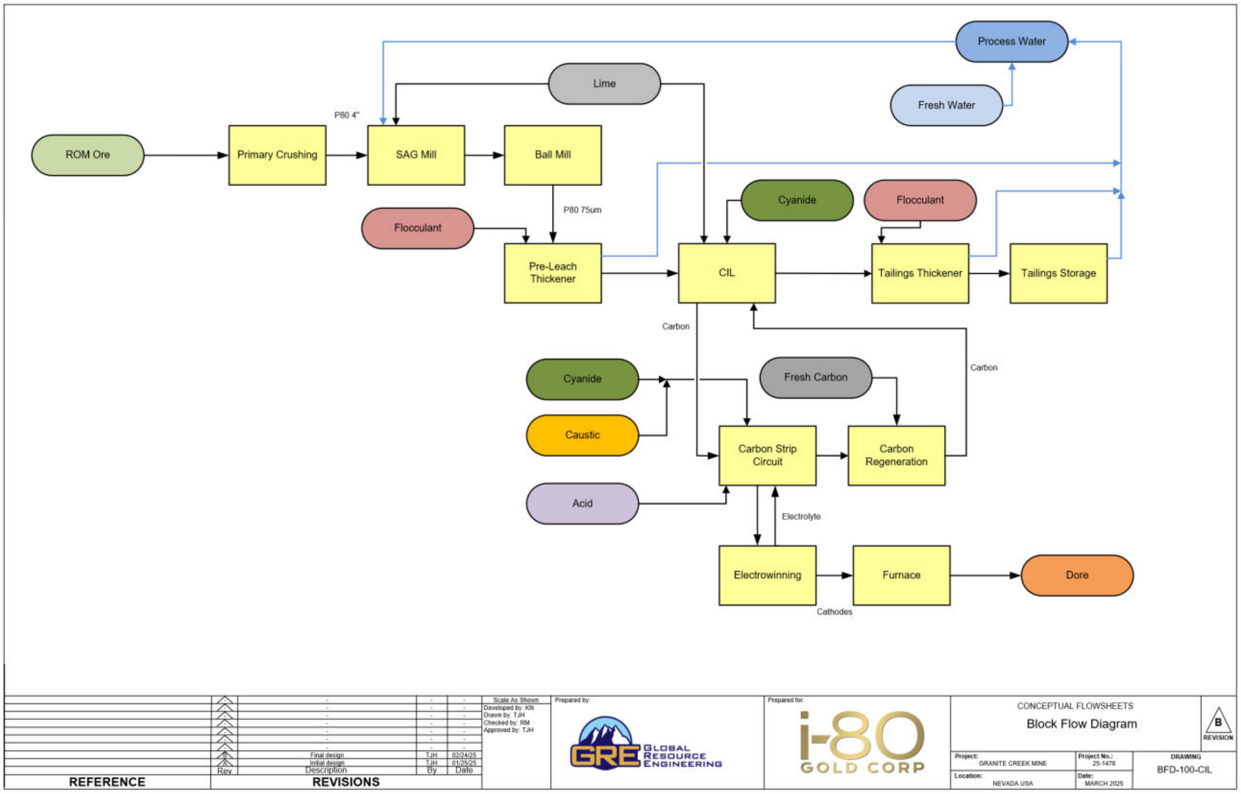
A pre-leach thickener is then used to thicken the ground material and flocculant is added to improve settling rates. The thickener underflow is then pumped to a series of CIL tanks, where the slurry flows countercurrent to the activated carbon. Cyanide and lime are added in the CIL circuit as required. The thickener overflow reports to the process water tank.

Gold is extracted via cyanidation and quickly adsorbed onto the active carbon. Carbon flows counter-current to the slurry and is recovered from the first tank. Inter-tank screens prevent the carbon from leaving the CIL tanks and recessed impeller pumps advance the carbon.

The loaded carbon extracted from tank 1 is transferred to a large column vessel. This carbon is then treated with a dilute acid wash to remove any calcium deposits. After water rinsing, the carbon is then eluted with cyanide and caustic to remove the adsorbed gold and silver. The resulting pregnant solution is then pumped to the electrowinning process and the precious metal cathodes are smelted into doré bars.

The eluted carbon is transferred to a regeneration kiln. Thermal regeneration is used to reactivate the barren carbon from the elution column. This reactivated carbon is pumped back into the last CIL tank. Fresh carbon may be added to this stream as well. Tailings from the CIL tanks are pumped to a tails thickener with the underflow reporting to the tailings storage facility (TSF). The overflow reports to the process water tank. Figure 14-1 shows the conceptual flowsheet.

Figure 14-1: Conceptual Flowsheet



14.2.1 Crusher Circuit

The crusher is designed to process approximately 555 metric tonnes per hour on an 18-hour basis. The crusher capacity is designed for 10,000 metric tonnes per day (tpd) at 75% availability. The jaw crusher is 1.22x1.52 meters (48x60 inch) with a 220 kW (300 HP) motor.

The run of mine feed passes over the static grizzly to reject oversize. The crusher is equipped with a hydraulic rock breaker. The feed then passes to a vibrating grizzly with a 150-mm (6 inch) opening. The undersize reports directly to the jaw crusher discharge conveyor (CV-110) while the oversize feeds the jaw crusher. The jaw crusher would crush to a nominal 100-mm (4 inch), with the crushed product reporting to a live stockpile.

The crusher discharge conveyor has a metal detector, magnet, and weigh scale. The stockpile has a live capacity of approximately 10,000 tonnes and has two vibrating feeders that feed the mill feed conveyor.

14.2.2 Grinding Circuit

The mill feed conveyor (CV-150) feeds the SAG mill. Lime is added to the belt from a silo with a screw feeder. The conveyor has a magnet, metal detector, and weigh scale. The conveyor feeds the primary semi-autogenous mill (SAG). Process water is added to the feed to achieve the desired percent solids.

The selected SAG mill is 9.75m diameter by 3.35 long (32' diameter by 11' long) with a 5,000 kW (6800 HP) motor. The SAG mill has a trommel to discharge oversize material (scats) out of the circuit. No provision has been made for a pebble crusher to reprocess the scats in this design.

The SAG mill discharge reports to a common mill sump. The mill sump material is pumped to a cyclone pack (10 x 0.66-meter diameter cyclones) with the cyclone underflow reporting to the ball mill and the overflow reporting to the grind thickener. No gravity circuit has been included in the design.

The ball mill selected for this circuit is 7.31-meter diameter by 12.19-meter long (24-foot diameter by 40-foot long) equipped with a 7,134 kW (9700 HP) motor. The target P80 of the cyclone overflow is 75 um.

The cyclone overflow reports to the stock tank which is then pumped to a 63-meter diameter grind thickener. The grind thickener increases the solids density to reduce the size of the CIL tanks and provide improved density control.

14.2.3 Carbon in Leach (CIL) Circuit

Grind thickener underflow is pumped to the CIL circuit consisting of six mechanically agitated tanks operating in series designed to provide 48 hours of retention time. Each tank has a live volume of approximately 5,920 m^3 with an 85% volume utilization. Slurry flows sequentially through tanks with the activated carbon retained in each tank by inter-tank screens. Carbon is advanced from the end of the train to the front of the train sequentially using recessed impeller pumps. The carbon flows countercurrent to the slurry. As gold is extracted from the ore, it is adsorbed onto activated carbon. The loaded carbon is extracted from tank 1 via pumping slurry across an external vibrating screen. The slurry returns to the tank and the carbon is then transported to the acid wash column.

To replace the loaded carbon removed in tank 1, regenerated or fresh carbon will be pumped to CIL tank 6. Slurry discharging from tank 6 gravitates to a carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. The slurry then proceeds to the tailings thickener.

14.2.4 CIL Strip Circuit

The screened loaded carbon from the first CIL tank is pumped to the acid wash column. The loaded carbon is acid washed with dilute hydrochloric to remove calcium and adsorbed metals. The spent acid is neutralized and disposed of. After acid washing, the carbon is rinsed with water before gold and silver elution.

Elution is conducted by the modified ZADRA system at a rate of 18 tonnes/day. A solution of caustic and cyanide is passed through the elution column to remove the adsorbed gold. The rich electrolyte is pumped to electrowinning cells, where the gold and silver are recovered on the cathodes. The cathodes are washed, and the recovered sludge is refined in a conventional induction furnace after drying. The circuit is designed to conduct two daily strip cycles with a total metal recovery of approximately 740 oz per day (90% gold). The doré produced is assayed and stored in a vault before being shipped off-site for payment.

Barren carbon from the elution column is returned to the CIL circuit after passing across a carbon sizing screen. Fine carbon from the screen underflow is stockpiled and sent for separate off-site recovery. Approximately 50% of the barren carbon reports to an indirect fired kiln for thermal regeneration. The regenerated carbon reports to a quench tank before being pumped to the carbon sizing screen. Fresh makeup carbon is first sent to an attrition tank for fines removal before being pumped to the carbon sizing screen. The fine carbon from the screen underflow is captured in a plate and frame filter.

Tailings from the CIL circuit pass to a 63-meter diameter thickener and are thickened to 60% solids density. The thickener underflow reports to the tailings storage facility (TSF) and the overflow reports to the process water tank. Cyanide destruction may be required on this solution before it is circulated back to the circuit. The comminution circuit does not employ cyanide because of variable TOC grades.

14.3 Refractory Processing

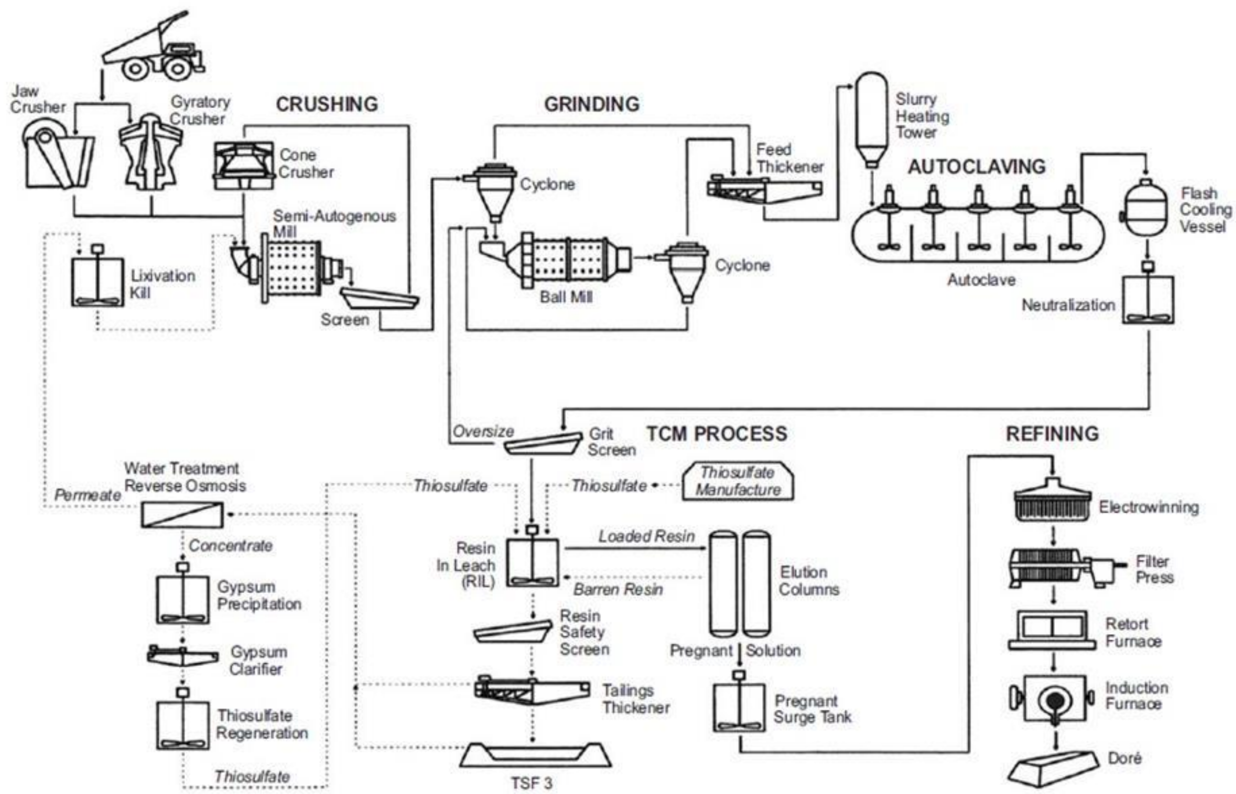
14.3.1 Third Party Processing

The third party autoclave circuit processes 4 - 5 million tons per year and consists of primary crushing, two parallel semi-autogenous grinding (SAG) Mill-Ball Mill grinding circuits with pebble crushing, five parallel autoclaves capable of acid pressure oxidation (POX) and three of which are capable of alkaline POX, two parallel calcium thiosulphate (CaTS) leaching circuits with resin-in-leach (RIL), electrowinning for gold recovery, and a refinery producing doré bullion from both autoclave and roaster circuits.

Gold recovery estimates are based on both testwork and operational history at both facilities with curves utilized for both depending on operating strategy and ore characteristics.

The current autoclave LOM has an average recovery of 50% when running solely alkaline ore (one SAG-Ball Mill circuit and 3 autoclaves to process 4.0 million tons per year) and an average of 73% after converting the RIL circuit to CIL and running single refractory ore. The average LOM gold recovery is 65%. The simplified POX flowsheet is shown in Figure 14-2.

Figure 14-2 Third Party POX Simplified Flowsheet



14.3.2 Lone Tree Pressure Oxidation Facility

i-80 Gold plans to process single refractory mineralization from Granite Creek at their Lone Tree Mill in a hub and spoke arrangement.

14.3.2.1 Lone Tree Mill Historic Processing

The Lone Tree Mine is located immediately adjacent to I-80, approximately 12 miles west of Battle Mountain, 50 miles east of Winnemucca, and 120 miles west of Elko. Mining commenced at Lone Tree in April 1991 with the first gold pour in August of 1991. In 1993, a POX circuit was added to the facility, which included a SAG / ball mill circuit, followed by a thickening circuit, the POX process for refractory gold ores, and finally CIL, carbon stripping, and refining.

In 1997, a 4,500 tpd flotation plant was constructed to make concentrate to supplement the feed to the POX circuit, as well as to ship excess concentrate to Newmont's Twin Creeks POX plant or to its Carlin roaster. The Lone Tree processing facilities were shut down at the end of 2007. Since that time, the mills have been rotated on a regular basis to lubricate the bearings. In general, the facility is still in place with most of the equipment sitting idle.

i-80 Gold Corp's objective is to refurbish and restart the POX circuit and associated unit operations, including the existing oxygen plant, as it was operating before the shut-down, while meeting all new regulatory requirements. The flotation circuit is not being considered for restart. The POX circuit will have the capability to operate under either acidic or basic conditions.

In order to restart the process plant, new environmental regulations in relation to allowable mercury emissions must be met. In February 2011, the NDEP and the EPA brought about new standards to limit mercury emissions to 127 lb of mercury for every million tons of ore processed. In order to meet this requirement, the Lone Tree facility will require several environmental upgrades prior to restarting.

14.3.2.2 Lone Tree Facility Block Flow Diagram

A block flow diagram for the Lone Tree Mill facility is included in Figure 14-3. The block flow diagram contains the following major processing areas:

Ore Reclaim, Grinding and Thickening and Acidulation

Pressure Oxidation

POX Off-gas Treatment and Quench Water Loop

Neutralization, Carbon-in-Leach, and Cyanide Destruction

Tailings Thickening and Filtration

Acid Wash, Carbon Stripping, and Carbon Regeneration

Electrowinning and Refinery

Plant and Instrument Air

Oxygen Plant

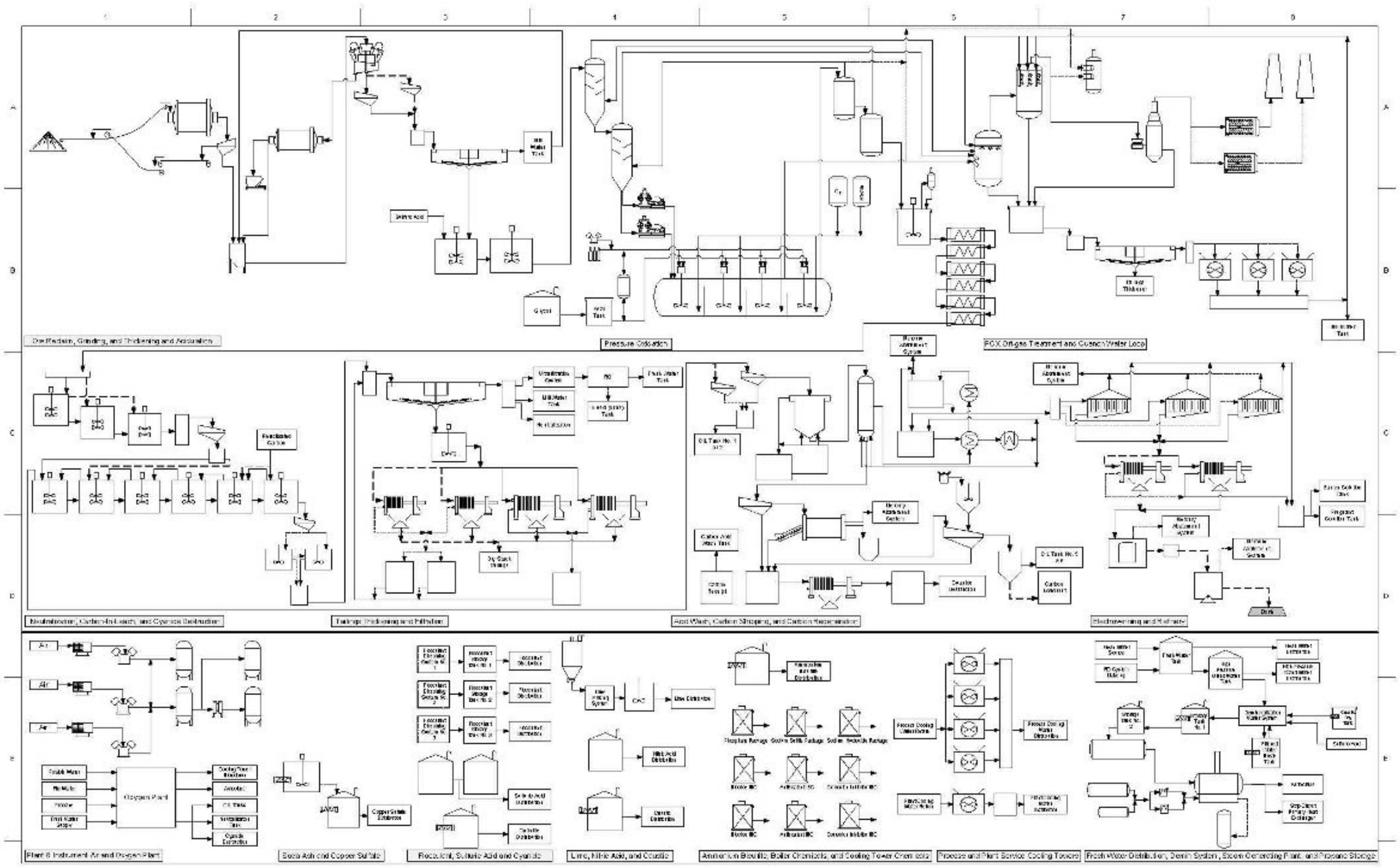
Reagent Preparation and Storage

Process and Plant Service Cooling Towers

Water Distributions

Steam Generating Plant and Propane Storage.

Figure 14-3 Lone Tree Facility Block Flow Diagram



Source: i80Gold (2025)

14.3.3 Key Design Criteria

The Lone Tree Pressure Oxidation (POX) Facility restart will have minimal changes made from the 1993 Process Design Criteria (PDC). A new PDC was developed based on the expected production sources as defined by i-80.

Key process design criteria are summarized in Table 14-1.

Table 14-1: Summary of Key Process Statistics

Criteria	Units	Value
Annual Mill Throughput	tons	912,500
Daily Throughput (per calendar day)	**tons**	**2,500**
Operating Throughput of Ore to Autoclave Circuit (LTH feed)	tph	122.5
Operating Time / Availability	%	85
Design Sulfur Treatment Rate	tph S	2.7
Gold Recovery	%	*Varies*
Silver Recovery	%	*Varies*

14.3.4 Lone Tree Facility Description

14.3.4.1 Mill Feed Reclaim

The purpose of the Mill feed reclaim area is to store and reclaim material for processing, which has been shipped to the lone tree processing facility via highway ore trucks.

Run of mine (ROM) crushed material is delivered to the stockpile area. Material from various mining locations, namely Granite Creek, Cove, and Archimedes, is dumped at designated locations within the storage area and blended into facility feed stockpiles.

The stockpile area will have the capacity to store multiple days of mined and crushed material to accommodate the production shipment schedule to site. Additionally, the reclaim area is utilized for feed blending for the POX circuit. This blending will be used to manage the sulfide sulfur concentrations, gold grades, and carbonate grades through the autoclave to ensure stable circuit operation within the design window for the plant.

14.3.4.2 Comminution

The purpose of comminution area is to reduce the particle size of the feed ore to the target autoclave circuit feed size for sufficient sulfide oxidation kinetics and gold recovery within the autoclave. The comminution area contains an SABC circuit with a dedicated SAG (semi-autogenous grinding mill) and ball mill to reduce the feed particle size to the target grind size. The SAG mill is fed via a conveyor from the dump hopper. The ball mill cyclone overflow is directed to the POX feed thickening conveyor.

14.3.4.3 Thickening and Acidulation

The purpose of the thickening area is to prepare the slurry for autoclave process by densifying the product of the grinding circuit to improve storage capacity of the downstream slurry storage tanks, improve the autoclave heat balance by reducing the water transferred to the autoclave and improving the possible solids flow through the autoclave feed pumps. The dense slurry is stored in two acidulation tanks that provide a combined storage / acidulation retention time of 12 hours. The acidulation tanks ensure a continuous feed to the autoclave plant, unaffected by upstream throughput variations.

14.3.4.4 Pressure Oxidation

The POX autoclave circuit includes the slurry pre-heaters, autoclave feed, autoclave, and the POX ancillary services: autoclave agitator seal system, oxygen supply, high pressure cooling water, and high-pressure steam. The Lone Tree Facility restart includes provisions to operate the circuit in alkaline or acidic modes depending on the feed carbonate concentration among other factors.

14.3.5 Slurry Heaters

The purpose of the slurry heaters is to capture excess energy discharged from the autoclave and pre-heat the feed slurry prior to the autoclave process reducing the total energy input required to operate the autoclave. The heating is achieved in two stages consisting of a series of two refractory lined counter-current splash slurry heater vessels. The heat source is flashed steam released from the autoclave discharge slurry during the pressure letdown process. The splash slurry heaters are direct contact heat exchanger and provide a means of heat recovery via steam condensation. This reduces the off-gas load on the downstream off-gas equipment and reduces the required input steam.

14.3.6 Autoclave Feed

The purpose of the autoclave feed area is to increase the pressure of the pre-heated slurry to above the autoclave operating pressure to facilitate transfer into the autoclave at the required pressure using the autoclave feed pumps.

14.3.7 Autoclave

The purpose of the autoclave is to oxidize the refractory sulfide minerals under acidic or alkaline conditions to liberate the gold trapped in the sulfide sulfur minerals. The autoclave at Lone Tree is designed to operate at 389 °F and 297 PSI(g) with a slurry residence time of 40 - 50 minutes and consists of 4 compartments. The design expects a 78% - 97% cumulative sulfide sulfur oxidation through the autoclave depending on operating conditions. In either operating condition high purity oxygen is introduced to all four compartments of the autoclave at controlled rates to oxidize the fed sulfide minerals. Due to the low sulfur grades steam is required to be continuously fed to the autoclave to maintain the kinetically required oxidation rates to achieve the sulfide sulfur oxidation extent. The autoclave slurry is discharged through a level control choke valve and is fed to the high pressure flash vessel.

14.3.8 Flash System

The purpose of the flash system is to reduce the pressure and temperature of the autoclave discharge, making it suitable for subsequent unit operations downstream. The oxidized slurry undergoes a controlled pressure and temperature reduction process as it passes through two stages of flashing vessels located downstream of the last autoclave compartment.

14.3.8.1 POX Off-gas Treatment

The purpose of the POX off-gas treatment area is to effectively eliminate particulate matter present in the POX vent stream, while simultaneously reducing the temperature and volume of the vent gas through direct contact condensation. This process serves to alleviate the burden imposed on downstream equipment, ensuring their optimal performance, and mitigates the environmental impact by minimizing emissions. The off-gas treatment circuit also includes a mercury removal step to minimize autoclave mercury emissions to the environment.

14.3.8.2 Slurry Coolers

The purpose of slurry coolers is to reduce the temperature of the incoming slurry from the lowpressure flash vessel to prepare it for the downstream neutralization and CIL circuits through a series of water cooled shell and tube heat exchangers.

14.3.8.3 Neutralization

The purpose of neutralization circuit is to neutralize all free acid in the slurry, precipitate the heavy metals as their hydroxides and raise the pH to approximately 10 to ensure cyanide stability in the CIL circuit for personnel safety and process optimization. The neutralization circuit is dosed with lime slurry to raise the pH of the autoclave discharge slurry. The neutralized slurry from this circuit is then fed to the CIL circuit for gold recovery.

14.3.8.4 Carbon-in-Leach

The purpose of CIL circuit is to leach and extract gold and silver from the oxidized slurry from neutralization using cyanidation and carbon adsorption. The CIL circuit provides retention time of 24 to 28 hours. The CIL circuit consists of 6 mechanically agitated tanks arranged in a series. The agitators prevent solid settlement and maximize contact time to improve gold and silver recovery. The carbon flows counter current to the slurry flows and the loaded carbon is sent to an elution circuit for carbon stripping and regeneration. Unloaded carbon is fed into the last tank of the CIL circuit. The leached slurry is transferred from to the cyanide destruction circuit.

14.3.8.5 Elution

The purpose of the elution circuit is to elute precious metals from the loaded carbon and transfer the resulting loaded solution of high gold concentration (pregnant eluate) to the refinery to generate doré.

14.3.9 Carbon Acid Wash

The purpose of acid wash is to rinse the loaded carbon form CIL with dilute nitric acid solution prior to the carbon stripping process. Carbonate scale builds up on the activated carbon during the CIL process and fouls the carbon's adsorption properties by depositing a layer of scale. If left intact, over time the scale will limit the adsorption capacity of the carbon and will cause softening of the carbon in the regeneration kiln. The loaded carbon from CIL is first treated within the carbon acid wash vessel prior to treatment within the carbon stripping vessel.

14.3.10 Carbon Stripping

The purpose of the carbon strip circuit is to strip the cleaned loaded carbon from the acid wash vessel of the adsorbed gold using a Pressure ZADRA Strip scheme. The ZADRA strip uses several bed volumes of a recirculated solution to strip the precious metals off the loaded carbon. The cyanide solution is buffered by caustic to assist with gold elution. The stripped carbon is then sent to carbon regeneration circuits. The loaded solution is next processed in the electrowinning circuit.

14.3.11 Elution Mercury Abatement

The purpose of elution mercury abatement system is to condition the off gas leaving the pregnant and barren solution tank to remove fine particulate, solution aerosols and condensed and gas phase mercury.

14.3.11.1 Carbon Regeneration

The purpose of the carbon regeneration circuit is to restore the activated carbon's ability to recover gold from the cyanidation circuit solutions. The circuit also permits the introduction of new carbon to the process and removes carbon fines from the process.

14.3.12 Carbon Regeneration Kiln

As carbon is used in the CIL and elution circuits, the surface and internal pore structure becomes contaminated with organic species. The organics foul the carbon, slow the gold adsorption rate, and decrease the gold loading capacity of the carbon. The carbon reactivation electric kiln is a horizontal rotary kiln that is specifically designed for this purpose.

14.3.13 Carbon Fines Handling

Carbon fines are transferred by gravity from the reactivated carbon vibrating screen, carbon reactivation feed vibrating screen, kiln feed hopper, and carbon reactivation electric kiln. The carbon fines are dewatered in a filter press and discharged into supersacks for external sale.

14.3.13.1 Refinery

The purpose of the refinery circuit is to recover gold cyanide solutions via electrowinning and produce doré bullion bars.

14.3.14 Electrowinning

The purpose of the electrowinning (EW) circuit is to recover gold from the pregnant solution by applying a voltage across electrodes immersed in the pregnant solution. Rich solution from the pregnant solution tank is transferred through the EW cells to electrowin the gold.

14.3.15 Refining

The purpose of the refining process is to produce doré bars void of other contaminants including but not limited to mercury.

The sludge from the EW cells is first processed in a mercury retort oven to remove the co-captured mercury from the precious metals recovery steps. The retorted gold sludge is then processed in a melt furnace to produce the final mine grade doré bars.

14.3.15.1 Cyanide Destruction

The purpose of the cyanide destruction circuit is to effectively reduce the concentration of cyanide in the final tail discharge and the recycled process water, ensuring compliance with predefined environmental standards and regulations and improving the safety of the operation by reducing cyanide concentrations outside of the CIL and elution circuits. The circuit targets a specific concentration limit of 2.5 mg/L of residual weakly acid-dissociable cyanide (CN_{WAD}). This reduction is accomplished through the application of the SO_2/air cyanide destruction process, which oxidizes the cyanide to meet the required concentration level. The cyanide destruction circuit is fed directly from the slurry discharge from the CIL circuit.

14.3.15.2 Tailings Preparation

The purpose of the tailings circuit is to increase the density of the detoxified tailings to aid with dry stacking of tailings residue. Additionally, this circuit produces process water for internal use within the facility. The tailings preparation circuit consists of a thickener as a first stage of solids densification. The thickener underflow is then fed to a tailings filtration circuit which dewaters the tailings sufficiently to support tailings dry stacking. The de-watered tailings from the filter presses are then dry stacked at the tailings storage facility.

The water removed from the tailings slurry is used as process water within the facility to offset water requirements. Excess process water is processed via a reverse osmosis circuit to provide supplemental permeate water to offset fresh water requirements.

14.3.15.3 Water Distributions

There are eight types of defined water services at Lone Tree:

Fresh water – generally used for reagent make-up and water washing streams.

Gland water – used to supply gland water to slurry pumps.

Mill water – used to provide dilution water within the milling circuit.

Potable water – used for safety showers and sanitary uses.

Demineralized water – primarily used to supply the steam generating plant.

Process water – used for washing and slurry dilutions. Additionally, generally feeds the reverse osmosis circuit to generate permeate water.

Quench water – used within the POX off-gas circuit as the source of direct cooling water.

Excess water – discharged from the main processing facility to the existing heap leach facility for treatment.

14.3.15.4 Solution Cooling

The purpose of the cooling area is to reject heat absorbed within the process to atmosphere. The solution cooling area includes the process service cooling circuit and the plant service cooling circuit. The process cooling circuit rejects the heat from the autoclave cooling circuit and the elution circuit heat exchangers. The plant service cooling circuit provides trim heat rejection from various equipment support systems throughout the design.

14.3.15.5 Reagents

Each set of compatible reagent preparation and storage systems is located within dedicated containment areas to prevent erroneous mixing of reagents. Storage tanks are equipped with level indicators, instrumentation, and alarms to reduce the risk of spills during normal operation. Appropriate ventilation, fire and safety protection, safety shower stations and Safety Data Sheet stations are located throughout the facility.

14.3.16 Oxygen Plant

High purity oxygen is primarily used for oxidation of sulfide during the POX process, of iron conversion from ferrous to ferric in the neutralization circuit, and of cyanide to cyanate in cyanide destruction. Furthermore, during cyanidation, the addition of oxygen maximizes the rate of gold dissolution. At Lone Tree, a cryogenic ASU produces high purity oxygen. The unit uses pressure swing adsorption technology for front end purification and production of high-pressure oxygen at 95% purity.

14.3.16.1 Instrument and Plant Air

The Lone Tree facility includes separate instrument and plant air systems to support the facilities air requirements.

14.3.17 Utilities Consumption

The plant consumptions for water and power are provided for the average processing case below and consider the design blend of material to be processed within the Lone Tree Facility for the design life of operation.

14.3.17.1 Water Consumption

Table 14-2 provides a summary of the water consumption by type for the Lone Tree processing facility.

Table 14-2: Lone Tree Facility Water Consumption by Type

Type	Consumption (gpm)
Mill Water	1 550
Fresh Water	570
Permeate Water	195
Low Pressure Gland Water	105
High Pressure Gland Water	170
Demineralized Water	110
Potable Water	15

14.3.17.2 Electrical Power Requirements

The estimated annual electrical energy requirements for the Lone Tree processing facility are summarized by area in Table 14-3.

Table 14-3: Lone Tree Facility Energy Usage by Area

Area	Annual Energy Consumption (MWh/y)
000 – General Plant Wide	2 250
180 – Water System	930
181 – Potable Water	240
182 – Process Water (RO and Process Water Tank)	4 900
210 – Ore Reclaim	770
240 – Refinery	2 310
241 – POX Grinding	26 920
242 – POX Grinding Thickening and Acidulation	1 890
244 – Neutralization and CIL and Acid Storage	6 540
245 – Carbon Stripping	4 090
247 – CND	690
248 – Reagents	2 640
249 – Plant Air and Propane	3 310
250 – Pressure Oxidation (POX) and POX Utilities	15 540
251 – POX Demineralized Water System	2 660
275 – Tailings Filtration	13 690
300 – Plant Wide Electrical and Instrumentation	4 000
305 – ABS and CN Storage	160
320 – POX Mercury Abatement	900

Area	Annual Energy Consumption (MWh/y)
340 – Quench Water Treatment	4 020
255 – Oxygen Plant	40 090
099 – Existing Plant Areas	3 570
Total	**142 090**

15 Infrastructure

15.1 Operations Dewatering

There are four dewatering wells operating (APW-1, BPW-3, BPW-5 and GCW-6), three in place at the time of purchase. one of which was deepened, and one new drilled by well i-80 in 2022. A fifth well exists but does not benefit the current mining and is not in operation. The wells are pumping from the CX block hydrogeologic unit at a combined average rate of approximately 1,410 gpm with an additional pumping of approximately 1,000 gpm collected from sumps in the underground mine workings (Figure 7-25). Water is discharged to the rapid infiltration basins (RIBs) for infiltration back into the downgradient alluvial basin aquifer. A pipeline has been constructed to connect the dewatering circuit to the treatment circuit so the dewatering water can be treated if required, as well operations requirements for dust suppression and backfill material. The dewatering well pump parameters are listed in Table 15-1.

15.2 Operations Monitoring Wells and VWPs

Monitoring wells and VWPs are used to collect hydrogeological data in support of mining operations. Currently, there are 41 active monitoring wells and 15 active vibrating wire piezometers across 5 locations (Figure 7-25). Construction and recent water level data are provided in Table 15-2..

15.3 Operations RIBs

Water from the dewatering wells that is not utilized for operations is currently discharged to Rapid Infiltration Basins (RIBs) on the east side of Getchel Mine Road through HDPE pipelines. Two of the four permitted RIBs (NEV2005102) have been constructed to date, with discharge to one of the two cells at any given time (Figure 7-25). When RIB maintenance is required, discharge is routed to the dormant cell. Current dewatering efforts are well under the permitted 6,900 gpm threshold of the RIBs and the RIB infiltration is sufficiently limiting surface ponding in the active cell.

A portable rental water treatment plant capable of 800gpm is in use currently to treat water from the two deepest wells (BPW-5 and GCW-6) which do not meet minimum requirement quality for direct discharge to the RIBS. The mag pit serves as a diversion route for plant upsets, but is limited by a total number of gallons discharged.

Figure 15-1 Water Treatment Plant

15.4 Operations Water Supply

Well WW-8, east of the Getchell road supplies potable water well to the Project (Figure 7-25). The well is completed in basin alluvial deposits to a depth of 580 ft and equipped with a pump capable of supplying 60 gpm.

15.5 Underground Development

The mine is accessed through either of two portals, and dual egress has been established for most areas of the mine. Over 9,000 feet (2,743 meters) of underground workings have been completed Where dual egress is not possible, rescue chambers have been installed. Equipment is repaired in an underground mine shop. Air doors and a ventilation fan provide required air supply to the workings in compliance with Mine Safety and Health Administration (MSHA) standards.

15.6 Other Infrastructure

Existing infrastructure at the Project includes an office building, dry and warehouse facilities, and a lined stockpile area on the surface. Landline telephone and digital subscriber line service are available at the Project site. Cellular phone service is also available, but is dependent on the strength of receiving antennas, topography, and lines of sight. A fiber optic line provides wifi throughout surface infrastructure and key areas of the underground to support phone, radio, and process control instrumentation.

Table 15-1 Dewatering Well Completion Details

	Mine Coordinates		Elevation	Casing Diameter	Well Depth	Static Water Level	Screened Interval	Average Pumping Rate	Pump Power	Pump Set-Depth
Well Identifier	Easting (ft)	Northing (ft)	(ft amsl)	(inches)	(ft bgs)	(ft bgs)	(ft bgs)	(gpm)	(hp)	(ft bgs)
APW-1	9890.2	10154.3	4722.3	18	620	380	120-140 160-180 200-600	210	100	574
BPW-3	10188.9	9474.8	5057.1	18	1391	780	500-540 580-620 660-700 740-1380	750	400	1307
BPW-5	10387.1	11126.9	5093.9	18 12	2222	708	679 - 1380 1400 - 2222	380	200	1290
GCW-6	10310.9	11742.9	5153.2	14	2093	742	803-2083	110	150	1950

Table 15-2: Summary of Locations, Construction Information and Water Levels for Dewatering Wells, VWPs, Monitoring Wells and Piezometers

Identifier	Local Mine Coordinates Easting (ft)	Local Mine Coordinates Northing (ft)	Elevation of Land Surface (ft amsl)[a]	Year Completed	Inclination (degrees)[b]	Open Interval of Well or VWP Setting Depth (ft bls)[c]	Open Interval of Well or VWP Setting Elevation (ft amsl)	Geologic Unit(s)	Static Water Level Depth (ft bls)[d]	Static Water Level Elevation (ft amsl)[e]	Static Water Level Date (dd/mm/yyyy)	Comments
Dewatering Wells												
APW-1	9889.60	10152.60	4722.28	2005	-90	120 to 600	4602 to 4122	Ocl	308.1	4414.18	27/08/2024	CX Pit (active)
BPW-2	9804.81	10554.13	4762.98	2008	-90	200 to 920	4563 to 3843	Ocl	385.4	4377.58	23/11/2024	CX Pit (inactive)
BPW-3	10188.94	9474.81	5057.14	2008	-90	500 to 1380	4557 to 3677	Ocl	662.53	4394.61	10/07/2024	South of CX Pit (active)
BPW-4	10806.54	9132.50	5011.94	2008	-90	540 to 1380	4472 to 3632	Ocl	627.5	4384.44	07/10/2024	South of CX Pit (inactive)
BPW-5	10387.06	11126.94	5093.90	2008 / 2024	-90	679 to 2222	4415 to 2872	Ocu; Ocl	703.06	4390.84	17/12/2024	Between Mag and CX West Pits (active)
GCW-06	10310.90	11742.90	5153.20	2022	-90	803 to 2083	4350 to 3070	Ocu; Ocl	1852.3	3300.90	27/12/2024	Between Mag and CX Pits (active); deepened 2025
Water Supply Wells												
WW-8	15141.92	8899.17	4756.11	1987	-90	210 to 560	4546 to 4196	Qal	190	4566.11	31/08/2000	East side of county road (inactive); no sounding port for manual DTW
VWPs[a]												
iGS22-17	11081.00	11264.70	4830.70	2022	---	---	---	Ocu				Mag Pit (active)
iGS22-17D_4188	10801.60	11478.90	---	2022	-62	736	4188	Ocu	---	---	---	
iGS22-17C_3735	10588.50	11598.00	---	2022	-62	1250	3735	Ocl	---	---	---	
iGS22-17B_3391	10428.30	11688.90	---	2022	-62	1640	3391	Ocl	---	---	---	
iGS22-17A_3233	10354.60	11732.70	---	2022	-62	1820	3233	Ocl	---	---	---	
iGS22-25	10411.80	11448.50	5104.00	2022	---	---	---	Ocu				Between Mag and CX Pit (active)
iGS22-25D_4281	10081.30	11747.90	---	2022	-63	937	4281	Ocl	---	---	---	
iGS22-25C_4190	10049.50	11779.40	---	2022	-64	1038	4190	Ocl	---	---	---	
iGS22-25B_4077	10010.20	11819.40	---	2022	-63	1164	4077	Ocl	---	---	---	
iGS22-25A_3978	9975.60	11855.50	---	2022	-63	1275	3978	Ocl	---	---	---	
iGS22-26	11247.10	12508.00	5092.00	2022	---	---	---	Qal				North of Mag Pit (active)
iGS22-26D_4199	11147.30	12590.70	---	2022	-83	902	4199	Ocu	---	---	---	
iGS22-26C_3983	11124.00	12609.50	---	2022	-82	1120	3983	Ocu	---	---	---	
iGS22-26B_3635	11087.10	12638.80	---	2022	-82	1472	3635	Ocu	---	---	---	
iGS22-26A_3384	11061.10	12659.10	---	2022	-80	1725	3384	Ocl	---	---	---	
iGS23-10A	11005.50	12407.30	5111.00	2023	---	---	---	Ocu				North of Mag Pit (active)
iGS23-10A_4412	10935.20	12481.40	---	2023	-81	707	4412	Ocu	---	---	---	
iGS23-10A_3861	10857.40	12517.30	---	2023	-81	1264	3861	Ocu	---	---	---	
iGS23-10A_3700	10830.50	12526.90	---	2023	-81	1428	3700	Ocu	---	---	---	
iGS23-10A_3579	10811.70	12533.50	---	2023	-81	1552	3579	Ocl	---	---	---	

iGS23-02A	11091.40	11430.30	4826.60	2023	---	---	---	Ocu				Mag Pit (active)
iGS23-02A_4128	10792.40	11906.90	---	2023	-52	898	4128	Ocu	---	---	---	
iGS23-02A_3747	10634.20	12132.80	---	2023	-52	1371	3747	Ocu	---	---	---	
iGS23-02A_3641	10592.10	12195.10	---	2023	-52	1500	3641	Ocl	---	---	---	
Monitor Wells												
GMWCX-1	9844.60	12588.10	5258.10	1997	-90	505 to 545	4753 to 4713	Ocl	382.8	4875.30	11/12/2024	
GMWCX-2	8409.50	11701.20	5580.20	1997	-90	465 to 505	5115.2 to 5075.2	Kgd	278.92	5301.28	05/11/2024	
GMWCX-3	8235.70	10619.10	5321.30	1997	-90	278 to 318	5043.3 to 5003.3	Cpy	274.45	5046.85	05/11/2024	
GMWCX-4	7785.30	7991.50	5335.50	1997	-90	610 to 670	4725.5 to 4665.5	Cpy	506	4829.50	05/11/2024	
GMWCX-5	10341.30	8970.70	5073.00	1997	-90	490 to 523	4583 to 4550	Ocl	DRY	DRY	11/05/2024	
GMWCX-5D	10008.58	8670.70	5109.77	2008	-90	1000 to 1080	4109.77 to 1029.77	Ocl	711.68	4398.09	23/12/2024	Replacement for GMWCX-5
AMW-1	9857.09	10652.34	4762.31	2005	-90	900 to 960	3862.31 to 3802.31	Ocl	383.55	4378.76	23/12/2024	
BPZ0802	11070.54	9224.37	4995.80	2008	-90	---	---	Qal	DRY	DRY	---	
BPZ0803	9329.24	11014.22	5194.53	2008	-90	---	---	Ocl	678.8	4515.73	16/05/2024	
GMW-HLMW-1	14221.21	4917.50	4692.27	1989	-90	254	---	Qal	264.06	4428.21	27/09/2023	
GMW-RCH-588	15614.11	10449.75	4788.60	1995	-90	505	---	Qal	193.94	4594.66	28/09/2023	
GMW-W7A	20480.58	10155.92	4657.70	1995	-90	110	---	Qal	125.94	4531.76	28/09/2023	
MW 8	15140.35	8999.35	4757.01	1998	-90	212	---	Qal	154.87	4602.14	02/10/2023	
RCH-1101	11440.58	12178.09	5060.51	1992	-90	264	465	Ocu	270.71	4789.80	05/06/2024	
RCH-1280	13419.41	9893.00	4850.00	1991	-90	---	---	Qal	211.25	4638.75	23/12/2025	
RCH-1305	10859.00	10333.45	5012.79	1992	-90	300	---	Ocu	DRY	DRY	---	Records indicate well has collapsed
RCH-1308	10981.17	9783.77	4996.49	1992	-90	285	---	Ocu	DRY	DRY	---	Records indicate well has collapsed
RCH-1309	10642.73	10744.50	5048.88	1991	-90	337	---	Ocu	DRY	DRY	---	Records indicate well has collapsed
RCH-1515	12602.88	9750.12	4895.68	1993	-90	219 to 465	4676.68 to 4430.68	Qal	258.1	4637.58	05/06/2024	
RCH-1516	12395.45	10333.65	4904.70	1993	-90	229	---	Qal	DRY	DRY	---	Records indicate well has collapsed
RCH-1517	12458.51	10705.76	4915.72	1993	-90	229	---	Qal	DRY	DRY	---	Records indicate well has collapsed
RMW2NE	15915.24	7792.24	4718.68	2005	-90	178 to 218	4540.68 to 4500.68	Qal	104.16	4614.52	15/02/2024	
RMW2SE	15902.63	7292.08	4711.48	2005	-90	158 to 198	4553.48 to 4513.48	Qal	91.35	4620.13	15/02/2024	
RMW2W/GMWMW2A	13432.00	9194.40	4838.20	1992	-90	140	500	Qal	203.91	4634.29	20/03/2024	
RMW3NE	15621.68	6584.88	4710.37	2005	-90	158 to 198	4552.37 to 4512.37	Qal	90.52	4619.85	15/02/2024	
RMW3SE	15488.44	6250.25	4706.13	2005	-90	158 to 198	4548.13 to 4508.13	Qal	87.59	4618.54	15/02/2024	
RMW3W	13681.94	7080.52	4804.22	2005	-90	278 to 318	4526.22 to 4486.22	Qal	179.19	4625.03	15/02/2024	
WELL 10	10454.99	11200.32	5084.57	1992	-90	242	542	Ocu	DRY	DRY	16/05/2024	Records indicate well has collapsed

WELL 2A	13452.52	9148.66	4839.80	1992	-90	144 to 450	4695.8 to 4389.8	Ocu	204.15	4635.65	05/06/2024	
WELL 6	8381.24	9858.41	5168.70	1998	-90	274	---	Cpy	248.75	4919.95	16/05/2024	
WSW-W#11	10907.57	12300.59	5117.49	1998	-90	105 to 505	5012.49 to 4612.49	Ocu	DRY	DRY	---	
WSW-W#2	13024.00	9420.00	4861.90	1992	-90	251 to 555	4610.9 to 4306.9	Ocu	228.83	4633.07	05/06/2024	
WSW-W#9B	10323.65	11966.86	5173.91	1998	-90	264 to 617	4909.91 to 4556.91	Ocl	DRY	DRY	---	
Rib Piezometers												
RPZ2E	14833.50	8020.25	4768.41	2005	-90	58 to 138	4710.41 to 4630.41	Qal	90.03	4678.38	15/02/2024	
RPZ2N	14732.84	8348.93	4771.00	2005	-90	58 to 138	4713 to 4633	Qal	128.5	4642.50	15/02/2024	
RPZ2S	14595.45	7762.05	4773.27	2005	-90	58 to 138	4715.27 to 4635.27	Qal	81.05	4692.22	15/02/2024	
RPZ2W	14496.42	8090.13	4785.56	2005	-90	58 to 138	4727.56 to 4647.56	Qal	88.58	4696.98	15/02/2024	
RPZ3E	14388.32	6615.05	4763.06	2005	-90	58 to 138	4705.06 to 4625.06	Qal	DRY	DRY	15/02/2024	
RPZ3N	14377.55	6939.79	4772.25	2005	-90	58 to 138	4714.25 to 4634.25	Qal	DRY	DRY	15/02/2024	
RPZ3S	14140.12	6414.30	4766.55	2005	-90	58 to 138	4708.55 to 4628.55	Qal	DRY	DRY	15/02/2024	
RPZ3W	14124.26	6741.97	4778.56	2005	-90	58 to 138	4720.56 to 4640.56	Qal	DRY	DRY	15/02/2024	

a) feet above mean sea level; for wells, elevation of land surface at surface casing; for VWPs elevation of surface casing at land surface is provided;

b) degrees from horizontal at bottom of well or depth of VWP along inclined borehole using IDS survey

c) feet below land surface for wells; feet along inclined borehole for VWPs based on IDS inclination survey and Leapfrog Geologic Model positioning

d) feet below land surface for wells

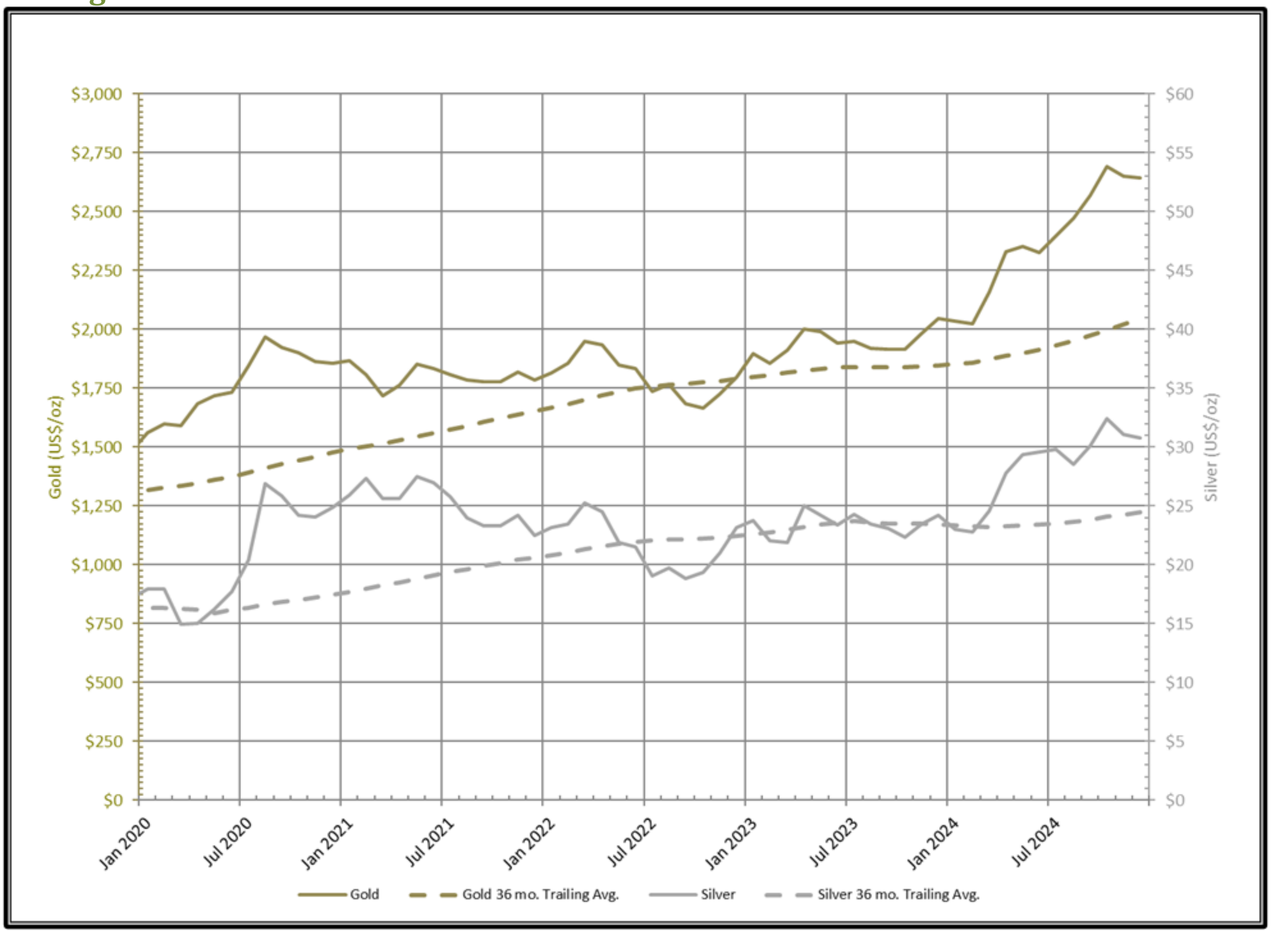
e) feet below land surface for wells; depth to water subtracted from collar elevation

16 Market Studies and Contracts

16.1 Precious Metal Markets

Gold and silver are fungible commodities with reputable smelters and refiners located throughout the world. The price of gold has reached all-time highs in 2024 with December's price averaging 2,644 per ounce. As of December, 2024 the three-year trailing average gold price was $2,044 per ounce and the two-year trailing average price was $2,166 per ounce. The three -year and two-year trailing average prices for silver in December 2024 were $24.50 and $25.88 per ounce respectively. Historical plots for both are shown in in Figure 16-1.

Figure 16-1 Historical Monthly Average Gold and Silver Prices and 36 Month Trailing Average



Issuers may also rely on published forecasts from reputable financial institutions. The current long term price forecast by CIBC is $2,169 and per ounce and $27.61 per ounce for gold and silver respectively (CIBC., 2025).

Commodity prices for Mineral Reserves are chosen not to exceed financial institution forecasts or the three-year trailing average price. Commodity pricing for the estimation of Mineral Resources can be 10% to 20% higher than that used for Mineral Reserves. The gold price selected for estimating Mineral Resources disclosed in this technical report is $2,175. The silver price selected is $27.25 per ounce.

16.2 Contracts

16.2.1 Orion and Sprott Financing Package

The Company entered into a financing package with OMF Fund III (F) Ltd. an affiliate of Orion Mine Finance (collectively “Orion“) on December 31, 2021, and a fund managed by Sprott Asset Management USA, Inc. and a fund managed by CNL Strategic Asset Management, LLC (“Sprott”) on December 9, 2021 (together the “Finance Package”).

The Financing Package in its aggregate consists of:

a. $50 million convertible loan (the “Orion Convertible Loan”)

b. $10 million convertible loan (the “Sprott Convertible Loan” and together with the Orion Convertible Loan, the “Convertible Loans”)

c. $45 million gold prepay purchase and sale agreement entered into with affiliates of Orion (the “Gold Prepay Agreement”), including an accordion feature potentially to access up to an additional $50 million at i-80 Gold’s option

d. $30 million silver purchase and sale agreement entered into with affiliates of Orion (the “Silver Purchase Agreement”), including an accordion feature to potentially access an additional $50 million at i-80 Gold’s option and an amended and restated offtake agreement entered into with affiliates of Orion (the “A&R Offtake Agreement”)

e. 5,500,000 warrants of the Company issued to Orion (the “Orion Warrants” and together with the Orion Convertible Loan, Gold Prepay Agreement, Silver Purchase Agreement and the A&R Offtake Agreement, the “Orion Finance Package”).

Under the Gold Prepay Agreement, i-80 Gold was due to deliver to Orion 3,000 troy ounces of gold for each of the quarters ending March 31, 2022 and June 30, 2022, and thereafter, 2,000 troy ounces of gold per calendar quarter until September 30, 2025 in satisfaction of the

$45 million prepayment, for aggregate deliveries of 32,000 troy ounces of gold. i-80 Gold may request an increase in the $45 million prepayment by an additional amount not exceeding $50 million in aggregate in accordance with the terms of the Gold Prepay Agreement.

The final Gold Prepay Agreement includes an amendment to adjust the quantity of the quarterly deliveries of gold, but not the aggregate amount of gold, to be delivered by the Company to Orion over the term of the Gold Prepay Agreement. Under the amended Gold Prepay Agreement, commencing on the date of funding, the Company is required to deliver to Orion 1,600 troy ounces of gold for the quarter ending March 31, 2022, 3,100 troy ounces of gold for the quarter ending June 30, 2022, and thereafter 2,100 troy ounces of gold per calendar quarter until September 30, 2025, in satisfaction of the $45 million prepayment, for aggregate deliveries of 32,000 troy ounces of gold, subject to adjustment as contemplated by the terms of the Gold Prepay Agreement. As the funding from Orion did not occur until April 2022, payment for the delivery of 1,600 ounces for the quarter ending March 31, 2022 was offset against the $45 million of proceeds received from Orion.

Under the Silver Purchase Agreement, commencing April 30, 2022, i-80 Gold will deliver to Orion 100% of the silver production from the Granite Creek and Ruby Hill projects until the delivery of 1.2 million ounces of silver, after which the delivery will be reduced to 50% until the delivery of an aggregate of 2.5 million ounces of silver, after which the delivery will be reduced to 10% of the silver production solely from the Ruby Hill Project. Orion will pay i-80 Gold an ongoing cash purchase price equal to 20% of the prevailing silver price. Until the delivery of an aggregate of 1.2 million ounces of silver, i-80 Gold is required to deliver the following minimum amounts of silver (the "Annual Minimum Delivery Amount") in each calendar year: (i) in 2022, 300,000 ounces, (ii) in 2023, 400,000 ounces, (iii) in 2024, 400,000 ounces, and (iv) in 2025, 100,000 ounces. Upon a construction decision for the Ruby Hill project, comprised of one or both of the Ruby Deep or Blackjack Deposits, which construction decision is based on a feasibility study in form and substance satisfactory to Orion, acting reasonably, i-80 Gold will have the right to request an additional deposit from Orion in the amount of $50 million in aggregate in accordance with the terms of the Silver Purchase Agreement.

Both the Gold Prepay Agreement and the Silver Purchase Agreement were funded on April 12, 2022 with i-80 Gold receiving net proceeds of $71.6 million after netting the aforementioned March 31, 2022 gold delivery and closing costs as further described in Note 10 and Note 24 in the Company’s Financial Statements.

The main amendments reflected in the A&R Offtake Agreement include the increase in the term of the agreement to December 31, 2028, the inclusion of the Granite Creek and Ruby Hill projects, and the increase of the annual gold quantity to up to an aggregate of 37,500 ounces in respect of the 2022 and 2023 calendar years and up to an aggregate of 40,000 ounces in any calendar year after 2023. During the year ended December 31, 2022, Orion assigned all of its rights, title and interest under the A&R Offtake Agreement to TRR Offtakes LLC, now Deterra Royalties Limited.

On September 20, 2023, the Company entered into an Amended and Restated ("A&R") Gold Prepay Agreement with Orion, pursuant to which the Company received aggregate gross proceeds of $20 million (the "2023 Gold Prepay Accordion") structured as an additional accordion under the existing Gold Prepay Agreement.

The 2023 Gold Prepay Accordion will be repaid through the delivery by the Company to Orion of 13,333 troy ounces of gold over a period of 12 quarters, being 1,110 troy ounces of gold per quarter over the delivery period with the first delivery being 1,123 troy ounces of gold. The first delivery will occur on March 31, 2024, and the last delivery will occur on December 31, 2026. Obligations under the A&R Gold Prepay Agreement, including the 2023 Gold Prepay Accordion, will continue to be senior secured obligations of the Company and its wholly-owned subsidiaries Ruby Hill Mining Company, LLC and Osgood Mining Company, LLC and secured against the Ruby Hill project in Eureka County, Nevada and the Granite Creek project in Humboldt County, Nevada.

The remaining terms of the A&R Gold Prepay Agreement remain substantially the same as the existing Gold Prepay Agreement. The Company may request an increase in the prepayment by an additional amount not exceeding $50 million in aggregate in accordance with the terms of the A&R Gold Prepay Agreement.

In connection with the 2023 Gold Prepay Accordion, the Company issued to Orion warrants to purchase up to 3.8 million common shares of the Company at an exercise price of C$3.17 per common share until September 20, 2026, and extended the expiry date of 5.5 million existing warrants by an additional 12 months to December 13, 2025.

16.2.2 Orion Offtake

In February of 2025, i-80 Gold and Orion entered into an offtake agreement (the "Orion Offtake Agreement"). The Orion Offtake Agreement has similar terms to the current A&R Offtake Agreement with Deterra Royalties Limited and will commences upon its expiry. The Orion Offtake Agreement expires on December 31, 2034

Under the terms of the Offtake Agreement, the Company agreed to sell, and Orion agreed to purchase (i) an aggregate of 29,750 ounces of refined gold for 2021, and (ii) up to an aggregate of 31,500 ounces of refined gold annually (the "Annual Gold Quantity") from the Company's Eligible Projects until March 1, 2027. The Company's Eligible Projects include the South Arturo Project, the McCoy-Cove Project, and the Granite Creek Project. The final purchase price to be paid by Orion will be, at Orion's option, a market-referenced gold price in U.S. dollars per ounce during a defined pricing period before and after the date of each sale. In the event that the Company does not produce the Annual Gold Quantity in any given year, the obligation is limited to those ounces actually produced.

16.3 Previous Financing Agreements

16.3.1 South Arturo Purchase and Sale Agreement (Silver)

The Company entered into a Purchase and Sale Agreement (Silver) (the "Stream Agreement") with Nomad, which was connected to South Arturo, whereby the Company will deliver to Nomad (i) 100% of the refined silver from minerals from the main stream area, and (ii) 50% of the refined silver from the exploration stream area. Nomad will pay an ongoing cash purchase price equal to 20% of the silver market price on the day immediately preceding the date of delivery and will credit the remaining 80% against the liability. Following the delivery of an aggregate amount of refined silver equal to $1.0 million to Nomad under the Stream Agreement, Nomad would continue to purchase the refined silver at an ongoing cash purchase price equal to 20% of the prevailing silver price. The liability for the Stream Agreement was included in the net asset value in connection with the asset exchange with Nevada Gold Mines LLC (“NGM”) discussed in the “Lone Tree and Ruby Hill Acquisition”, and therefore, is no longer impacting the Financial Statements as of December 31, 2021.

16.3.2 Autoclave Mineralized Material Purchase Agreement

The company has negotiated an agreement with a third party to sell up to 1,000 tons per day mineralized material for a fixed recovery of 58% of the contained gold at the average gold price during the month. In exchange there are no processing costs, refining or sales costs deducted from the purchase price. Transportation of the material to the processing site remains the responsibility of i-80. This agreement will apply to all refractory material mined from Granite Creek prior to the restart of the Lone Tree facility in 2028.

16.3.3 Contract Mining

Granite Creek mining is performed by a qualified contractor. The contract is structured to pay on footage advance with no allowance or additional payment for overbreak. Additional items are ground support required in addition to primary ground support and hourly rates for labor or equipment when work outside the scope is requested. There are no monthly fixed administration costs added.

16.3.4 Other Contracts

The company also intends to negotiate contracts for underground mine development, production mining, and over-the-road haulage with reputable contractors doing business in northeast Nevada.

At the time of this report these negotiations have not been initiated. From time to time the company enters into other contracts for goods and services as a routine course of business.

17 Environmental Studies, Permitting and Plans, Negotiations or Agreements with Local Individuals or Groups

17.1 Environmental Setting

The site is a producing underground operation built on a historic mine site that has been impacted by operations and exploration since the 1940s. The majority of disturbances have been reclaimed. In the valley to the east of the mine, there are several center-pivot irrigation systems raising hay. These adjacent water users may be beneficially impacted by the mine contributing groundwater to the RIBs. They are far enough away (and in the opposite direction of the prevailing wind) thus making them unlikely to be impacted by noise or dust from operations.

17.2 Geochemistry

The site has had limited geochemical characterization throughout history. Most of the geochemical test work was performed by WMCI in 1998. This study involved acid-base accounting, metals enrichment by acid-digestion and ICP-MS, and kinetic tests. 51 rock samples were tested statically, and 15 of those samples were selected for kinetic cell testing. Samples were selected from a variety of lithologies and locations but were designed to primarily focus on the Mag and CX future pit wall material (WMC Consultants, 1998).

Results from the ICP-MS analysis showed that major element abundance was controlled by rock type, with calcium abundant in carbonate-bearing rocks, and silica and aluminum concentrations abundant in silici-clastic rocks. Arsenic was found to be elevated in three samples and associated with hydrothermal deposits. ABA results indicated that the rock had low acid-generating potential (AP). Tested rocks had low sulfur, neutral paste pH, and abundant neutralization potential (NP) that resulted in 49 out of 51 samples being classified as non acid-generating based on the CANMET standards of NP/AP >3 (WMC Consultants, 1998).

Kinetic cells were run for a minimum of 20 weeks, with some running for a total of 28 weeks. Rates of ARD generation were tracked weekly, and the change in acidity and alkalinity was used to provide a quantitative estimate of whether the retained alkalinity would outlast the acid generation. Only 1 cell showed potential for acid generation: an argillite with 0.47% sulfur. This sample had consistently acidic pHs (4 – 2.2) with sulfate in the hundreds to thousands of mg/L range. All other kinetic cells had neutral to basic pH (7-9), alkalinity between 20-40 mg/L, and no quantifiable acidity. Monthly leachate analyses largely confirmed the weekly results. Metals in leachate were generally within the reference values, but some samples showed elevated levels of antimony and arsenic multiple times after the initial stabilization period. Additionally, the 2 cells that were uncertain under the acid-generation calculation showed levels of aluminum, antimony, arsenic, copper, iron, lead, and thallium after the initial stabilization period that were above Nevada Profile I reference values at the time. While the remaining cells were not over the standard at the time, the arsenic and antimony reference values were lowered in 2006 to 0.001 and 0.01 mg/L, which makes all the cells retrospectively over the standard. However, there was not a particular rock type that was consistently exhibiting acid generating or metal leaching potential (WMC Consultants, 1998).

Since 1998, the site performed periodic sampling characterizing waste rock authorized for disposal by backfill to the bottom of the CX pit. In the sample set taken from 2005 to 2022, the NP/AP ratio of this rock has varied from 2.5 to 568, confirming the presence of limestone layers within the Comus Sediments (see Section 7.0), that will readily neutralize any acid generated from the dissolution of sulfide minerals (Osgood Mining Company LLC, 2023). In addition to ABA tests, Meteoric Water Mobility Procedure (MWMP) tests have been performed on the waste rock deposited in the bottom of the CX pit. The weighted average of MWMP values based on volume of rock placed, the minimum, maximum, and geometric mean of constituents in exceedance of Nevada Profile I reference values is tabulated in Table 17-1 (Stantec Consulting Services, 2023).

Table 17-1 Weighted Average Concentrations of MWMP Results of Rock Placed in CX Pit 2005 - 2022

Analyte	NDEP Profile I Reference Value	Weighted Average	Maximum	Minimum	Mean
Antimony (mg/L)	0.006	0.190	0.070	0.001	0.184
Arsenic (mg/L)	0.010	0.510	2.200	0.015	0.423
Nitrate (mg/L)	10	51.3	150.0	0.6	46.7
Sulfate (mg/L)	500	151	800	3	184
TDS (mg/L)	1000	590	1,900	27	601

It is important to note that all the rock deposited in the CX pit will be covered with an engineered evapotranspiration (ET) cover (see Section 17.3). This will be protective of water quality impacts from rock leachate.

The project, with the current available geochemical data, does not appear to pose ARD risk and only appears to pose minimal metal leaching (ML) risk in regards to antimony and arsenic release. To mitigate this ML risk, the mine operates a water treatment plant (see Section 17.1.4). GRE suggests that the project expand upon the current geochemical characterization to include a thorough characterization of future waste rock and tailings, as this will be required for future permitting efforts (See Section 23).

17.2.1 Onsite Water Quality

Water is sampled from many sources:

- Underground dewatering wells (APW1, BPW3, BPW5, and GCW-06)
- Background groundwater wells,
- Underground mine sumps,
- Surface water,
- Mag pit lake,
- RIBs, and
- Influent and effluent from the Water Treatment Plant (WTP).

Bedrock groundwater from the underground dewatering wells has variable chemistry, which is reflective of the variable groundwater flow between different mineralized and unmineralized geologic units. In general, the groundwater from unmineralized blocks has lower arsenic and antimony than from mineralized blocks. The pH of bedrock groundwater ranges between 6.8 to 8.4, with total dissolved solids from 180 to 1,500 mg/L and alkalinity between 74 to 134 mg/L. Some bedrock groundwater also exceeds the Nevada Profile I reference values for metals, particularly for arsenic, cadmium, iron, manganese, nickel, and zinc (Osgood Mining Company LLC, 2023).

The site has maintained continuous bedrock groundwater quality monitoring for the purposes of compliance with Water Pollution Control permits (WPCP). A summary of the most recent water quality data collected is compiled in Table 17-2. The Nevada Profile I reference values for antimony and arsenic are 0.006 mg/L and 0.01 mg/L respectively.

Table 17-2 Water Quality April 2023-Jan 2025

Location ID	Antimony mg/L Range	Arsenic mg/L Range
APW1	0.0025-0.0041	0.029 - 0.059
BPW3	0.0025	0.02 – 0.027
BPW5	0.0025	0.012 – 0.037
GCW-06	0.0035 – 0.0085	0.33 – 1
RIB Distribution Pipeline	0.0025	0.023 – 0.028

WTP Effluent	0.0025 – 0.003	0.005 – 0.015

Most natural water meets Nevada reference values for antimony. The majority of bedrock groundwater onsite exceeds Nevade Profile I reference values for arsenic, however, the WTP effluent water quality demonstrates the WTP's ability to meet arsenic reference values. An arsenic attenuation study is ongoing to address the high arsenic concentrations found in most groundwater.

Underground mine water is most impacted by metal leaching with analyzed samples collected from April to August 2024 exhibiting Nevada Profile I reference value exceedances for arsenic (ranging from 0.03 mg/l – 0.375 mg/l), antimony (ranging from 0.015 mg/l – 0.115 mg/l) and thallium (ranging from 0.003 mg/l – 0.014 mg/l). Range-front background alluvial groundwater quality has historically been relatively consistent over the period of record, with few exceedances of Nevada Profile I reference values, and some indication that natural chemical attenuation is occurring (Enviroscientists, Inc. Water Management Consultants, 2005). Alluvial groundwater on the project is monitored by numerous wells and is well understood. Onsite alluvial groundwater generally meets Nevada Profile I water quality reference values, with most trace metals at or below analytical laboratory standards (Osgood Mining Company LLC, 2023).

Granite Creek is located adjacent to the site and flows ephemerally during the spring and summer in response to snow melt and precipitation events. It is currently diverted through a series of pipes and culverts around the southern rim of the CX Pit to the original stream channel location downgradient of the pit. The water quality is consistently good, with all constituent concentrations below the Nevada reference values for surface water (Osgood Mining Company, LLC, 2020).

The Mag pit lake water quality has been monitored consistently since 2015. Samples have been taken from the top, middle, and bottom of the water column to establish any chemical differences in water quality with depth. Over 10 years of sampling, the average arsenic surface concentration of the Mag pit is 0.029 mg/L, the middle of the water column in the Mag pit has 0.029 mg/L arsenic, and the bottom of the Mag pit has 0.032mg/L arsenic. All layers show consistently high total dissolved solids (around 1000 mg/L) and sulfate (490-580 mg/L). The bottom of the Mag pit also appears to be elevated in manganese up to maximum of 0.53 mg/L in 2021 (LRE Water, 2024).

17.2.2 Pit Lake Future Water Quality

The mine plan assumes that the CX pit will be backfilled and will not create a pit lake; however, a pit lake study exists. GRE believes that this study retains its relevance because it is the best-available study at present to discuss the water quality of the future MAG pit lake which will form after mining ceases. Similar to the CX pit study, ongoing work from LRE (unpublished as of the effective date) also shows that the Mag pit will be a hydrologic sink.

A thorough investigation of future CX pit wall material and backfill was conducted by WMC in 1998. This was followed up by a pit lake model to predict future pit lake water quality. WMC found that the majority of future pit wall rock were acid-neutralizing, with paste pH >7 and average sulfur content of 0.053% weight. Most rock had significant neutralization potential, with an average of 115 tons $CaCO_3$/1000 tons rock.

The lake will behave as a hydrologic sink with no discharge of impounded waters to the surface or groundwater. The waste rock backfilled to the bottom of the CX pit will be inundated by rising post-mining pit lake waters to an estimated minimum depth of approximately 30 feet. Inundation of the backfill will cut off the oxygen supply and reduce or eliminate the potential for the backfill to generate acidic conditions. Pit lake predictive modeling reports indicate that long-term post-mining lake water will be in compliance with WPCP NEV2005103, with many metals concentrations less than the analytical laboratory reporting limit, significantly below Nevada reference values with the exception of arsenic (0.019 mg/l). In the event that post mining CX pit lake arsenic concentrations rise above acceptable levels, the modeling predicts that the addition of ferric sulfate solution ($Fe_2(SO_4)_3$) at a rate of 0.23 grams per gallon would reduce the concentrations of arsenic to below the analytical laboratory reporting limit in the short- and long-term (Osgood Mining Company LLC, 2023). The model result is well-supported by the ferric sulfate dosing program that was tested at the CX pit lake in 2001 to help reduce arsenic concentrations reported in the pit lake at that time (Beale & Feehan, 2005).

17.2.3 Water Treatment Plant

The project currently has an 850 gpm capacity Veolia mobile water treatment plant (WTP). All the WTP storage and treatment components are contained within HDPE lined containment areas and consist of three areas: Surge Tank, Water Treatment Plant, and Sludge Dewatering System. The surge tank has a 10,000-gallon capacity and is used to blend water from mine contact water (200-250 gpm), BPW5 (400 gpm), and GCW-06 (120 gpm) to obtain adequate flow.

Water is treated for arsenic and antimony. Treated water is then discharged to the RIBs or diverted to the MAG pit which are regulated under WPCP NEV2005102.

The water treatment plant utilizes ferric iron to precipitate antimony and arsenic from solution. It is made up of the following components:

- Metal Precipitation Reactor
    - Addition of ferric sulfate or ferric chloride and sodium hydroxide during aeration with blowers

- Actiflo Clarifier
  - Solids settle out of solution and are sent to sludge dewatering system

The Sludge Dewatering System consists of a splitter box that splits the sludge from the water, pumps that pump water back to the Actiflo clarifier, and a 1,500-gallon HDPE sludge tank. Sludge is then deposited in geobags, and the supernatant is recirculated back to the metals precipitation reactor. The geobag containment system consists of a two-layer HDPE liner system, with the bottom liner used for leak detection. Material from the geobags is mixed with waste rock and placed in the CX pit (approved by NDEP-BMRR).

An additional water treatment plant (a modular twin of the existing system) will be designed and built to accommodate the greater water disposal needs of the project.

Water Volumes

Water treatment needs are variable over time. At its peak, the underground mine is expected to produce 2900 gpm at its maximum dewatering extent. 80% (2300 gpm) is expected to come from dewatering wells, and 20% (600 gpm) is expected to come from contact water from underground sumps. The current IA considers that 2900 gpm of underground water is treated. However, there is an opportunity to create an improved water balance and to greatly decrease the treatment requirements (see Section 27).

Furthermore, the MAG pit will require dewatering. 2 years prior to open pit operations, the mine must commence dewatering the MAG pit at a rate of 450 gpm. This dewatering is anticipated to take 4 months to evacuate the volume of 69M gallons. MAG pit water has ~0.035 mg/L arsenic, which will be allocated to the TSF pond for forced evaporation.

17.3 Environmental Studies and Issues

The estimated cost to close and reclaim the Granite Creek Project is approximately $3 million (Osgood, 2023). This amount includes closure of all permitted surface and underground mining and exploration related disturbance at the Project and is calculated using standardized reclamation cost estimators that assess the following:

- Exploration drill hole abandonment
- Exploration roads and pads
- Waste rock dumps
- Heap leach pads
- Roads
- Pits
- Foundations and buildings

- Other demolition and equipment removal
- Sediment and drainage control
- Process ponds
- Landfill
- Yards
- Waste disposal
- Well abandonment
- Underground related infrastructure and portals closure
- Miscellaneous costs
- Monitoring
- Construction management
- Mobilization and demobilization.

Bonds in the amount of approximately $3 million are held by the BLM to address surface and underground reclamation and closure related activities (Osgood, 2023). There are no other known environmental liabilities
associated with pre-Project operations (Osgood, 2024).

17.4 Social or Community Impacts

The following information on community relations and stakeholder consultation has been provided by Osgood Mining personnel (2024).

Historical mining activities date back to the 1940's at the Granite Creek property (a.k.a., Pinson Mine), with intermittent periods of operation continuing to the present day. Over its history, the operation has contributed significantly to the economic development of Humboldt County through job creation and both direct and indirect economic benefits.

Osgood Mining Company periodically hosts Town Hall meetings in Winnemucca, Nevada, to offer operational updates to local stakeholders. The company also collaborates with the Nevada 95-80 Regional Development Authority during its annual economic development conference and works closely with the Humboldt County School District and Great Basin College, supporting various educational initiatives that benefit local students and the community at large.

Beyond these partnerships, Osgood Mining places a high priority on maintaining positive, long-term relationships with local government officials, ranchers, and neighboring landowners, ensuring that all parties are heard, respected, and included in the development process. Through these efforts, Osgood Mining strives to be a responsible and engaged community partner, prioritizing the well-being of the area and its residents throughout the duration of the Granite Creek Project.

17.5 Permits

Osgood Mining Company is currently permitted to carry out mining operations and reclamation activities at the Project site. This permitting allows it to carry out the exploration, geotechnical and metallurgical field work recommended in this Report. Specific permits related to site activities are presented in Table 17-3.

Table 17-3 Granite Creek Mine Project Significant Permits

Permit Name	Agency	Permit Number
Plan of Operations Granite Creek Mine Project	BLM	NVN-064101
Class II Air Quality Operating Permit	NDEP-BAPC	AP1041-3086.02
Mercury Operating Permit to Construct	NDEP-BAPC	MOPTC AP1041-3089 (De Minimis)
Water Pollution Control Permit - Rapid Infiltration Basins	NDEP-BMRR	NEV2005102
Water Pollution Control Permit - Granite Creek Mine	NDEP-BMRR	NEV2005103
Mine Reclamation Permit	NDEP-BMRR	0047
Granite Creek UG Mine Reclamation Permit	NDEP-BMRR	0242
Mining Stormwater General Permit	NDEP-BWPC	NVR300000: MSW-42365
Onsite Sewage Disposal System	NDEP-BWPC	GNEVOSDS09S0177
Hazardous Materials Storage Permit	Nevada State Fire Marshal	12441012106
Waters of the United States Jurisdictional Determination	USACE	Request for Approved Jurisdictional Determination (AJD) submitted to USACE November 2022

17.6 Water Use Permits

Water rights at the Granite Creek Mine have a total combined duty of 9,853 acre-feet annually (AFA), of which 1,149 AFA is for consumptive use (Osgood, 2024).

Water usage for the Project is managed via three certified water rights and ten permits, three of which are block permits. All water rights are subject to State Engineer's Order 1087 (Block Order).

All use from the mine, including consumptive and non-consumptive use, is reported monthly on a site pumpage report and the specific meter readings are recorded and subsequently uploaded monthly to NDWR's online meter database.

17.7 Future Permitting Requirementss

The permits discussed above allow for the current ongoing underground operation at Granite Creek. Major permit revisions, as well as additional permits, will be required for the proposed plan of operations in this IA. The following sections details the anticipated new permits, permit revisions, and permitting efforts that Granite Creek will need to face for the mining plan described in this IA.

17.7.1 National Environmental Policy Act (NEPA)

The National Environmental Policy Act (NEPA) is likely the largest single permitting hurdle that the project will face. The NEPA process is required when disturbances are anticipated to take place on federal lands and non-patented mining claims. It is reasonable to expect that this NEPA permitting effort will require the completion of an Environmental Impact Statement (EIS) through the BLM. An EIS is usually a lengthy process involving:

- A large database of baseline data (prior to the anticipated mining impact)
- A Plan of Operations (PoO) amendment describing the mining plan in detail
- An assessment of the environmental impacts as a result of operations
- A discussion of mitigation measures
- An evaluation of the effectiveness of mitigation measures
- A wide variety of supporting and supplemental environmental reports

The EIS is prepared by a third party hired by the BLM (not the mining company, and not the consultants who prepare the PoO amendment and baselines studies) but paid for with mining-company funds. The EIS is submitted to the BLM, where it is given a public comment period. After a process that often takes years from the commencement of baseline data collection, the BLM provides a Record of Decision (ROD), which acts as the permit.

At minimum, because the site has never had a full EIS, the following supplementary reports will be required:

- Geochemistry of tailings and waste rock
- MAG pit lake study
- Backfill study for the mine waste below the water table in the CX pits
- Groundwater and surface water resource studies and water quality studies
- Seepage and groundwater quality studies for the TSF
- Noise and vibration
- Air quality
- Wildlife and impacted biology
- Archeology and cultural resources

17.7.2 State Permits

State permits are required for air quality protection, groundwater protection, surface water protection, and water rights. All of the permits presented in Table 17-3 will require revision with the amended PoO. Several key state permits are described below

17.7.2.1 Water Pollution Control Permit

The WPCPs are granted by the state of Nevada (NDEP-BMRR) and cover any potential discharge of water to surface or groundwater. This permit will require revision to be consistent with the amended PoO. An arsenic attenuation study will likely need to be conducted as part of the efforts to mitigate arsenic metal leaching risk and to establish an alternative discharge standard for the RIBs (see Section 20.1.4).

17.7.2.2 Reclamation Permits

Reclamation permits are overseen by NDEP-BMRR with the BLM providing supporting input. Existing reclamation permits for surface mining and underground mining will have to be revised. A new closure bond must be calculated and provided in anticipation of the new mining impacts in accordance with the PoO proposed in the IA. Section 20.3 discusses the closure plan and the reclamation cost estimate.

17.7.2.3 Other State Permits

Sewage disposal permits, stormwater permits, and air quality permits must be updated to be consistent with the PoO amendment specifics. It is important to note that the MAG pit dewatering is currently permitted under the exploration permit, and it can commence prior to acquiring other permits and prior to the ROD on the EIS.

17.7.3 Monitoring Requirements

Currently the site has been executing all environmental monitoring requirements required to maintain the WPCPs, air quality permits, reclamation permits, and other state permits associated with the small-scale underground operation. Samples of surface and groundwater are required quarterly, and quarterly and annual reports are provided to the regulators. Reclamation permits require annual disturbance reports and bond updates every three years. The stormwater permit requires quarterly inspections, an annual report, and an annual fee. All permit monitoring requirements are up-to-date. Widespread additional monitoring will be required in support of the NEPA permitting process, as well as after the ROD and well into closure.

17.8 Mine Closure

The mine closure cost estimate was derived using the Standardized Reclamation Cost Estimator Version 1.4.1, developed by the Nevada Division of Environmental Protection. This software is used by the Nevada regulators to calculate closure bond and closure cost requirements, and as a result, it is the only tool that could be reasonably applied for this estimate.

GRE has expanded, modified and updated prior closure bond calculations provided by i-80 Gold. These included the version 1.4.1, 2025 update of the costs to close the current configuration of the Granite Creek Mine. These prior bond calculations were augmented and expanded to include the elements that would be constructed in the mine plan evaluated by this IA.

17.8.1 Mine Closure Design Criteria

The following design criteria and assumptions were applied to closure:

- All key regulatory closure requirements will be met.
- All regrading of mine waste structures is performed during operations. This includes creating tailings and waste rock facilities at a closure-ready 3:1 slope.
- Some regrading of the waste rock dump and TSF will be required to recontour benches and to create a more-effective surface water drainage pattern.
- Excess water on the TSF must be evaporated using an enhanced evaporation method. The surface will be allowed to sit dry for one year to consolidate and stabilize to a "trafficable" surface condition. It will take approximately one year to evaporate the tailings pond using the forced evaporator systems which were purchased for the MAG pit dewatering (see Section 17.1.4 above).
- Waste rock and pit backfill is to be covered by 12 inches of cover and 5 inches of growth media. This is consistent with the prior closure cover.
- An approved rangeland grass mix will be used. Grassland and wildlife habitat with grazing is the anticipated post-mining land use.
- Because of the ML risk, the TSF, pit backfill, and WRSF will be covered with an Evapotranspiration Cover (ET Cover). In arid climates, ET covers have been shown to perform as well as HDPE covers in preventing mine waste leachate. For the PEA, the ET cover is assumed to be 18 inches thick.
- All buildings, power lines, and other infrastructure will be removed.
- The RIBs will be filled-in and reclaimed.
- Four stream channels will be reclaimed, allowing precipitation in the hills to have a reclaimed and restored streambed in which it can flow through the reclaimed mine down into the valley. Granite Creek will be restored across the pit backfill and adjacent to the waste dumps to a channel similar to the pre-mining flowpath.
- The water quality in the MAG pit lake will unlikely meet Nevada water quality standards due to elevated arsenic standards (see Section 17.1.1). However, because the lake will be a terminal sink for water, this IA does not consider it a long-term risk to groundwater quality. GRE assumes that an ecological risk assessment will likely conclude that the marginal arsenic concentrations (estimated at 35 ppb) will not be a significant risk to migratory waterfowl.

- GRE assumes that seepage from the CX pit backfill will not have a significant impact on groundwater quality and be contained within the evaporative cone-of-depression for the MAG pit lake and/or will be protected by the ET cover.
- With an ET cover, GRE assumes that the WRDs will not create a surface or groundwater quality impact.
- After dewatering and consolidation and with an ET cover, GRE assumes that the TSF will not cause a long-term surface water or groundwater seepage water quality issue.
- Other than the issues discussed above, there are no other potential water quality or water quantity impacts at Granite Creek upon closure.

17.8.2 Closure Costs

The closure costs are summarized in **Error! Reference source not found.**.5

Table 17-4 Mine Closure Cost Summary

Category	Cost (Millions , USD)	Notes
Earthwork Recontouring	8.68	Minimal recontouring is required because waste facilities are constructed at final closure grade.
Revegetation/Stabilization	0.706	Follows example of previous successful revegetation on site
Detoxification/Water Treatment/Disposal of Waste	0.430	Includes anticipated waste disposal and evaporation of water in TSF
Structure, Equipment and Facility Removal, and Misc.	1.57	Includes plant removal, new fence around MAG pit lake upon closure.
Monitoring	1.63	Assumes more monitoring wells due to larger footprint.
Construction Management and Support	0.28	Calculated by the Reclamation Spreadsheet based on Nevada-based experience.
Contingency and Indirect Costs	4.68	Recommended indirect costs with contingency as set in the SRCE model.
Total	17.98	This value is entered into the cost model.

17.8.3 Closure Cost Limitations

The closure calculation is preliminary. Additional studies are required to confirm the design criteria are correct. Nearly all these studies are ultimately part of the EIS and permitting effort (see Section 17.2).

17.9 Local Procurement and Hiring

i80 gold has a specific corporate policy for local procurement and hiring. This program will be described in greater detail in subsequent SK-1300 reports.

18 Capital and Operating Costs

18.1 Open Pit Capital Cost Estimate

The open pit capital cost estimate has been prepared for the IA under the assumption of processing of open pit mined material at 10,000 tpd through a CIL. Project costs were estimated using first principles, cost data from Infomine (Infomine, 2024), and the experience of senior staff. The estimate assumes that the project will be operated by a contractor; therefore, no mining equipment capital costs were included as this equipment would be provided by the contractor.

The initial capital costs are incurred in the years prior to production. GRE's QP expects there will be three to five years of continued exploration, engineering, and permitting prior to a production decision.

Initial capital costs are defined as all costs until a sustained positive cash flow is reached. This includes labor and development costs in the pre-production years. Sustaining capital is defined as the capital costs incurred in the periods after a sustained positive cash flow is achieved through the end of the mine life.

All capital cost estimates cited in this Report are referenced in US dollars with an effective date of December 31, 2024.

Table 18-1: Granite Creek Open Pit Mine Project Capital Costs

Item	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Capital Cost (millions)
Open Pit Mining Equipment	$0.0	$0.0	$0.0	$0.0	$0.9	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.9
Capitalized Waste	$0.0	$23.3	$6.8	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$30.2
CIL Process	$73.3	$73.3	$0.0	$0.0	$0.0	$4.6	$4.6	$4.6	$4.6	$4.6	$0.0	$0.0	$0.0	$0.0	$169.6
Infrastructure	$4.7	$4.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$9.4
G&A	$0.0	$4.8	$0.0	$0.0	$0.1	$0.0	$0.0	$0.1	$0.0	$0.0	$0.1	$0.0	$0.0	$0.0	$5.1
Sustaining	$0.0	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.1
Working	$0.0	$0.0	$10.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	($10.6)	$0.0
Reclamation	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$9.0	$9.0	$18.0
Permitting	$5.0	$5.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$10.0
Contingency	$19.5	$21.9	$1.7	$0.0	$0.2	$1.2	$1.2	$1.2	$1.2	$1.2	$0.0	$0.0	$0.0	$0.0	$49.2
Total	**$102.5**	**$133.1**	**$8.5**	**$0.0**	**$1.2**	**$5.8**	**$5.8**	**$5.8**	**$5.8**	**$5.8**	**$0.1**	**$0.0**	**$9.0**	**$9.0**	**$292.4**

A contingency of 25% was applied to all capital costs.

Initial capital costs total $235.6 million, as detailed in Table 18-2.

Table 18-2: Granite Creek Open Pit Mine Project Initial Capital Costs

Item	Initial Capital Cost (millions)
Open Pit Mine	$0.0
Capitalized Waste	$23.3
Plant	$146.6
G&A	$4.8
Infrastructure	$9.4
Working	$0.0
Sustaining	$0.1
Reclamation	$0.0
Permitting	$10.0
Contingency	$41.4
Total	**$235.6**

18.1.1 Sustaining

Sustaining capital costs are set at 10% of the average yearly owner's mobile equipment operating costs, or $0.1 million, and are incurred in year -1.

18.1.2 Facilities

All buildings and associated infrastructure installed on the property on a permanent or semi-permanent basis are considered facilities. They include material and installation costs. These costs are incurred in years -2 and -1.

Each item's capital cost was estimated based on knowledge of nearby mine operations or senior engineers' experience. Table 18-3 shows total cost for each facility item.

Table 18-3: Granite Creek Open Pit Mine Project Facilities Capital Cost

Item	Capital Cost (millions)
Haul Roads	$0.5
Office	Existing
Warehouse	$1.2
Mine Shop	$4.1
Fuel Bay	$0.1
Wash Bay	$0.2
4x4 Pickup	$0.3
Security and Fencing	Existing
Surface Water Management	$0.6

Item	Capital Cost (millions)
Water Well with Pump	Existing
New Well Pump	Existing
Back Up Gen Set	$0.4
Sub-Station	Existing
Power Line 33KV	$2.1
Total	**$9.4**

18.1.3 Process Plant

Costs for the CIL plant are incurred in years -2 and -1. Costs for tailings dam expansions are incurred in years 4 through 8. The plant capital costs are summarized in Table 18-4.

Table 18-4: Granite Creek Open Pit Mine Project Plant Capital Costs

Item	Capital Cost (millions)
CIL	$146.6
Tailings Expansion	$23.0
Total	**$169.6**

18.1.4 Mine Equipment

Because the project was assumed to be contractor-operated, no mine equipment capital costs were included, with the exception of pit dewatering pumps and evaporators, which are incurred in year 3 and total $0.9 m.

18.1.5 G&A Capital

General and administrative (G&A) capital costs include computers, software, technical support equipment, and office equipment. Initial capital costs for computers are $50k, occurring in year -1, with replacement costs occurring every three years for the life of the project. Initial capital costs for software are estimated at $150k, occurring in year -1, with supplemental costs of $15,000 every year for the life of the project. The total G&A capital costs are summarized in Table 18-5.

Table 18-5: Granite Creek Open Pit Mine Project G&A Capital Costs

Item	Capital Cost (millions)
Computers	$0.2
Software	$0.3
Tech Equipment	$0.1

Item	Capital Cost (millions)
Office Equipment	$0.3
Metallurgical/Geotechnical Drilling and Assaying	$0.8
Total	**$1.7**

18.1.6 Working Capital

Working capital is the necessary cash on hand for the next period's operating cost. The estimated total is $10.6 million. This cost is recovered at the end of production.

18.1.7 Closure

Closure costs are estimated over two years at the end of production for closure and covering of waste storage facilities and the TSF. Total cost for site closure is $18.0 million. Additional details on closure costs are presented in Section **17.8**.

18.2 Open Pit Operating Cost Estimate

The operating costs assume contractor operation. A 10% contractor's premium was applied to all operating unit costs, labor unit rates, and supplies. Operating costs are summarized in Table 18-6.

Table 18-6: Granite Creek Open Pit Mine Project Operating Cost Summary

Item	Total Operating Cost (millions)	Unit Operating Cost	Unit
Mining	$637.0	$1.93	$/tonne mined
Processing	$343.6	$9.86	$/tonne processed
G&A	$53.3	$1.58	$/tonne processed
Contingency	$206.8		
Total	**$1,240.7**		

18.2.1 Labor

Hourly labor for the project is based on the number of people needed to operate and support equipment for each shift in a day plus additional crew to fill in for absences. Salaried labor in the project is based on job positions filled regardless of production changes or equipment units needed. Table 18-7 through Table 18-10 show the required labor, and Table 18-11 shows the estimated mining and G&A labor costs by year. Processing labor costs are built into the processing unit costs of $9.86/tonne.

Table 18-7: Granite Creek Open Pit Mine Project Hourly Laborers by Year

Position	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Drill Operator	12	24	28	24	24	20	24	20	16	8
Blaster	6	12	14	12	12	10	12	10	8	4
Blaster Helper	6	12	14	12	12	10	12	10	8	4
Haul Truck Driver	4	36	56	68	68	80	64	36	36	12
Loader/Shovel Operator	4	20	24	24	28	24	24	16	16	8
Dozer Operator	8	16	16	16	16	16	16	16	16	12
Loader Operator	4	4	4	4	4	4	4	4	4	4
General Equipment Operator	13	13	13	13	13	13	13	13	13	13
Water Truck Driver	8	8	8	8	8	8	8	8	8	8
Lube Truck Driver	8	8	8	8	8	8	8	8	8	8
Laborer	8	8	8	8	8	8	8	8	8	8
Heavy Duty Mechanic	24	48	59	62	63	65	60	45	44	29
Light Duty Mechanic	4	4	4	4	4	4	4	4	4	4
Tire Man	4	4	4	4	4	4	4	4	4	4
Total	**113**	**217**	**260**	**267**	**272**	**274**	**261**	**202**	**193**	**126**

Table 18-8: Granite Creek Mine Open Pit Mine Project Salaried Workers, Mine Management

Position	Number Each Year
Mine Superintendent	1
Mine Engineer	1
Geologist	1
Surveyor/Tech	1
General Foreman	1
Shift Supervisor	4
Total	**9**

Table 18-9: Granite Creek Mine Open Pit Mine Project General and Administrative Positions by Year

Position	Number Each Year of Active Open Pit Mining	Number Each Year of Stockpile Processing
General Manager	1	1
Purchasing Manager	0	0
Purchaser	1	1
Chief Accountant	1	1
Accounting Clerk	2	1
Human Resources/Relations Manager	1	1
Human Resources/Payroll Clerk	1	1
Security/Safety/Training Manager	1	1
Safety Officer	2	1
Environmental Supervisor	0	1
Environmental Technicians	2	1
Logistics Administrator	0	0
IT Manager	0	0
Warehouseman ON SITE	2	2
Accounts Payable Clerk	1	0
Receptionist/Secretary	1	0
Guards	4	4
Drivers	1	0
Laborers / Janitorial On Site	2	1
Total	**23**	**17**

Table 18-10: Granite Creek Mine Open Pit Mine Project Processing Positions by Year

Position	Number Each Year of Processing - CIL
Metallurgical Staff	
Superintendent	1
General Foreman	1
Maintenance Foreman	1
Shift Foreman	4
Chief Assay Chemist	1
Sr Metallurgist	1
Metallurgist	1
Process Technician	0
Instrument Technician	0
Subtotal	**10**

Position	Number Each Year of Processing - CIL
Laboratory	
Sample Prep	4
Assayers	2
Analytical	0
Subtotal	**6**
Crusher	
Operator	4
FEL Operator	4
Maintenance	1
Electrical	1
Subtotal	**10**
Grinding	
Operator	8
Maintenance	2
Electrical	2
Subtotal	**12**
CIL	
Operator	8
Maintenance	2
Electrical	1
Subtotal	**11**
Strip Circuit	
Operator	24
Maintenance	2
Electrical	2
Subtotal	**6**
Tailings	
Operator	0
Maintenance	0
Electrical	0
Subtotal	**0**
Total	**39**

Table 18-11: Granite Creek Open Pit Mine Project Labor Costs by Year (millions)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Open Pit Hourly Labor	$3.5	$19.7	$23.7	$24.3	$24.7	$24.0	$23.7	$18.2	$17.4	$5.6	$184.7

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Open Pit Salaried Labor	$0.5	$1.4	$1.4	$1.4	$1.4	$1.3	$1.4	$1.4	$1.4	$0.7	$12.3
G&A Labor	$1.0	$2.8	$2.8	$2.8	$2.8	$2.7	$2.8	$2.8	$2.8	$2.3	$25.2
Total	**$5.0**	**$23.8**	**$27.8**	**$28.4**	**$28.9**	**$28.0**	**$27.9**	**$22.4**	**$21.5**	**$8.6**	**$222.3**

18.2.2 Mining Equipment and Consumables

Mining equipment includes production equipment and support equipment. Mining production equipment hours are calculated using the equipment productivity estimates and the number of tonnes required to be moved. It was assumed that all mining will be contractor-operated. GRE included a 20% surcharge on the estimated operating costs for account for contractor markup.

Mining support equipment hours are calculated using the number of shifts that the equipment is operated per day, the number of pieces of equipment, and the operating hours per day. The operating hours per day are calculated assuming utilization of 85%, availability of 90%, and two twelve-hour shifts per day. Table 18-12 and Table 18-13 summarize the mining costs by year.

Table 18-12: Granite Creek Open Pit Mine Project Mining Equipment Costs by Year (millions)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Open Pit Production Equipment	$7.0	$25.2	$38.3	$34.9	$31.7	$33.4	$45.5	$27.2	$25.5	$4.9	$273.6
Open Pit Support Equipment	$2.1	$6.0	$6.0	$6.0	$6.0	$5.8	$6.0	$6.0	$6.0	$3.0	$52.6
Total	**$9.1**	**$31.2**	**$44.3**	**$40.8**	**$37.7**	**$39.1**	**$51.5**	**$33.1**	**$31.5**	**$7.8**	**$326.2**

Table 18-13: Granite Creek Open Pit Mine Project Blasting Costs by Year (millions)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Explosives	$2.3	$2.3	$6.3	$8.5	$6.6	$6.3	$5.7	$8.4	$6.2	$5.0	$0.7
Primers	$0.3	$0.3	$0.8	$1.0	$0.8	$0.8	$0.7	$1.0	$0.7	$0.6	$0.1
Material Control/ Sample Testing	$0.6	$0.6	$1.8	$2.4	$1.9	$1.8	$1.6	$2.4	$1.8	$1.4	$0.2
Misc	$0.2	$0.2	$0.5	$0.5	$0.5	$0.5	$0.5	$0.5	$0.5	$0.5	$0.3
Total	**$3.4**	**$3.4**	**$9.4**	**$12.5**	**$9.7**	**$9.3**	**$8.4**	**$12.4**	**$9.2**	**$7.5**	**$1.2**

18.2.3 Process Plant

The processing operating costs include labor, reagents and consumables, and power. The unit rate for processing is $9.86/tonne of material processed. In addition, $0.83/tonne was included for rehandling of material from stockpiles. A summary of the process operating costs is provided in Table 18-14.

Table 18-14: Granite Creek Mine Project Processing Costs by Year (1000s)

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
CIL Processing	$20.7	$33.2	$33.2	$33.2	$33.2	$33.2	$33.2	$33.2	$33.2	$33.2	$11.1	$330.2
Rehandle	$0.1	$1.4	$1.2	$1.9	$2.2	$1.3	$0.1	$0.0	$1.4	$2.9	$1.0	$13.4
Total	**$20.9**	**$34.6**	**$34.4**	**$35.0**	**$35.3**	**$34.4**	**$33.2**	**$33.2**	**$34.6**	**$36.0**	**$12.0**	**$343.6**

18.2.4 Taxes and Royalties

GRE prepared a generalized tax computation for the Granite Creek Mine Project. The following is a summary of tax elections incorporated into this tax computation:

- The Granite Creek Open Pit Mine Project consists of a single mine and property
- The Granite Creek Open Pit Mine Project will elect to treat mine development costs as incurred as deferred expenses
- The Granite Creek Open Pit Mine Project will elect out of bonus depreciation.
- The Granite Creek Open Pit Mine Project will elect to depreciate long-lived assets under the unit of production basis and all other assets will be depreciated using either 7-year or 15-year straight line depreciation
- The Granite Creek Open Pit Mine Project will elect to deduct reclamation costs under Section 468.

Royalties were included in the cost estimation on a block by block basis. The total royalty applied totaled approximately 5.7% of the gross revenue.

18.2.5 General and Administrative

General and administrative costs were estimated for four phases of the mine plan: open pit production operating and stockpile operating. The G&A costs include both salaried and hourly labor, supplies, office equipment, and anticipated regular expenses. Open pit production years have a G&A cost of $5.2 million per year; stockpile operations years have a G&A cost of $0.9 million per year.

18.3 Granite Creek Underground

18.3.1 Capital Costs

Capital cost estimates are based on past actual, supplier costs, and internal estimates. Capital contingencies are calculated at 15% of capital development and delineation drilling and 25% on everything else. Capital costs estimates are within a range of accuracy of +/- 50% and are suitable for an Initial Assessment evaluation.

Table 18-155 Capital Cost Estimates ($000's)

	2025	2026	2027	2028	2029	2030	Total
Capital Development	$19,553	$20,760	$12,755	$4,617	$600	$600	$58,885
Delineation Drilling	$6,000	$2,000	$2,000	$2,000	$2,000	$2,000	$16,000
Water Treatment Plant	($3,300.0)	($3,500.0)	($2,646.0)				$9,446
Feasibility Study	$1,000						
Underground Electrical	($500.0)	($1,000.0)	($500.0)	($500.0)	($500.0)	($500.0)	$3,500
Fans/Ventilation	($100.0)	($500.0)	($100.0)	($100.0)	($100.0)	($100.0)	$1,000
Contingency	($5,058)	($4,664)	($3,025)	($1,142)	($540)	($540)	$14,969
Total	($35,511)	($32,424)	($21,026)	($8,359)	($3,740)	($3,740)	$104,800

Unit mine development costs are derived from actual expenditures and work done over the period January through August 2024.

Table 18-16 Mine Development Unit Costs

Description	$/ft
Primary Drifting (15 ft x 17 ft)	$2,300
Secondary Horizontal Access (15 ft x 15 ft)	$2,300
Raise Bore (10 ft dia.)	$4,000

Excludes contingency

18.3.2 Closure and Reclamation

Total reclamation costs are estimated at $7.4M or $17.69/ounce produced. Reclamation costs are only for the underground mine-related disturbance. Legacy reclamation costs are included in the open pit estimates. Table 18-1 show reclamation cost detail.

Table 18-17 Annual Bonding, Reclamation and Closure Costs ($000s)

	2025 - 2032	2033–2037	2038-2042	Total
Reclamation Bonding	515	-	-	4,120
Reclamation	-	403	-	2,015
Closure and Monitoring	-	-	250	1,250
Total	515	403	250	7,385

18.3.3 Underground Mine Operating Costs

Operating unit costs are summarized in Table 18-1. The unit cost accuracy is within a +/- 50% range and are suitable for an Initial Assessment evaluation. The mining and trucking costs are derived from analysis of actual cost and production data over the period January through August 2024. They include all contractor charges, owner supplied materials and services.

Table 18-18 Unit Operating Cost Estimates

Item	Unit Cost	Units
Stope Attack Ramps	$110.59	$/ ton
Drift and Fill	$110.59	$ /ton
Cemented Backfill	$37.93	$ /fill ton
Gob Fill	$13.00	$ /fill ton
Expensed Waste	$110.59	$ /waste ton
Lone Tree Acid POX Processing	$106.00	$/ton
Lone Tree Alkaline POX Processing	$70.81	$/ton
Trucking to Twin Creeks	$8.16	$ /ton
Trucking to Lone Tree	$17.08	$ /ton

Total cost, cost per ton and cost per ounce are shown in Table 18- and Table 18- for the with and without inferred mineral resource cases respectively.

Table 18-19 Total and Unit Operating Costs (With Inferred Mineral Resources)

	Total Costs ($M)	Unit Cost ($/ton milled)	Cost per Ounce ($/oz Au)
Mining	$331.7	$208.7	$794

	Total Costs ($M)	Unit Cost ($/ton milled)	Cost per Ounce ($/oz Au)
Transportation & Processing	$98.8	$62.1	$237
G&A, Royalties & Net Proceeds Tax	$140.0	$88.1	$335
By-Product Credits			
Total Operating Cost/Cash Costs	$570.5	$358.9	$1,366
Closure & Reclamation	$7.4	$4.6	$18
Sustaining Capital	$88.8	$55.9	$213
All-in Sustaining Costs (1)	$666.6	$419.4	$1,597

Excludes Resource Conversion Drilling

Table 18-20 Total and Unit Operating Costs (Without Inferred Mineral Resources)

	Total Costs ($M)	Unit Cost ($/ton milled)	Cost per Ounce ($/oz Au)
Mining	$171.2	$208.7	$923
Transportation & Processing	$51.7	$63.1	$279
G&A, Royalties & Net Proceeds Tax	$92.4	$112.6	$498
By-Product Credits			
Total Operating Cost/Cash Costs	$315.3	$384.3	$1,699
Closure & Reclamation	$7.4	$9.0	$40
Sustaining Capital	$88.8	$108.2	$479
All-in Sustaining Costs (1)	$411.5	$501.6	$2,217

Excludes Resource Conversion Drilling

18.3.4 Cutoff Grade

Cutoff grades vary depending upon process location and recovery. Cutoff grades for both refractory process locations are shown in Table 18-.

Table 18-21 Cutoff Grades for Lone Tree POX and Mineralization Sales Agreement

	Acid POX	Alkaline POX	NGM
Gold Price ($/oz)	$2,175		
Nevada Commerce and Excise Tax	1.151%		

	Acid POX	Alkaline POX	NGM
Refining and Sales ($/oz)	$1.85		
Royalty	6%		
Recovery[1]	90%	82.5 – 94.2%	58%
Process Capacity (tpd)	2500	2,500	1000
Mine Capacity (tpd)	1000		
Mining Costs ($/ton)	$208.70		
Haulage Cost	$17.08	$17.08	$8.16
Process Cost	$106.00	$70.81	-
Incremental Cutoff Grade (opt)	0.067	0.053 - 0.052	0.050
Mine Limited Cutoff Grade (opt)	0.182	0.177 - 0.136	0.185
Fixed Costs ($ /ton)$	$39.49		
Process Limited Cutoff Grade (opt)	0.203	0.201 – 0.159	0.218

19 Economic Analysis

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under S-K 1300. This IA is preliminary in nature and 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. Readers are advised that there is no certainty that the results projected in this IA will be realized.

19.1 Taxes

19.1.1 Federal

The United States Government tax rate on corporations is 21% of taxable income. Taxable income is determined by offsetting revenue with depreciation, amortization, and depletion deductions. Unused depreciation and amortization deductions can be carried forward to the following year. The carryforward balance for the Granite Creek project at the beginning of 2025 is $91.9M.

19.1.2 Nevada

19.1.2.1 Net Proceeds Tax

Net mining proceeds are taxed at a rate of up to 5%. Net proceeds are generally defined as revenue less the costs of production. Capital investments are deductible using straight line depreciation over a 20-year period.

19.1.2.2 Excise Tax

The State legislature enacted an excise tax that went into effect in 2022. The tax applies to gross revenue from the extraction of gold and silver. The tax is two-tiered. Revenues greater than $20,000,000 and less than $150,000,000 are taxed at 0.75% while revenues above $150,000,000 are taxed at 1.1%.

19.1.2.3 Sales and Use Tax

Equipment and supplies for use in mining are subject to the sales and use tax. The tax rate for Eureka County is 6.85%.

19.1.2.4 Commerce Tax

The commerce tax is imposed on businesses with annual revenue exceeding $4,000,000. The commerce tax rate for mining companies is 0.051% of revenue above $4,000,000.

19.1.2.5 Modified Business Tax

All employers subject to Nevada Unemployment Compensation are also subject to the Modified Business Tax (MBT) on total gross wages less employee healthcare benefits paid. The MBT rate is 1.378%. The first $50,000 of gross wages is exempt from MBT.

19.1.3 Property Taxes

Property or ad valorem taxes are based on the value of the property, both real and personal. The Nevada constitution caps the property tax rate at five dollars for every $1,000 of assessed value. It is also capped by statute at $3.64 per $100 of assessed value. The assessed value in Nevada is 35% of the taxable value. Real and personal property taxes attributable to Real and personal property taxes attributable to Osgood Mining Company are summarized in Table 19 1 below. The total tax due for the 2023-2024 tax year is $100,996.60 (Humboldt County Assessor 2024).

Table 19-1 Real and Personal Property Taxes

Location	APN	Taxable Value	Assessed Value	Tax Rate	Annual Tax
Osgood 38N 42E 21	07-0121-05	$27,000			$210.79
Murphy/Osgood 38N 42E 28	07-0121-07	$11,000			$85.88
Osgood 38N 42E 29	07-0121-06	$27,000			$161.36
Premier 38N 42E 21	07-0121-08	$27,000			161.36
Osgood 38N 42E 33	07-0121-33	$27,000			$6,615.54
Real Property Total					$7,234.63
Ruby Hill Mine	Kelly Creek Groundwater				$6,404.45
Personal Property Mining Equipment	MM000015				$94,594.15
Total					$100,996.60

19.2 Granite Creek Underground

19.2.1 Cash Flow

Granite Creek underground is an operating mine and is in the initial production stage and ramping up to full production in 2025. Positive cash flow occurs early in the economic model with corresponding high IRR and short payback times.

A constant dollar cash flow analysis combining the mine production schedule including inferred mineral resources presented in Section 13.2 combined with the commodity pricing of Section 16.1 and the capital and operating costs of Section 18.3 is presented in Table 19-2 and Table 19-3 and graphically in Figure 19-1 and Figure 19-2. Results for the production plan excluding inferred mineral resources is presented in Table 19-4 and Table 19-5 and graphically in Figure 19-3 and Figure 19-4. Financial statistics from both cases are presented side by side in Table 19-6.

Table 19-2 Income Statement (Includes Inferred Mineral Resources)

	Production									
	2025	2026	2027	2028	2029	2030	2031	2032	2033	Total
Total Revenue	$88	$103	$96	$165	$176	$130	$107	$42	$ -	$908
Mining Cost	($44.3)	($41.8)	($45.3)	($48.2)	($54.5)	($40.4)	($33.3)	($11.8)	$ -	($319)
Haulage and Processing	($1.7)	($1.7)	($1.8)	($24.3)	($27.7)	($20.3)	($16.1)	($5.1)	$ -	($99)
Electrical Power	($1.3)	($1.4)	($1.6)	($1.7)	($1.7)	($1.7)	($1.5)	($1.2)	$ -	($12)
Site Administration	($8.6)	($8.6)	($7.7}	($7.7)	($7.7)	($7.7)	($7.7)	($7.7)	$ -	($64)
Refining and Sales	($0.1)	($0.1)	($0.1)	($0.1)	($0.1)	($0.1)	($0.1)	$0.0	$ -	($1)
Royalties & NV Taxes	($6.8)	($8.6)	($7.5)	($14.8)	($15.5)	($10.7)	($8.7)	($2.9)	$ -	($76)
Total Cash Cost	($63)	($62)	($64)	($97)	($107)	($81)	($67)	($29)	$ -	($570)
Cash Cost per Ounce[1] ($/oz)	$1,551	$1,307	$1,455	$1,275	$1,328	$1,357	$1,363	$1,479	$ -	$1,366
EBITDA	$25	$41	$32	$68	$68	$49	$40	$14	$ -	$338
Reclamation Accrual	($1)	($1)	($1)	($1)	($1)	($1)	($1)	($0)	$0	($7)
Depreciation	($19)	($26)	($27)	($49)	($53)	($41)	($34)	($13)	$ -	($262)
Total Cost	($82)	($89)	($92)	($147)	($162)	($123)	($102)	($43)	$ -	($840)
Income Tax	($3)	($5)	($2)	($7)	($6)	($3)	($3)	($1)	$ -	($29)
Net Income	$3	$10	$2	$12	$8	$3	$2	($1)	$ -	$39

Table 19-3 Cash Flow Statement (Includes Inferred Mineral Resources)

	Production									
	2025	2026	2027	2028	2029	2030	2031	2032	2033 - 2042	Total
Net Income	$3	$10	$2	$12	$8	$3	$2	-$1	$0	$39
Depreciation	$19	$26	$27	$49	$53	$41	$34	$13	$0	$262
Reclamation	$0	$0	$0	$1	$1	$1	$0	$0	$0	$0
Working Capital	-$7	$0	$0	-$4	-$1	$3	$2	$4	$3	$0
Operating Cash Flow	$15	$36	$29	$58	$61	$48	$38	$17	$3	$301
Total Capital	-$36	-$32	-$21	-$8	-$4	-$4	$0	$0	$0	-$105
After Tax Cash Flow	-$21	$4	$8	$49	$57	$44	$38	$17	$3	$197
Cumulative Cash Flow	-$21	-$17	-$9	$40	$97	$141	$180	$197	$200	

Table 19-4 Income Statement (without Inferred Mineral Resources)

	Production									
	2025	2026	2027	2028	2029	2030	2031	2032	2033	Total
Total Revenue	$39	$46	$43	$74	$78	$58	$48	$19	$ -	$404
Mining Cost	($22)	($21)	($23)	($24)	($27)	($20)	($17)	($6)	$ -	($159)
Haulage and Processing	$0	$0	$0	($10)	($11)	($8)	($7)	($4)	$ -	($52)
Electrical Power	$0	$0	$0	($10)	($11)	($8)	($7)	($4)	$ -	($12)
Site Administration	($9)	($9)	($8)	($8)	($8)	($8)	($8)	($8)	$ -	($64)
Refining and Sales	($0)	($0)	($0)	($0)	($0)	($0)	($0)	($0)	$ -	($0)
Royalties & NV Taxes	($3)	($3)	($3)	($6)	($6)	($4)	($3)	($1)	$ -	($28)
Total Cash Cost	($35)	($35)	($36)	($51)	($56)	($44)	($37)	($21)	$ -	($315)
Cash Cost per Ounce[1] ($/oz)	$1,971	$1,648	$1,824	$1,514	$1,566	$1,648	$1,687	$2,438	$ -	$1,699
EBITDA	$4	$11	$7	$22	$22	$14	$11	($2)	$ -	$88
Reclamation Accrual	($1)	($1)	($1)	($1)	($1)	($1)	($1)	($0)	$ -	($7)
Depreciation	($19)	($26)	($27)	($49)	($53)	($41)	($34)	($13)	$ -	($262)
Total Cost	($55)	($62)	($63)	($101)	($111)	($86)	($72)	($35)	$ -	($585)
Income Tax	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Income	($16)	($16)	($21)	($28)	($33)	($28)	($24)	($16)	$ -	($181)

Table 19-5 Cash Flow Statement (Without Inferred Mineral Resources)

	Production									Total
	2025	2026	2027	2028	2029	2030	2031	2032	2033 - 2042	
Net Income	($16)	($16)	($21)	($28)	($33)	($28)	($24)	($16)	$0	($181)
Depreciation	$19	$26	$27	$49	$53	$41	$34	$13	$0	$262
Reclamation	$0	$0	$0	$1	$1	$1	$0	($0)	($3)	($0)
Working Capital	($4)	$0	($0)	($2)	($1)	$1	$1	$2	$3	$0
Operating Cash Flow	($1)	$11	$6	$20	$21	$15	$11	($1)	($3)	$81
Total Capital	($30)	($30)	($19)	($6)	($2)	($2)	$0	$0	$0	($89)
After Tax Cash Flow	($36)	($22)	($15)	$12	$17	$11	$11	($1)	($36)	($24)
Cumulative Cash Flow	($36)	($58)	($73)	($61)	($44)	($33)	($22)	($23)	($24)	

Table 19-6 Financial Statistics

	With Inferred	Without Inferred
Gold price (US$/oz)	$2,175	
Silver price (US$/oz)	$27.25	
Mine life (years)	8	
Average mineralized mining rate (tons/day)	435	225
Average grade (oz/t Au)	0.339	0.292
Average gold recovery (autoclave %)	78%	78%
Average annual gold production (koz)	52	23
Total recovered gold (koz)	418	186
Sustaining capital (M$)	$88.8	$88.8
Cash cost (US$/oz) [1]	$1,366	$1,699
All-in sustaining cost (US$/oz) [1,2]	$1,597	$2,217
Project after-tax $NPV_{5\%}$ (M$)	$155	($30)
Project after-tax $NPV_{8\%}$ (M$)	$135	($33)
Project after-tax IRR	84%	-12.7%
Payback Period	3.2 Years	NA
Profitability Index $_{5\%}$[3]	12.6	0.7

Notes:

7. *Net of byproduct sales;*
8. *Excluding income taxes, resource conversion drilling, corporate G&A, corporate taxes and interest on debt;*
9. *Profitability index (PI), is the ratio of payoff to investment of a proposed project. It is a useful tool for ranking projects because it allows you to quantify the amount of value created per unit of investment. A profitability index of 1 indicates breakeven;*
10. *This IA is preliminary in nature, it 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 IA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability;*
11. *Inferred mineral resources constitute 50% of mass and 56% of gold ounces of all mineral resources. The "Without Inferred" statistics presented are a gross factorization of the mine plan without any redesign of mine excavations or recalculation of productivities and costs. Capital costs are the same for the "With Inferred" and "Without Inferred" scenarios. The "Without Inferred" scenario is presented solely to illustrate the project's dependence on inferred mineral resources;*
12. *The financial analysis contains certain information that may constitute "forward-looking information" under applicable Canadian and United States securities regulations. Forward-looking information includes, but is not limited to, statements regarding the Company's achievement of the full-year projections for ounce production, production costs, AISC costs per ounce, cash cost per ounce and realized gold/silver price per ounce, the Company's ability to meet annual operations estimates, and statements about strategic plans, including future operations, future work programs, capital expenditures, discovery and production of minerals, price of gold and currency exchange rates, timing of geological reports and corporate and technical objectives. Forward-looking information is necessarily*

based upon a number of assumptions that, while considered reasonable, are subject to known and unknown risks, uncertainties, and other factors which may cause the actual results and future events to differ materially from those expressed or implied by such forward looking information, including the risks inherent to the mining industry, adverse economic and market developments and the risks identified in Premier's annual information form under the heading "Risk Factors". There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information. All forward-looking information contained in this Presentation is given as of the date hereof and is based upon the opinions and estimates of management and information available to management as at the date hereof. Premier disclaims any intention or obligation to update or revise any forward-looking information, whether as a result of new information, future events or otherwise, except as required by law.

Figure 19-1 Gold Production and Cost per Ounce (With Inferred)

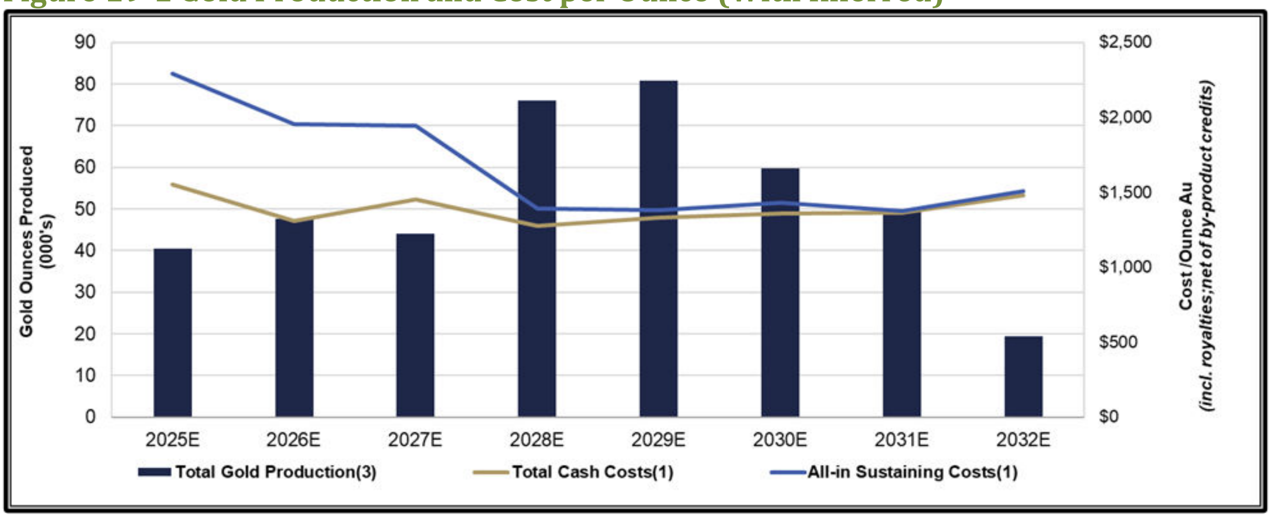


Figure 19-2 Cash Flow Waterfall Chart (With Inferred)

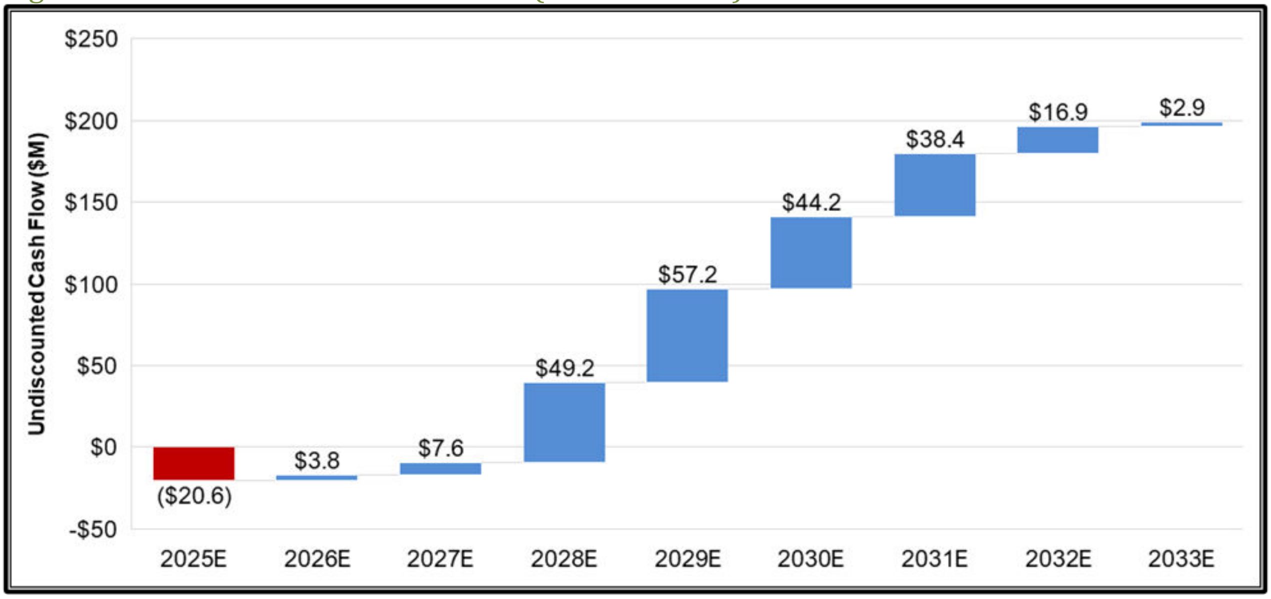


Figure 19-3 Gold Production and Cost per Ounce (Without Inferred)

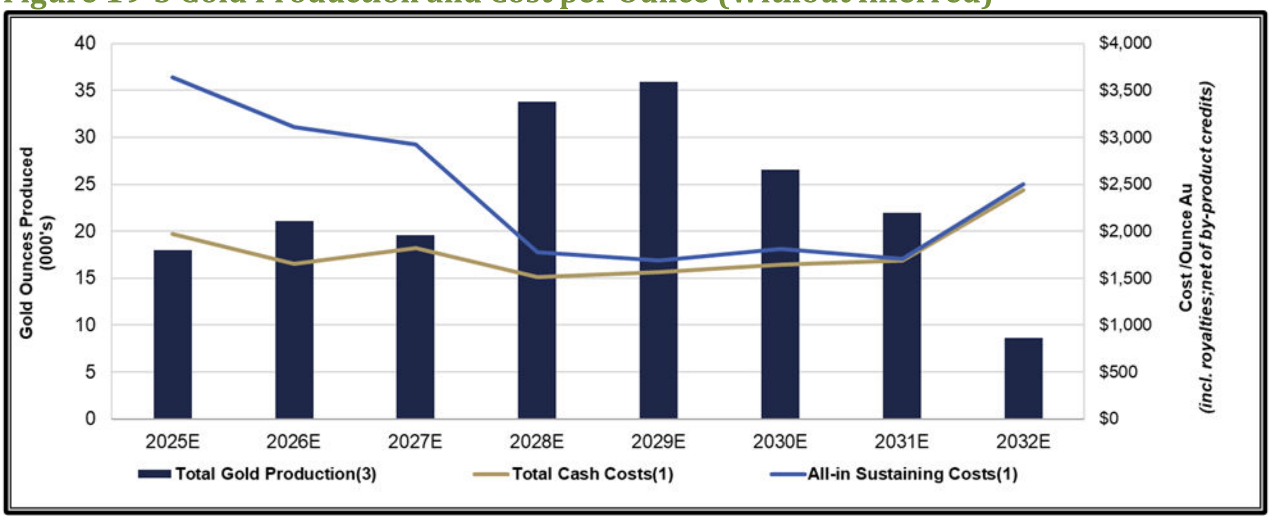
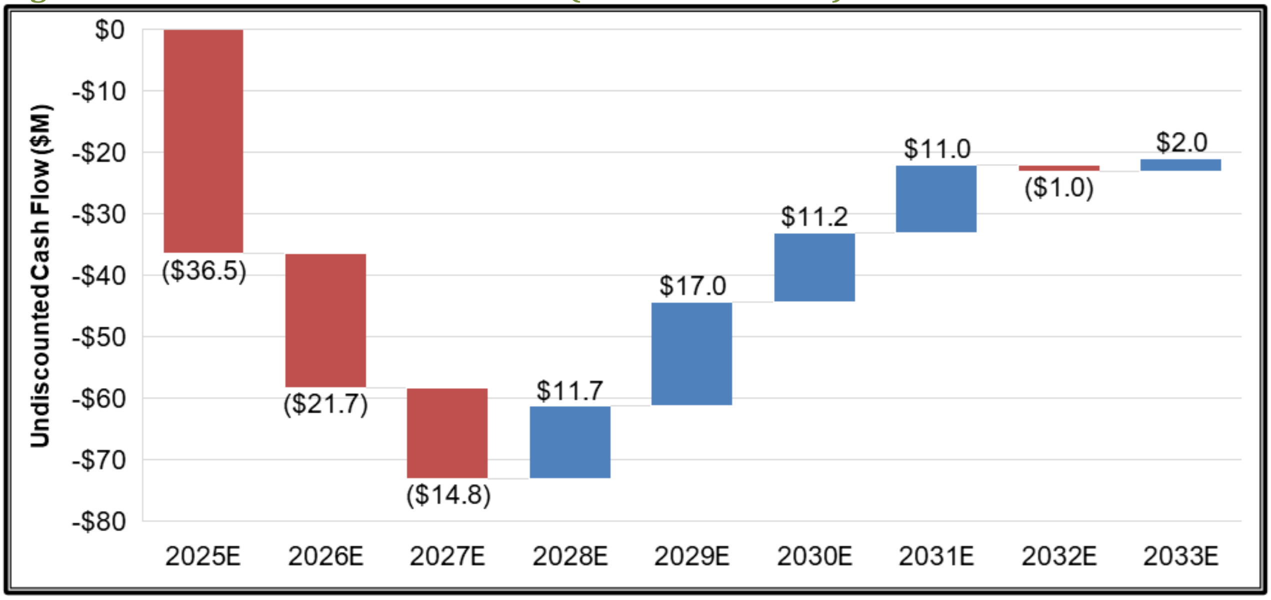


Figure 19-4 Cash Flow Waterfall Chart (Without Inferred)



19.2.2 Sensitivity Analysis

The sensitivity of NPV, IRR and profitability index are shown in Figure 19-5 through Figure 19-6. The Granite Creek underground mine's transition from development to production stage along with sustaining capital comprising 100% of all remaining capital expenditures provides the mine resilience to negative variance in gold price, operating costs and capital costs. The mine is most sensitive to gold price fluctuations. The gold price can decline to $1,565 per ounce or 28% before the after tax cash flow turns negative.

Figure 19-5 NPV 5% Sensitivity

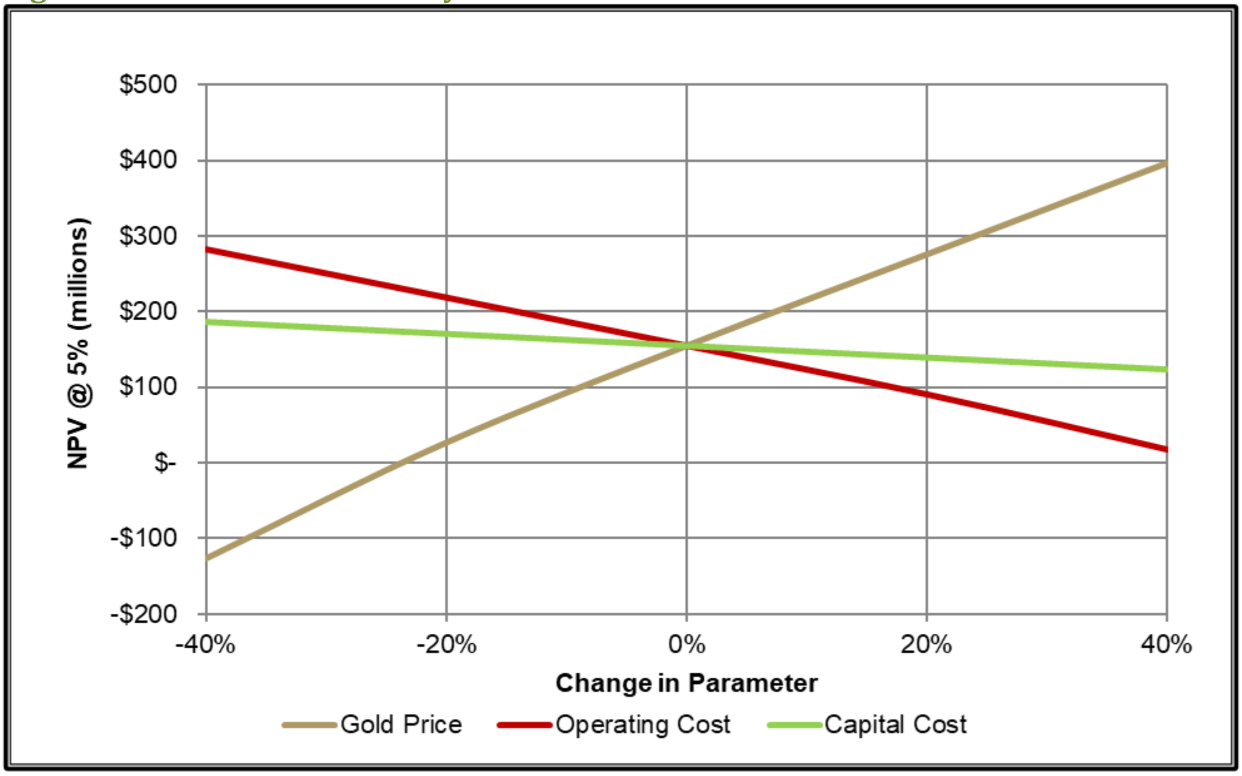


Figure 19-6 NPV 8% Sensitivity

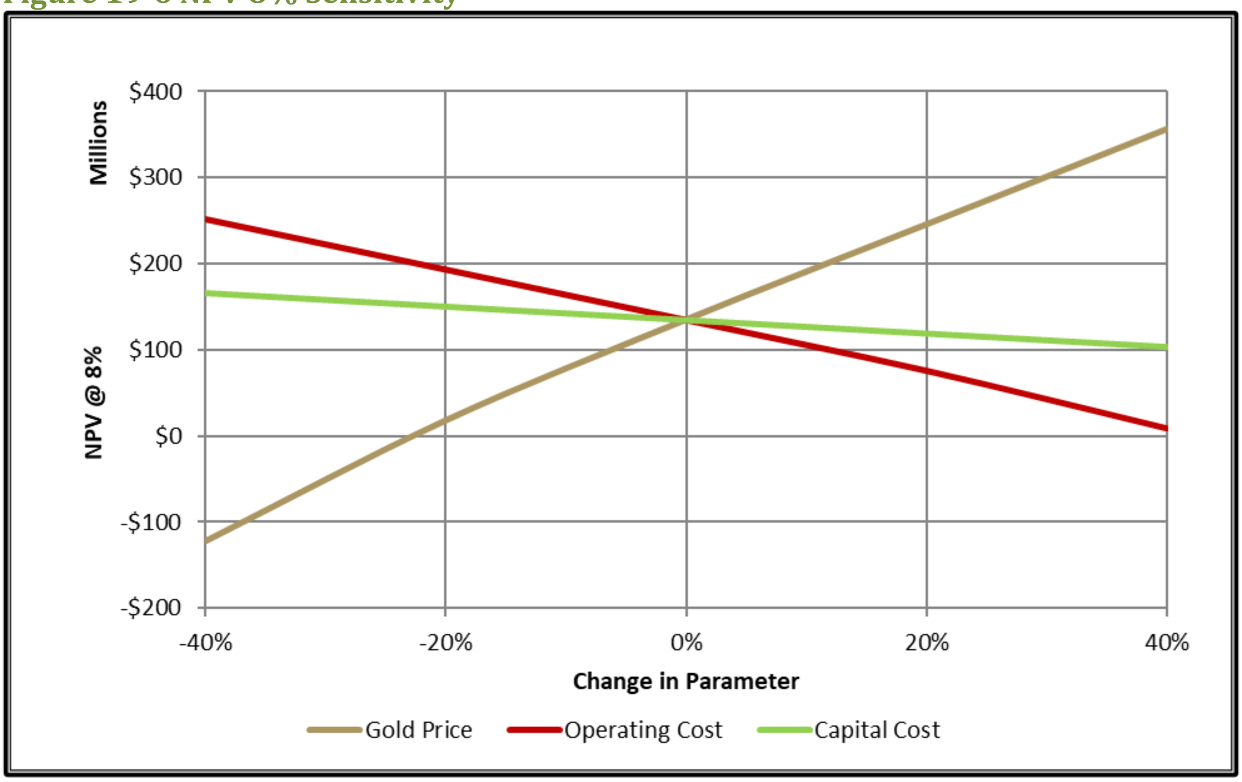


Figure 19-7 IRR Sensitivity

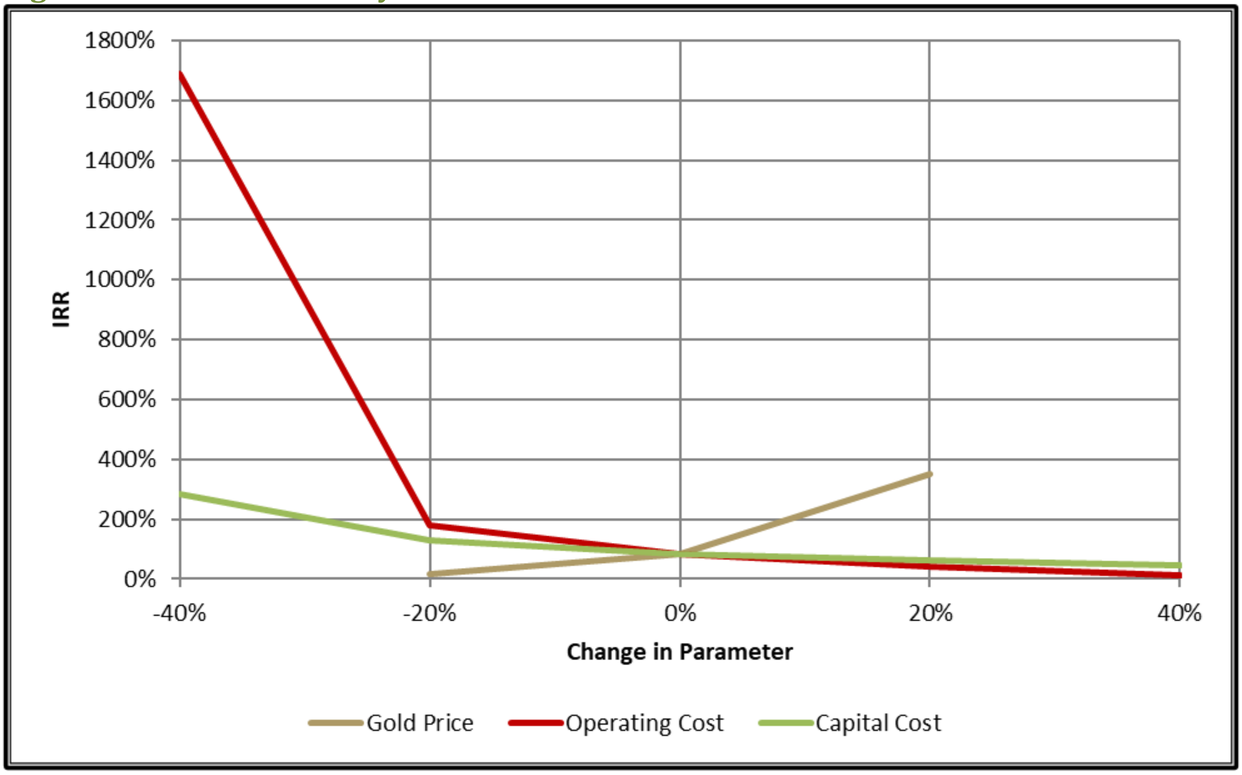
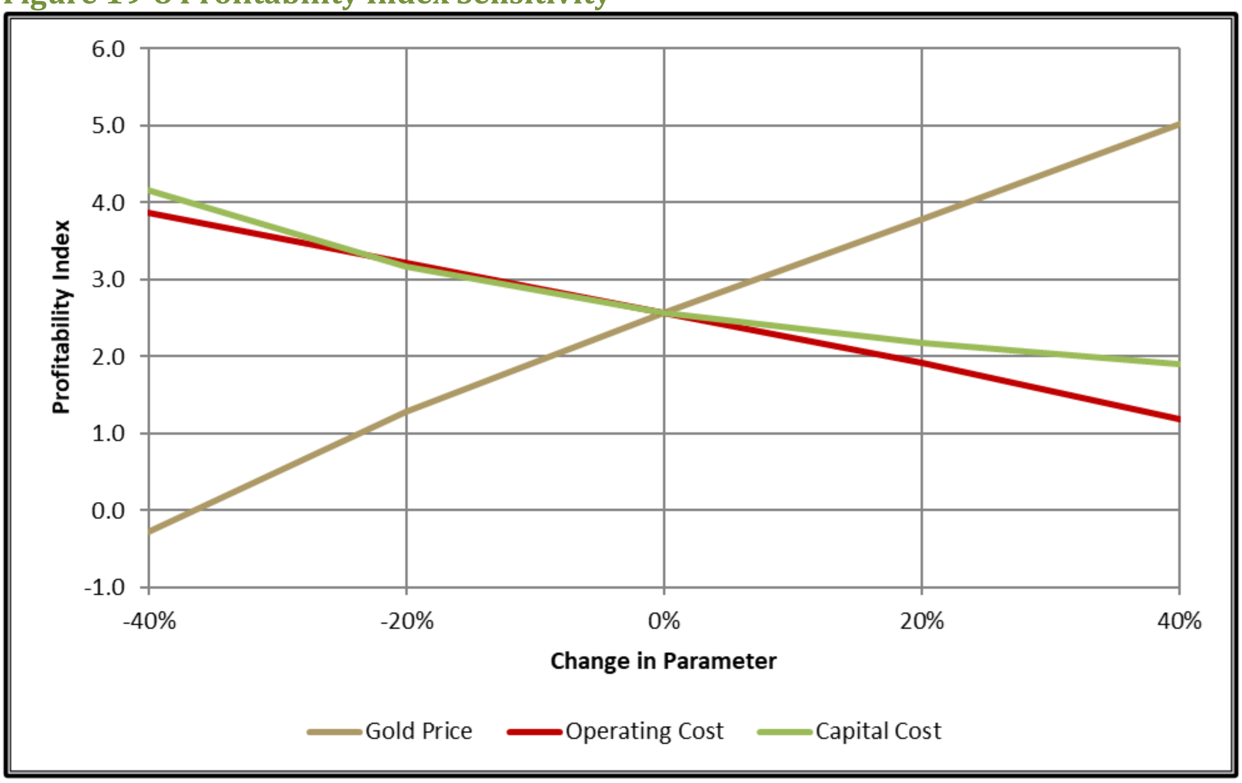


Figure 19-8 Profitability Index Sensitivity



19.3 Open Pit

19.3.1 Model Cases

A multi-scenario analysis method was used to analyze the economic performance of the project by varying the cutoff grade used, processing method or combination of methods, owner vs. contractor operations, tailings type, and equipment sizes.

GRE evaluated the following options:

- High grade cutoff grades of 0.35, 0.4, 0.45, 0.5, 0.55, 0.6, 0.65, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, and 1.0 g/t. All cases also included low-grade material between 0.25 g/t and the high grade cutoff grade.
- Three processing options: both CIL and heap leach, heap leach only, and CIL only.
- Two tailings types: conventional and dry stack
- Multiple sets of equipment sizes.

After analyzing the economic results of all cases considered, GRE selected the CIL only case with 0.85 g/t high grade cutoff, contractor operation, conventional tailings, and 133-tonne haul trucks and 21.9-tonne loaders as the base case as it results in the best overall economic results.

19.3.1.1 Economic Analysis

GRE performed an economic analysis of the project by building an economic model based on the following assumptions:

- Federal corporate income tax rate of 21%
- Nevada taxes:
  - Proceeds of Minerals Tax – variable, with a maximum of 5% of Net Proceeds
  - Property tax – 2.5605%
  - Nevada gold and silver mine royalty – variable, with a maximum of 1.1% of gross revenue
- Sales and use taxes are not included in the model
- Equipment depreciated over a straight 7 or 15 years and has no salvage value at the end of mine life
- Loss carried forward
- Depletion allowance, lesser of 15% of net revenue or 50% of operating costs
- Gold price of $2,175 per troy ounce, selected based on the CIBC January 2025 Consensus Gold Price Estimate (see Section 19 for more information)
- Gold recovery calculated as detailed in Section 13

- Block by block royalties totaling 5.7% of gross revenue, and a 10% net profit royalty

19.3.2 Results

GRE considered the following key economic parameters to determine the best scenario: NPV@5%, IRR, payback period, mine life, and initial capital cost. Table 19- summarizes the results of the economic model.

Table 19-7: Granite Creek Open Pit Mine Project Summary of Economic Model

Item	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Production															
CIL Tonnes Processed ('000s)	0.0	0.0	2,187.5	3,500.0	3,500.0	3,500.0	3,500.0	3,500.0	3,500.0	3,500.0	3,500.0	3,500.0	1,167.0	0.0	34,854.5
Contained CIL Au Oz Processed ('000s)	0.0	0.0	131.7	144.5	142.8	115.6	99.0	166.6	196.9	184.3	136.2	59.8	19.9	0.0	1,397.2
Recovered CIL Au Oz ('000s)	0.0	0.0	113.8	125.8	123.8	100.2	85.6	145.4	170.7	159.5	117.6	51.0	17.0	0.0	1,210.5
Revenue															
Gross Revenue (M$)	$0.0	$0.0	$247.6	$273.6	$269.3	$217.9	$186.2	$316.3	$371.3	$346.8	$255.8	$110.9	$37.0	$0.0	$2,632.7
Refining/Selling Cost (M$)	$0.0	$0.0	$0.6	$0.6	$0.6	$0.5	$0.4	$0.7	$0.9	$0.8	$0.6	$0.3	$0.1	$0.0	$6.1
Transportation Charges (M$)	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Royalty (M$)	$0.0	$0.0	$10.0	$13.1	$12.6	$10.4	$8.2	$16.4	$26.1	$26.2	$19.0	$6.2	$2.1	$0.0	$150.3
Net Smelter Revenue (M$)	**$0.0**	**$0.0**	**$237.0**	**$259.9**	**$256.1**	**$207.0**	**$177.6**	**$299.2**	**$344.4**	**$319.8**	**$236.1**	**$104.5**	**$34.8**	**$0.0**	**$2,476.4**
Total Operating Costs (M$)	$0.0	$0.0	$101.2	$152.5	$142.5	$140.4	$144.4	$165.9	$137.0	$131.9	$64.6	$44.8	$15.5	$0.0	$1,240.7
Owners Royalty Add Back Pre-Tax	$0.0	$0.0	$0.0	$0.1	$0.8	$1.2	$0.2	$3.1	$13.6	$15.2	$10.6	$1.4	$0.5	$0.0	$46.7
Pre-Tax Operating Cash Flow	**$0.0**	**$0.0**	**$135.8**	**$107.5**	**$114.5**	**$67.8**	**$33.4**	**$136.3**	**$221.0**	**$203.1**	**$182.2**	**$61.0**	**$19.7**	**$0.0**	**$1,282.4**
Taxes															
Federal Taxes (M$)	$0.0	$0.0	$15.9	$11.6	$11.4	$5.4	$2.4	$15.0	$28.7	$24.5	$23.5	$5.7	$1.5	$0.0	$145.5
State Taxes (M$)	$0.0	$0.0	$7.9	$6.5	$6.3	$4.3	$3.1	$7.5	$11.6	$10.5	$9.2	$2.7	$0.8	$0.0	$70.4
After-Tax Operating Cash Flow	**$0.0**	**$0.0**	**$112.1**	**$89.3**	**$96.7**	**$58.2**	**$28.0**	**$113.8**	**$180.7**	**$168.1**	**$149.5**	**$52.7**	**$17.5**	**$0.0**	**$1,066.6**
Nevada Property Taxes (M$)	$0.0	$0.0	$3.0	$2.4	$2.0	$1.6	$1.2	$0.9	$0.8	$0.8	$0.7	$0.0	$0.0	$0.0	$13.3
Total Capital Costs (M$)	$102.5	$133.1	$8.5	$0.0	$1.2	$5.8	$5.8	$5.8	$5.8	$5.8	$0.1	$0.0	$0.0	$0.0	$274.4
Net Cash Flow After Tax (M$)	**($102.5)**	**($133.1)**	**$90.0**	**$86.1**	**$82.2**	**$44.2**	**$17.7**	**$93.8**	**$153.4**	**$142.7**	**$131.5**	**$46.7**	**$6.5**	**$1.6**	**$660.9**
Cumulative Net Cash Flow After Tax (M$)	($102.5)	($235.6)	($145.6)	($59.5)	$22.7	$66.9	$84.6	$178.4	$331.8	$474.5	$606.0	$652.7	$659.3	$660.9	

Table 19- summarizes the key economic results for the base case.

Table 19-8: Granite Creek Open Pit Mine Project Key Economic Results

After Tax Economic Measure	Value
After Tax NPV@5% (millions)	$417.2
After Tax IRR	28.7%
Initial Capital (millions)	$254.7
Payback Period (years)	3.72
All-in Sustaining Cost ($/oz Au Produced)	$1,227.4
Cash Cost ($/oz Au Produced)	$1,180.5

19.3.3 Sensitivity Analyses

GRE evaluated the after-tax NPV@5% and IRR sensitivity to changes in gold price, capital costs, and operating costs. For this analysis, GRE used a base case gold price of $2,175/oz. The results indicate that the after-tax NPV@5% and IRR are most sensitive to gold price and grade, moderately sensitive to operating cost, and least sensitive to capital costs (Figure 19- and Table 19- for NPV@5%, and Figure 19- and Table 19- for IRR).

Figure 19-9: After Tax NPV@5% Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

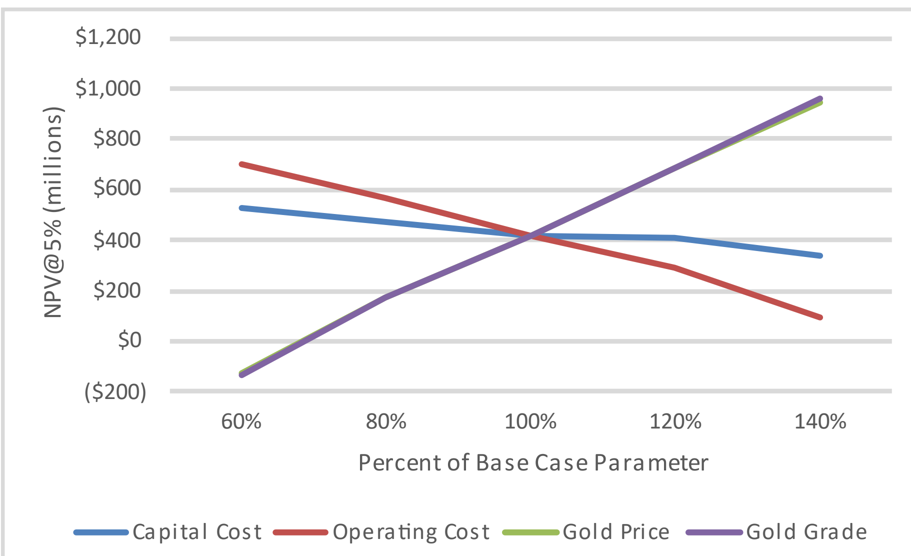


Table 19-9: After Tax NPV@5% (1000s) Sensitivity to Gold Price, Capital Costs, and Operating Costs

Variable	% of Base Case				
	60%	80%	100%	120%	140%
Capital Cost	$527.5	$474.0	$417.2	$406.1	$339.4

Operating Cost	$703.2	$566.2	$417.2	$290.6	$94.4
Gold Price	($128.8)	$174.4	$417.2	$681.5	$943.6
Gold Grade	($136.6)	$169.1	$417.2	$688.7	$960.4

Figure 19-10: IRR Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

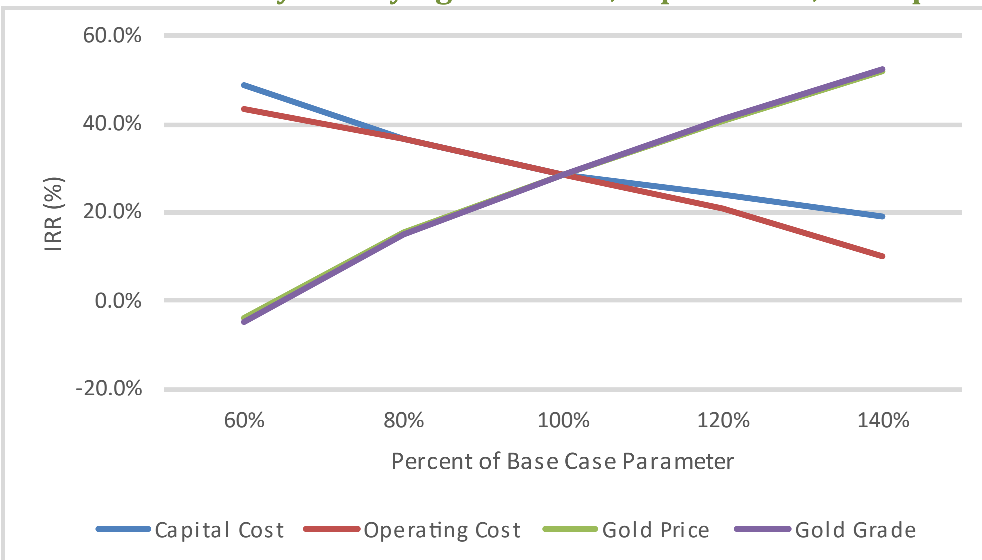


Table 19-10: IRR Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

Variable	% of Base Case				
	60%	80%	100%	120%	140%
Capital Cost	48.7%	36.8%	28.7%	24.1%	19.2%
Operating Cost	43.4%	36.5%	28.7%	21.1%	10.3%
Gold Price	-4.0%	15.5%	28.7%	40.9%	51.8%
Gold Grade	-4.6%	15.2%	28.7%	41.2%	52.6%

19.3.4 Inferred Mineral Resource Impacts on Economics

This economic evaluation includes inferred mineral resources, which are considered to have a level of geological uncertainty too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability. The quantity of inferred mineralized tonnes included in the evaluation is 1,891.5 ktonnes, and the quantity of inferred Au ounces is 63.5 koz, representing 5.4% of the mineralized tonnes and 4.5% of the Au ounces.

As the percentages of inferred mineralized tonnes and Au ounces are quite low, the impact of including them in the economic evaluation is low. Removing the inferred tonnes and ounces would result in an after-tax cash flow of $600 million, an after-tax NPV@5% of $378 million, and an IRR of 27.4%.

19.3.5 Conclusions of Economic Model

The project economics shown in the IA are favorable, providing positive NPV values at varying gold prices, capital costs, and operating costs. The IA is preliminary in nature and 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

20 Adjacent Properties

Most of the mineral rights surrounding Granite Creek are owned or controlled by Nevada Gold Mines. The only reported mining activities adjacent to Granite Creek were conducted by Nevada Gold Mines at the Turquoise Ridge Complex (TRC) where approximately 28 Moz Au have been produced since 1938 (Table x-x). The TRC is in Humboldt County, approximately 64 km northeast of Winnemucca and approximately 10 km north of the Granite Creek Mine. Historic and current production at TRC includes open-pit and underground mining.

Deposits that compose the TRC are Carlin-type, structurally and stratigraphically controlled and sediment-hosted. They containing disseminated micrometer-sized gold occurring on arsenic-rich pyrite rims, primarily within decalcified, carbonaceous rocks. Preferred host lithologies for gold mineralization are the Comus Formation, followed by the Valmy and Etchart Formations. Sub-microscopic gold mineralization is associated with arsenian pyrite, quartz, calcite, realgar, and orpiment. Gold mineralization is likely Eocene in age, and it is overprinted in some areas by a late stage of realgar, orpiment, and calcite. Gold-bearing zones can be located close to granodiorite and dacite dikes and beneath basaltic sills, evidencing the importance of rheologic contacts to mineralization (Barrick., 2024).

Practical Mining is unable to verify the information on production, mineral resources and mineral reserves included for the adjacent properties. This type of adjacent property information is not necessarily indicative of the mineralization at Granite Creek. In addition, the QP is not aware of any declared mineral resource that might have an impact on Granite Creek's Mineral Resources, Mineral Reserves or mining operations.

21 Other Relevant Data and Information

The authors are not aware of any other relevant data or information.

22 Interpretation and Conclusions

22.1 Drilling

Drilling programs completed at the Property between 2005 and 2015 have included QA/QC monitoring programs that have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams.

In 2021, QP GRE reviewed all of AMC's work on available QA/QC data between 2005 and 2015 (AMC, 2020). In 2025, i-80 provided all QA/QC data from surface exploration holes drilled in 2021 and 2022 to GRE, and GRE reviewed all of them and found no material errors. GRE also reviewed and checked QA/QC Procedures and the database provided by i-80. GRE confirmed discussions and recommendations made in prior technical reports and noted the following:

- Formal, written procedures for data collection and handling should be developed and made available to PMC field personnel. These should include procedures and protocols for fieldwork, logging, database construction, sample chain of custody, and documentation trail. These procedures should also include detailed and specific QA/QC procedures for analytical work, including acceptance/rejection criteria for batches of samples.
- A detailed review of field practices and sample collection procedures should be performed on a regular basis to ensure that the correct procedures and protocols are being followed.
- Review and evaluation of laboratory work should be an on-going process, including occasional visits to the laboratories involved.

In general, the QA/QC sample insertion rates used fall below general accepted industry standards. For future exploration campaigns, standards, blanks, and duplicates including one standard, one duplicate, and one blank sample should be inserted every 20 interval samples, as is common within industry standards.

CRM samples show a reasonable level of accuracy but poor to moderate precision when using standard deviations provided by the CRM supplier. A maximum of three to five different CRM samples would be adequate to monitor laboratory performance at the approximate cut-off grades, average grades, and higher grades of the deposits.

Blank sample results are considered acceptable and suggest no systematic contamination has occurred throughout the analytical process.

Duplicate sample results show suboptimal performance, which may be a result of the heterogenous nature of mineralization, uncrushed samples, and sampling variance. Overall duplicate samples appear to be positively biased, with duplicate results returning higher grade than original samples.

Previous reporting suggests that umpire sampling has been completed at the Property. The results of this sampling were not available in the drillhole database and therefore the QP was not able to assess accuracy of the primary laboratory.

Although it is not possible to guarantee that there are no material impacts on the local scale, overall, based on the checking and reviewing the previous technical report dated 2020, GRE considers the assay database to be acceptable for Mineral Resource estimation.

22.2 Environmental

The environmental, permitting and social conditions are favorable for the project. The site is a producing underground operation built on a historic mine site that has been impacted by operations and exploration since the 1940s.

The project, with the current available geochemical data, does not appear to pose ARD risk and only appears to pose minimal metal leaching (ML) risk with respect to antimony and arsenic release. To mitigate this ML risk, the mine operates a water treatment plant (see Section 17.2356). GRE suggests that the project expand upon the current geochemical characterization to include a thorough characterization of future waste rock and tailings, as this will be required for future permitting efforts (See Section 23).

The site has all relevant State and Federal permits to allow for the current ongoing underground operation at Granite Creek (see Section 17.3). Major permit revisions, as well as additional permits, will be required for the proposed plan of operations in this IA. Section 17.3 details the anticipated new permits, permit revisions, and permitting efforts that Granite Creek will need to face for the mining plan described in this IA. Due to the favorable jurisdiction, and despite the challenges of the National Environmental Policy Act (NEPA) permitting process, it is anticipated that the permits can be acquired in ~3 years.

The project has sufficient water rights to operate and will be self-sufficient for water due to the underground and open pit dewatering requirements.

22.3 Open Pit Conclusions

22.3.1 Metallurgical Conclusions

22.3.1.1 Sample Representativity

Within each zone, drilling has been localized to relatively small portions of the mineralized domains, as seen in Figure 10 1 and Figure 10 2The samples' metallurgical response is likely to represent the zone's general behavior, but additional sampling of each zone to confirm the metallurgical response will reduce uncertainty. The lack of this metallurgical drilling remains a risk to the project.

22.3.1.2 Test Work on Open Pit Samples

Cyanide leach bottle roll tests and column leach tests were completed on samples from both the Mag and CX open pits. Both Homestead and Atna commissioned these tests.

The test work demonstrated that many of the Mag Pit samples had high preg-robbing factors due to carbonaceous material in the feed. Due to the variable preg-robbing characteristics of the feed material, a higher degree of representativity of the Mag Pit should be evaluated.

Bottle roll tests were conducted on Mag Pit samples using NaOH as an alternative to hydrated lime, as a method of treating material with preg-robbing characteristics. These tests demonstrated that raising the pH improved gold recovery and decreased cyanide consumption.

A column leach test on a Mag Pit sample showed that there was no gold recovery benefit in using NaOH rather than lime (at the equivalent pH).

Test work on ground materials showed that Mag Pit materials were amenable to CIL methods. CIL treatment showed low impact from the TOC. Gold recoveries ranged from 83% to 94%.

Column leach tests on the Mag Pit samples achieved gold recoveries in the range of 19% to 82%.

Column leach tests on the CX Pit samples achieved gold recoveries of 82%.

22.3.2 Mineral Resource Conclusions

Mineral Resources for the Granite Creek open pit mine project have been prepared to industry best practices and conform to the resource categories defined by the SEC in S-K 1300. In the option of the QP, the resource evaluations reported herein are a reasonable representation of the Granite Creek open pit deposits.

In the opinion of the QP, the Mineral Resource model presented in this report is representative of the informing data, which is of sufficient quality and quantity to support the Mineral Resource estimate to the classifications applied.

22.3.3 Mining

The open pit mine plan for the is based on conventional mining techniques, reasonable production assumptions, and consideration of risks to achieving the mine plan.

22.3.4 Economics

The open pit project economics shown in the IA are favorable, with an after-tax NPV@5% of $417.2 million and IRR of 28.7%. The sensitivity analysis shows that the economics are most sensitive to gold price and grade, then operating costs, and then capital costs.

The IA is preliminary in nature and 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 under CIM Definition Standards. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

22.4 Underground Conclusions

22.4.1 Metallurgy

1. Granite Creek underground samples were refractory with baseline CIL gold recoveries ranging from 9% to 46%, averaging 31%;
2. Shake flask tests with gold cyanide spikes were used to determine preg robbing index. The average preg-robbing index was 17.9%, ranging from 4.4% to 54.1%.;
3. Bench top autoclave batch pressure oxidation tests were completed on all samples with 2 sets of acid conditions and four sets of alkaline conditions. Acid conditions resulted in resulted in higher sulfur oxidations and higher gold recoveries;
4. Three continuous pressure oxidation runs were completed with two acid and one alkaline sets of conditions based on the batch results. The continuous results followed the results of the batch tests with the acid conditions producing the higher sulfur oxidations and gold recoveries;
5. Overall gold recoveries increased with increasing sulfur oxidation;
6. Cyanide destruction tests on CIL tailings using the SO2/air process reduced weak acid dissociable cyanide concentrations to below 50 ppm using established reagent addition rates and retention time;
7. Thickening and filtration tests on CIL tailings showed unacceptable thickening properties and filtration rates. Thickening and filtration of pressure oxidation streams is not recommended.
8. Arsenic concentrations in the samples averaged 0.29%, largely occurring as arsenian pyrite with only trace amounts of arsenopyrite.
9. Sulfide minerals were predominantly pyrite with some marcasite.
10. Mercury concentrations ranged from 31 ppm to 138 ppm, averaging 81 ppm. These concentrations will require mercury capture and abatement equipment in the process flowsheet.

22.4.2 Mining and Infrastructure

1. Mine infrastructure has been completed.
2. Production ramp up has reached approximately 400 tons per day.
3. The mining contractor is in place with the full complement of equipment and personnel.
4. Decline development has accessed 700 vertical feet of mineralization of the Otto and Ogee zones. Development has reached the top of the South Pacific zone allowing additional active production stopes.
5. The drill lateral drift over the South Pacific zone has been completed.
6. Reconciliation of the model to mill indicates process head ounces exceed model by 19%. This appears to be from mining in a larger low grade halo around high grade core.
7. Processed grade is lower than the life-of-mine planned grade due to extensive mining of marginal mineralization below the economic cutoff grade.

22.4.3 Economics

At $2,175 per ounce gold price the Granite Creek underground mine provides an 84% IRR due to the advanced stage of the underground mining with infrastructure in place. The projects NPV5% is $155M.

23 Recommendations

23.1 Granite Creek Open Pit

The QP's recommend the following items and budget to advance the Granite Creek project towards production (Table 23-1).

Table 23-1 Estimated Costs to Complete the 2-Year Program

Exploration Cost Area	Total
Drilling (Exploration QA/QC, Metallurgy, and Geotechnical)	$5,000,000
Metallurgical Testing	$400,000
Permitting, including all baseline studies	$9,000,000
Engineering	$750,000
Total	$15,150,000

23.1.1 Metallurgical Recommendations

The following recommendations have been put forward:

23.1.1.1 Test Work Recommendations

A metallurgical drilling program should be undertaken to collect samples within the various zones representing the spatial, mineralogical, and grade difference. The collected samples should be tested for the following:

- Paired fire assays and cyanide soluble assays to define cyanide solubility.
- Bottle roll tests with and without carbon to predict reagent consumption as well as amenability to CIL treatment and to evaluate the impact of sulfide sulfur on the CIL performance.
- Column leach tests at various sizes to predict field recovery for material to be heap leached. This should be performed on those materials with a cyanide solubility of greater than 50%. Recovery by size fraction should be completed as part of the testing program.
- Conduct SAG and ball mill testing to determine the work index.
- Additional autoclave pretreatment of underground materials should be completed, especially for those materials that showed lower gold extraction.
- Infill the drill hole database with TOC and S= assays.
- Conduct arsenic and mercury assays on all samples employed for metallurgical testing.

23.1.1.2 Geometallurgy Recommendations

The geometallurgical work completed as part of this technical report should be expanded using the planned metallurgical test program results. The intent will be to confidently define those materials that can be treated by heap leaching or CIL methods and those that require autoclave treatment.

23.1.2 Environmental Recommendations

At the IA stage, the environmental recommendations focus on the efforts required to get state and federal permits. These recommendations are summarized below.

23.1.2.1 Requirements for the EIS

As mentioned in Section 17.7, the NEPA permitting process is expected to result in the need to complete an EIS. This process can take many years (even in a favorable jurisdiction like Nevada). As a result, the site will need baseline studies and supplemental environmental reports to prepare itself for the permit process.

To begin, the site needs several baseline reports. These will likely be:

- Air Quality
- Biological Resources
- Surface Water
- Groundwater
- Geochemistry (including mine waste rock leachate, TSF leachate, TSF supernatant, and pit lakes)
- Archeological and cultural resources

Because the kinetic geochemical tests have a year-long duration, the geochemistry study may be the critical path for permitting (and possibly production).

SERs are part of the NEPA process. Mr. Breckenridge of GRE recommends the commencement of the following

- Geochemistry study
- Pit lake study for the MAG pit
- Backfill study for the mine waste below the water table in the CX pit
- RIB water quality impact study

Because the site has never had a full EIS, it may require the following additional SERs:

- Noise and vibration
- Visual impacts
- Air quality
- Biology
- Archeology

These additional reports should be started as soon as possible so they do not slow the critical path to production.

Some of these studies, such as the hydrogeologic study predicting the underground mine dewatering and the potential impact of underground dewatering on water resources, were currently underway as of the effective date of this study.

23.2 Granite Creek Underground

23.2.1 Recommendations

1. Metallurgical Testing
   a) Establish sampling using the most recent mine plan to select samples to evaluate pressure oxidation with CIL cyanidation under Lone Tree conditions. Testing should also include baseline CIL tests and roasting testing as a comparison.
   b) Testing should attempt to establish head grade and extraction relationships for use in more detailed resource modelling;
   c) Mineralogy impacts need to be established and geologic domains within each resource need to be determined;
   d) Additional comminution data should be collected to assess hardness variability within the zones and any potential impacts on throughput in the Lone Tree process plant..
   e) The resource model should be advanced to include arsenic, TCM, TOC, mercury, as these will be important for predicting grades if toll process offsite is used and potentially for estimating extractions within the resources;
   f) The estimated cost for the suggested next phase metallurgical program is to $350,000 based on current market pricing.
2. Feasibility Study
   a) Complete the resource conversion drilling program in the South Pacific zone;
   b) Update the mineralization models, and;
   c) Complete the feasibility study.
3. Dilution Control
   a) Align stope drift excavation direction parallel to strike wherever possible;
   b) Minimize mining outside the high grade 0.10 opt grade shells, and;

c) Reduce stope drift widths when the mineralized high grade zone is less than 15 feet.

23.2.2 Underground Feasibility Study Work Program

The recommended work program for the underground mine is listed in Table 23-2

Table 23-2 Feasibility Study Work Program

Task	Cost $M
South Pacific Infill Drilling	$ 6.0
Metallurgical Testing	$ 0.5
Feasibility Study -	$ 0.5
Contingency	$ 1.15
Total -	$ 8.15

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25 Reliance on Information Provided by the Registrant

i-80 provided information regarding the following

- Status of surface and mineral estate, Section 3;
- Geologic setting and mineral deposits, Section 6;
- Hydrogeologic study and reports prepared by LRE Water, Section 7.4, 15.1 through 15.3;
- 2024 Processing statistics. Section 11.7.2;
- Lone Tree POX facility description and flow sheet. Section 14.3.
- CIBC market study. Section 16.1;
- Summary of financing arrangements. Section 16.2;
- Status of permitting, environmental liabilities and closure estimates. Section 17.
- Underground mine cost accounting reports for 2024, Section 18.3;
- Underground mine production data for 2024, Section 11.72 and 18.3;

The Registrant provided information appears reasonable and is within industry norms or information is from reputable third party organizations.