UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 8-K

CURRENT REPORT
Pursuant to Section 13 or 15(d) of the Securities Exchange Act of 1934

Date of Report (Date of earliest event reported): **February 10, 2025**

U.S. GOLD CORP.
(Exact name of registrant as specified in its charter)

Nevada	**001-08266**	**22-1831409**
(State or other jurisdiction of incorporation)	(Commission File Number)	(I.R.S. Employer Identification Number)

1910 E. Idaho Street, Suite 102-Box 604 Elko, NV	**89801**
(Address of principal executive offices)	(Zip Code)

Registrant's telephone number, including area code: **(800) 557-4550**

(Former name or former address, if changed since last report)

Check the appropriate box below if the Form 8-K filing is intended to simultaneously satisfy the filing obligation of the registrant under any of the following provisions:

☐ Written communications pursuant to Rule 425 under the Securities Act (17 CFR 230.425)

☐ Soliciting material pursuant to Rule 14a-12 under the Exchange Act (17 CFR 240.14a-12)

☐ Pre-commencement communications pursuant to Rule 14d-2(b) under the Exchange Act (17 CFR 240.14d-2(b))

☐ Pre-commencement communications pursuant to Rule 13e-4(c) under the Exchange Act (17 CFR 240.13e-4(c))

Securities registered pursuant to Section 12(b) of the Act:

Title of each class	Trading Symbol(s)	Name of each exchange on which registered
Common stock	**USAU**	**Nasdaq Capital Market**

Indicate by check mark whether the registrant is an emerging growth company as defined in Rule 405 of the Securities Act of 1933 (§ 230.405 of this chapter) or Rule 12b-2 of the Securities Exchange Act of 1934 (§ 240.12b-2 of this chapter).

Emerging growth company ☐

If an emerging growth company, indicate by check mark if the registrant has elected not to use the extended transition period for complying with any new or revised financial accounting standards provided pursuant to Section 13(a) of the Exchange Act. ☐

Item 8.01 Other Events

U.S. Gold Corp. (the "Company") has completed a Technical Report Summary detailing the results of its updated pre-feasibility study for the CK Gold Project, titled "Technical Report Summary CK Gold Project for U.S. Gold Corp." and effective as of February 10, 2025. The Technical Report Summary was prepared by the Company and Samuel Engineering, Inc. in accordance with Subpart 1300 of Regulation S-K. A copy of the Technical Report Summary is attached as Exhibit 96.1 to this Current Report on Form 8-K.

Item 9.01 Financial Statements and Exhibits.

(d) Exhibits.

Exhibit No.	Description
96.1	Technical Report Summary of CK Gold Project for U.S. Gold Corp., Laramie County, Wyoming, USA, effective as of February 10, 2025.
104	Cover Page Interactive Data File (embedded within the Inline XBRL document)

SIGNATURES

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned hereunto duly authorized.

	U.S. GOLD CORP.	
Date: February 14, 2025	By:	*/s/ Eric Alexander*
	Name:	Eric Alexander
	Title:	Chief Financial Officer

Exhibit 96.1

TECHNICAL REPORT SUMMARY
CK GOLD PROJECT

FOR

U.S. GOLD CORP.

Responsible Company	Signature & Date
AKF Mining	
Drift Geo	
John Wells	
Samuel Engineering, Inc.	
Tierra Group International, Ltd. (TGI)	
U.S. Gold Corp (Registrant)	

PREPARED BY

U.S. Gold Corp.
1807 Capitol Avenue
Cheyenne, WY. 82001

Samuel Engineering, Inc.
8450 East Crescent Pkwy. Ste. 200
Greenwood Village, CO 80111-2816
303.714.4840

SE Project No. 24128-01, Rev A
February 10, 2025

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1.0 EXECUTIVE SUMMARY

1.1 PROPERTY SUMMARY AND OWNERSHIP

Samuel Engineering, Inc. (Samuel) was commissioned by U.S. Gold Corp. (U.S. Gold) to prepare a Pre-Feasibility Study (PFS) for the CK Gold Project (Project). This is a Technical Report Summary (TRS) summarizing the findings of the PFS in accordance with Securities Exchange Commission Part 229 Standard Instructions for Filing Forms Regulation S-K subpart 1300 (S-K 1300). This TRS aims to report mineral resources, mineral reserves, and economics for the CK Gold Project. The effective date of this report is February 10, 2025.

The CK Gold Project is in Laramie County, Wyoming, in the southeastern portion of the state, approximately 20 miles west of Cheyenne. It is centered in the north half of Section 36, T14N, R70W. The property encompasses approximately 1,120 acres of mineral leases on Section 36, the south half of Section 25, and the northeast quarter of Section 35. Additionally, to accommodate the mine footprint for facilities, primarily the tailings storage facility, which cannot be accommodated on State Section 36, an option agreement for a further 712 acres on portions of Section 25 and Section 31 has been secured with the private landowner. Unless otherwise specified, all units are U.S. Customary and U.S. dollars.

1.2 MINERAL RESOURCE STATEMENT

This section remains largely unchanged from the *"S-K 1300 Technical Report Summary CK Gold Project,"* dated December 1, 2021. However, the modeled mineral resources have been updated to include datasets from the 2021 drilling program conducted by U.S. Gold.

Mark Shutty, CPG, is the Qualified Person (QP) responsible for the mineral resource estimation. The updated estimate was prepared using Leapfrog and MinePlan software, utilizing the geologic database accumulated throughout the project's history. The 2021 drilling program contributed to enhancing resource classification and expanding the modeled mineralized domain and contained resources. Gold (Au), copper (Cu), and silver (Ag) mineralization observed in the 2021 drill intercepts closely aligned with previously modeled host lithologies, exhibiting consistent metal grades with neighboring drill hole intercepts and the estimated grades in the block model.

In the QP's opinion, the updated resource estimate represents a reasonable representation of the in-situ resources for the CK Gold Project based on all available data as of the effective date. Mineral resources are reported using a gold-equivalent (AuEq) cut-off grade, incorporating metal recovery and pricing assumptions. The cut-off grade varies with the expected recovery of different material types but averages 0.010 ounces per short ton (oz/st) AuEq, equivalent to 0.35 grams per metric tonne (gpt) AuEq. The resources are constrained within an optimized pit shell that, in combination with the cut-off grade, provides reasonable prospects for economic extraction.

Table 1.1 and Table 1.2 present a detailed tabulation of the Mineral Resources, inclusive of Reserves, for the CK Gold Project as of the effective date of this report.

Table 1.1: Mineral Resource Statement									
	Mass	Gold (Au)		Copper (Cu)		Silver (Ag)		Au Equivalent (AuEq)	
	Tonnes (000's)	Oz (000's)	oz/st	lbs (million)	%	Oz (000's)	oz/st	Oz (000's)	oz/st
Measured	36,400	608	0.0167	138	0.189	1,703	0.047	975	0.0268
Indicated	51,200	544	0.0106	163	0.159	1,901	0.037	1,001	0.0195
M + I	**87,600**	**1,152**	**0.0131**	**301**	**0.172**	**3,604**	**0.041**	**1,976**	**0.0226**
Inferred	34,900	334	0.009	112	0.161	1,073	0.031	653	0.0187

1. Mineral resources are estimated using Ordinary Kriging, constrained by geological domains based on lithology and mineralization controls. The underlying datasets supporting the resource estimate have been reviewed, validated, and verified by the Qualified Person (QP).

2. Mineral resources are reported in short tons within an optimized pit shell, using a breakeven gold equivalent (AuEq) cut-off grade of 0.011 oz/st for Oxide and Mixed material and 0.010 oz/st for Sulfide material. The overall average AuEq cut-off grade for all reported resources is 0.010 oz/st. No dilution or mining recovery factors have been applied.

3. The AuEq cut-off grade is calculated using realized metal prices of $1,860.10/oz Au, $3.92/lb Cu, and $22.52/oz Ag, with average metallurgical recoveries by oxidation type as follows:

    - Gold (Au): 55% (Oxide/Mixed), 64% (Sulfide)
    - Copper (Cu): 30% (Oxide), 78% (Mixed), 87% (Sulfide)
    - Silver (Ag): 61% (Oxide/Mixed), 70% (Sulfide)

4. The optimized pit shell was generated using the Lerchs-Grossman method, incorporating all classified resources, realized metal prices, $2.50/ton mining costs, $9.20/ton processing costs, a 50° slope angle, and varying metallurgical recoveries as detailed in Table 11.12.

5. No dilution or mining recovery factors have been applied to the resource estimate.

6. No known legal, environmental, or permitting issues impact the reported resources.

7. Resources are reported within the company's permitted land tenure/exploration license boundaries.

8. Mineral resources are classified in accordance with S-K 1300 definitions and are reported inclusive of mineral reserves.

9. Rounding may result in minor discrepancies in tonnage, grade, and contained metal totals.

10. There is no guarantee that mineral resources will be converted to mineral reserves.

11. The mineral resource estimates were prepared, reviewed, and validated by Mark Shutty, CPG, the independent Qualified Person (QP) for these estimates, in accordance with S-K 1300 Definition Standards adopted on December 26, 2018.

12. The mineral resource estimate effective date is January 6, 2025.

Table 1.2: Mineral Resource Statement (Metric)									
	Mass	Gold (Au)		Copper (Cu)		Silver (Ag)		Au Equivalent (AuEq)	
	Tonnes (000's)	Oz (000's)	gpt	Tonnes (000's)	%	Oz (000's)	gpt	Oz (000's)	gpt
Measured (M)	33,000	608	0.57	62.4	0.189	1,703	1.60	975	0.92
Indicated (I)	46,500	544	0.36	74.0	0.159	1,901	1.27	1,001	0.67
M + I	**79,500**	**1,152**	**0.45**	**136.4**	**0.172**	**3,604**	**1.41**	**1,976**	**0.77**
Inferred	31,600	334	0.33	50.9	0.161	1,073	1.06	653	0.64

1. Mineral resources are estimated using Ordinary Kriging, constrained by geological domains based on lithology and mineralization controls. The underlying datasets supporting the resource estimate have been reviewed, validated, and verified by the Qualified Person (QP).

2. Mineral resources are reported in short tons within an optimized pit shell, using a breakeven gold equivalent (AuEq) cut-off grade of 0.39 g/t for Oxide and Mixed material and 0.34 g/t for Sulfide material. The overall average AuEq cut-off grade for all reported resources is 0.35 g/t. No dilution or mining recovery factors have been applied.

3. The AuEq cut-off grade is calculated using realized metal prices of $1,860.10/oz Au, $3.92/lb Cu, and $22.52/oz Ag, with average metallurgical recoveries by oxidation type as follows:

   - Gold (Au): 55% (Oxide/Mixed), 64% (Sulfide)
   - Copper (Cu): 30% (Oxide), 78% (Mixed), 87% (Sulfide)
   - Silver (Ag): 61% (Oxide/Mixed), 70% (Sulfide)

4. The optimized pit shell was generated using the Lerchs-Grossman method, incorporating all classified resources, realized metal prices, $2.50/ton mining costs, $9.20/ton processing costs, a 50° slope angle, and varying metallurgical recoveries as detailed in Table 11.12.

5. No dilution or mining recovery factors have been applied to the resource estimate.

6. No known legal, environmental, or permitting issues impact the reported resources.

7. Resources are reported within the company's permitted land tenure/exploration license boundaries.

8. Mineral resources are classified in accordance with S-K 1300 definitions and are reported inclusive of mineral reserves.

9. Rounding may result in minor discrepancies in tonnage, grade, and contained metal totals.

10. There is no guarantee that mineral resources will be converted to mineral reserves.

11. The mineral resource estimates were prepared, reviewed, and validated by Mark Shutty, CPG, the independent Qualified Person (QP) for these estimates, in accordance with S-K 1300 Definition Standards adopted on December 26, 2018.

12. The mineral resource estimate effective date is January 6, 2025.

The estimates of Mineral Resources may be materially affected if mining, metallurgical, or infrastructure factors change from those currently anticipated at the CK Gold Project. Estimates of Inferred Mineral Resources have significant geological uncertainty, and it should not be assumed that all or any part of an Inferred mineral resource will be converted to the Measured or Indicated categories. Mineral Resources that are not Mineral Reserves do not meet the threshold for reserve modifying factors, such as estimated economic viability, that would allow for conversion to Mineral Reserves.

1.3 MINERAL RESERVE STATEMENT

Mineral Reserves are based on an open pit mine design and production schedule using reasonable design parameters. AKF Mining Services Inc. (AKF) performed economic pit-limit analysis using Vulcan's Pit Optimizer software, which uses the Lerchs–Grossmann (LG) algorithm to determine an economic excavation limit based on input optimization parameters. Antonio Loschiavo, P. Eng., is the QP responsible for the Mineral Reserves statement.

Measured Mineral Resources within the mine design and schedule convert to Proven Mineral Reserves , and Indicated Mineral Resources within the mine design convert to Probable Mineral Resources. Metal prices used for the cut-off grade calculation and economics are $1,755/oz gold, $3.77/lb. copper, and $23/oz silver. The Mineral Reserves are reported above a value per ton cut-off threshold of $0.01/st, as recovery varies by material type.

Table 1.3 contains the tabulation of the Mineral Reserves for the CK Gold Project as of the effective date of this report. Mineral reserves are reported inside a detailed pit design using suitable parameters for the site.

Table 1.3: Mineral Reserves Statement

	Mass	Gold (Au)		Copper (Cu)		Silver (Ag)		Au Equivalent (AuEq)	
	Tons (000s)	Oz (000s)	oz/st	M lbs	%	Oz (000s)	oz/st	Oz (000s)	oz/st
Proven (P1)	34,500	595	0.017	133	0.192	1,591	0.046	909	0.026
Probable (P2)	38,800	426	0.011	127	0.164	1,417	0.037	763	0.020
P1 + P2	**73,200**	**1,022**	**0.014**	**260**	**0.177**	**3,008**	**0.041**	**1,672**	**0.023**

1. Reserves tabulated above a VPT cut-off value of $0.01/st (see text).
2. Note: Only three significant figures are shown, and the sum may not be due to rounding.

No known relevant factors materially affecting the estimation of Mineral Reserves are discussed in this report.

AKF completed the 2021 Pre-Feasibility Study (PFS) design pits based on the 2021 PFS metal prices and operating costs. During the 2024 PFS update, metal prices and operating costs increased, which triggered a review of the Mineral Reserves by rerunning the LG optimizations based on the latest metal prices and operating costs. As a result, the comparison between the 2021 and 2024 PFS Mineral Reserves shows a 3% increase in ore tons and a 5% increase in waste tons.

1.4 GEOLOGY AND MINERALIZATION

The Silver Crown Mining District, where the Project is located, is underlain by Proterozoic rocks that make up the southern end of the Precambrian core of the Laramie Range. Metavolcanic and metasedimentary rocks of amphibolite-grade metamorphism are intruded by the approximately 1.4-billion-year-old Sherman Granite and related felsic rocks. Within the project area, foliated granodiorite is intruded by aplitic quartz monzonite dikes, thin mafic dikes, and younger pegmatite dikes. Shear zones with cataclastic foliation striking N60°E to N60°W are found in the southern part of the Silver Crown district, including at CK Gold. Copper and gold mineralization at the Project occurs primarily in unfoliated to mylonitic granodiorite. The granodiorite typically shows potassium enrichment, particularly near contact with quartz monzonite. Mineralization is associated with a N60°W-trending shear zone.

CK Gold mineralization has been interpreted as a shear-zone controlled, disseminated, and stockwork gold-copper deposit in Proterozoic intrusive rocks. Most mineralization is in granodiorite, with lesser amounts in quartz monzonite and thin mafic dikes. Hydrothermal alteration is overprinted on retrograde greenschist alteration and includes a central zone of silicification, followed outward by a narrow potassic zone, surrounded by propylitic alteration. Higher grade mineralization occurs within a central core of thin quartz veining and stockwork mineralization surrounded by a zone of lower grade disseminated mineralization. Disseminated sulfides and native copper with stockwork malachite and chrysocolla are present at the surface, and chalcopyrite, pyrite, minor bornite, primary chalcocite, pyrrhotite, and native copper are present at depth. Gold occurs predominantly associated with chalcopyrite and a minor proportion of free gold.

1.5 METALLURGICAL TESTING

Several metallurgical testwork programs have been completed on multiple samples of mineralization from the CK Gold Project. The work dates to 2008 when Saratoga Gold Company (Saratoga) first contracted SGS Lakefield (SGS) to perform fundamental characterization work and scoping level separation tests (flotation and cyanide leaching) on several composites of sulfide and oxide mineralization. This established that flotation was the most suitable method to process CK Gold Project mineralization to recover copper, gold, and silver into a high value concentrate.

No further work was completed until 2020 when U.S. Gold commenced a drilling program that included several holes designed to generate sufficient sample material for a metallurgical test work update. The metallurgical program that followed commenced in December 2020 at Kappes, Cassiday, and Associates (KCA) Laboratory in Reno, Nevada, before it transitioned over to Base Metals Laboratory (BML) in Kamloops, Canada. Several metallurgical programs have been completed at BML, including further flotation characterization, grindability, mineralogy, and dewatering. This work has confirmed flotation as the most suitable processing method. The most recent test work program concluded at BML in August 2022, and the overall body of testwork is now judged to be of suitable depth and quality to act as a valid reference for the Pre-Feasibility Study process design.

Although the current QP (John Wells) was not directly involved with historical work pre-2020, the reports were reviewed, resulting in general concurrence with the conclusions. Whilst the 2021 PFS work incorporated process plant designs that were based on early SGS test work and 2021 BML results, this PFS also includes more recent work from BML. The most recent test work program concluded at BML in August 2022, and the overall body of testwork is now judged to be of suitable depth and quality to act as a valid reference for the Pre-Feasibility Study process design.

Three composites, each 200-300 kg, were prepared for test work: a High-Grade Oxide composite, an Oxide composite, and a Sulfide composite. A narrow band of "mixed" material is between the oxide and sulfide zones. As this only represented a small component of the drill core it was included in the Sulfide Composite. However, as results subsequently showed, the impact was significant. The mineralogy indicated 10-15% of the copper minerals in this "sulfide" composite were not sulfide. A second Sulfide Composite was prepared from core more remote from the mixed zone and tested at BML in July 2021. This resulted in significantly better copper, gold, and silver recoveries.

Sub-samples of core from each composite were provided to Hazen Research in Denver for comminution test work. This showed the material to be of medium hardness but relatively competent. This supports the selection of a SAG-Ball-Pebble crusher grinding circuit. A primary grind P_{80} of 90 µm appears to be close to optimal.

Open circuit flotation of the High-Grade composite was successful at KCA, providing high recoveries of copper (55%), gold (69%), and silver (40%) to a 25% Cu flotation concentrate. Locked Cycle Tests (LCTs) at BML confirmed these results.

Flotation of the Oxide Composite proved to be more challenging. Flotation was moderately successful in that open circuit rougher and cleaner tests produced a low-grade but high value copper concentrate, 10-15% Cu, that contained over 150 gpt gold and 100 gpt silver. This material constitutes about 6-8% of the deposit. The mine plan could see this material treated on a campaign basis or combined with sulfide.

The sulfide zone constitutes most of the deposit. LCTs on two sulfide composites gave high recoveries of copper (82-88%) and gold (67-74%) to a 21-25% Cu, 76 gpt Au, 82 gpt Ag concentrate. Silver recovery was 59%.

Seven variability samples were selected for test work. With less non-sulfide copper, the copper recovery was over 80%. Gold and silver recoveries showed significant variation.

1.6 MINE DESIGN, OPTIMIZATION, AND SCHEDULING

The CK Gold Project is planned as an open pit mine with a production life of approximately 10.3 years. Two independent mine planning and sequencing studies have been accomplished and show broad agreement. U.S. Gold contracted AFK Mining to develop a mine design and schedule for the Project. Lerchs-Grossmann pit optimization analysis was performed using reasonable design and economic parameters, and the result was used to guide the mine design process. The final mine design is comprised of four phases, and material movement is scheduled on an annual basis. Pit design parameters are based on a geotechnical drilling program and detailed stability analysis and are suitable for the mining equipment selected.

Surface mining is a cyclical process where the four main tasks, including drilling, blasting, loading, and haulage, occur concurrently at different areas of the property. In areas to be excavated, vertical blast holes are drilled in a regular pattern and charged with blasting agents. The material will be shot, loaded into 100 st class rigid frame haul trucks, and transported based on material type to one of four different locations: run-of-mine (ROM) Crusher Stockpile, Tailings Facility, Ore Stockpile, or Waste Rock Facility. Wherever possible Crusher Stockpile ore will be directly dumped into the primary crusher at the process plant.

Owner operator mining has been selected as the preferred method for this PFS. The owner will also operate the mine planning, ore control, process plant, and general site administration (G&A). This decision is due to the location of the Project, local mining, and the availability of potential labor within 30 miles of the site (Laramie and Cheyenne, Wyoming). Hybrid owner/contractor operations are still being evaluated to leverage the regional mine contractor expertise and possibly reduce project capital costs.

The primary driver of the mine schedule is the production of sufficient ore, which drives the excavation of waste and other materials to ensure sufficient ore is exposed for mining. The nominal ore production rate was set at 20,000 stpd or 7.3M stpy (18,100 tpd, or 6.6 Mtpy) of ore delivered to the crusher. In the first year, ore production is 90% of full capacity to account for the commissioning of the concentrator. Pre-production stripping is scheduled to start two years before production begins (Year -2 Q1) which consists of 1.200,000 st of material. Pit mining life is approximately eight years with almost another two additional years of ore stockpile processing.

1.7 MINERAL PROCESSING

The CK Gold Project processing facility has been designed to process 20,000 stpd of gold/copper sulfide ore. The processing facility and the unit operations therein are designed to produce a concentrate at 17.0% Cu or greater, with an average gold grade of 41 g/st.

The process facility will consist of a ROM crushing circuit, crushed ore storage, a semi-autogenous grinding (SAG) mill/ball mill comminution circuit, rougher flotation, regrind circuit, and cleaner flotation to liberate, recover, and upgrade the copper and gold from the ROM ores. Flotation concentrate will be thickened, filtered, sent to a concentrate load out bin, and bagged for subsequent shipping.

Tailings generated in the flotation process will be filtered to an optimum low moisture content to produce “dry stack” tailings. This will maximize water conservation and structural strength and avoid the need for a conventional tailings dam and the associated environmental and safety risks.

The tailings slurry produced by flotation will first be thickened for initial water recovery, and thickener overflow water will be returned to the process for reuse. The thickened slurry will be pumped to storage tanks ahead of a large pressure filtration plant comprising multiple large pressure filters that further reduce the water content to <15% (typically 14%). This will leave the solids as a compressed “cake” material that will be dropped from the press onto a conveyor going to a tailings bin where the dry filtered cake will be loaded into haul trucks for transportation to the dry-stack Tailings Management Facility (TMF).

The process plant will consist of the following unit operations and facilities:

- Coarse ore receiving and storage area from the open pit mine.
- Jaw crushing system, crushed ore stockpile, and stockpile reclaim system to convey crushed ore to the process.
- SAG/Ball mill circuit incorporating cyclones for classification.
- SAG mill pebble crushing circuit.

- Rougher flotation circuit.
- Rougher concentrate regrinding circuit.
- Cleaner flotation circuit incorporating three flotation stages and cleaner scavenger flotation cells.
- Concentrate thickening and filtration circuit, including a concentrate bin and bagging station.
- Tailings thickening and filtration circuits.
- Tailings disposal at a dry-stack storage facility.
- Reagent handling, utilities, process water, and fresh-water systems.

1.8 INFRASTRUCTURE

An access road approximately 4.2 miles long and 26 feet wide will be constructed, generally centered along a 60-foot-wide right-of-way outside the project site boundary.

The TMF is sited east of the process plant within a valley formed by the ephemeral South tributary of Middle Crow Creek. It begins to the east of the South Crow Creek water transmission pipeline easement. The basin's topography contains and directs the placement of tailings towards the northeast.

The filtered tailings will be co-deposited with waste rock to provide structural buttresses for stability and a cover to protect against weathering and wind erosion. The TMF will be developed in three phases, each consisting of a prepared subgrade, underdrain collection system, composite liner system (CLS), seepage collection system, tailings, and waste rock. The tailings will be placed in the TMF in 10-to-20-foot lifts, and the waste rock buttress and shell will be installed in 10- to 20-foot lifts as the tailings increase in elevation. Processed tailings will be hauled to and placed in the TMF until Year 8.25. After that, the remaining tailings produced will be hauled to and placed in the open pit.

Designs were prepared for the mine maintenance area, administration and warehouse building area, and other supporting facilities. The civil grading designs utilized 3H:1V to 5H:1V slopes to balance the cut and fill areas, address stormwater runoff, and reduce erosion.

Electrical power for the CK Gold Project will be supplied by a local utility company, Black Hills Energy (BHE), under an Industrial Contract Service Agreement. The power demand for the Project requires that a new 115 kV power line be constructed for the Project by BHE. The power line would be constructed from BHE's West Cheyenne substation, located approximately 16 miles east of the Project, to a new BHE owned, built, and operated 115 kV distribution substation (including transformer) near the mine. The estimated construction costs for the proposed power line, easement cost, and substation can be amortized in addition to the base power unit rate charged.

The Project will operate in a net water deficit situation, given that the mean annual evapotranspiration exceeds the mean annual precipitation. The total average Project water consumption will be 562 gallons per minute (gpm). Water to meet processing, mining, and potable water demand has been identified, and potential well sites have been investigated. A contract to supply water with the Board of Public Utilities (BOPU) in Cheyenne, Wyoming, has been executed, outlining water sourced from the Lone Tree Creek well field south of the site. However, under an agreement with the Ferguson Ranch, the surrounding landowner, a water exploration program has successfully identified a nearby source in the Red Canyon approximately 1-mile north of the project. The Red Canyon water will be less costly to develop and less costly to purchase under the agreement with the Ferguson Ranch, and adjustments to the identified "source and use" specified in the two main project permits (ISC and MOP) will be made to reflect the Red Canyon water supply once final development has been completed. Regardless of the source, water purchased will be used to make up the water deficit. Local consultants conducted preliminary engineering to confirm the feasibility and costs associated with the Red Canyon supply. Following studies by TGI, water generated from pit dewatering, surface runoff, and waste rock and tailings seepage will be recycled for use in mineral processing and/or dust suppression, reducing the volume of make-up water.

1.9 ENVIRONMENTAL, PERMITTING, AND COMMUNITY IMPACT

Environmental studies began in October 2020 to establish the pre-mining site conditions and fulfill the requirements for permitting. The environmental study reports, including baseline, groundwater modeling, seepage modeling, and geochemical characterization, have been submitted to the State as part of the permitting process. Applications for the principal state have been granted the Industrial Siting Permit (ISP0, May 2023, and the Mine Operating Permit (MOP) in April 2024. The MOP was conditional on a water discharge permit (WYPDES), furnishing a reclamation bond, and an Air Quality Permit (AQP), and these conditions were met in May, June, and November, respectively. The Project will occupy state-owned and private land. Permitting is primarily at the state and local level; no major federal permits are required.

Mining projects in Wyoming that are not located on Federal Land fall under the jurisdiction of the Wyoming Department of Environmental Quality, Land Quality Division (DEQ-LQD), which issues the Mine Operating Permit (MOP). This is an operating permit, but it is needed to start construction. The Project initially applied for the MOP in September 2022. The Project application went through two rounds of comments. The MOP was granted to the Project in April 2024.

The DEQ-LQD has permitted the Project's exploration activities to date. The Project has posted bonds to guarantee the reclamation of surface disturbance caused by the development of exploration drill pads, test pits, and some roads. All such surface disturbance has been reclaimed, including revegetation. Bond release is currently pending based on the re-establishment of revegetated areas.

In February 2021 the US Army Corps of Engineers (USACE) issued an Approved Jurisdictional Determination, under which two surface water bodies and associated wetlands in the Project area are considered Waters of the United States and subject to USACE jurisdiction and permitting for discharging of dredged or fill materials. There are no plans for project discharges or dredge or fill material deposition in these surface waters. Therefore, no further USACE permitting was anticipated. The USACE provided the Project with a no permit required letter in April 2024.

The Project required an Air Quality Permit to Construct and Operate issued by the DEQ's Air Quality Division (DEQ-AQD). This permit was approved in November 2024 with a New Source Review, including the development of the Project's air emission inventory. Electrical power will be supplied from a local utility rather than on-site generators (an on-site standby generator will be used in case of power interruptions). It is expected that the Project will be classified as a Minor Source. Title V of the Clean Air Act is not expected to apply. The permit application was submitted and underwent agency review and a public comment period before the final agency review. The air quality permit was granted in November 2024.

The Project also required an Industrial Siting Construction Permit issued by the DEQ's Industrial Siting Division (ISD). This permit is required for projects exceeding $253.8 million in construction costs. The application, including a socioeconomic and environmental impact assessment, was submitted in February 2023, following public notifications to affected local government agencies and two public informational meetings in Laramie County and the adjacent Albany County. DEQ-ISD granted the Industrial Siting Construction Permit to the Project in June 2023.

The State Engineer's Office (SEO) issues permits to appropriate water for beneficial use, as well as permits to construct and operate water related infrastructure such as wells, mine dewatering systems, and reservoirs, including stormwater or sediment control structures. SEO permits to construct and abstract water from the Project's surface water diversion channels and detention ponds were received in 2022 and 2023. Applications for permits to abstract groundwater flowing into the mine pit and to install a proposed on-site potable water well were also approved in 2023.

The DEQ Water Quality Division, State Fire Marshall, and Laramie County will require several other permits. Additionally, the US Environmental Protection Agency has jurisdiction over public water supply systems in Wyoming and requires a permit to supply potable water from the proposed on-site well. These permits will entail significantly less time and effort than the principal state permits granted.

In addition to government agencies' permitting requirements, the project's development will require certain agreements with private local entities. Agreements with Ferguson Ranch were negotiated for surface use rights, easements, and temporary rights to on-site water sources. Planning for a power supply agreement is also ongoing with Black Hills Energy. Beyond the extensive outreach during the ISP, U.S. Gold has and continues to reach out to and provide project information to various additional local public and private entities that may be affected by and/or interested in the Project. Procurement of goods and services and hiring of personnel are governed by the Project's policy of prioritizing local and State of Wyoming sources.

Environmental requirements and associated planning focus on avoiding or mitigating environmental impacts throughout the project life cycle. Waste rock and tailings generated during mining and mineral processing will be deposited in engineered facilities on the project site. Geochemical testing of mine rock and tailings using industry standard methods on representative samples indicates a limited probability of producing acid rock drainage (ARD) and/or metal release to water. Static geochemical testing on tailings samples produced by locked cycle laboratory testing indicates that the tailings are not acid generating. Static geochemical testing of waste rock samples indicates only a small percentage of waste rock is potentially acid generating (PAG). Confirmatory kinetic and leach test results show no or low production of acidic water or metal release for all tested samples.

The tailings will be filtered to extract as much moisture as feasible prior to their deposition, maximizing their structural strength and geotechnical stability, thereby avoiding the need for a tailings dam and the associated stability and seepage risks. Filtered tailings also maximize the amount of water that can be recycled to mineral processing, reducing make-up water requirements and minimizing overall water consumption. The tailings will be co-deposited in a Tailings Management Facility (TMF) with waste rock to provide structural buttresses and a retention shell for stability. Slope stability analyses of the TMF under static, pseudo-static, and post-peak loading conditions, including liquefaction assessment, were performed to verify that acceptable safety factors were obtained.

Runoff and seepage from the TMF will be collected in detention ponds at the downstream toe. A liner will limit seepage to the subsurface. A seepage collection drain installed above the liner will maintain a low hydraulic head in the bottom of the tailings mass and promote free drainage of the tailings, minimizing tailings saturation. The seepage collection drain will discharge to the detention pond downstream of the TMF.

To minimize fugitive dust emissions from the TMF, the top of the tailings surfaces will be compacted as quickly as feasible following tailings deposition, spreading the tailings by dozers using a smooth roller compactor to seal the surface. Once the final tailings slope and elevation have been achieved, the waste rock retention shells will be placed over the exposed tailings slopes. Speed limits will be imposed and enforced for mobile equipment operating on and around the TMF. Water will be sprayed on active surfaces to control fugitive dust emissions as required.

Waste rock will be used for construction of haul roads, erosion control features, and buttresses forming the outer shell of the TMF. Surplus waste rock will go into the West and East Waste Rock Facilities. These facilities are designed to have a slope angle of 3H:1V, which is substantially flatter than the rock's angle of repose, inherently providing an acceptable safety factor for geotechnical stability. Runoff and seepage will be collected in sedimentation ponds constructed at the downstream toe of the waste rock facilities. While kinetic testing on waste rock resulted in no ARD/metal leaching, the Project proposes segregating and isolating PAG waste rock, as determined by NAG pH testing, representing less than 11% of the total waste rock to be excavated and handled. PAG waste rock is proposed to be deposited in the interior of the lined TMF, as space allows, and, if needed, in the open pit after Year 8.

Extensive hydrogeologic site characterization has been completed to support the development of a regional groundwater flow model. The model simulates pre-mining conditions and hydrologic changes during mining and post-mining. Predicted mine-induced groundwater drawdown decreases rapidly away from the pit. The 5-ft drawdown will generally remain within the Project site boundary. The nearest domestic wells are 2,000 ft from the predicted 10 ft drawdown area and are not expected to experience discernable effects. Likewise, the effects on surface water flow in nearby streams will be negligible. The average annual groundwater pit inflow is expected to be less than 15 gpm, which will be captured using passive, in-pit sumps. After mining, groundwater and precipitation flowing into the backfilled pit will cause a gradual rebound of the groundwater level. A pit lake is not expected to form since evaporation losses will keep the groundwater level below the top of the backfill. This will result in the pit being a hydraulic sink with no groundwater outflows.

The Project site will be in a net water deficit situation, given that the mean annual evapotranspiration exceeds the mean annual precipitation. To minimize the overall demand for water from external sources, the Project will implement the following water conservation measures:

- Tailings filtration maximizes the amount of water recycled back into the flotation process, thereby avoiding the need for a tailings dam where much of the water would be lost to seepage and evaporation.
- Pit inflow collection in a sump to use for dust control in the pit.
- Surface runoff and seepage collection from waste rock facilities, TMF, and other facilities to use for dust control on site.
- Conversion of an existing on-site irrigation ditch providing water during the spring season.
- On-site potable water supply well.
- Truck wash water recovery and reuse for dust control.
- Recycling of water used for in-pit and primary crusher dust control.

The Project submitted a Reclamation Plan as part of the MOP application. The closure objective is to reclaim the site to enable the resumption of its current use of cattle grazing and mule deer winter range. A reclamation cost estimate has been developed for the reclamation bonding process. Concurrent reclamation will be practiced during the life of the mine to reclaim portions of the project site as soon as feasible before the end of mining, securing corresponding early releases in bonding obligations. At the end of operations, the process plant and supporting facilities will generally be demolished, and their footprints will be regraded. The disturbed areas, including the waste rock facilities and TMF, will be covered in topsoil and revegetated. Micro-topographical undulations and rock outcroppings will be created in the TMF slope for wildlife habitat and to promote revegetation. After the pit is fully excavated, it will be backfilled with tailings produced during the last two years of post-mining mineral processing. With a combination of blasting and earthmoving, the pit rim will be dozed into the pit to create a 3H:1V final pit wall slope covering the tailings. To help increase the local area's long-term water storage capacity, discussions have begun with BOPU about the possibility of converting the post-mining open pit into a water storage reservoir.

1.10 CAPITAL COSTS, OPERATING COSTS, AND FINANCIAL ANALYSIS

An after-tax, discounted cash flow model was developed to assess the economic performance of the CK Gold Project. This analysis relies on this report's mining schedule, capital and operating cost estimates, and recovery parameters. The model assumes 100% equity funding, a 5% discount rate, a gold price of $2,100/oz, copper price of $4.10/lb. and silver price of $27/oz. The results of the analysis are shown in Table 1.4 and Table 1.5. The positive economic outcome of the financial analysis is used to delineate the CK Gold Mineral Reserve.

Table 1.4: Economic Model Results

Key Project Indicators	Value US$M
Pre Tax Economics	
IRR	36.0%
Cash Flow (Undiscounted)	$693.2
NPV 5% Discount Rate	$459.2
Payback (years)	1.7
After Tax Results	
IRR	29.5%
Cash Flow (Undiscounted)	$556.9
NPV 5% Discount Rate	$355.9

Table 1.5: Project Details	
Key Project Indicator	**Value**
Gold Ounces Sold (000's)	663
Copper Sold (Million Lbs.)	196
AuEq Ounces Sold (000's)	1,069
Initial Capital ($ Million)	$272.8
Sustaining Capital ($ Million)	$16.6
Avg. Cash Cost of Production ($/oz AuEq)	$922.0
All in Sustaining Cost ($/oz AuEq)	$937.0

A sensitivity analysis on metals pricing indicates additional potential for this project at higher metals pricing, Table 1.6. Additionally, the sensitivity indicates the robustness of the project with positive economic outcomes at reduced metals pricing.

Table 1.6: Metal Price Sensitivity							
Metal Pricing		**Pre-Tax**			**Post-Tax**		
Gold Price	**Copper Price**	**NPV**	**IRR**	**Payback**	**NPV**	**IRR**	**Payback**
Au/oz	**Cu/lb**	**$M**	**%**	**Years**	**$M**	**%**	**Years**
$1,300	3.80	$35	8.1%	5.55	($13)	3.8%	6.98
$1,700	3.90	$240	23.0%	2.71	$177	18.4%	3.44
$2,100	4.10	$459	36.0%	1.73	$356	29.5%	2.12
$2,500	4.30	$678	47.6%	1.37	$532	39.4%	1.63
$3,000	4.50	$945	60.4%	1.10	$745	50.3%	1.31

1.11 CONCLUSIONS AND RECOMMENDATIONS

1.11.1 General Recommendations

U S Gold's CK Gold Project focuses on the historical Copper King deposit in the Silver Crown Mining district, which has been the subject of sporadic mining activity for over 100 years. The CK Gold Project demonstrates a very low waste to ore ratio, the absence of a large pre-strip period to expose mineralization, simple low-cost mineral extraction, and proximity to key infrastructure and support services, which all favor positive project economics.

With a life of mine cash cost per equivalent gold ounce of $922/oz, the margin compared to both the study price, set at $2,100 per gold ounce, and the gold price at the time of writing of approximately $2,885 per gold ounce, indicates robust project economics. The fact that the bulk of the revenue is split between sales of gold and copper suggests that the project may be less sensitive to cyclical swings in the prices of either individual metal. A unique feature of the CK Gold Project is its proximity to growing population centers and infrastructure, which may further offer opportunities to bolster revenue through the sale of waste rock as aggregate. Investigations have proven the non-mineralized rock to be of very good quality for aggregate products. The aggregate potential has not been included in this study; however, the Burgex Study, August 2024, investigates the regional market conditions and summarizes the testwork concluded to date showing the non-metal bearing rock as an excellent source of aggregate and rail ballast. The study concludes that up to 1 million tons could be sold into the local market utilizing truck haulage at a significant margin. Greater tonnage could be sold if a rail haulage scenario could be arranged. More than 40 million tons of rock will be set aside and reclaimed under the current plan; however, putting this rock to beneficial use can benefit the CK project, provide additional royalty payments to the State, and reduce the project footprint and closure costs. The potential of creating a separate business unit to benefit from the rock mined will be evaluated in follow-up studies.

U.S. Gold elected to focus on data capture to support a Pre-feasibility study and permit application with its 2020 and 2021 field season activities. The resource model shows that there are potential extensions to the mineralization at depth and to the southeast of the deposit and these should be investigated. Additionally, there is uncertainty as to the genesis of the mineralization, with the deposit not neatly fitting a porphyry or Iron oxide copper gold (IOCG) type depositional model. The company is set to support study work with the University of Wyoming, and we recommend that efforts continue to better understand the geological setting and assess district potential.

In reviewing the Project, we conclude that the type of mining, rate of mining, and mineral processing technology selected in the PFS study is appropriate. While evidence suggests that improved gold recoveries can be readily obtained through the implementation of flotation, followed by cyanidation of the flotation tailings, other factors and considerations make the application of such technology difficult to assess. Not least of these considerations is the public perception of the use of cyanide gold recovery. With the potential to recover an additional 180,000 gold ounces with the addition of a cyanide circuit, we recommend that trade-off studies be conducted but tend to agree with U.S. Gold management that further studies and permitting be advanced without the inclusion of a cyanide circuit, under current price assumptions.

1.11.2 Specific Work Plan

The goal of the Pre-feasibility Study is to provide information to the directors of U.S. Gold so that they can make an informed decision about the future development of the Project. To advance the CK Gold Project, it is recommended that a feasibility study be commenced to advance project engineering and planning. The estimated budget to complete this is $3MM to develop the appropriate level of detail and proceed with the recommendations provided in this report.

To further assess the project's viability and feed into a more accurate and comprehensive assessment, plans for an EPCM strategy, construction, and operations to support project development should be created. The development of a detailed owner's team for development, contracting strategy and transitional plan to operations should be identified.

1.11.2.1 Mine Planning

At the initial stage of this study, 20 and 30-foot benches were used as the basis for the mine bench design. Since the production rate of plant throughput has since increased to 20,000 stpd and larger mining equipment would be used to accommodate the higher plant throughput, it is recommended to re-design the mine using possibly 35 ft or 40 ft bench heights. This would reduce unit costs for drilling, blasting, loading, and hauling.

1.11.2.2 Future Metallurgical Test Work

The geometallurgical models prepared for the Pre-Feasibility Study highlight recovery relationships with head grade and oxidation level. Additional variability testing, together with larger scale work on lower grade samples, would be useful.

Specialty test work with the vendors of tailings and concentrate filtration, including the oxide and sulfide components. Specific test work on the mixed ore zone.

Additional vendor specific test work on flotation and regrinding is needed to quantify possible recovery and/or grade improvements utilizing different flotation and grinding technologies.

1.11.2.3 Mineral Processing

- Conversations with equipment vendors indicate that additional investigation into the utilization of alternative flotation and/or regrind technologies could reduce the plant footprint, which would result in structural cost savings. The alternative technologies have the potential to reduce operating costs due to a reduction in electrical power usage. Realization of these savings would be contingent on positive results from the metallurgical test work.

1.11.2.4 Design and Engineering

- Additional discrete studies should be pursued to develop possible capital and operating improvements. A comprehensive list is in Section 23. Priority items would be coarse ore flotation tests for capital and process improvements, Run of Mine ore testing for comminution improvements, additional engineering to improve quantity takeoffs, and layout optimization. Also, alternative building types and water management could decrease costs, and earthwork balancing could be improved by layout optimizations.

1.11.2.5 Environmental, Permitting, and Social

- Continue activities needed to obtain the required state and local permits.
- Continue disclosing project information and consulting with local stakeholders, especially focusing on project impact assessment, local project benefits, and impact mitigation measures.
- Concluded a wildlife mitigation agreement with the Wyoming Game and Fish Department.
- Concluded the needed land use agreements with the Office of State Lands and Investments and the affected private landowner.
- Identify and secure a potential alternative backup water supply source.
- Continue engagement with the City of Cheyenne regarding the potential post-mining conversion of the pit to a water storage reservoir serving the city.
- Develop and implement a Project Environmental Management System (EMS) consisting of site-specific plans and procedures governing the environmental management of project activities causing potential environmental impacts during construction, operations, closure, and post-closure.

2.0 INTRODUCTION

2.1 TERMS OF REFERENCE AND PURPOSE

Samuel Engineering, Inc. (Samuel) was commissioned by U.S. Gold Corp (U.S. Gold) to prepare an updated Pre-Feasibility Study (PFS) for the CK Gold Project ("CK Gold Project" or the "Project"). This report is a Technical Report Summary (TRS), which summarizes the findings of the PFS following the Securities Exchange Commission Part 229 Standard Instructions for Filing Forms Regulation S-K subpart 1300 (S-K 1300). This TRS aims to report mineral resources, mineral reserves, and economics for the CK Gold Project. The effective date of this report is February 10th, 2025.

The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in Samuel's services based on the following:

- Information available at the time of preparation.
- Data supplied by the client.
- The assumptions, conditions, and qualifications outlined in this report.

Any opinions, analyses, evaluations, or recommendations issued by Samuel under this report are for the sole use and benefit of U.S. Gold. Because there are no intended third-party beneficiaries, Samuel (and its affiliates) shall have no liability to any third parties for any defect, deficiency, error, or omission in any statement contained in or in any way related to its deliverables provided under this Report.

The regional geologic setting of the CK deposit within the Cheyenne suture belt is significant, as is the nature of occurrence of sulfide mineralization as disseminations in undeformed granodiorite and alignment with foliation in foliated to mylonitized granodiorite. Based on the available data and information to date, we suggest that Klein's (1974) description of the CK deposit as a "structurally controlled base and precious metal deposit hosted in a Precambrian shear zone" is essentially correct if you want further refinement. While Klein's description does not present a conventional deposit model, it does provide a reasonable interpretation on which to base plans for future exploration. Future drilling exploration (and petrographic and/or mineralogical analysis) should be carefully planned to test Klein's interpretation and target data useful in further developing an appropriate deposit model for the CK Project, whether conventional or not.

2.2 SOURCES OF INFORMATION

The information, opinions, conclusions, and estimates presented in this report are based on the following:

- Information and technical data provided by U.S. Gold.
- Review and assessment of previous investigations.
- Assumptions, conditions, and qualifications as outlined in the report.
- Review and assessment of data, reports, and conclusions from other consulting organizations and previous property owners.

These sources of information are presented throughout this report and in the References section. The Qualified Persons are unaware of any material technical data other than that presented by U.S. Gold.

2.3 DETAILS OF INSPECTION

Below is a list of the qualified persons involved in preparing this TRS and details of their property inspection.

Antonio Loschiavo, P.Eng visited the CK Gold Project property and site facilities, which include core logging facilities, pit and waste dump areas, tailings facility, and plant location, on multiple occasions between 2022 and 2024. The last site visit was the week of June 3rd, 2024, touring the Komatsu Equipment suppliers (Power Motive).

Mark Shutty, CPG, visited the CK Project site and US Gold's logging and sample storage facilities in Cheyenne on July 26–27, 2021, and again on July 11, 2024. Mr. Shutty has reviewed the drillhole datasets and geological information supporting the Mineral Resource Estimate.

John Wells visited the core storage and metallurgical labs on multiple occasions listed below throughout validating the metallurgy:

- 2021-CORE SHED AND SELECTION OF SAMPLES.
- 2021-Kappes Cassidy Lab-RENO.
- 2022 and 2024, Base Metals Lab-KAMLOOPS, CANADA.

Samuel Engineering had Eric Brunk and Peter Clarke visit the site to assess topography and constructability, understand the site access, and confirm proximity to existing infrastructure.

Tierra Group visited the property on 19 April 2022 to assess general site topography, visible geology, and other site conditions.

Kevin Francis, Vice President of Exploration and Technical Services with the registrant has management responsibility over the CK Gold project and visits the site, logging and storage facilities frequently. The last visit was January 16, 2025 when Mr. Francis toured the project site.

Responsible Company	QP Individual(s)	Responsible Section
AKF Mining	Antonio (Tony) Loschiavo, P.Eng., President, AKF Mining & Mineral Services Inc.	12, 13, 15.2, 15.3.4, 17.2.1.1
Drift Geo	Mark C. Shutty, CPG, Drift Geo	9 & 11
John Wells	John Wells	10
Samuel Engineering, Inc.	Cameron Wolf, Steve Pozder, Matt Boling, Richard Morris, Jim Sorensen	1, 2, 14, 15.4, 15.3, 15.5.1, 18, 19, 21, 22, 23, 24, 25
Tierra Group International, Ltd. (TGI)	Various	15.1.2, 15.2.1, 15.3.2, 15.3.3, 17.1.3, 17.2.1.2, 17.2.3.3, 17.2.3.2

U.S. Gold Corp (Registrant)	Kevin Francis, SME-RM, VP, U.S. Gold Corp.	3, 4, 5, 6, 7, 8, 15.1.1, 15.3.1, 15.5.2, 16, 17.1, 17.1.1, 17.1.2, 17.1.3, 17.1.4, 17.2, 17.2.1, 17.2.2, 17.2.3,17.2.3.1, 17.2.3.4, 17.2.3.5, 17.3, 17.4, 17.5, 17.6, 17.7, 20

2.4 PREVIOUS REPORTS ON THE PROJECT

The first TRS U.S. Gold submitted for the CK Gold Project was the Gustavson Associates report, "S-K 1300 Technical Report Summary CK Gold Project", dated Dec. 1, 2021. The authors are unaware of any other TRS submitted by prior owners of the Project. However, U.S. Gold did publish a Technical Report and Preliminary Economic Assessment for the CK Gold Project in December 2017. This previous Technical Report did disclose a mineral resource for the Project under the reporting requirements of the Canadian Securities Administrators National Instrument 43-101 (NI-43-101). The CK Gold Project was formerly referred to as the Copper King Project.

2.5 LIST OF ABBREVIATIONS AND UNITS

All measurement units are in the US Customary System (USCS), and currency is expressed in US dollars unless otherwise noted. Certain specific abbreviations not listed here will be defined within the report text. The following abbreviations may be used in this report:

Abbreviation	Unit or Term
A	ampere
AA	atomic absorption
AACE	Association for Advancement of Cost Engineering
ABA	acid-base accounting
Ac	Acre
Ai	Abrasion index
AJD	Approved Jurisdictional Determination
AML	Abandoned Mine Lands Division
amsl	above mean sea level
ARD	Acid Rock Drainage
Ag	silver
Au	gold
AuEq	gold equivalent grade
bcy	bank cubic yards
BLM	US Bureau of Land Management
BOPU	Cheyenne Board of Public Utilities
C$	Canadian dollar
°C	Centigrade degrees
cfm	cubic feet per minute
cfs	cubic feet per second
CLS	composite liner system

Abbreviation	Unit or Term
cm/s	centimeters per second
COG	cut-off grade
CRM	certified reference material
Cu	copper
CV	coefficient of variance
CWA	Clean Water Act
°	degree (degrees)
D	day
DEQ	Wyoming Department of Environmental Quality
EWRF	East Waste Rock Facility
EMS	environmental management system
°F	Fahrenheit degrees
FEL	front end loader
FA	fire assay
fasl	feet above sea level
FIBCs	Flexible Intermediate Bulk Containers, e.g., bulk bags
FS	Feasibility Study
FSR	freight, smelting, and refining
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
Ga	billions of years
g	gram
gal	gallon
GD	granodiorite
GDK	potassic-altered granodiorite
G&A	general and administrative
gpm	gallons per minute
gpt	grams per tonne (metric)
h	hour
HCT	humidity cell testing
HDPE	high-density polyethylene
hp	horsepower
HTW	horizontal true width
ICP-MS	inductively coupled plasma mass spectrometry
ID^2	inverse-distance squared
ID^3	inverse-distance cubed
IRR	internal rate of return
k	thousands
kg	Kilogram (metric)

Abbreviation	Unit or Term
koz	thousand troy ounces
kst	thousand short tons
kstpd	thousand short tons per day
kstpy	thousand short tons per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
lb	pound
LG	Lerch-Grossman
LME	London Metal Exchange
LCT	locked cycle test
LOM	Life-of-Mine
LQD	Land Quality Division
M	million
Ma	million years
MD	mafic dikes
mi	mile
min	minute
Moz	million troy ounces
ms	millisecond
MSED	metasediment unit
Mst	million short tons
Mstpy	million short tons per year
MW	megawatt
MWMP	Meteoric Water Mobility Procedure
MYL	Mylonite
NAG	net acid drainage
NEPA	National Environmental Policy Act
NI-43-101	Canadian Securities Administrators National Instrument 43-101
NPAG	not potentially acid generating
NPV	net present value
NRHP	National Register of Historic Places
NRCS	Natural Resource Conservation Service
oz	troy ounce
opt	troy ounce per short ton
OSLI	Office of State Lands and Investments
P_{80}	80-percent passing size
PEG	pegmatites
PFS	Prefeasibility Study or Pre-Feasibility Study
pH	negative log of hydrogen ion concentration

Abbreviation	Unit or Term
ppb	parts per billion
ppm	parts per million
QA/QC	quality assurance/quality control
QC	quaternary cover
RC	rotary circulation drilling
ROM	run of mine
ROW	right of way
RQD	Rock Quality Designation
SAG	semi-autogenous grinding
SD	standard deviation
sec	second
SG	specific gravity
SHPO	Wyoming State Historic Preservation Office
SPMDD	Standard Proctor Maximum Dry Density
sq mi	square mile
st	short ton (2,000 pounds)
stph	short tons per hour
stpd	short tons per day
stpy	short tons per year
TRS	Technical Report Summary
TMF	Tailings Management Facility
µm	micron or microns
USACE	United States Army Corps of Engineers
US$	U.S. dollar
USFWS	US Fish and Wildlife Service
USGS	United States Geological Survey
V	volts
W	watts
WB	Water Balance
WGFD	Wyoming Game and Fish Department
WQD	Water Quality Division
WWRF	West Waste Rock Facility
y	year
yd^2	square yard
yd^3	cubic yard

3.0 PROPERTY DESCRIPTION

3.1 PROPERTY LOCATION

The CK Gold Project is in Laramie County, Wyoming, in the southeastern portion of the state, approximately 20 miles west of Cheyenne. It is centered in the north half of Section 36, T14N, R70W. The property footprint is approximately 1119 acres (453 hectares), subject to surface disturbance. It includes the S ½ of Section 25, the NE ¼ of Section 35, all of Section 36, and north 2/3 of Section 31. A regional and local map is shown in Figure 3.1.

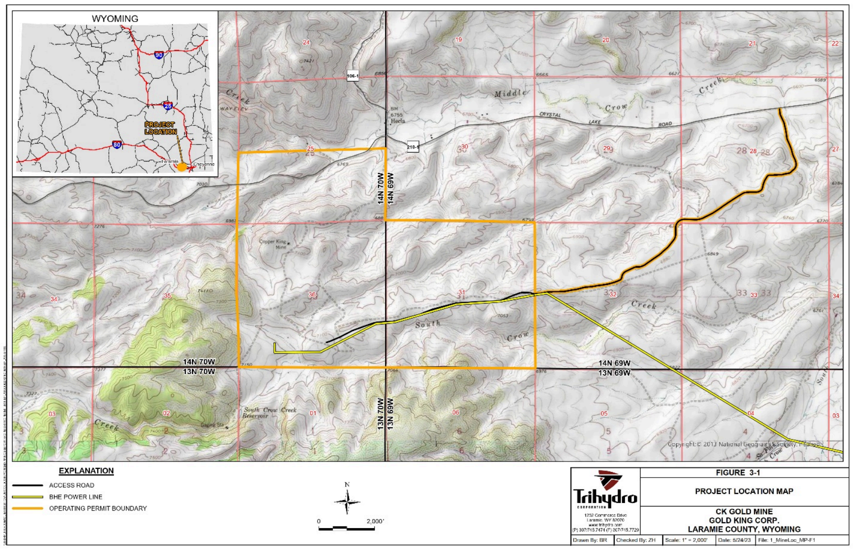


Figure 3.1: Regional and Local Map

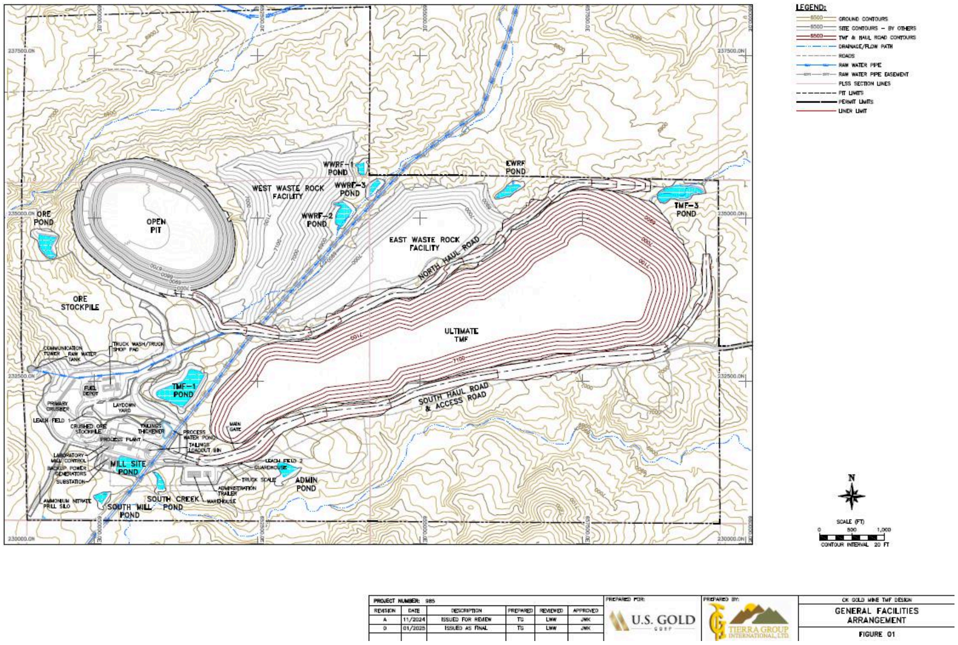


Figure 3.2: Project Map

3.2 MINERAL TITLES, CLAIMS, RIGHTS, LEASES, AND OPTIONS

The CK Gold property consists of two State of Wyoming Metallic and Non-metallic Rocks and Minerals Mining Leases listed below. Both mineral leases listed can be renewed for successive 10-year terms if certain conditions are met.

Lease #0-40828 for 640 acres (259 hectares), which includes all of Section 36, T14N, R70W, is a 10-year renewable lease that expires February 1, 2033. The current annual rental is $2.00 per acre, $1,280 total.

Lease #0-40858 for 320 acres (130 hectares), which includes S½ Section 25 T14N, R70W and 160 acres within NE¼ Section 35, T14N, R70W. The current annual rental is $2.00 per acre, $1,280 total. The lease is a 10-year renewable lease that expires February 1, 2034.

Surface Lease Option Agreement Section 31 and Section 25. In August 2021, an option agreement to lease surface rights for project development was executed, contemplating the use of a portion of 712 acres (288 hectares) for project development activities.

The surface of S½ Section 25 and NE¼ Section 35 is privately owned. An easement agreement providing access has been negotiated with Ferguson Ranch Inc. on the S½ Section 25, T14N, R70W, as well as the W½ Section 31, T14N, R69W. The original access easement was first signed in November 2006 but replaced and superseded by one effective May 1, 2009; the agreement is for one year and is renewable annually. Annual payments on the easement agreement are $5,000 for the first year and $10,000 for the next four years if the agreement is renewed. U.S. Gold reports that the agreement has been renewed for the current year. Additionally, a new temporary easement preferred by the landowner was established and celebrated in 2021. This new easement follows the same path as the proposed project access and is subject to the Option Agreement on the land lease and Right-of-Way (ROW).

The surface of Section 36 is owned by the State of Wyoming and is leased for agricultural use to Ferguson Ranch Inc. As part of the terms for its surface-use lease option agreement with Ferguson Ranch Inc., U.S. Gold has an arrangement to compensate the Ferguson Ranch for the loss of grazing. Prior to mining development, upon the celebration of the Option Agreement and exercising the Lease for the land, annual payments identified in the Option Agreement would be split between the State of Wyoming and the surface lessee based on a sliding scale (per current agreement based on a formula provided by the Wyoming Office of State Lands and Investments).

Various private owners own the surface of Sections 25 and 35. While the open pit expands onto a small portion of the southern part of Section 25, there is no planned activity on Section 35 besides the placement of a freshwater header tank and communications equipment. U.S. Gold owns 28 hectares (70.2 acres) immediately west of Section 36 in the NE ¼ Section of Section 35 and the water tank and communications equipment will be placed on U.S. Gold property. There will be a minor amendment in the project description associated with the current permit to incorporate this land into the project area. Otherwise, the land on Section 35 will serve as a buffer between the mine and other residents in the area.

OTHER PROPERTIES

In 2021 and 2022, the Company acquired two parcels of land immediately west of and adjacent to Section 36 T14N 70W on Section 35. The two parcels, totaling approximately 70.2 acres, lie outside of Cheyenne city limits, and property tax payments are current. The Company owns the surface rights and leases the mineral rights from the state of Wyoming. The Company believes that these parcels may be used for later project development other than described above and are presently viewed as an investment.

3.3 ENVIRONMENTAL IMPACTS, PERMITTING, OTHER SIGNIFICANT FACTORS, AND RISKS

Since 2017, U.S. Gold has conducted a field exploration program for drilling, soils, geotechnical, and hydrological investigations. This program is fully permitted, and the CK Gold Project currently holds a DEQ-issued Exploration Permit # DN0440, TFN 7 3/064, which includes cumulative bonding presently totaling $155,000. In addition, an exemption of Stipulation 5 of U.S. Gold's mineral lease 0-40828 has been obtained from the Wyoming Game and Fish Department, addressing mineral lease terms that exclude activity in sensitive big game habitats between November 15th and the end of April each year. Negotiations with Wyoming Game and Fish have been held to outline measures that can be taken if the project proceeds to contribute to the enhancement of wildlife habitat. Discussions identified that mitigation measures are reasonable to accomplish, such as programs to install wildlife-friendly fencing, invasive species (e.g., cheatgrass) mitigation, and land swaps. Currently, U.S. Gold is contemplating a $300,000 mitigation effort agreed to, in coordination with Wyoming Game and Fish, along with recognition that measures such as "game friendly" fence installation will be adopted during project development.

The current surface disturbance from exploration activities, including roads and test sites, is 40 acres. Costs associated with the reclamation of the exploration disturbance are bonded through cash payments to the state and recoverable upon inspection and release by the DEQ.

3.4 ROYALTIES AND AGREEMENTS

The CK Gold Project is subject to a production royalty of 2.1%, payable to the Office of State Lands and Investments (OSLI) for the State to fund appropriate education trust accounts. The royalty payment in the original lease package was negotiated, and the 2.1% royalty supersedes the prior royalty provision as per the 2023 amendment. The royalty is calculated based on the gross sales value of the product sold, less applicable deductions for costs incurred for processing, transportation, and related costs beyond the point of extraction from the open pit mining operation. Once the Project is in operation, the Board of Land Commissioners has the authority to reduce the royalty payable to the State. Before commercial production, a royalty of $2.00 per acre is payable to the OSLI. In addition to the permitting requirements and associated interaction with the DEQ and other state and local agencies, the development of the CK Gold Project will require exercising certain agreements with other local entities, including (1) Ferguson Ranch for land use rights and easements for access road, power line and water supply well(s) and pipeline; (2) Negotiating a water pipeline route across private property, (3) an agreement for a power line easement; and (4) a power supply agreement with Black Hills Energy, a subsidiary of Black Hills Corporation.

4.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 TOPOGRAPHY, ELEVATION, AND VEGETATION

The CK Gold Project is located on the eastern flank of the Laramie Range between the Rocky Mountains and High Plains sections of the Great Plains physiographic province. The Laramie Range is an approximately 130-mile-long mountain range between Laramie and Cheyenne, WY, that trends north from the Colorado-Wyoming border towards Casper, WY. The Laramie Range consists of granite/granodiorite peaks and rolling hills bound to the east non-conformably by shallow eastward dipping sedimentary rocks of the White River Formation. East of the Project area, towards Cheyenne, WY, the topography transitions to flatter plains along the western margin of the Great Plains physiographic province.

The gradually sloping sedimentary deposits on the flank of the Laramie range created what was referred to as a land bridge, allowing the main east-west rail line to pass the area, avoiding difficult mountainous terrain. Elevations within the Laramie Range in the vicinity of the property reach over 8,000 ft above mean sea level (amsl), while the city of Cheyenne, located on the western edge of the Great Plains Province, is at an elevation of 6,100 ft amsl. The Project property has elevations ranging from 6,625 ft to 7,311 ft amsl with generally low to moderate relief. The exception is the northwest portion of the property, which covers a moderate to steep, northwest-facing slope that bottoms at 6,900 ft elevation in a northeast-flowing intermittent stream drainage. The Project mineral resource area elevation ranges from 6,950 ft to 7,172 ft amsl. The currently identified mineral resource is exposed at the surface along a west-northwest trending ridge, and the topography is conducive to open-pit mining methods.

The Project area consists primarily of rolling grassland/herbaceous habitat with forested and shrub/scrub-covered drainages. Most of the project site consists of prairie grasslands, with some areas of xeric forest and sparse areas of foothills, sagebrush shrublands, and riparian vegetation.

4.2 ACCESSIBILITY AND TRANSPORTATION TO THE PROPERTY

The Project is approximately 20 miles west of Cheyenne and is accessible from the paved State Road 210 (a.k.a Happy Jack Road) to the County Road 210 (a.k.a. Crystal Lake Road), a maintained gravel road. The Project site access entryway is approximately two miles off the pavement to the west on County Road 210 and crosses Ferguson Ranch land, subject to a Right-of-Way Option Agreement. From the County Road 210 entryway to Section 31 in the Project site area, approximately four miles of single-track gravel road will be upgraded and maintained for the Project's life.

4.3 CLIMATE AND OPERATING SEASON

Based on data compiled from the CK Gold Project site weather station and other surrounding stations (the latter over at least ten years), the daily average temperature ranges from about 25° F in February to about 70° F in July. The average low temperature is -11° F in February, and the average high is 90° F in July.

The Project site is in a net water deficit condition. The average annual precipitation is about 17 inches, while the annual evaporation is about 53 inches, as determined by the on-site meteorological station. May is the wettest month, with an average of about 3 inches; January is the driest, with an average of about 0.6 inches. Snowfall typically occurs from September to May.

The site experiences relatively strong winds, with an average monthly wind speed ranging from about 8 mph in July to about 17 mph in December. For those same months, the average maximum wind speeds are 43 and 63 mph, respectively, with peak wind speeds of 55 and 75 mph. The predominant wind direction is westerly.

The lease terms for Section 36 have been renegotiated to enable unrestricted full-time, year-round project construction, mining, and mineral processing activities.

4.4 LOCAL INFRASTRUCTURE AVAILABILITY AND SOURCES

Given the proximity to Cheyenne, the state capital of Wyoming, and the Front Range metropolitan area, personnel needs, delivery of consumables, and infrastructure needs are available locally and regionally. This should not pose a material negative impact to the Project; on the contrary, the infrastructure allows relatively easy access to major mine supply centers, the closest being Denver, Colorado, Salt Lake City, Utah, and Gillette, Wyoming. The area has access to Union Pacific and Burlington Northern Santa Fe (BNSF) railroad lines, the intersection of two major interstate highways, I-80 and I-25, and a regional airport.

Electrical power for the Project will be supplied by a local utility company, Black Hills Energy (BHE), under an Industrial Contract Service Agreement. The power demand for the Project requires that a new 115 kV power line be constructed for the Project by BHE. The power line would be constructed from BHE's West Cheyenne substation, located approximately 16 miles east of the Project, to a new BHE owned, built, and operated 115/13.8 kV distribution substation (including transformer) adjacent to the mine. The powerline alignment would take advantage of existing easements and planned county roads near the Project. The alignment would require easements from the City of Cheyenne, the State of Wyoming, and local ranches. BHE will acquire the easements, construct the power line for the project at their expense, and recoup the capital cost through demand charges added to the standard industrial mine power cost.

5.0 HISTORY

The CK Gold Project was originally known as the Copper King Mine. It was first discovered in 1881, along with the Climax and Potomac lodes, by James Adams. The deposit was developed, and a 160 ft (48 m) shaft was sunk, along with the construction of a mill and smelter by the Adams Copper Mining and Reduction Company. No production figures are available from this period; however, modest-sized waste dumps around the shaft indicate that the underground mining wasn't extensive. The Ferguson Ranch, which presently owns or leases most of the surface land in the CK Gold Project's area, was homesteaded in 1874 by the first native-born children of settlers to the area (Angus Journal, 1996).

The Copper King Mine was noted as idle by the State Geologist in 1890 when Wyoming attained statehood and assumed ownership of the associated section of land (Section 36). In 1911, C.E. Jamison, the State Geologist of Wyoming, mentioned several active copper and gold mines within the Silver Crown Mining District (SCMD) and near the CK Gold Project, including the Dan-Joe Prospect, Comstock Mine, Fairview Mine, Louise Mine, Little London Mine, Bull Domingo Prospect, and several additional unnamed prospects.

Mineral rights transferred several times over the next century, starting with the Otego Mining Company in 1907, followed by the Hecla Mining Company until about 1910. By 1910, production at the Copper King Mine had reached 316 st (287 t), producing 27 ounces (oz) of gold, 483 oz of silver, and 25,782 lbs. (11,700 kg) of copper. From 1890 to 1938, there were at least eight drilling campaigns totaling 37,500 ft. (11,430 m) of drilling. Excavation of numerous prospect pits and developing two adits also likely occurred during this time.

The American Smelting and Refining Company (ASARCO) acquired the property in 1938 and performed the first major drilling campaigns on the project site. It was subsequently acquired by the Copper King Mining Company in 1952. ASARCO re-optioned the property in 1970. Henrietta Mines Ltd gained rights to the property in 1972. At some point before 1987, Henrietta's interest was folded into Wyoming Gold, Inc., which William C. Kirkwood and Caledonia Resources Ltd., the parent company of Henrietta, jointly owned. Royal Gold, Inc. entered an option agreement to buy Wyoming Gold in 1989. Compass Minerals Ltd. then acquired the property in 1993. Saratoga bought it in 2006. Strathmore acquired the issued and outstanding shares of Saratoga in 2012, which were subsequently purchased by Energy Fuels. Energy Fuels then sold the property to U.S. Gold in 2016.

5.1 HISTORICAL EXPLORATION AND PRODUCTION

ASARCO completed five exploration holes for 1,400 ft (427m) in 1938, two of the holes yielding significant gold and copper mineralization. Copper King Mining then completed six more holes in 1952-54 for 2,630 ft (802 m) of drilling, partially subsidized by the U.S. Bureau of Mines. When ASARCO took control again in 1970, they conducted soil geochemical sampling, geologic mapping, IP and aeromagnetic surveys, and eight additional core holes totaling 3,263.1 ft (874 m).

Henrietta completed the first reserve and resource estimate in 1973 after they had completed an 11-hole drilling campaign for 3,766 ft (1,148 m) of drilling, a control survey, geologic mapping, IP and vertical-intensity magnetic geophysical surveys, geochemical soil sampling, re-logging of historical core holes, and preliminary metallurgical studies.

John Nelson of Kirkwood Oil and Gas did a second reserve estimate around 1986. It does not appear any additional drilling was done before this estimate; however, the company did collect 228 surface geochemical samples in 1982, and the Colorado School of Mines Research Institute had done some metallurgical work on the property in 1980.

Caledonia undertook a new drilling campaign in 1987 of 25 holes for 9,980 ft (3,042 m), designed to improve confidence and prove reserves within the known extent of the deposit. They also funded a three-sample preliminary metallurgical study that year. Results were used to create a preliminary resource estimate published in the Wyoming State Geological Survey Bulletin 70. Tenneco Minerals Company then produced a reserve estimate in 1988. In 1989, both FMC Gold Company and Royal Gold, Inc. funded metallurgical studies and produced reports that discussed small exploration campaigns, which were likely completed in that year but whose results were unavailable. The FMC study was completed by Kappes, Cassiday & Associates (KCA) and references some work done to collect and test mine dump samples in 1986 and 1987. It is believed that the Royal Gold report, completed by Hazen Research, Inc. in 1989, used the same metallurgical sampling composites in its study. It also includes two holes drilled for 505 ft (154 m) that year; however, this data is also lost.

Compass funded an aeromagnetic survey over the area and 25 new drill holes for 9,202 ft (2,805 m) in 1994. They also conducted two metallurgical studies in 1994 and 1996 by Metallurgy International and a preliminary resource study by Mine Development Associates (MDA).

Mountain Lake Resources then funded a ground magnetometer and VLF-EM geophysical survey, drilled eight holes for 4,740 ft (1,445 m), including two metallurgical test holes, and a metallurgical study by the Colorado Minerals Research Institute in 1998.

MDA completed a technical report in 2006. 27 holes for 18,296 ft (5,577 m) were drilled during the spring and summer of 2007, and MDA created an updated report to include these results through October 31, 2007. Saratoga completed another eight holes in 2008 for 7,167 ft (2,185 m).

Saratoga commissioned further work focused on flotation methods to extract gold and copper, as reported in 2009 by SGS, Canada Inc. In a report dated December 8th, 2010, a test program was conducted on oxide material from the Copper King deposit to determine a flotation flowsheet to maximize recoveries of Au and Cu. The oxide portion of the resource is minor; however, the work was completed to follow on from the successful results obtained on sulfide samples where a 26% copper concentrate was produced containing 98 grams per ton of gold. The oxide concentrate produced was reported as being expected to be marketable. However, further work was identified to support these conclusions.

Gustavson (WSP) completed a PFS in December 2021, including RC drilling by U.S. Gold of two holes in 2017 and eight holes in 2018, totaling 12,040 ft (3,670 m). Both programs were designed to investigate magnetic and IP anomalies generated by geophysical surveys. Also included was U.S. Gold drilling from 2020, comprising 25 drill holes totaling 20,449 ft. The PFS resulted in favorable economics, the first mineral reserve, and a recommendation to advance to a feasibility study.

6.0 GEOLOGICAL SETTING, MINERALIZATION, AND DEPOSIT

6.1 REGIONAL GEOLOGIC SETTING

The CK Gold Project area is located on the eastern flank of the southern Laramie Mountains, within the terrane of the Colorado Province and just south of a northwest-trending crustal suture zone known as the Cheyenne Belt (Figure 6.1). The Cheyenne Belt represents the margin along which the island-arc terrane of the Colorado Province (or Colorado orogen) accreted to the southern edge of the Wyoming Craton during the Paleoproterozoic. As a result of this collision, older Archean rocks of the Wyoming Province were intensely deformed and metamorphosed for at least 75 km inboard of the suture, which is marked today by the Laramie Mountains (Sims et al., 2001).

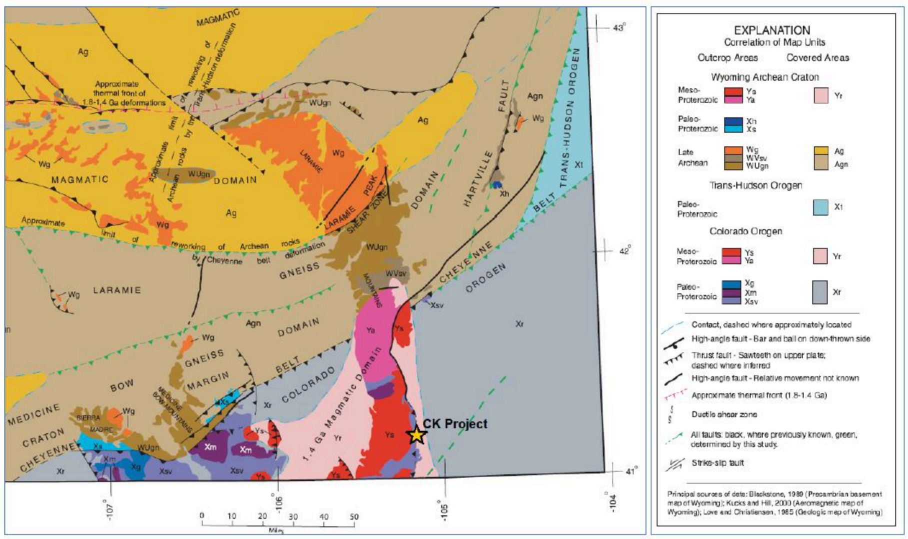


Figure 6.1: Regional Geologic Setting of the CK Project Area. Source: Sims et. Al (2001)

The Laramie Mountain Range is an asymmetrical Laramide uplift that exposes a core of Precambrian rocks that extends for approximately 140 miles from north to south. The mountain range is segmented by steeply dipping shear zones and regional-scale thrust faults. The northern portion of the range is comprised of terrane belonging to the Archean Wyoming Province, while rocks of the Proterozoic Colorado Province core the southern portion. Near the CK Gold Project area, the Laramie Mountains are bound to the east by an unconformity between overlying Mesozoic sedimentary rocks and underlying Proterozoic igneous and metamorphic rocks of the Colorado orogen. The Colorado orogen consists of metasedimentary-metavolcanic rocks and granitic-gabbroic rocks of island-arc affinity (Sims et al., 2001). In the Laramie Mountains, the metavolcanic and metasedimentary rocks are modified by batholithic intrusions of two discrete generations, ~1.7 and ~1.4 Ga (Tweto, 1987).

The oldest (~1.7 Ga) and most abundant intrusions are mainly intermediate composition, foliated hornblende-biotite granodiorite, or monzogranite of calc-alkalic affinity. These intrusions are generally synchronous with regional deformation attributed to the Colorado orogeny, with U-Pb zircon ages in the 1.75-1.65 Ga (Reed et al., 1987; Reed et al., 1993). A second major intrusive episode is represented by the Mesoproterozoic (~1.4 Ga) Laramie Anorthosite Complex (northern Laramie Range) and the ilmenite-bearing Sherman Granite, which outcrops immediately north of the CK Project area (Figure 6.2). Both anorthosite and granite transect the Cheyenne Belt and intrude crystalline rocks of the Wyoming Province. These intrusions comprise the northernmost segment of a wide belt of 1.4 Ga granitic intrusions throughout the Colorado orogen (Sims et al, 2001).

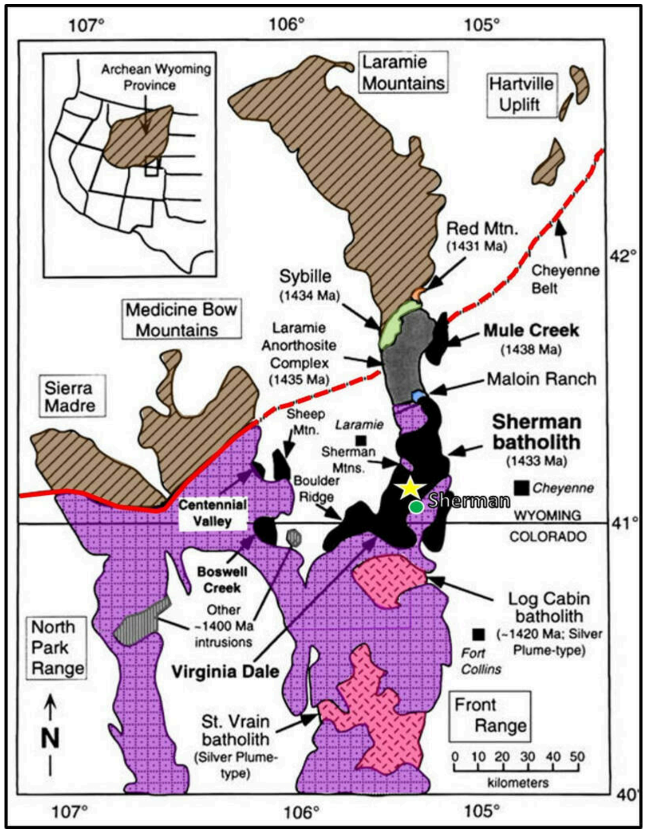


Figure 6.2: Mesoproterozoic intrusive within the Cheyenne suture zone.

The red circle is the approximate vicinity of the Project area; the yellow star denotes the location of Vedauwoo. The basement (brown diagonally lined) north of the Cheyenne Belt is the Archean Wyoming Province; the basement (purple squares with dots) south of the Cheyenne Belt is the Paleoproterozoic Colorado Province. *Source: Edwards and Frost (2000).*

6.1.1 Local and Property Geology

Bedrock geology in the vicinity of the Project area has been described in some detail in various previous reports (Brady, 1949; Hausel, 1982, 1989, 1997, and 2012; Klein, 1974; McGraw, 1954; MDA, 2017, etc.). Most of these existing reports rely solely on surface investigation, though a few discuss observations of historical drill core. While somewhat dated, reports by Klein (1974) and McGraw (1954) are particularly useful as they provide the results of petrographic analysis in conjunction with detailed field measurements and observations. The following discussion draws partly from work completed during previous studies but is largely based on first-hand field observations and careful examination of a combined total of more than 50,000 ft of historical and modern drill core.

6.1.2 Lithology

Within the Project area, bedrock is largely comprised of Proterozoic metasedimentary and intrusive granitic rocks, both of which are unconformably overlain by the Tertiary White River Formation (Figure 6.3). The metasedimentary rocks are exposed in outcrop in the far eastern half of the project area, and these rocks generally consist of interlayered metagraywacke, quartz-biotite schist, and greenschist, all widely variable in grain size and degree of foliation. Trace amounts of very fine-grained, disseminated pyrite are commonly observed in metasedimentary drill core.

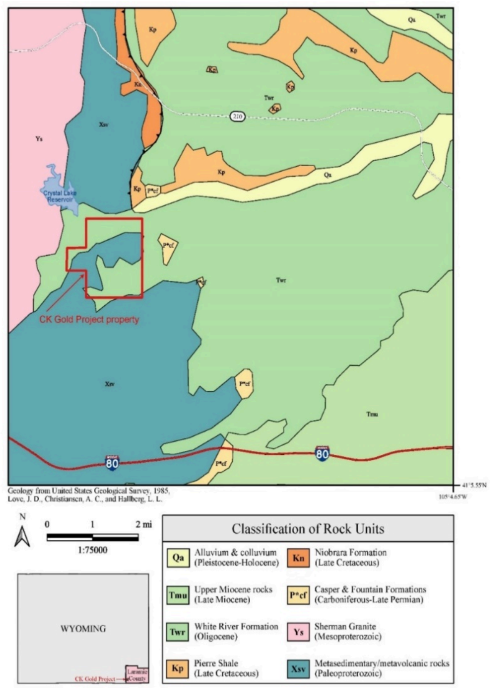


Figure 6.3: Bedrock geology in the vicinity of the CK Gold Project area

A typical cross-section illustrating the lithologic relationships in Figure 6.4

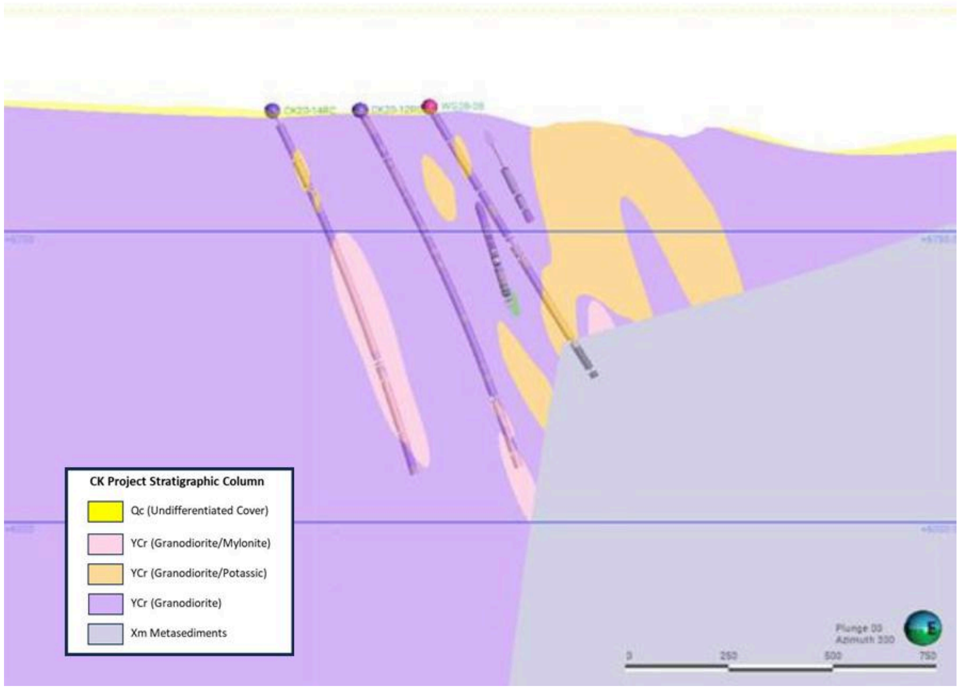


Figure 6.4: Typical Lithologic Cross Section

The metasedimentary rocks are intruded by granodiorite that displays a range of textures from primary igneous (Figure 6.5) to intensely mylonitic (Figure 6.6). These textures are often wildly variable over very short drilling intervals. Undeformed granodiorite is typically hypidiomorphic-granular with subhedral-to-euhedral hornblende and feldspar phenocrysts, generally less than 1 inch in diameter. Porphyritic granodiorite with hornblende and/or feldspar phenocrysts in a fine-grained hornblende, feldspar, biotite, and quartz matrix is also common. Deformed granodiorite varies considerably from proto-mylonitic/weakly foliated to ultra-mylonitic and fine-grained. Sulfide mineralization, predominantly disseminated pyrite and chalcopyrite in the matrix or as inclusions in hornblende and feldspar, are associated with undeformed and deformed granodiorite. Undeformed granodiorite exhibits primarily disseminated sulfide mineralization; however, blebs, sulfide veins, and veinlets also occur. In weakly foliated- to-mylonitic granodiorite, sulfide crystals are commonly aligned with foliation and locally exhibit clustering and/or veinlet-type mineralization. The intrusive contact between granodiorite and metasedimentary rocks is not exposed within the project area but was encountered during drilling in drill holes CK20-18c, CK21-08c, and CK21-09c.

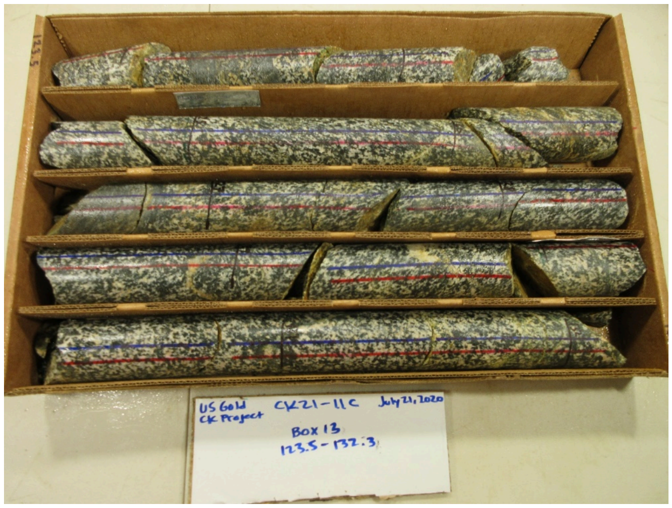


Figure 6.5: Relatively undeformed granodiorite

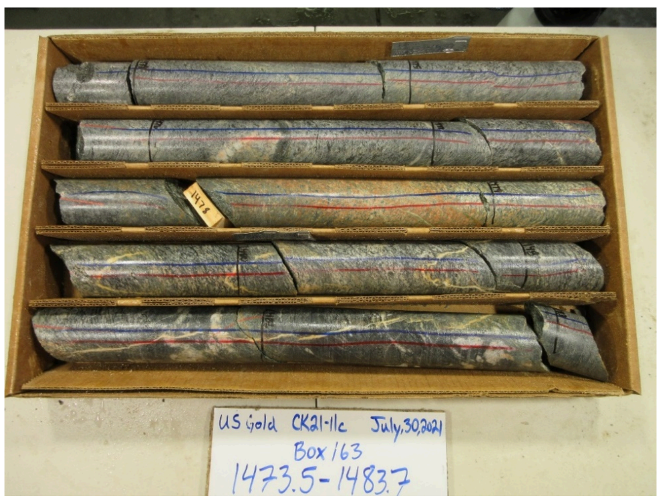


Figure 6.6: Mylonitized granodiorite

All crystalline rocks in the Project area are locally crosscut by pegmatitic to aplitic dikes (Figure 6.7) and very fine-grained mafic dikes (Figure 6.8). Based on the drill core and field exposures, the felsic dikes range in width from inches to roughly 30 feet, while the mafic dikes are generally less than 10 feet in width. Occasional zones of potassic enrichment and/or local pyrite mineralization occur within the felsic and mafic dikes. Potassic-alteration halos of highly variable width and intensity are common along pegmatitic/aplitic margins.

Figure 6.7: Felsic (pegmatite) dike (top row) within granodiorite

Figure 6.8: Typical mafic dike (center of photo) intruding granodiorite

The Sherman Granite is exposed immediately to the north of and adjacent to the CK Gold Project area. The Sherman Granite has been dated at 1430 +/- 20 Ma by the Rb-Sr whole-rock method (Zielinski et al., 1981). Aleinikoff (1983) obtained a U-Pb upper-intercept age of 1412 +/- 13 Ma on zircons separated from different host minerals of the Sherman Granite and, because of possible Pb loss, interprets this as a minimum age. The Sherman intrudes the host granodiorite, which is presumed to be of the ~1.7 Ga generation of regional intrusive events. The dominant rock type of the Sherman Batholith is coarse-grained, biotite hornblende granite, a distinctly reddish-orange rock that commonly weathers deeply to a thick grus. The Sherman Granite is sub-porphyritic, with a seriate, hypidiomorphic-granular texture. Local augen gneiss within the Sherman indicates some late-stage deformation (Houston and Marlatt, 1997). Major phases are microcline, plagioclase, quartz, hornblende, biotite, and ilmenite, while accessory phases are zircon and apatite with rarer allanite and fluorite (Houston and Marlatt, 1997). The contact between the Sherman Granite and granodiorite appears gradational on the order of 5 to 20 ft (Klein, 1974), and (rare) dikes of Sherman Granite within the host granodiorite are exposed in the field near the contact between the two.

6.1.3 Alteration

Several alteration types are observed in the crystalline rocks within the Project area, both in the outcrop and the drill core. The most prevalent type of alteration is potassium enrichment in host granodiorite with the replacement of primary plagioclase feldspar and hornblende by alkali feldspar and secondary biotite. The extent of potassic alteration throughout the granodiorite is variable in terms of intensity and nature of occurrence. In drill core, weak to moderate potassic alteration (Figure 6.9) is typically splotchy to highly localized (i.e., halos around minor veins), while zones of pervasive, moderate to extreme potassic alteration (Figure 6.10) are encountered over intervals of several to more than 100 ft. Potassic alteration occurs independent of deformation (or lack thereof) within the granodiorite, and while it is certainly locally associated with aplitic and pegmatitic dikes, the origin of or driving force behind the more pervasive and extensive zones of potassic alteration is unclear. Klein (1974) has suggested that these zones are a product of fluid transfer during the emplacement of the Sherman Granite, which intrudes the granodiorite just north of the Project area. This seems a reasonable presumption, and particularly so if the aplitic and pegmatitic dikes prove to be distal intrusive extensions of the Sherman Pluton, which should be discernable via age determinations on the granodiorite and the felsic dikes in comparison to existing age data on the Sherman Granite.

Figure 6.9: Moderate, localized potassic alteration in granodiorite

Figure 6.10: Intense, pervasive potassic alteration in granodiorite

Potassic alteration also occurs in mafic dikes within the granodiorite and the metasedimentary rocks, though to a much lesser extent than within the granodiorite proper. In general, intensely potassically altered granodiorite appears to be depleted of sulfide mineralization, with only local, trace amounts of pyrite and extremely rare to no visible chalcopyrite mineralization. Potassic alteration is frequently accompanied by epidote veining (Figure 6.11 and Figure 6.12), and less so by minor propylitic alteration. Propylitic alteration consists of the texturally preserved replacement of plagioclase and hornblende with epidote and is visually much more prevalent in mafic dikes and metasedimentary rocks, particularly in greenschist and discrete quartzite lenses in quartz-biotite schist and metagraywacke. Pyrite grains with epidote halos are occasionally encountered in the granodiorite and, more frequently, in the mafic dikes and metasedimentary rocks.

Figure 6.11: Intense potassic alteration with associated stockwork epidote veining

Figure 6.12: Localized weak potassic alteration with associated epidote veining

While much less prevalent than potassic alteration, phyllic alteration and silicification are also observed in drill core. Again, the extent and intensity of these alteration styles vary across and within the individual crystalline rock types. Phyllic alteration (Figure 6.13) is most often observed in intensely mylonitized granodiorite but also occurs in metasedimentary rocks, particularly near intrusive contact and in significant structural zones. Phyllic alteration is indicated by fine-grained white mica (sericite), chlorite, pyrite, and quartz, and often occurs together with silicification, though the two are not necessarily codependent. In some instances, phyllic alteration identified in the drill core may be a product of cataclasis rather than hydrothermal alteration, wherein the rock has undergone dynamic recrystallization and alignment of sheet silicates during shearing to produce an extreme grade of cataclastic rock known as phyllonite. Phyllonites are often associated with major (crustal) structural zones and typically retain a penetrative cleavage oriented parallel to the fault plane.

Figure 6.13: Phyllically altered mylonite (phyllonite?)

Silicified domains (Figure 6.14) exhibit blurred grain boundaries, moderate to extensive hairline quartz veining, and strong induration. Silicified intervals are generally rich in relatively pure, microcrystalline quartz veins, with apparent associated silica flooding and replacement within the local crystalline groundmass. So-called ‘stockwork’ quartz veining is rare and is generally limited to local zones of brecciation rehealed by quartz or, more commonly, a combination of quartz and calcite.

Figure 6.14: Silicified mylonite

6.1.4 Mineralization

Copper and gold mineralization is largely disseminated, and based on available information to date, occurs solely within the granodioritic plutonic body. Secondary copper minerals, primarily chrysocolla, cuprite, and trace malachite and azurite, as well as secondary iron minerals (hematite, limonite, and jarosite), chalcocite and native copper (flecks and veins) are observed on the surface and define an oxide or supergene zone that extends to depths up to 100 ft below the topographic surface and deeper in fractured or faulted localities. This surficial oxide zone is essentially devoid of magnetite. An intermediate oxide-sulfide or 'mixed' zone observed in drill core is characterized by secondary copper and iron minerals as well as primary pyrite and trace chalcopyrite. The mixed zone transitions to a sulfide-dominant zone at depths ranging from 100 to 300 ft, with a significant decrease in oxide mineral content, increase in occurrence of disseminated pyrite and chalcopyrite, and the appearance of magnetite. Within the sulfide zone, sulfide minerals are typically disseminated and very fine-grained, though occasional sizeable pyrite and or chalcopyrite blebs and minor veins and veinlets are observed in drill core.

Sulfide content is modally highest in granodiorite and mylonitic granodiorite and generally ranges, based on visual analysis, from trace amounts to less than 5% of whole-rock content. In addition to pyrite and chalcopyrite, bornite, covellite, molybdenite, and pyrrhotite are also present, as well as trace amounts of very fine-grained native gold, 10 to 250 microns in size (Mountain Lake Resources Inc., 1997). Assay data indicates a significant, if not direct, relationship between metal concentration and sulfide content, particularly chalcopyrite. Copper-sulfides are virtually restricted to granodiorite, though trace amounts of chalcopyrite are observed both in mafic dikes within the granodiorite and in the metasedimentary rocks immediately adjacent to the east. Trace to weight-percent amounts of pyrite is also observed in drill core in metasedimentary rocks, aplitic dikes, pegmatites, and mafic dikes, all within the sulfide zone.

Gold mineralization at the CK Gold Project occurs in the west central portion of the Project area and is distributed in plan-view in an elongate ovoid pattern trending roughly N60°W (Figure 6.15). The orientation of the mineralized zone is generally coincident with the local trend of shear as interpreted by Klein (1974) and McGraw (1954) based on field measurements of exposed structural fabrics (cataclastic foliation) and fault planes. The primary known mineralized zone is essentially vertical and “keel-like” in shape, as represented by the 1 gpt Au cut-off grade shell with a surface length of 400 ft along strike, width of approximately 200 ft, and depth (thickness) of 600 ft (Figure 6.16). This higher-grade, central core is surrounded by a halo of lower grade mineralization with an overall length of roughly 760 ft along strike, an average width of approximately 500 ft, and thickness of at least 1,100 ft. Low-grade (<0.5 gpt Au) gold mineralization is open and uniform along strike, both to the northwest and southeast, as well as at depth.

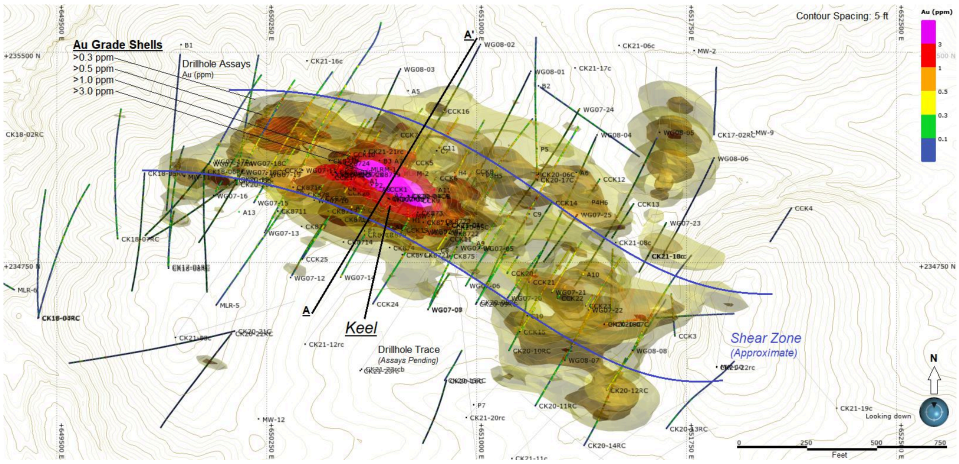


Figure 6.15: Oblique view of the distribution of gold mineralization, CK Gold Project

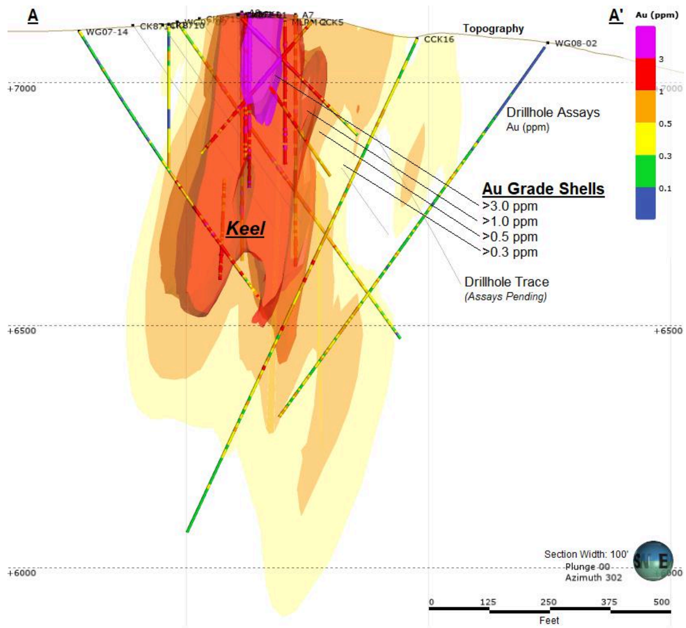


Figure 6.16: Cross-sectional view central to the primary zone of mineralization

The mineralized zone is crudely bound to the north and to the east by the Northwest fault and the Copper King Fault, respectively (Figure 6.17). The Northwest Fault is interpreted based on a combination of drillhole data, geophysical data, and downhole televiewer data from 2020 and 2021 drilling. The Northwest Fault strikes west-northwest and dips steeply to the northeast along the northern margin of the mineralized zone. The fault represents an apparent structural control of the CK deposit, as copper-gold mineralization is essentially restricted to south of the fault.

The Copper King Fault trends roughly N30°E along the eastern extent of the CK deposit, truncating known mineralization in that direction. Host granodiorite occurs to the west of the fault, and unmineralized metasedimentary and metavolcanic rocks occur to the east. Drillhole intercepts indicate that the Copper King Fault dips somewhat steeply to the west, and that primary displacement along the fault plane is reverse with the western hanging wall riding up to the east. This contradicts previous interpretations of the fault as normal with a down-to-the-east, nearly vertical, dip slip offset (Hausel, 2012). Based on examination of exposures in prospect pits north and east of the deposit, the Copper King Fault is thought to be Laramide or younger, though it may represent remobilization along a much older, existing fault plane. Further investigation of the Copper King Fault, including orientation measurements on all available surface exposures as well as additional drilling targeted to intercept the fault at depth, should be considered to verify the orientation of the structure and to evaluate the direction and magnitude of offset. While the fault is presently considered a post-mineral structural control, a better understanding of the direction and scale of offset may provide valuable insight for use during planning of future drilling exploration.

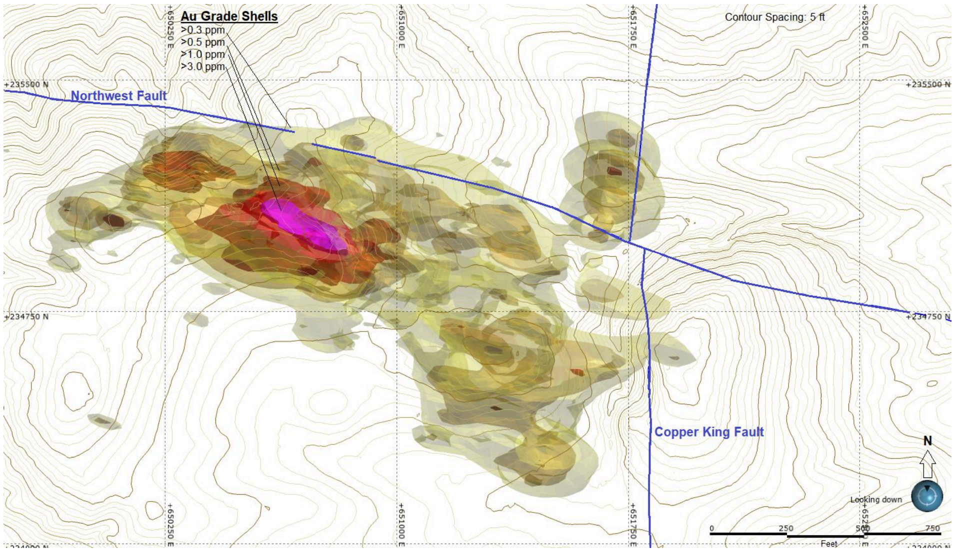


Figure 6.17: Plan view of the location and trend of the Northwest and Copper King Faults

A variety of other faults have been interpreted within the Project area based largely on surface expression, indications in drill core, and televiewer data. As noted by Klein (1974), many local structures are generally concordant with the trend of Precambrian shear and may represent more recent (Laramide or younger), shallow depth rejuvenation along previously existing fault planes. Several local structures are discordant with the Precambrian trend of shear, and these are also generally thought to be Laramide or younger based on a lack of cohesion and recrystallization in the faulted material (Klein, 1974). The significance of these structures relative to the CK deposit is likely limited to an associated increase in intensity and/or depth of oxidation and supergene copper mineralization, and potential, small-scale physical displacement of copper-gold mineralization at depth.

6.2 DEPOSIT TYPE

6.2.1 Discussion

Gold mineralization at the CK Gold Project occurs within a steeply dipping to near-vertical, brittle-ductile shear zone presumably generated during Paleoproterozoic orogenesis of the Colorado Province. As previously stated by Klein (1964), the localization of metallic mineralization at the Project is a product of both structural and lithologic control. The dominant structure appears to be the nearly east-west trending zone of Precambrian shear and cataclasis, and, lithologically, mineralization is virtually confined to the granodiorite plutonic body. Visual examination of barren to high grade drill core intervals shows that gold mineralization (or lack thereof) is not restricted to any specific textural variation within the granodiorite, nor is it strictly associated with any type or intensity of alteration, except for consistently low grades in zones of moderate to intense potassic alteration.

Mylonitic rocks return the highest gold and copper assay values on average. Mylonites form under specific circumstances at significant crustal depths below brittle faults, in continental and oceanic crust (Figure 6.18). Mylonites are the result of extreme plastic deformation, with original textures modified by dynamic recrystallization while the parent rock remains chemically unaltered.

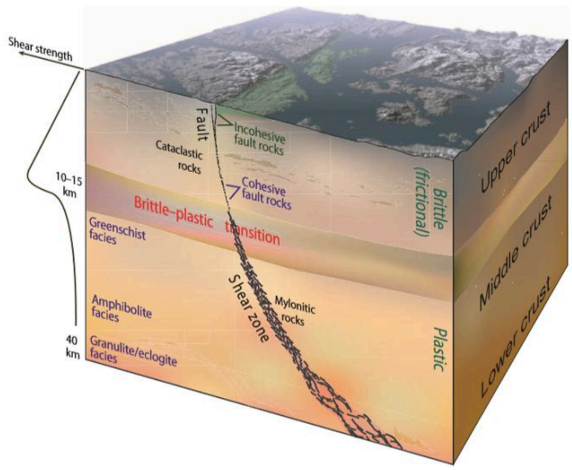


Figure 6.18: Schematic illustration of the transformation of brittle to ductile deformation in granitic rocks at depth (Fossen, 2016)

Gold mineralization in the mylonitized granodiorite occurs in close association with sulfide minerals, which are largely disseminated, but also frequently occur as veinlets or stringers aligned with mylonitic foliation (Figure 6.19). On a microscopic scale, pyrite in mineralized intervals is often broken, indicating some deformation during or after mineralization. Sulfide minerals in the surrounding granodiorite are widely disseminated, typically occur within igneous hornblende and plagioclase, and occasionally occur as clusters and stringers which also tend to parallel weak to moderate foliation, where present.

Figure 6.19: Pyrite +\- chalcopyrite aligned with mylonitic foliation

The mineralogical setting and physical character of the sulfide minerals in both the mylonitized and undeformed granodiorite suggests a primary igneous origin, wherein mineralization occurred during magmatic crystallization, and syn-magmatic or post-magmatic mylonitization due to brittle-ductile shearing served as physical means of concentrating metals simply via shortening of the host granodiorite. The metasedimentary rocks intruded by the granodiorite may have served as a sulfur source to the crystallizing pluton, catalyzing base, and precious metal mineralization through sulfur saturation of the magmatic fluid.

Emplacement and crystallization of the host granodiorite was followed by a regionally extensive, felsic intrusive event represented by the Sherman Granite and the Laramie anorthosite complex. Regional circulation of high temperature, potassium-rich magmatic-hydrothermal fluids exsolved during emplacement of the Sherman Granite is indicated within the host granodiorite by alteration aureoles and halos along pegmatitic/aplitic dike margins and alkali feldspar-quartz veins and by intense potassic alteration associated with significant brittle deformation features. Hydrothermal alteration associated with post-mineral, brittle deformation attributed to emplacement of the Sherman Granite apparently contributed to some degree of gold redistribution, as evidenced by the typically low gold grades within zones of moderate to intense potassic alteration and occasional anomalous gold grades within silicified sample intervals.

Long after formation of the CK deposit, during the Laramide orogeny (55-80 Ma), the host granodiorite was uplifted and exposed to erosion. Reactions between hypogene sulfide minerals and descending, acidic meteoric waters resulted in the supergene enrichment (oxidized) zone exposed at the modern topographic surface. The enriched zone is characterized by the presence of iron oxides, secondary copper minerals, and rare native copper. Pervasive oxidation is typically encountered to a depth of about 100 to 150 ft, though locally (near fault structures) is known to extend to depths approaching 300 ft.

6.2.2 Interpretations and Conclusions

The CK copper-gold deposit does not neatly fit into any specific category or class of conventional deposit models, in part because of the wide array of variability and overlap of assigned deposit model parameters such as geochemical signatures, geologic setting and time frames, and the origin and mechanisms of emplacement of metal-bearing solutions.

Previous authors (Hausel, 1997, 2012; Carson, 1998) have postulated that the CK deposit represents some portion of a copper (Au-Cu) porphyry system, largely based on observations of the nature and occurrence of hydrothermal alteration assemblages exposed in outcrop. According to the U.S. Geological Survey's Porphyry Copper Deposit Model (John et. Al, 2010) and Preliminary Model of Porphyry Copper Deposits (Berger et. Al, 2008), porphyry deposits consist of disseminated copper minerals and copper minerals in veins and breccias that are relatively evenly distributed in large volumes of rock forming high tonnage, low to moderate grade ores. The USGS model descriptions further provide the following (select) characteristics common to known porphyry copper deposits:

- Host rocks are altered and genetically related to granitoid porphyry intrusions and adjacent wall rocks.
- Deposits are centered in high-level intrusive complexes that commonly include stocks, dikes, and breccia pipes, which generally form in the upper crust (less than 5–10 km depth) in tectonically unstable convergent plate margins.
- Wall-rock alteration is intimately linked to narrow veins, commonly 0.1 to 10 cm in width, that typically make up less than 1 to 5% volume of ore but also are present in other alteration zones.
- Copper-bearing sulfides are localized in a network of fracture-controlled stockwork veinlets and as disseminated grains in the adjacent altered rock matrix.
- Hydrothermal wall-rock alteration minerals and assemblages (namely potassic, sericitic, argillic, and propylitic) are zoned spatially and temporally, with kilometer-scale vertical and lateral dimensions.
- Zones of phyllic-argillic and marginal propylitic alteration overlap or surround a potassic alteration assemblage.
- Potassic and sericitic alteration are invariably associated with sulfide mineralization and generally are temporally, spatially, and thermally zoned with respect to one another.
- Potassic alteration tends to be more centrally located, deeper, higher temperature, and earlier compared to sericitic alteration.
- Owing to the shallow depths of deposit formation (1–4 km), preserved deposits are predominantly Mesozoic and Cenozoic.

While the alteration assemblages encountered within the CK deposit are indeed like those associated with porphyry copper deposits, hydrothermal alteration zones at CK decidedly lack kilometer-scale vertical and lateral dimensions, and potassic and sericitic alteration are clearly not invariably associated with sulfide mineralization, nor are they necessarily temporally, spatially, and thermally zoned with respect to one another. The Proterozoic age of the CK deposit's host granodiorite and apparent pre- or syn-deformational mineralization further preclude it from classification as a sensu-strictu porphyry deposit.

The CK deposit also exhibits a variety of characteristics that are, individually or in combination, like those of known intrusion related, iron oxide copper-gold (IOCG), and even orogenic deposits. In each instance, however, the similarities (i.e., age, structural setting, geochemical signature, alterations styles, etc.) are either outweighed by significant differences or are too limited, at present, to support a decisive association with the deposit model.

The regional geologic setting of the CK deposit within the Cheyenne Suture belt is significant, as is the nature of occurrence of sulfide mineralization as disseminations in undeformed granodiorite and in alignment with foliation in foliated to mylonitized granodiorite. Based on the available data and information to date, we suggest that Klein's (1974) description of the CK deposit as a "structurally controlled base and precious metal deposit hosted in a Precambrian shear zone" is essentially correct if you want further refinement. While Klein's description does not present a conventional deposit model, it does provide a reasonable interpretation on which to base plans for future exploration. Future drilling exploration (and petrographic and/or mineralogical analysis) should be carefully planned to test Klein's interpretation and target data useful in further developing an appropriate deposit model for the CK Gold Project, whether conventional or not.

7.0 EXPLORATION

7.1 SUMMARY OF EXPLORATION ACTIVITIES

The CK Gold Project was reportedly discovered in 1881, high-graded, and saw limited mining. The first exploration work reported was drilling by ASARCO in 1938. Several additional rounds of drilling have been conducted since that time. In 1972, Henrietta Mines Ltd. acquired the property and completed a comprehensive exploration and development program. In addition to drilling, an induced polarization (IP) survey, geologic mapping, geochemical sampling, and metallurgical testing were conducted (Nevin, 1973). Drilling campaigns were conducted by Saratoga since 2006 and Strathmore since 2012, with a hiatus in drill exploration until the acquisition by U.S. Gold from Energy Fuels in 2016. U.S. Gold conducted drilling programs in 2017, 2018, 2020, and 2021. Drilling in 2021 focused on data collection to support post-PFS and PFS updates in 2022.

7.2 EXPLORATION DRILLING

The drilling record prior to 1997 is incomplete and much of the historical core has been lost. Contemporary drilling reports as well as comparisons to recent drilling have been used to support the use of the pre-1997 drilling. In 2020, historical drill hole collars were located, surveyed and the results compared closely to their location in the historical drilling database.

Figure 7.1 indicates a total of 173 holes with a total drill length of 98,415 ft (29,997 m) have been drilled on the CK Gold property. Figure 7.1 shows the location of all holes within the CK Gold mineral resource area. An additional six historic holes totaling 3,560 ft (1,085 m) are in the database but outside of the current resource area.

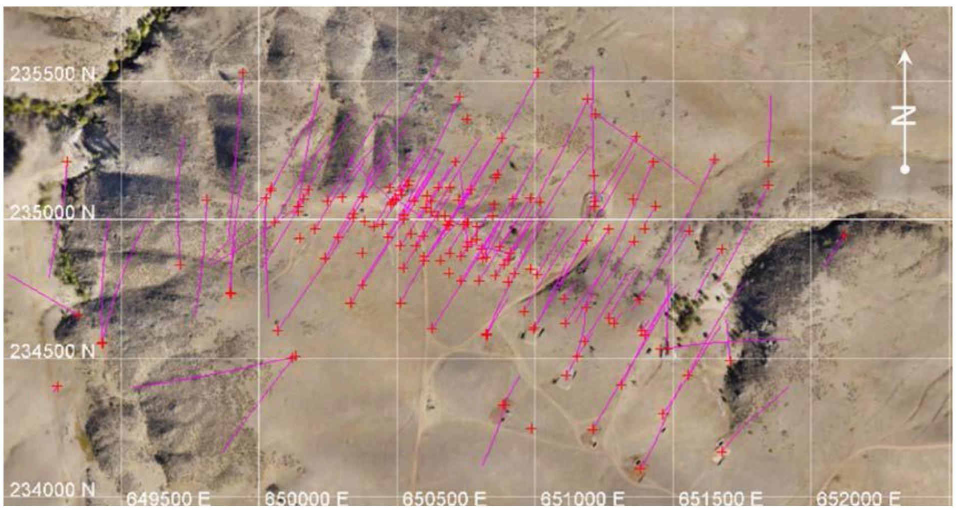


Figure 7.1: Drill hole Map

7.2.1 U.S. Gold 2021 Drilling Campaign

U.S. Gold began a drilling campaign in July of 2021 consisting of 48 holes and 40,930 ft (12,475 m), comprised of reverse circulation, rotary, and core drilling. The primary purposes of this campaign were to continue to refine hydrologic and geotechnical subsurface conditions, and minor exploration immediately southeast of the proposed project. Thirteen monitoring wells totaling 5,600 ft were proposed for sub- surface groundwater studies. Results from this campaign were compared visually to the existing model, and a model was estimated using the previous parameters and including the new holes. There is no material change in the mineral resource or mineral reserve estimate. There have been no findings or observations following the 2021 exploration and data gathering program that materially affected the findings of this study.

7.2.2 U.S. Gold 2020 Drilling Campaign

In October 2020, U.S. Gold conducted a drill program at the Project. Part of that work included surveying new and historical drill hole collars that U.S. Gold could locate in the field and flag.

All historical collar coordinates (pre-2020) were loaded into a handheld GPS unit and visited in the field. Those identifiable (cement, tags, drill pipe, etc.) were flagged with lath and flagging, with the hole name on the lath. These collars were then surveyed at the same time as the 2020 holes, on October 21st, 2020.

Surveying was completed by Topographic Land Surveyors of Casper, WY, and the results were certified by Professional Land Surveyor Aaron Money, #14558. The survey method was Real-Time Kinematic GPS using a Trimble R10 GNSS GPS system.

Drill hole collars from the historical programs dating back to 1938 were identified in the field and resurveyed, confirming the locations recorded in the drilling database.

Comparison of the new-collar surveys with the old coordinates showed small variability in X and Y coordinates, typically less than 5 ft and around 25 ft at most, and a bit more in elevation (around 25 ft at most).

Two permanent survey control points were placed on the Project for future use.

7.2.3 U.S. Gold 2020-2017

U.S. Gold completed two RC drilling programs in 2017 and 2018. RC drilling comprised four holes in 2017 and eight in 2018, totaling 12,040 ft. (3,670 m). Both programs were designed to investigate magnetic and IP anomalies generated by geophysical surveys. Drilling was completed by AK Drilling of Butte, Montana, using a Foremost MPD 1500 RC drill. Samples were collected at 5 ft (1.5 m) intervals from the discharge of a rotary splitter attached to the drill. A chip tray was also filled from cuttings for geologic logging and archived. Samples were delivered to Bureau Veritas of Sparks, Nevada, for analysis.

A rotary, reverse circulation, and diamond core drill program was begun in September 2020, and 30 drill holes totaling 21,810 ft (6,647 m) were completed by early December 2020. Core drilling totaled 10,561 ft (3,219 m), and rotary drilling totaled 10,538 ft (3,312 m). The focus of U.S. Gold’s work was to generate metallurgical composites, collect geotechnical data, and expand mineral resources.

Alford Drilling completed core drilling using an LF90 drill rig. HQ core was recovered using a split tube core barrel system to minimize core damage. Holes are monumented using braided steel cable and a tag embedded in a concrete pad at the drill hole collar.

7.2.4 Saratoga 2007 – 2008

Saratoga's drilling campaign focused on expanding the mineralized body outlined in previous campaigns and providing material for metallurgical testing and future geotechnical studies. The diamond drill program began in 2007, paused over winter, and was completed in 2008. 35 holes were completed for a total length of 25,462 ft (7,760 m). Logan Drilling, based in Nova Scotia, Canada, was the drilling contractor, and a Longyear Fly 38 skid rig drilling NQ-size core (4.76 cm diameter) was used.

7.2.5 Historical Drilling

There is limited information on drilling and sampling procedures for the ASARCO, Copper King Mining, and the U.S. Bureau of Mines (USBM) drill programs. The original geology logs are not available, although Nevin (1973) provides summary geology logs for all but the ASARCO 1938 drilling and assay sheets for these drill programs. The assay sheets include collar coordinate information, bearing and dip of hole, sample intervals, and Au, Ag, and Cu assay data. Defense Minerals Exploration Administration documents (0647_DMA) include identical logs for the ASARCO which only contain assays and recoveries for ASARCO diamond drill holes A-1 through A-5 and state they were assayed by Federal Mining and Smelting Co Wallace Testing Plant in Wallace, Idaho.

Previous attempts to locate the drill core from ASARCO's, and the USBM drill programs that had been housed at the USBM in Denver were unsuccessful. According to Mountain Lake Resources Inc. (1997), the core collected from Henrietta's holes was destroyed.

Soule (1955) reported that the USBM's drilling was done by contract and that all three holes were core holes, but his report provided no further information.

Henrietta Mines drilled seven rotary holes totaling 482 m and six core holes totaling 666 m. Several of the holes were started as rotary and finished as core. Boyles Brothers Drilling Company of Golden, Colorado, was the drilling contractor.

Compass Minerals drilled 21 rotary holes and five diamond core holes. Hole CCK-16 was drilled rotary to a depth of 152 m and then cored with NX core to a total depth of 341 m. Notes on the geologic log indicate the core was split before logging. Hole CCK-19 was cored for its entire length with HQ core. Holes CCK-24 and CCK-25 were both started with RVC drilling, changing to NX core at 136 m and 136 m, respectively. Hole CCK-26 was cored completely with NX core. There are no further details about Compass's drilling program.

There are few details on the Caledonia or Mountain Lake drill programs. No drill logs are available for the Caledonia holes; the collar locations were taken from a map. The Caledonia holes ranged from 220 ft (65 m) to 550 ft (170 m) in depth and were intended to confirm the results of prior drilling. A report by Gemcom (1987) describes the Caledonia drilling as spaced 50 ft (15 m) apart through the mineralization, sampled every 10 ft (3 m), and assayed for gold. Gemcom entered and verified the Caledonia drilling data.

Drill logs of the Mountain Lake holes are available which do contain collar and drill orientation data. Summary geology from the Mountain Lakes drill holes was entered into the database.

As mentioned above, Henrietta's core hole H-1 does not show evidence that any of the other holes drilled on the Copper King property were downhole surveyed.

There is inherent risk associated with these legacy drilling programs (pre-2007 drilling), and limited information is available. These risks include errors in collar location, downhole orientation, assay grade precision and accuracy, and database transcription errors. Comparisons to recent infill drilling continue to support the use of the legacy holes. To acknowledge the risk, no legacy holes are used in the classification of measured resources.

7.3 NON-DRILLING EXPLORATION ACTIVITIES

7.3.1 Geophysics

Magnetic and two IP surveys were completed in the early 1970s. The magnetic survey measured vertical intensity using a Jalander instrument on 200 ft (60 m) line spacing and stations. Two significant positive anomalies are present. One, about 800 ft (245 m) wide and 1,500 ft (460 m) long in a northwest direction, has a magnitude of 500 gammas above the background and coincides with the principal mineralization direction. The anomaly is believed to be caused by the presence of magnetite in the mineralized rock.

The initial IP survey showed a resistivity high extending northeast through the CK deposit, following a trend of thin overburden and chargeability high of 18 ms against a background of 6 ms. The second IP survey was by McPhar Geophysics Inc. using a Scintrex I.P. R-7 unit over the principal mineralized area. Line spacing was 300 to 800 ft (90 m to 240 m). Five north-south lines and two east-west lines were run. Dipole spacing was 200 ft (60 m). An anomaly, principally a moderate to shallow metal factor anomaly, was detected, trending east-northeast to the principal mineralized area. Both IP surveys established that the ore does not respond well to IP chargeability, and frequency effects for the two methods are low and do not duplicate each other as expected.

In 1994, Pearson deRidder & Johnson, Inc. conducted an aeromagnetic survey on the property for Compass Minerals. Flight lines were flown at a nominal altitude of 300 ft (90 m) above ground level, with north-south lines spaced 660 ft (200 m) apart and east-west lines spaced 1,320 ft (400 m) apart. Several major magnetic trends and features were observed. The primary mineralized area around the Copper King Mine is identified as a magnetic high.

In 1997, Gilmer Geophysics, Inc. supervised and interpreted a ground magnetic survey and a VLF-EM survey. The ground survey was laid out using GPS and total survey technologies with principal directions oriented N33E and N57W. This orientation was chosen to cross-mapped features at right angles. Line spacing was 200 ft (60 m) between the N33E lines. Total field ground magnetometer data were obtained using two GEM Systems GSM-19 units used in "walking mag" mode, obtaining data every two seconds, resulting in station spacings of 2 ft to 10 ft (0.5 m to 3 m) along survey lines. The VLF-EM data was obtained using an IRIS T-VLF instrument.

In June 2017, Magee Geophysical Services, supervised by Jim Wright of Wright Geophysics, completed a ground magnetic survey over the CK Gold Project. 70 line miles (113 km) of magnetic data were surveyed using real-time corrected differential GPS and Geometrics Model G-858 magnetometers. Lines were spaced 160 ft (50 m) apart and oriented N30E across the project. Magnetometers were mounted on a backpack with data collected every two seconds. Data interpretation by Jim Wright essentially duplicated the 1997 Gilmer survey. A strong magnetic anomaly was demonstrated over the CK Gold deposit along with several magnetic anomalies to the east and south of the deposit. A prominent anomaly at the southeast corner of the project called the Fish Anomaly, was tested by RC drilling in 2017, along with a couple of others to the east of the CK Gold deposit.

In October 2017, an IP survey was completed over the CK Gold Project area by Zonge International and interpreted by Wright Geophysics. A total of eleven lines were completed using a standard 9-electrode dipole-dipole array with a dipole length (a-spacing) of 1,082 ft (330 m) as designed by Wright Geophysics. Data were acquired in the time-domain mode using a 0.125 Hz, 50% duty cycle transmitted waveform. Data were acquired along eleven north-south oriented lines. Stations were located using a Garmin hand-held GPS, model GPSMAP 64CSx. The GPS data were differentially corrected in real time using WAAS corrections. Accuracy of the GPSMAP 60CSx typically ranges from 6 ft to 16 ft (2 m to 5 m) line control in the field utilized UTM Zone 13N NAD27 datum. Measurements were made for continuous line coverage at n-spacing of 1 through 7. Data were acquired in the time-domain mode using a 0.125 Hz, 50% duty cycle transmitted waveform. Chargeability values (IPm) represent the Newmont Window with integration from 450 to 1100 milliseconds after transmitter turnoff. A discussion of the time-domain acquisition program is presented with the digital data release. IP anomalies identified to the west of the CK Gold deposit were tested by RC drilling in 2018.

7.3.2 Geochemical

Nevin (1973) reports the results of soil geochemistry. Forty-four soil geochemical samples were taken on 100 ft and 200 ft (30 m and 60 m) centers in widely separated traverses as a pilot study. All were analyzed for copper and arsenic, and some were analyzed for gold, zinc, silver, and mercury. Three copper populations were sampled. The absolute background has values of about 20 ppm; a high background population in proximity to the mineralized rock has values of about 500 ppm; four samples taken in thin soil directly over the mineralized rock returned values of more than 1,000 ppm. Gold values appear to be a useful indicator of mineralization. Zinc, silver, and arsenic had little contrast between mineralized and unmineralized areas. Mercury was found to have good contrast and was recommended for further investigation.

7.4 GEOTECHNICAL DATA, TESTING, AND ANALYSIS

Prior to 2020, no previous geotechnical work was completed on the Project. U.S. Gold retained Piteau Associates of Reno, Nevada, to design, complete, and analyze a geotechnical program that included field outcrop mapping, on-site geotechnical core logging, rock testing and sampling, televiewer data validation, and interpretation. Four days were spent reviewing existing drill core and mapping surface outcrops at the CK Gold Project. Surface mapping focused on joint and fracture set characterization for integration with sub-surface derived data.

Five geotechnical core holes (CK20-16c to 20c) totaling 4,685 ft (1,428 m) were completed. Core from these holes was logged on-site, run by run, in a designed-for-purpose logging trailer by Piteau staff or consultants. Geologists completing the geotechnical logging also completed needed rock characterization testing and selected geomechanical samples for third-party testing. Logging parameters included core recovery, hardness, RQD, RMR, fracture frequency, joint condition, and angle, degree of breakage, and degree of alteration.

Piteau staff completed point load index (PLI) testing in the field on the five geotechnical core holes and two metallurgical holes (CK20-06c and 07c). During geotechnical logging, 1,065 PLI tests were completed on the whole core.

Geomechanical samples were collected at chosen intervals by Piteau staff during logging. These samples were utilized for the characterization of the intact rock strength. 13 samples were collected for uniaxial compressive strength, 15 for triaxial compressive strength, 11 for indirect tensile strength, and 25 for discontinuity direct shear testing. Sample testing was completed at the Wood Group, PLC Rock Mechanics Laboratory in Hamilton, Ontario, Canada. In addition, one fault gouge sample from CK20-16c was taken and tested at Golder Associates Geotechnical Laboratory in Denver, Colorado. Piteau Associates integrated the results of this testing into their mine design recommendations.

Piteau Associates also validated, processed, and interpreted down-hole televiewer data from 13 holes completed in 2020, including the five geotechnical core holes and holes CK20-01c, 03c, 04cB, 05c to 07c, 09rc, and 21c. For major faults and contacts, Ken Coleman with U.S. Gold completed initial processing and structure picking, followed by Piteau work for joint and fracture set characterization. Televiewer surveys were completed by either COLOG or DGI Geoscience.

7.5 HYDROGEOLOGY

No previous hydrogeologic work was completed at the Project prior to 2020. During its 2020 drilling program, U.S. Gold and its consultants, Neirbo Hydrogeology (Neirbo) and Dahlgren Consulting, completed a limited water characterization and hydrogeology program. Several designed-for-purpose drill holes were completed, and data were collected from holes designed primarily for other uses.

Seven water characterization wells (MW-xx series) were drilled and completed in 2020, five by DrillRite Drilling of Spring Creek, Nevada, and two by McRady Drilling of Cheyenne, Wyoming. DrillRite drilling was completed using reverse-circulation methods and McRady work was completed using conventional rotary methods. A total of 2,755 ft (840 m) was drilled and completed. Holes were completed as water wells, screened, and cased at proper intervals with a locking cover and monuments placed at the surface. These wells are checked regularly for water levels and water quality.

Eight core and RC holes designed for metallurgical resource expansion and geotechnical purposes were also utilized for hydrogeologic purposes. These holes totaled 7,511 ft (2,289 m) and consisted of two metallurgical core holes, one RC resource expansion hole, and five geotechnical core holes. The two metallurgical core holes (CK20-04cB and CK20-06c) were kept open, cased, and capped, similar to the water characterization wells. These two holes are utilized for water quality sampling and obtaining water levels. Televiewer surveys were completed in these two holes as well to aid in hydrologic and geotechnical studies.

Three geotechnical core holes (CK20-17c, 18c, 19c) and one RC hole (CK20-09rc) had vibrating wire piezometers (VWPs) installed in them. Packer testing and televiewer surveys were also completed on the core holes. The two remaining geotechnical core holes, CK20-16c and 20c, only had packer testing and televiewer surveys completed.

Packer testing was completed by Alford Drilling under the supervision of a Neirbo consultant. VWP installation was completed and supervised by Call & Nicholas, Inc. of Tucson, Arizona. Televiewer surveys were completed by staff of either COLOG or DGI Geoscience at the same time as downhole gyroscopic surveying at the end of drilling each hole. Additional details on the current program are available in Section 13.3.

8.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

8.1 SAMPLING

8.1.1 U.S. Gold 2021 - 2017

Ordinarily core was collected by the geologist four times per 24-hour shift and returned to the core logging facility. The core processing steps were as follows:

- Core is washed and scrubbed.
- Core is aligned in the box to represent the original condition of the core as accurately as possible (i.e., all fractured/broken ends are matched and rotated to fit back together.)
- Core is washed and scrubbed again.
- Beginning and ending depths are marked on the inside core boxes while the core dries.
- When the core is dry, it is marked top to bottom with blue and red orientation lines, blue on the left, and red on the right, depths are marked and labeled in black on one-foot increments
- Core is logged for recovery, RQD, and fracture frequency per run, and this information is recorded on the log sheet, along with any structural features significant enough to be recorded at the resolution of the log sheet.
- Gross lithology breaks are identified and recorded in the graphic lithology log column.
- Core is inspected in greater detail as sample intervals are selected on a nominal 5-foot sample interval within consistent lithologies, and sample breaks on lithologic (or other appropriate, i.e., significant variation in alteration type or intensity) contacts with a minimum sample interval of 1 ft.
- Assay sample intervals are marked in green, with a line perpendicular to the core axis indicating the top and bottom of the interval, and the sample ID marked on the core (if possible) parallel to the core axis.
- Sample IDs are scribed on silver sample tags, which are stapled to the core box on the left-hand side of the core.
- Detailed information is recorded for each sample interval on the core log sheet (rock type, oxidation, alteration, mineralization, sulfide content, mineral content, veins, fracture, etc.
- Magnetic susceptibility meter measurements.
- Assay samples are recorded on the lab's assay sample inventory form. The log sheet indicates the core boxes in which each assay interval is contained (sample intervals often cross box boundaries).
- Logged core is transferred from the logging table to the photo station, re-wetted, and photographed.
- Photographed core boxes are reunited with their lids and moved either to the back of a waiting truck for transport to the pick-up area at the back of the lot or to a secondary staging area near the garage entrance to be moved to the back of the lot later.

8.1.2 CK Gold – 2021-2017

RC samples were collected in five-foot intervals from the discharge of a rotary splitter attached to the drill and then delivered to ALS in 2021 and to Bureau Veritas lab in Sparks, Nevada, in 2017 and 2018 for analysis. U.S. Gold staff labeled and inserted commercial QA/QC samples.

A red cut line was drawn along the midline of the core by a geologist, and a blue line, which indicates the core direction, was drawn next to it. During 2021, the core was sawn in half by U.S. Gold personnel. During 2017-2018, the core was sawn by Bureau Veritas in Reno, NV, and the half core containing the blue line was sampled. Sample tags were affixed to the inside of each core box, and the sample number was written on the core. Typically, samples were 5 ft (1.5 m) long, broken at lithologic or important geologic feature contacts.

Ordinarily, the geologist collected the core four times per 24-hour shift and returned it to the core logging facility. The core was housed in the garage of a residential home in Cheyenne, WY, or placed in the backyard prior to shipping. In 2021, all core was moved to a secured facility in Cheyenne, WY. Shipping was by a commercial carrier using the chain of custody documents and delivered to the assay lab facilities in Elko and Reno, NV.

8.1.3 Saratoga

The core from the 2008 drill program was logged in the spring/summer of 2008, contemporaneous with the drilling, though sampling was delayed until the fall of 2009 due to budgetary constraints.

Saratoga sampled the 2007 and 2008 drill core on approximate 5 ft (1.5 m) intervals, although sample intervals did range from 1 to 10 ft (0.3 to 3 m) as warranted by the geology. Due to the pervasive alteration and potential for mineralization observed throughout all drill holes, the core was continuously sampled with no gaps in the sample sequence. The samples were collected principally by sawing the core in half, though some intervals, due either to the hardness of the rock or the unavailability of the saw, were split with a hydraulic splitter. In those cases where the sample intervals were fractured, and many of the core pieces were too small to either saw or split, the sampling technician sampled the core using a trowel, a small shovel, or by hand. One half of the core was bagged and sent for assay, while the remaining half was placed back into the core box and put into storage.

The geologic logging process for the first 15 core holes of the 2007 drill program included core photography and geotechnical rock quality (RQD) measurements, along with structural and lithologic determinations. However, core-recovery data recording was missing.

For the remaining 2007 core holes and all the 2008 drill holes, core photography, RQD, and core recovery measurements, geologic logging, and sampling were conducted in an open-sided shed. Due to the limited covered space, some of the core was exposed to the weather.

The proposed drill hole locations were in the field by Western Research and Development (Western), a professional survey company based out of Cheyenne, Wyoming. Western used a LYCA XLS 1200 GPS survey instrument, which has a <0.5 ft (0.15 m) accuracy. Upon completion of the drill program, Western returned to the project site and re-surveyed the actual drill collars.

8.1.4 Historical Exploration

According to Soule (1955) and the photocopied data provided to MDA, the ASARCO 1938 core samples were sampled at 5 ft (1.52 m) intervals, while the Copper King core holes were sampled at 10 ft (3.1 m) intervals. The 1970 ASARCO sampling was variable, though most sample lengths were 10 ft (3.1 m).

Soule's (1955) report briefly described the USBM's sampling procedures. For their three holes, all core and necessary sludge samples were delivered to the USBM's engineer. All core samples were logged and split, with one split half sent to the USBM's Salt Lake City laboratory for analysis. Sludge samples were taken when core recovery was less than 85-90%. All sludge samples from holes B-1 and B-2 were saved until the end of the project; most from hole B-1 were analyzed, but only a few from hole B-2 were analyzed. No sludge samples from B-3 were saved because core recovery was generally excellent. The USBM drill holes were sampled on variable length intervals ranging from approximately 3 ft to 16 ft (1 m to 5 m) with most sample lengths between 6 ft and 10 ft (2 m and 3 m).

Henrietta's drill holes were sampled and assayed at about 10 ft (3.1 m) intervals for gold and copper and occasionally for silver and acid-soluble copper (Nevin, 1973). The core was split, with one half sent for assay and the other half stored on site. For the dry intervals of the rotary holes, a box and cyclone in series were used for sampling with splitting by a Jones riffle. Nevin (1973) estimated that about 1 to 2% of the sample was lost as very fine dust. For the wet drilling, cuttings were split in a long, metal sluice box equipped with a longitudinal baffle set to retain about a 10% fraction for assay. Rejects were stored on site.

According to Clarke (1987), Caledonia's drill holes were sampled every 10 ft (3 m) and assayed for gold, but the historic data included only composite intervals ranging from 3 m to >50 m.

The Compass RVC holes were sampled at 5 ft (1.5 m) intervals, while the core holes were sampled at 10 ft (3.1 m) intervals. The Mountain Lake drill holes were all samples at 5 ft (1.5 m) intervals. MDA has no further information on the Compass or Mountain Lakes drill sampling.

8.2 ANALYTICAL PROCEDURES

8.2.1 U.S. Gold 2021 Campaign

For the 2021 drilling campaign, Hard Rock Consulting (HRC), sub-contracted through Gustavson, conducted field activities, logging, core sawing, and initial sample selection. ALS were selected to conduct assaying, and selected samples, along with standards and blanks, were sent off to the laboratory by HRC. The program was initiated to provide additional data to support a FS and included the test necessary for both the hydrological and geotechnical studies. There have been no material findings to date that would support a departure from the findings in the PFS.

8.2.2 U.S. Gold 2017 – 2020 Campaign

2020 samples were logged, and sample intervals were selected and passed along with cut sheets to Bureau Veritas (BV). BV cut the core and analyzed a sample from the half core, with the other half returned to the core boxes for storage and reference. The retained half core and sample rejects were initially stored in the warehouse at BV while assaying was conducted and have been subsequently moved for storage in a facility in Cheyenne near the Project. During the sample submission process, a contract geologist, M. C. Newton, was on hand at the BV facility to receive core, discuss and inspect procedures, on an intermittent basis as part of the chain of custody and QA/QC check procedures.

BV inserted commercial blanks and standard reference materials from cut sheets determined by U.S. Gold. Throughout 2017 – 2020, BV of Reno, NV, was the primary laboratory responsible for cutting the core, sampling, preparation, and assaying. Some compromises were needed during the 2020 COVID-19 outbreak as access to the BV lab and personnel was restricted. Video and careful consultation with laboratory staff satisfied the role of the consulting geologist in verifying that correct handling and procedures were followed.

8.2.3 2007 – 2008 Saratoga Campaign

The Saratoga core samples from the 2007 drill program were shipped to ALS Chemex (Chemex) in Elko, Nevada for sample preparation and then on to the Chemex facility in Sparks, Nevada, for analysis for gold and a 33-element geochemical suite. Results were received in December 2009. The Chemex sample preparation and analysis methods requested by Saratoga were “AA23” for gold and “ME-ICP61” for the geochemical suite. Both methods employ the same sample preparation methods, which include crushing the whole sample to 70% passing -2mm and then pulverizing 250 g to 85% less than 75 microns (-200 mesh). The “AA23” gold analysis consists of splitting out a 30 g pulp sample and then using fire assay techniques followed by an atomic absorption (AA) finish. The detection level for this analysis is 5 ppb Au, while the upper precision level is 10 ppm Au. Samples assaying over 10 ppm are re-assayed using a fire assay with a gravimetric finish technique (Chemex lab code “Au-GRA21”), which has an upper precision level of 1,000 ppm Au. The “ME-ICP61” analytical procedure consists of a four-acid digestion and analysis by inductively coupled plasma (ICP) followed by atomic emission spectroscopy (AES). The reported range for copper values using this technique is between 1 and 10,000 ppm Cu. Samples with initial values over 10,000 ppm Cu are re-run using the same analytical techniques optimized for accuracy and precision at high concentrations (Chemex lab code “CU-OG62” with an upper precision of 40% Cu).

The core samples from the 2008 drill program were shipped in the fall of 2009 to American Assay Laboratories (American Assay) in Sparks, Nevada for sample preparation and analysis for gold and copper only. The results were received in September 2009. The American Assay sample preparation and analysis methods requested by Saratoga were “FA30” for gold and “D2A” for copper. Both methods employ the same sample preparation methods, which include crushing the whole sample to 70% passing -2mm and then pulverizing 300 g to 85% less than 105 microns (-150 mesh). The “FA30” gold analysis consists of splitting out a 30 g pulp sample and then using fire assay techniques. The detection level for this analysis is 3 ppb Au, while the upper precision level is 10 ppm Au. Samples assaying over 10 ppm are re-assayed using a fire assay with a gravimetric finish technique (American Assay lab code “Au-GRAV”), which has an upper precision level of 1,000 ppm Au. The “D2A” analytical procedure for copper consists of an aqua regia digestion and analysis by AA. The reported range for copper values using this technique is between 1 and 10,000 ppm Cu. Samples with initial values over 10,000 ppm Cu are re-run using the same analytical techniques optimized for accuracy and precision at high concentrations (lab code “Cu Ore Grade”) with an upper precision of 40% Cu.

After the analyses were completed and temporary storage at Chemex, Saratoga retrieved all of the pulps and selected coarse reject samples from mineralized intervals and is currently in storage in Elko, Nevada.

The drill crew, upon filling a core box, placed a wooden top over the core, and the box was secured using strapping tape. At the end of each drill shift, the core was transported by the drill crew into Cheyenne, WY, about 20 miles (32 km), and placed in a locked commercial storage unit. The storage unit is located within a secure, gated facility. About once per week, the core was transported on a trailer to the logging and sampling facility in Casper, Wyoming, 200 miles (320 km).

Logging and sampling of the first 13 core holes drilled in 2007 were completed in a large, converted garage located on leased private property outside of Casper, Wyoming. The property was fenced off and kept securely locked when personnel were not on-site. After being logged and sampled, the remaining half-core was placed in a locked storage unit within a secure, commercial storage facility in Casper.

Saratoga's lease on the Casper logging facility ended on August 31, 2007, and the remaining 2007 core holes were transported 200 miles (320 km) to Dubois, Wyoming, for storage and further core processing. Sampling was conducted within an open-sided ranch shed on private property owned by Norm Burmeister, an officer with Saratoga. The core facility was within a fenced area. After sampling was complete, the core was transported to a commercial storage facility and stored on racks in a locked storage unit. These same procedures were used for the 2008 drilling.

The half-core samples to be shipped to the lab were given non-referential sample ID numbers. The individual bagged samples were placed into larger shipping bags, which were securely closed using heavy wire ties and kept inside the logging facility awaiting shipment via a commercial trucking company to Chemex in 2007, and Chemex and American Assay in 2008.

8.2.4 Legacy Campaigns

Very little is known about the sample preparation, assaying and analytical procedures of the sampling at the CK Gold Project except as described below. A table summarizing pre-1998 drilling on the property (Mountain Lake Resources Inc., 1997) gives detection limits for gold and copper assays for six of the drill campaigns. For both the 1938 and 1970 assays by ASARCO, the detection limits were 0.001 opt Au (0.034 gpt Au) and 0.01% Cu (Mountain Lake Resources Inc., 1997). For Copper King Mining's assays, the detection limit for gold was 0.01 opt Au (0.343 gpt Au), and the detection limit for copper was thought to be 0.10% (Mountain Lake Resources Inc., 1997).

For the three holes drilled by the USBM, analysis was done by the USBM's Salt Lake City laboratory (Soule, 1955). The detection limits were 0.005 opt Au (0.171 gpt Au) and 0.05% Cu as indicated by Mountain Lake Resources Inc. (1997). The USBM also prepared composite samples of the core from their three holes and analyzed them for molybdenum, tungsten, nickel, and for most of them, titanium. In addition, the USBM ran multi-element spectrographic analyses on five composite samples from hole B-1, and Copper King Mining ran the same on five composite samples from hole C-7 and one sample from hole C-8; results of these spectrographic analyses are reported in Soule (1955).

Skyline Laboratories Inc. and Hazen Research Inc., both of Denver, Colorado, assayed Henrietta samples (Nevin, 1973). The detection limits for the gold and copper assays were 0.005 opt Au (0.171 gpt Au) and possibly 0.001% Cu (Mountain Lake Resources Inc., 1997).

Little information exists regarding Caledonia's drill program other than that drill samples were only assayed for gold (Clarke, 1987).

MDA (2010) found assay certificates for Compass holes CCK-19 and CCK-24 that showed the assays were performed by Barringer Laboratories Inc., in Reno, Nevada, using fire assay with an atomic absorption ("AA") finish for gold and AA for copper. It was not evident from the data reviewed by MDA whether Barringer assayed all of Compass's holes. The detection limits for Compass's assays were 2 ppb gold and 5 ppm copper (Mountain Lake Resources Inc., 1997).

Assaying of the samples for Mountain Lake was performed by Barringer Laboratories Inc. in Reno, Nevada. MDA has seen no assay certificates for Mountain Lake's drill holes but did find a spreadsheet with the assays, which were entered into the database for Mountain Lake's eight drill holes. The detection limits were 2 ppb gold and 5 ppm copper (Mountain Lake Resources Inc., 1997). Metallurgical testing of bulk composite samples from holes MLRM-1 and MLRM-2 was conducted by the Colorado Minerals Research Institute of Golden, Colorado.

8.3 RESULTS, QC PROCEDURES AND QA ACTIONS

8.3.1 U.S. Gold 2021 Campaign

As described above, the data derived from the 2021 drilling program that commenced in August 2021 has not been included in support of the PFS study which relies on 2020 and prior data. The purpose of the 2021 data collection was primarily to support additional geotechnical and hydrological studies. There have been no material observations that would affect the PFS study as written. The 2021 drilling results shown in Table 8.1 have been reviewed in the context of the existing resource and they are not material.

Table 8.1: 2021 Drilling Program Results

Standard	Au (ppm)		Cu (ppm)		Ag (ppm)	
	Expected	StdDev	Expected	StdDev	Expected	StdDev
CDN-BL-10	0.0064	0.0069	29.3511	5.5799	0.0316	0.0124
CDN-CM-19	2.11	0.11	20200	350	2.6414	0.2038
CDN-CM-37	0.171	0.012	2120	60	1.17	0.135
CDN-CM-38	0.942	0.036	6860	160	6	0.2
CDN-CM-47	1.13	0.055	7240	140	69	3
MEG-Au.17.01	0.38	0.015	723	19	6.525	0.203
MEG-SiBlank.17.12	0.0059	0.0164	3.0223	3.234	0.0148	0.0136

8.3.2 U.S. Gold 2017 – 2020

U.S. Gold's QA/QC program implemented for the 2017, 2018, and 2020 drilling campaigns included the analysis of certified reference materials (CRMs), blanks, coarse rejects, and pulp duplicates inserted regularly into the sample stream. A random selection of samples from mineralized intervals was also submitted to an umpire laboratory.

U.S. Gold geologists evaluated the control sample results. When the control samples returned values outside of acceptable limits, the assay laboratory was contacted, and the batch of samples was re-assayed.

Gustavson compiled and reviewed the 2020 control sample results and found assay accuracy and precision acceptable for resource estimation. No significant bias was observed in the gold, copper, or silver CRM results. Check assays showed no significant bias between Bureau Veritas original assays and ALS check assays. No significant carryover contamination was observed in the blank results.

Three standards were used for the 2020 drilling program, CDN-CM-43 and CDN-CM-38 from CDN Resource Laboratories Ltd., and MEG-Au.17.01 and MEG-Au.17.10 from MEG, Inc. The recommended values and standard deviations for Au, Cu, and Ag are found in Table 8.2.

Table 8.2: Sample Standards

Standards	gpt Au	Au_2SD	% Cu	Cu_2SD	gpt Ag	Ag_2SD
CDN-CM-38	0.942	±0.072	0.686	±0.032	6.0	±0.4
CDN-CM-43	0.309	±0.040	0.233	±0.012	-	-
MEG-Au.17.1	0.382	±0.015	0.0723	±0.0019	6.525	±0.203
MEG-Blank.17.10	<0.003	-	0.00015	-	0.9	

A commercial 99% quartz sand standard MEG-Blank.17.10 was used during the 2020 drilling campaign. Results are reasonable, and blank assay results exceed 90% less than two times the detection limit of .005 ppm gold. The blank has a reported average of less than 0.003 gpt. The same blank has a reported average of 1.5 ppm copper and although not a blank, it showed carryover on 5 occasions but well below any economic consideration. Silver was below detection 100% of the time. The blank samples demonstrate that the laboratory has reasonable control over sample cross-contamination.

The duplicate pulp performance of 64 pairs was greater than five times the gold detection limit, exceeding 90% of the pairs within a grade difference of 5%. These results are reasonable.

A subset of 110 randomly selected samples collected during the 2020 drilling campaign were submitted to ALS for umpire assay analysis. The paired Au and Cu data were analyzed and found to agree with the ALS checks. The correlation coefficient (r) of the raw data is 0.97 for Au, Figure 8.1, and 0.997 for Cu, Figure 8.2.

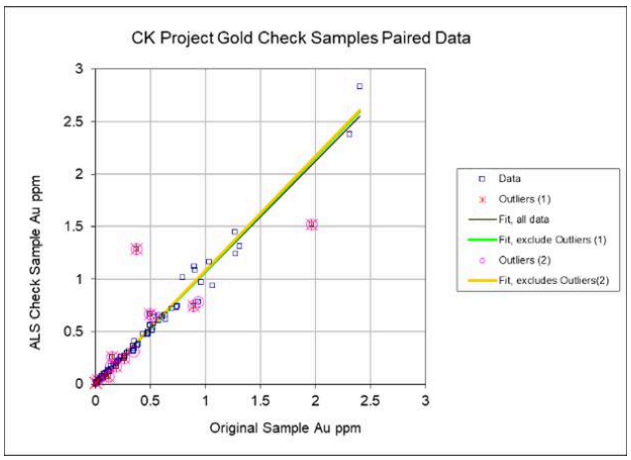


Figure 8.1: Umpire Analysis Au Correlation

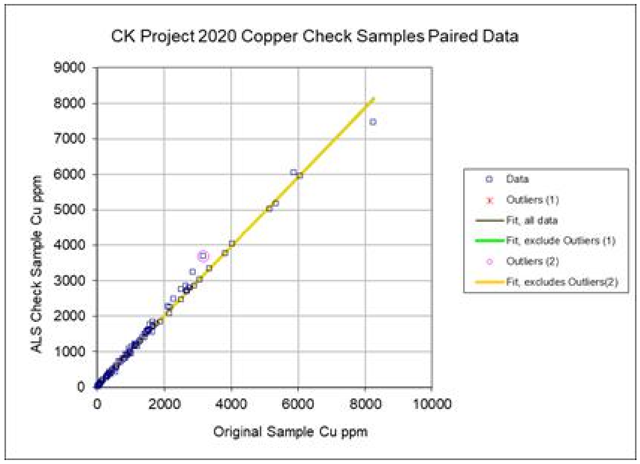


Figure 8.2: Umpire Analysis Cu Correlation

8.3.3 2007 – 2008 Saratoga

Details on QA/QC programs for the 2007 and 2008 drill campaigns can be found in Tietz (2010), Saratoga's QA/QC program implemented for the 2007 and 2008 drilling included 1) analytical standards and blanks inserted into the drill-sample stream, 2) duplicate assaying of selected coarse-reject samples by the primary assay laboratory, and 3) re-assaying of selected original pulps by an umpire laboratory. American Assay was used as the umpire laboratory for the 2007 drill program in which Chemex was the primary laboratory, while the roles were reversed for the 2008 drilling.

A total of 169 standard samples were submitted to Chemex and American Assay. One standard sample was inserted into the stream at an approximate rate of one standard for every 40 drill samples. Standards were also used in the duplicate pulp and pulp re-assay check assay programs at a higher rate, ranging from one standard per 10 to one standard per 25 samples. Five unique analytical standards were used. The standards were inserted into the drill core sample stream with the same sample ID designation, though as pulps, they were not blind to the lab.

Tietz found that the check assay analyses show good agreement between the Chemex duplicate pulp analyses on the original Chemex coarse rejects and between the Chemex pulp re-assays of the original American Assay samples. No significant biases or assay variability issues were found within these data. There are concerns, primarily within the copper analyses, with the December 2009 American Assay pulp duplicate and pulp re-assay check analyses. Further examination and follow-up analytical work is warranted to determine the specific problem within these data, though any resolution of these issues would not materially affect the resource model or stated resource.

8.4 OPINION OF ADEQUACY

The QP believes that the sampling procedures are adequate for mineral estimation purposes and for reporting mineral resources and reserves.

9.0 DATA VERIFICATION

9.1 PROCEDURES

Site visits by the Qualified Persons (QPs) authoring this report were conducted during the 2020–2021 exploration campaign and the Pre-Feasibility Study (PFS) development in 2024. Mark Shutty, CPG, visited the CK Project site and U.S. Gold's logging and sample storage facilities in Cheyenne on July 26–27, 2021, and again on July 11, 2024

During the site visits, the following observations and evaluations were made:

- Mineralization: Oxide copper mineralization was observed in outcropping granodiorite host rocks above the core of the modeled mineralization (Figure 9.1).

- Drilling Operations: Active drilling operations were reviewed in 2021, and monumented drill collars from the 2021 and earlier campaigns were inspected in 2024 (Figure 9.2).

- Geologic Facilities: Logging, sampling, and storage facilities were evaluated to confirm compliance with industry standards. Drill core sampling was conducted using sawn core methods, and storage facilities were found to be secure, well-organized, and inclusive of legacy core from previous operators.

Data Validation

Drill Collar Locations and Surveys:

- Drill collars were professionally surveyed, with elevations cross-checked against a digital terrain model (DTM). Historical collar elevations not conforming to the DTM were adjusted to match the digital surface, validated by observations of site disturbance, monuments, and historical maps.

Downhole Survey Data

- Downhole survey data were visually inspected in 3D and checked for deviations using modeling software tools. Errors were flagged for review and correction. A redundant true north correction factor (±7.0° E declination) was identified in 2021 downhole survey data but had minimal impact on the Inferred resources located below the constraining resource optimization pit and no impact on the Measured and Indicated resources located within the Reserves pit limit.

Quality Assurance/Quality Control (QA/QC):

- QA/QC procedures (see Section 8) ensured the reliability of analytical data. Control samples, including blanks, duplicates, and standard reference materials, were appropriately selected and used at suitable frequencies. Their performance was evaluated using statistical methods to ensure quality sample handling and analytical data accuracy.

- Digital analytical record handling was utilized in modern drilling campaigns, minimizing errors during data transfer from laboratories to the drill hole database and modeling systems.

Logging and Database Management

Logging Procedures:

- Comprehensive logging captured attributes required for modeling geology, oxidation, and structures. These data were securely stored in a detailed project database integrating historical and modern drilling information.

Database Compilation:

- U.S. Gold compiled a comprehensive Access database to preserve data quality while facilitating digital verification and analysis. Drill traces, logged geology, and assay data were independently reviewed in 3D.

Resource Dataset Overview

The CK Project drilling datasets, compiled over several decades, originated from multiple operators employing varied drilling, sampling, and analytical methods.

- Modern Era Drilling: Data generated since 2007 from U.S. Gold and Saratoga represent the most robust datasets, supported by comprehensive QA/QC protocols, digital analytical records, and thorough documentation. These datasets support most of the CK Project's Measured and Indicated Resources.

- Metal Distribution Validation: The deposit's well-defined metal distribution, with gradational Au, Cu, and Ag zonation in granodiorite host rocks, enables results compliance checking for modern and historical datasets.

Independent Verification

The QP independently verified analytical data by:

1. Reviewing and cross-checking unit conversions (e.g., ppm to opt for Au and Ag assays and ppm to percentage for Cu).

2. Calculating the AuEq variable for use in modeling and resource reporting.

3. Evaluating global and local metal grades by drill type for bias:

   - Diamond Core: 63% of resource drilling, well-dispersed across the deposit.

   - RC Drilling: 35%, primarily defining lower-grade margins.

   - Rotary Drilling: <2%, focused on the core of the deposit.

Figure 9.1: Oxide copper mineralization in outcropping granodiorite host rocks (2024).

Figure 9.2: U.S. Gold's CK21-11c drilling in-progress on July 11, 2021.

Observations and Compliance

- Surface disturbances from historical drill pads and access trails are well-preserved and/or have been reclaimed.
- Verification samples were not collected, but observed drilling, sampling, and data handling procedures were consistent with industry standards.
- Drill core recovery is excellent, as evident from core photographs, archived samples, and digital logs. Hard Rock Consulting re-logged core from the 2017–2018 campaigns, further validating geological observations.

In summary, the QPs confirm that the datasets used in the CK Project's Mineral Resource Estimate (MRE) meet industry standards for quality and reliability, providing a solid basis for resource modeling and reporting.

9.2 PREVIOUS AUDITS / OWNERS

9.2.1 Saratoga 2007 – 2008

Data verification of exploration activities before 2007 is not well documented, and there is no independent verification of the exploration, sampling, or laboratory procedures.

Drilling data from the 2007-2008 Saratoga drill programs was directly input from sources. Saratoga provided the original collar survey data files and the downhole survey driller's notebooks, while the assay data were digital data direct from the laboratories. After compiling, these data were audited against the sources by randomly checking values and specifically checking downhole survey data that appeared anomalous. Six individual down-hole surveys were removed from the database due to uncertain depths or atypical azimuth values. In all cases, the atypical azimuth values coincided with anomalously high magnetic field readings.

9.2.2 Historical Drilling

There was virtually no original historical data available to audit the database. Gustavson verified the drill-hole locations and values of those samples from ASARCO's holes A-1 through A-5, Copper King's holes C-6 through C-11, and the USBM's holes B-1 through B-3 by crosschecking values in the database with those reported in Soule (1955), but no original assay certificates were available for these or any other drill holes except Compass's holes CCK-19 and the cored portion of CCK-24. Gustavson verified the assay values in the database for Compass's holes CCK-19 and CCK-24 by crosschecking the values in the database with those shown on the assay certificates, and no errors were found. Gustavson verified gold values for the best gold intercepts in the holes drilled by Henrietta by crosschecking assays included on geologic logs against values in the database. Gustavson found spreadsheets with assays from Barringer Labs for Mountain Lake's eight drill holes and confirmed their values in the drilling database.

In 1996, Mountain Lake ran check assays on selected mineralized intervals from 12 of Compass's holes. The check analyses were conducted by Barringer Laboratories, Inc. Gold was analyzed by fire assay with an AA finish, and copper was analyzed by AA. A preliminary evaluation of the Mountain Lake check assay results by MDA in 2006 indicated general agreement between the original and the check assay Au values. The mean grades of gold and copper for the original and check assays are as follows: 3.46 gpt Au and 0.465% Cu and 3.29 gpt Au and 0.570% Cu, respectively. The absolute percent difference between the 185 check assays and originals averaged 16%, with a standard deviation of those absolute differences of 29%. Of the 20 check sample assays that showed a 30% (one standard deviation) or greater difference from the original assay, 14 were in the lower half of the grade range (<3.36 gpt Au), indicating greater variability within the lower-grade mineralization. In non-absolute terms, the average difference between the check and original assays was -1%.

9.3 DATA ADEQUACY

The QP considers that the drill data are generally adequate for resource estimation. There are no additional limitations to the exploration data, analysis, or exploration database for use in resource modeling and declaration of mineral resources and reserves.

10.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Several metallurgical testwork programs have been completed on multiple samples of mineralization from the CK Gold Project. The work dates back to 2008 when Saratoga Gold Company (Saratoga) first contracted SGS Lakefield (SGS) to perform fundamental characterization work and scoping-level separation tests (flotation and cyanide leaching) on several composites of sulfide and oxide mineralization.

No further work was completed until 2020 when U.S. Gold commenced a drilling program that included several holes designed to generate sufficient sample material for a metallurgical test work update. The metallurgical program that followed commenced in December 2020 at Kappes, Cassiday and Associates (KCA) Laboratory in Reno, Nevada before it transitioned over to Base Metals Laboratory (BML) in Kamloops, Canada. Several metallurgical programs have been completed at BML, including further flotation characterization, grindability, mineralogy, and dewatering.

Although the QP for this section was not directly involved with historical work, the reports were reviewed, and the conclusions were generally concurred with.

While the recent PFS incorporated process plant designs based on early SGS test work and 2021 BML results, this PFS also includes more recent work from BML.

The various testwork programs are described in the following section in chronological order.

10.1 SGS TESTWORK, 2008 - 2010

10.1.1 Program 11868-001 (2008 – 2009)

A preliminary metallurgical program was initiated by Saratoga in 2008, and this covered grindability, mineralogy, flotation testing, and environmental testwork on a master composite and four variability composites. Composite head analysis is summarized in Table 10.1 below. Comp 1 represents the oxide material that overlies the deposit, Comp 2 represents the deposit's relatively small but higher-grade core, and Comp 3 and 4 represent the east and west zones of the unoxidized volume within the deposit.

A “Master Composite” was compiled at SGS for flowsheet development by blending equal portions of Comp 2, Comp 3, and Comp 4 composite material. This Master Composite did not include material from the Comp 1 (Oxide) composite.

Table 10.1: SGS Composite Head Assay

Description	% Cu_T	% Cu_{CN}	gpt Au	gpt Ag	%S
Master Composite	0.28	<0.002	1.41	<10	0.25
Comp 1 (Oxide)	0.26	0.002	1.00	<10	0.02
Comp 2 (Mixed)	0.39	<0.002	1.96	<10	0.21
Comp 3 (Sulphide East)	0.22	<0.002	0.62	<10	0.21
Comp 4 (Sulphide West)	0.19	<0.002	0.56	<10	0.34

An initial grindability study included Bond rod (RWI) and ball mill (BWI) tests for the Master Composite, and Bond ball mill tests for the four variability composites. A Bond rod mill work index of 16.0 kWh/t (metric) was reported, along with a range of Bond ball mill work indices from 13.0 to 14.8 kWh/t (metric). The results point to a material that is slightly harder than average, compared to the population of results in SGS's database.

A QEMScan mineralogical program provided bulk mineralogy for each composite and identified several copper minerals across the sample set. Chalcopyrite dominated, with a range of secondary copper minerals (mainly chalcocite) also noted. No native copper was identified, and very low levels of pyrite were measured. Host minerals included feldspar (roughly 45%), quartz (roughly 25%), and micas (roughly 14%) with other oxides and clays making up the balance. Chlorites made up roughly 4-5% of each composite.

Flotation testing focused on the Master Composite. The work highlighted a general improvement in metallurgical performance at finer grinds (142 µm, 112 µm, 87 µm, and 65 µm were tested) although the test at 80% passing 65 µm did appear to suffer from the effects of lower mass pull. SGS metallurgists concluded that a primary grind of 80-90 µm was preferred for the remaining work.

Raising pulp pH with lime improved copper performance but had a very slight negative impact on gold recovery.

Early batch cleaner tests highlighted the need for a rougher concentrate regrind, and an initial study on the Master Composite suggested that a regrind target of approximately 80% passing 20 µm would be close to optimum.

An assessment of gangue depressant and/or dispersant reagents was completed, and it was concluded that these were unlikely to improve metallurgical performance.

Locked cycle testing (LCT) of the Master Composite used a conventional SGS flowsheet with rougher concentrate regrind, three-stage counter-current cleaning with cleaner one scavenging, and cleaner scavenger concentrate recycled back to the regrind mill. Two initial tests were completed at coarse grinds (80% passing 110µm), giving relatively inferior results. A third LCT was completed at a finer grind (80% passing 83 µm), showing a distinct copper and gold performance improvement. The third LCT concentrate graded 26% Cu and 89.7 gpt Au with overall recoveries of 77% Cu and 68% Au.

The SGS metallurgists performed an initial study of final concentrate Cu grade vs. overall Cu and Au recovery, concluding that a higher mass pull to concentrate could result in a Cu grade drop from 26% to 21% Cu, with an associated 1% increase in Cu and Au recovery.

A variability flotation program tested the response of Comp 2, Comp 3, and Comp 4 material to the Master Composite flowsheet and gave results that were generally in line with the Master Composite's performance.

An initial environmental testwork program was conducted on a flotation tailings sample taken from LCT-2 on Master Composite material. An acid-base accounting (ABA) test noted that acid generation would be highly unlikely given the sample's negligible sulfide content. A net acid generation (NAG) test determined the net acid generation potential to be zero, meaning that no acid was produced during the test. The tailing sample represented a very low acid generation and/or metal leaching risk.

10.1.2 Program 11868-002 (2010)

Saratoga approached SGS for a follow-up metallurgical program in the summer of 2010. The work was conducted on the Comp 1 (Oxide) material from the previous program with the objective of developing a flotation flowsheet for copper and gold recovery.

A variety of flotation activators and collectors were tested, and a locked cycle test was completed using the sulfide flowsheet as developed in the previous program. The testing was generally positive, with an acceptable recovery of gold (54.8%) despite a low copper recovery (8%). The copper concentrate grade was reasonable at 15.3% Cu, and the gold grade was excellent at 384 gpt Au. The performance was considered acceptable for preliminary inclusion of this material in the overall mine/processing plan should a sulfide mill be the selected process flowsheet for the Project.

10.2 KAPPES CASSIDAY TESTWORK, 2020-21

After a change in ownership from Saratoga to U.S. Gold, a new metallurgical program was commissioned in 2020 with the following objectives:

- Confirm the 2008/2010 SGS results using samples from a new drilling campaign.
- To develop a flotation flowsheet to improve the SGS results (specifically gold and copper recovery and concentrate grade) for the oxide and sulfide zones.
- To complete sufficient work to support PFS-level process engineering and to increase overall confidence in the results. The overall work for this program included:
  - Quantitative mineralogy to better characterize the deposit, especially the non-sulfide minerals and native copper, and provide gold deportment information.
  - Optimization of the primary grind and re-grind.
  - A more thorough investigation of flotation conditions and reagents.
- Variability testwork is used to ascertain the impact of depth, area, lithology, and grade.
- A more detailed evaluation of gravity recovery. The SGS test work was not successful in producing a gravity concentrate, although the report concluded that this required further investigation. Observation of the new core showed significant visually observable native copper in the high-grade oxide, and the recovery of this might justify the addition of a gravity circuit to the flowsheet.

Details of the metallurgical composites used for this work are given in the following section, but the objective at the onset was to develop the overall characterization of average-grade oxide and sulfide mineralization, and the composites reflect this. In addition, a High-Grade Oxide composite (similar to SGS “Comp 1”) was included in the scope for comparison with previous studies. This “Hole 4” composite was prepared using shallow samples (less than 80 ft depth) from CK20-04cA&B where a centrally located oxide zone was intercepted with average grades of 5.1 gpt Au, 0.98% Cu and less than 0.1% S, (assays of individual core sections). Below 80 ft, the gold and copper grades remained high in this area, but the sulfur grade increased to an average of 0.5% S.

The initial test work on the Hole 4 Oxide composite produced recoveries and concentrate grades that exceeded expectations based on historical SGS results. However, during April 2021 it became apparent that KCA were unable to reproduce the SGS results on the main Oxide and Sulfide composites. As a result, a second test program was initiated at BML in Kamloops, Canada (described in Section 10.3 below). BML was immediately able to duplicate and later improve upon the SGS results.

10.2.1 Sampling

During 2020, U.S. Gold carried out a major exploration drilling campaign that included seven holes drilled specifically to provide core for metallurgical test work. These metallurgical holes provided over 4,600 ft of mineralized core consisting of 1,100 sample intervals. The plan view location and orientation of the seven metallurgical holes is illustrated in Figure 10.1 below.

The metallurgical core was cut, prepared, and assayed at Bureau Veritas Analytical in Reno. One half was used for the assay, and the other half contributed to the preparation of metallurgical composite samples.

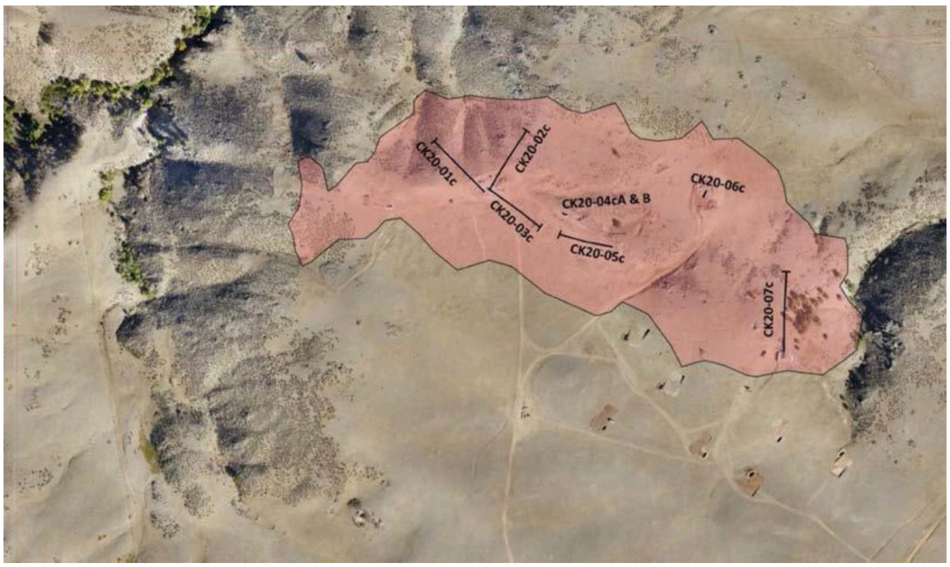


Figure 10.1: Location of Metallurgical Holes, highlighted area represents the approximate mineralized area

The three metallurgical composites are described below in Table 10.2, with names, masses, and head assays listed for reference.

Table 10.2: FLSmidth Mineralogical Analysis: Copper Deportment							
Ref	**Description**	**Mass, kg**	**% Cu_T**	**gpt Au**	**gpt Ag**	**%Fe**	**%S**
90104A	High-grade oxide, Upper Zone ("Hole 4")	203	0.99	4.88	4.83	6.42	0.02
90150B	Overall Oxide Zone, holes 1-3 and 5-7	235	0.28	1.14	2.10	3.59	<0.01
90151B	Overall Sulphide Zone, holes 1-7*	372	0.27	0.963	1.61	3.62	0.21

* Note: This composite included the small amount of material identified as "mixed " between the oxide and sulfide zones.

Visual inspection of the high-grade oxide Hole 4 core revealed that a significant proportion of the contained copper was native copper, much of which was coarse-grained. Thus, the testing program on Hole 4 commenced with both gravity and flotation in the flowsheet.

High Grade ("Hole 4") Composite

The Hole 4 Composite was prepared using material from three sources (described in detail within the KCA Report):

- 43.5 kg of crushed, blended split core.
- 92.5 kg of assay rejects from hole CK20-04cA.
- 67.7 kg of assay rejects from hole CK20-04cB.

Overall Oxide Composite

Samples were selected from six holes to make up an Overall Oxide composite. All the samples had sulfur assays less than 0.1% S. Gold grades ranged between 0.5 and 1.5 gpt Au. Copper grades ranged between 0.2 and 0.5% Cu. The average grade of this composite was 1.14 gpt Au and 0.28% Cu.

Sulfide Composite

Samples were collected from all seven holes to make up an Overall Sulfide composite. These samples had sulfur assays of more than 0.1% S and generally over 0.2% S. Gold grades ranged from 0.5 gpt to 1.5 gpt Au. Copper grades ranged from 0.25% Cu to 0.8% Cu. The average grade of the composite was 1.1 gpt Au and 0.3 % Cu.

Variability Samples

In addition to the overall composites described above, 24 oxide and 50 sulfide variability samples were selected, representing different grades, depths, and lithologies. Testing of the oxide variability samples commenced during the third quarter of 2021, while the sulfide samples were subsequently transferred to BML.

10.2.2 Mineralogy

An initial quantitative mineralogy (QEMScan) program was carried out at FLSmidth in Salt Lake City on samples of flotation feed and tailings from the high-grade oxide composite. The mineralogy indicates the probable limits for copper recovery and the need for fine primary and re-grinding.

Table 10.3: FLSmidth Mineralogical Analysis: Copper Deportment			
Description	**Recovery Potential**	**Oxide Head**	**90131 Tails (G+F)**
Native Copper	Y	0.346	0.001
Cuprite	Y	0.012	0.000
Chalcopyrite	Y	0.086	0.001
Bornite	Y	0.041	0.000
Chalcocite	Y	0.198	0.003
Covellite	Y	0.004	0.000
Cu/As/Sb Sulfides	Y	0.002	0.000
Cu-bearing clays	N	0.024	0.022
Cu/Chlorite	N	0.005	0.007
Cu/Biotite	N	0.004	0.003
Cu/Muscovite	N	0.009	0.007
Cu Wad	N	0.001	0.001
Fe Oxides	N	0.158	0.174
Fe Oxide / Chrysocolla	N	0.018	0.025
Chrysocolla	N	0.179	0.192
Other Cu	N	0.010	0.009

The data also illustrates that for oxide zones within the deposit, the best copper recovery by gravity and flotation combined would be about 60%, which is close to the actual test results.

In contrast to the initial SGS mineralogical assessment, the FLSmidth work also helped develop an understanding of gold and silver (electrum) mineralogy to some extent. Of note:

- Gold appears very fine-grained, most less than 10-20 µm.
- Gold is quite well liberated and is primarily not associated with copper minerals but located on grain boundaries, as gold or electrum.
- Gold association with pyrite appears minor.

With a relatively low pyrite content and the presence of acid consumers such as calcite, biotite, and chlorite noted in the samples, the tailings from this project are not expected to generate acid, confirming the initial environmental work by SGS.

10.2.3 Comminution

Sub-samples of half-core were selected from the metallurgical composite crushing/blending process at KCA and shipped to Hazen Research for comminution testing in early 2021. The Hazen work included SAG mill comminution (SMC), Bond ball mill work index (BWi) testing, and Bond abrasion index (Ai) testing. Results are summarized in Table 10.4 below.

Table 10.4: Comminution Test Work Results							
Ref	**Description**	**SG**	**BWi (kWh/t)***	**Ai (g)**	**Axb**	**t_a**	**SCSE (kWh/t)***
55432-1	High-grade oxide, Upper Zone, Hole 4	2.66	14.0	0.2008	37.5	0.36	10.1
55432-2	Overall Oxide Zone, holes 1-3 and 5-7	2.67	14.6	0.3430	33.3	0.32	10.7
55432-3	Overall Sulphide Zone, holes 1-7	2.71	15.1	0.4033	27.8	0.27	11.8

* Note: data is based on metric tonnes.

The A x b results (drop weight test results) give an indication of resistance to impact breakage and in this case, show that the Sulfide composite is slightly more resistant than the Oxide composites. The Sulfide composite is slightly more dense, more abrasive and more competent at the finer grind sizes also.

10.2.4 Gravity Concentration

One of the opportunities identified by SGS was the addition of a gravity circuit to the flowsheet, especially in the oxide zones, where significant native copper had been confirmed visually. Gravity tests using a bench-scale Knelson concentrator were scheduled on 40 lb samples of each composite.

The gravity tests on the overall Sulfide and Oxide composites were unremarkable, with low gold and copper recoveries and no obvious improvement opportunity. For the Hole 4 Composite (high-grade oxide); however, a similar test produced a gravity concentrate with a weight recovery of 1.6%, containing 51.5 gpt Au and 14.6% Cu, with recoveries of 15.4% Au and 22.7% Cu.

The tailing solids from the Hole 4 gravity test were stored for later flotation test work and overall circuit (gravity + flotation) performance comparisons.

In general, the flotation testwork was carried out on the gravity tailings to determine if the inclusion of a gravity circuit before flotation would provide better recoveries than by flotation alone. The results of this work are summarized in the below Table 10.5.

Table 10.5: Hole 4 Gravity + Flotation vs. Flotation Only, (KCA)		
Parameter	**Gravity + Flotation**	**Flotation Only**
Gravity concentrate grade	15.5 gpt Au 14.6% Cu	-
Recovery to gravity concentrate	15.4% Au 22.7% Cu	-
Overall flowsheet recovery: Gold Copper	70% 60%	70% 57%

The gravity test yielded recoveries of 15.4% gold and 22.7% copper. This was thought to generate higher recoveries for a gravity + flotation circuit. However, the gold recovery (at 70%) was the same. It is concluded that gold recovered by gravity would be recovered in the flotation circuit. The increase in copper recovery was 3%.

10.2.5 Flotation

Over 50 rougher flotation tests were carried out to investigate key flotation parameters (grind, reagents, pH, sulfidization, etc.) for each of the three composites. All lab flotation tests were completed on 2-kg test charges. The testwork is summarized in the following subsections and detailed in the KCA Report, "Copper King Test work for U.S. Gold," dated July 2021.

Hole 4

32 tests were completed for the Hole 4 Composite (90104). The best results were achieved in Test 90134, and this is summarized in Table 10.6. These conditions were carried into the cleaner flotation test program.

Table 10.6: Rougher Flotation, Test 90134 (Hole 4)

Parameter	Value
p_{80}	106 µm
pH	9.0
CaO, gpt	153
NaSH	n/a
F507 (frother), gpt	31
PAX, gpt	75
407, gpt	50
wt to rougher, concentrate, %	7.5
Gold Recovery, %	70
Silver Recovery, %	50
Copper Recovery, %	57

Overall Oxide

10 tests were completed for the Overall Oxide Composite (90150), investigating similar parameters to the Hole 4 work above. The best results were achieved in Test 90170, and this is summarized in Table 10.7.

Table 10.7: Rougher Flotation, Test 90170 (Oxide)

Parameter	Value
p_{80}	86 µm
pH	9.0
CaO, gpt	130
Frother, gpt	56
Collector, PAX; 407 gpt	76; 50
wt to rougher concentrate, %	7.0
Gold Recovery, %	61
Silver Recovery, %	18
Copper Recovery, %	21

The relatively low recovery of copper directly reflects the copper mineralogy (i.e., a high content of non-floating copper minerals such as chrysocolla).

Overall Sulfide

Twenty rougher flotation tests were carried out on the overall Sulfide Composite (90151) investigating P_{80}, pH, reagent addition and types. The best result was achieved in Test 90173, and this is summarized in the below Table 10.8.

Table 10.8: Rougher Flotation, Test 90173 (Sulfide)	
Parameter	**Value**
p_{80}	86 µm
pH	9.0
CaO, gpt	75
Frother, gpt	50
Collector, PAX; 407 gpt	51
wt to rougher concentrate, %	11.7
Gold Recovery, %	74
Silver Recovery, %	61
Copper Recovery, %	94

The relatively high copper recovery reflects the more favorable mineralogy (i.e., mainly chalcopyrite) described in the FLSmidth report.

10.2.6 Cleaner Flotation

Batch cleaner flotation tests on each composite used the optimized rougher flotation conditions achieved in the rougher flotation program discussed above.

Hole 4

A total of 13 cleaner tests were carried out, investigating the regrind P_{80} and a variety of reagents and addition rates. The best result was obtained in Test 90160, which was repeated for confirmation. The results are shown in Table 10.9.

Table 10.9: Cleaner Flotation, Test 90160 (Hole 4)	
Parameter	**Test 1**
Grind P_{80}, Primary Grind & Regrind	86/20 µm
pH	9.0
Total CaO, gpt	172
Total PAX, gpt	76
Total F-507, gpt	51
Concentrate Mass Pull, %	2.0
Concentrate Grade, % Cu	25.3
Concentrate Grade, gpt Au	186
Concentrate Grade, gpt Ag	90
Recovery Cu, %	53
Recovery Au, %	68
Recovery Ag, %	35

The rougher reagent suite used in this test is shown in Table 10.6. Reagents such as CMC, sodium silicate (Na2SiO3), and NaSH were not found to be of benefit. The only reagent used in cleaning was a small frother addition.

Overall Oxide

18 cleaner tests were carried out, first with no regrind, then with regrind P_{80} of 20 µm. These tests also investigated cleaner pH and various reagent suites, particularly gangue depressants. They were carried out throughout April 2021 to produce a saleable concentrate grade without unduly sacrificing recovery. These cleaner tests were unsuccessful, and the best result is summarized in Table 10.10. Subsequently, it was established that the collector additions to the rougher flotation were too high, leading to overpromotion.

Table 10.10: Oxide Composite Cleaner Flotation (KCA)	
Parameter	**Test 1**
Grind P_{80}, Primary Grind & Regrind	90/20 µm
Total PAX, gpt	14
Total 208, gpt	16
Concentrate Grade, % Cu	8
Concentrate Grade, gpt Au	188
Concentrate Grade, gpt Ag	87
Recovery Cu, %	7
Recovery Au, %	48
Recovery Ag, %	12

KCA decreased the collector addition, and a performance improvement was immediately realized. This reduced collector in the rougher circuit eliminated the need for depressants and/or dispersants.

At the end of April 2021, the highest copper concentrate grade achieved in the cleaner tests at KCA for Oxide was 8% Cu, significantly less than the 15% Cu achieved at SGS. The best grade for sulfide was less than 20% Cu. As a result, U.S. Gold sent 30 kg of all three composites to BML for cleaner flotation and Locked Cycle tests. This work commenced in the last week of April 2021.

Overall Sulfide

28 batch cleaner tests were carried out on the overall Sulfide composite to investigate the regrind P80, pH, and reagents. In the initial program, copper recovery to the cleaner concentrates was reasonable, but a commercial concentrate grade was difficult to achieve. The best cleaner result for the Overall Sulfide composite is shown in Table 10.11.

Table 10.11: Cleaner Flotation, (KCA)	
Parameter	**Value**
P_{80} Regrind	20 µm
pH	11.0
Concentrate Grade, % Cu	13.5
Concentrate Grade, gpt Au	34.6
Concentrate Grade, gpt Ag	55.0
%Cu Recovery	81%
%Au	62%
%Ag	74%

As a result, 40 kg of Sulfide composite material was sent to BML, Kamloops, for comparative testing. The results of the BML work are provided in Section 10.3.

KCA subsequently repeated this test using reduced collector addition (PAX, AF208 and 3418) and achieved a copper concentrate grade of 23% Cu, with recoveries of 83%, 64%, and 50% for copper, gold, and silver, respectively.

10.2.7 Locked Cycle Testing

Using the results of Cleaner Test 90160 as a guide, a single LCT was carried out on the Hole 4 Composite, with cleaner tails products recirculated counter-currently throughout the test.

The LCT could not produce a final copper concentrate of even 15% Cu, and the grade deterioration as the test progressed indicates that the test had not reached a stable state. Further analysis of test results suggested that the most likely reason for this LCT's failure was the addition of excessive collector reagent, resulting in over-promotion and a subsequent inability to reject low-grade middling in the cleaner circuit.

As a result, replicate Hole 4 Composite samples were shipped to BML in Kamloops, Canada for comparative rougher, cleaner and locked cycle testing. The BML testing achieved concentrate grades more than 30% copper, containing over 500 gpt Au and 300 gpt Ag. These are discussed in Section 10.3.4. BML generally uses 20-25% of the collector dosage used at KCA.

As a result, samples were transferred to BML for the remainder of the test work program.

10.2.8 Cyanidation on Flotation Tailing

Two 24-hour cyanidation tests were conducted on the Test 90139 flotation tailings at different cyanide strengths. These resulted in around 70% extraction of gold, with relatively low reagent consumption, as shown in Table 10.12.

Table 10.12: Cyanidation of Flotation Tailings

Test	P_{80}	NaCN gpL	Head gpt Au	Leach Time (h)	NaCN Cons. Kg/t	$Ca(OH)_2$ Kg/t	Extraction %
90139A	87	1.0	1.92	24	0.88	0.60	69
90139B	85	1.0	1.56	24	1.06	0.60	70
90139C	87	5.0	1.92	24	1.46	0.60	64
90139D	85	5.0	1.56	24	1.67	0.60	73

10.2.9 Tailing Thickening/Filtration

Samples of flotation tailing solids and solution from the Hole 4 locked cycle flotation test program were shipped to Pocock Industrial Inc. in Salt Lake City. Pocock's scope of work was to investigate flocculants, gravity sedimentation, pulp rheology, vacuum filtration, and pressure filtration. The objective of the test work was to provide data that could be used to assist in the selection and sizing of the tailing's thickener and filters.

Pocock conducted a size fraction analysis of the Hole 4 flotation tailings and established the P_{80} to be 65 µm. This is much finer than the primary grind used at KCA, which was 86 µm, but it is explained to some extent by the inclusion of the reground cleaner tailings.

Initial work focused on screening potential flocculant types. A medium/high molecular weight anionic polyacrylamide was selected based on overall performance, including overflow clarity, decantation rate, and underflow slurry viscosity characteristics. Two test methods were subsequently used to characterize the settling/thickening performance, namely static tests in 2L cylinders and dynamic tests in a bench-scale continuous unit. Pocock concluded that a conservatively sized high-rate thickener, using 55-60 gpt flocculant, with a heavy-duty rake mechanism and adequate feedwell dilution would be appropriate for CK, producing an underflow slurry density of up to 62% solids.

The apparent viscosity of underflow slurry collected from dynamic settling tests was measured across a range of solids concentrations and shear rates, confirming the maximum underflow density limitation of 62%.

Pocock investigated both vacuum and pressure filtration. The vacuum tests produced filter cakes with over 20% moisture at rates of 400-500 kg/m^2.hr. The pressure filtration tests achieved cakes with 12.8% moisture at rates over 2,000 kg/m^2.hr; on this basis, plate and frame filters (incorporating membrane squeeze) are recommended for CK.

10.3 BASE METALLURGICAL LABS (BL-0789, 2021)

The BL-0789 program at BML commenced in April 2021, when a shipment of ½ core oxide material was received from the KCA program described above. Subsequently, four sulfide and oxide material shipments were shipped as the metallurgical program developed.

This short program was intended to provide an initial comparison to the KCA flotation results; therefore, it excludes mineralogy or comminution programs.

10.3.1 Sampling

Samples were received in five shipments between April 10 and June 30, 2021, as summarized in Table 10.13.

Table 10.13: BL-0789 Composites

Shipment	Contents
1	12 samples of ½ core weighing 27.8 kilograms combined to make the Oxide Composite (High Grade).
2	16 samples in the form of ¼ core, weighing 55.3 kilograms. Combined with 23.2 kg of KCA 90151A (Shipment 3) to make the Sulfide Composite.
3	KCA Sample 90151A (Sulfide Composite) – 23.2 kilograms. KCA Sample 90150 (Oxide Composite) – 22.7 kilograms.
4	6 samples of ½ HQ Core, weighing a total of 20.9 kilograms. Combined with 22.6 kilograms of KCA 90150A (Shipment 3) to make Oxide Composite 2.
5	24 samples of ½ HQ Core, weighing 75.9 kilograms, combined to form Sulfide Composite 2.

Upon constructing the composites, the contents of each were stage crushed to pass 3.35 mm (6 mesh) and split into 2 kg test charges in preparation testing. Only 48 kg of the material shipped for Sulfide Composite 2 was prepared into test charges, with the remaining material kept and stored for future use.

It is worth noting that the original Sulfide Composite included some minor "mixed zone" material. As a result, between 10 and 15% of the copper minerals in this composite were non-sulfide (principally chrysocolla), which had a detrimental impact on the copper recovery. The second Sulfide Composite (Sulfide Comp 2) was prepared later in the program to rectify this issue and avoid core samples from or near the mixed zone.

The names and chemical composition of the four composites tested in the BL-0789 program are listed in Table 10.14 below.

Table 10.14: 2020 Metallurgical Composites Description

#	Description	% Cu_T	%CuOx	%CuCN	gpt Au	gpt Ag	%Fe	%S
1	Oxide Comp	1.15	0.43	0.15	5.95	3	6.6	0.06
2	Oxide Comp 2	0.31	0.17	0.02	1.36	1	3.7	0.04
3	Sulphide Comp	0.27	0.04	0.06	1.13	1	3.1	0.33
4	Sulphide Comp 2	0.35	0.004	0.02	0.92	1	3.3	0.47

The analysis of copper deportment provided by the sequential copper assays is instructive: the oxide and cyanide soluble copper assay as a fraction of the total copper assay is high in all but the Sulfide Comp 2 composite. This is expected in the oxide composites but has implications for the original Sulfide Comp (#3), which has only 63% of the total copper assay in recoverable (primary sulfide) form. On reflection, this composite might be more appropriately named “Mixed Comp,” with 10 and 15% of the copper content in oxide form. Performance expectations for the Sulfide Comp should be moderated relative to the Sulfide Comp 2, despite similar copper grades.

Sulphur values were relatively low for the oxide composites, particularly relative to copper, indicating the presence of only minor levels of sulfide minerals, namely pyrite.

10.3.2 Rougher Flotation

An initial set of 8 rougher flotation tests was completed on the Sulfide Composite as part of an investigation into the recent work at KCA. The tests evaluated different primary grinds and reagent recipes, including alternate collectors, sulfidizing agents, activators, and promotors. All tests used 2-kg test charges.

These preliminary tests gave results that were equal to the SGS results and significantly better than those achieved at KCA. Copper recovery to a rougher concentrate varied between 76.3 and 80.0%, whilst gold recovery into this concentrate ranged between 72.1% and 75.9%. Rougher concentrate mass pull varied between 5.3% and 7.8%.

At the same primary grind, many chemical additives had little impact on metallurgical performance. The copper and gold-specific collectors showed some promise at achieving higher overall recoveries, but generally at the cost of higher mass recovery.

10.3.3 Cleaner Flotation

A limited number of batch cleaner flotation tests were carried out on the four main composites, primarily to provide comparative data to the ongoing KCA flotation program. For these 2-kg tests, BML worked with a 90 µm primary grind and reduced the collector additions to "starvation" levels as compared to the KCA tests. This increased the concentrate grade to over 60% copper for the Oxide Composite and generated reasonable (>20%) copper grades for the other three. Results for the cleaner flotation tests are summarized in Table 10.15.

Table 10.15: Batch Cleaner Test Results

Composite	Cleaner Concentrate Grade			Cleaner Concentrate Recovery		
	% Cu	gpt Au	gpt Ag	Cu %	Au %	Ag %
Oxide Comp	62.2	1416	Not reported	12.9	49.9	Not reported
Oxide Comp 2	25.3 13.2	1232 393		4.8 6.2	50.2 43.5	
Sulfide Comp	30.2 19.9	110 65.2		64.2 69.0	55.2 59.1	
Sulfide Comp 2	23.1	61.9		83.9	66.5	

The test conditions noted to give superior results in the batch cleaner tests were subsequently carried through to the locked cycle program.

10.3.4 Locked Cycle Testing

Seven locked cycle tests examined the performance of each composite using batch test conditions, but with recycled slurry from the intermediate streams. 2-kg test charges were utilized for the work and the primary grind in all cases was 90 µm. Test conditions and results for each composite are summarized in Table 10.16 and Table 10.17. Note that PAX, AF208, PFSDB and PF7150 are all sulfide collectors.

Table 10.16: Locked Cycle Test Conditions

Composite	Regrind p_{80}	Lime gpt	PAX gpt	AF208 gpt	PFSDB	7150	MIBC	H57
Oxide Comp	26 µm	680	55	30	-	-	28	30
Oxide Comp 2	21 µm	310	51	26	-	-	28	60
Sulphide Comp	21 µm	465	8	8	-	-	56	30
Sulphide Comp 2	26 µm	305	-	-	3.5	3.5	105	10

Table 10.17: Locked Cycle Test Results - Master Composites						
Composite	**Final Concentrate Grade**			**Final Concentrate Recovery**		
	% Cu	**gpt Au**	**gpt Ag**	**Cu %**	**Au %**	**Ag %**
Oxide Comp	63.4	587	359	39	61	70
Oxide Comp 2	7.9	347	194	6	59	46
Sulphide Comp	25.0	76	82	75	66	47
Sulphide Comp 2	21.3	42	60	88	75	60

Generally, good concentrate copper grades were achieved, with a range of recoveries primarily dependent upon the copper mineral mix (i.e., CuOx:CuT). The original sulfide and oxide composites (i.e., matching those tested at KCA) performed very differently from the KCA work, with better results in most respects. The results also show that high recoveries of copper, gold, and silver can be achieved with "sulfide" material containing only minor "non-sulfide" minerals. These results also help to confirm the 90 µm primary grind. Collector addition is exceptionally low, in line with the low sulfide head grade. However, the high frother addition is to be noted.

10.3.5 Ancillary Testing

Gravity Recovery

Gravity recovery tests using a lab scale Kelson Concentrator and shaking table (referred to as "Pan") were carried out on LCT tailing samples from the Oxide Comp, Oxide Comp 2, and the Sulfide Comp.

Results for the Oxide Comp 2 and the Sulfide Comp were unremarkable, whilst the higher-grade Oxide Comp gave better results, as summarized in Table 10.18 below.

A significant amount of coarse native copper was observed in the higher-grade Oxide Comp LCT, which is apparent in the gravity concentrate.

Table 10.18: Gravity Test on High-Grade Oxide LCT Tailings					
	%Weight	**%Cu**	**gpt Au**	**Recovery %Cu**	**Recovery %Au**
Pan Conc.	1.0	5.74	23.4	10.4	9.1
Pan Tails	2.8	0.54	4.2	2.6	4.3
Knelson Tails	96.2	0.52	2.4	87.0	86.6

Note that the recovery data presented here represents recovery from the LCT tailings, not the original LCT mill feed. Calculating the contribution to overall recovery gives a 6% copper and 3.5% gold recovery. It was noted that most gangue material in the pan concentrate is magnetite. As discussed earlier, gravity has been eliminated from the flowsheet.

Cyanidation

BML also performed two cyanide leach tests on samples of flotation tailings from the Oxide Comp 2 LCT and the Sulfide Comp LCT. These 24-hour bottle roll tests used 1,000 ppm NaCN and 250 gpt PbNO3 dosage, resulting in the gold dissolution of 81% and 74% for oxide and sulfide, respectively, with cyanide consumption of 0.5 kg/t in both cases.

Cyanidation of LCT tailings effectively increased total gold recovery to over 90% for both samples.

10.4 BASE METALLURGICAL LABS (BL-0835/0882, 2021-2022)

The metallurgical work program continued at BML after the initial BL-0789 tests, including a variability program (BL-0835) that began in September 2021. This was later expanded into a locked cycle testing program (BL0882) that included product characterization (minor elements and tailings dewatering). The results of work completed under both contracts are described in a single BML report dated March 15th, 2022.

An additional 473 kg of drill core and crushed core sample were shipped to BML in three shipments before the commencement of BL-0835 in September 2021.

10.4.1 Sampling

BL-0835

58 variability samples were prepared for this program by combining the mass from two or three similar individual samples (detailed in Appendix A of the BL-0835 Report). From this initial suite of samples, ten were chosen for comminution, and 29 were selected for metallurgical assessment using the developed process. Head assays for the 58 samples are summarized in Table 10.19 below. Copper speciation is indicated by the Cu% Assay (Total Copper), the %CuOx assay (weak acid or oxide copper), and the %CuCN assay (cyanide soluble, or secondary/enriched sulfide copper). Oxide copper species do not recover well in a sulfide flotation environment.

Table 10.19: BL-0835 Variability Samples Head Assays

Sample ID	Assay						
	Cu %	Fe %	S %	Au g/t	Ag g/t	CuOx %	CuCN %
90153-A	0.11	2.5	0.12	0.28	0.4	0.027	0.023
90153-B	0.14	2.8	0.27	0.32	0.9	0.002	0.013
90153-C	0.20	2.2	0.80	0.56	2.1	0.004	0.016
90153-D	0.30	2.4	1.31	0.78	1.8	0.005	0.016
90153-E	0.26	2.9	0.27	2.00	7.3	0.009	0.052
90153-F	0.26	3.0	0.14	1.06	1.0	0.037	0.177
90153-G	0.11	2.3	0.23	0.32	0.6	<0.001	0.007
90153-H	0.37	3.6	0.32	1.20	2.1	0.022	0.141
90153-I	0.79	3.2	1.08	3.01	3.7	0.003	0.050
90153-J	0.22	3.2	0.27	0.78	1.3	0.009	0.045
90153-K	0.14	2.3	1.04	0.51	1.0	<0.001	0.013
90153-L	1.35	4.4	1.26	6.94	4.9	0.048	0.410
90153-M	1.41	5.0	0.40	5.89	5.0	0.012	0.220
90153-N	0.76	4.9	0.47	3.51	2.7	0.062	0.390
90153-O	0.62	3.4	0.50	3.60	2.5	0.039	0.250
90153-P	0.74	3.3	0.41	2.89	3.3	0.015	0.132
90153-Q	0.57	2.5	0.87	2.09	2.0	0.003	0.022
90153-R	0.18	2.8	0.34	0.56	0.8	0.003	0.014
90153-S	0.27	3.0	0.85	1.06	1.7	0.004	0.066
90153-T	0.58	2.6	0.73	2.25	2.9	0.004	0.097
90153-U	0.20	2.5	0.71	0.55	1.0	<0.001	0.012
90153-V	0.58	4.2	0.44	0.93	1.4	0.002	0.069
90153-W	0.28	2.9	0.13	0.52	1.0	0.003	0.028
90153-X	0.13	3.1	0.03	0.23	0.6	0.004	0.047
90153-Y	0.02	3.3	0.51	0.28	0.4	0.001	0.007
90153-Z	0.33	4.6	0.42	1.14	1.5	0.004	0.008
90153-AA	0.40	4.7	0.51	0.91	1.6	0.003	0.015
90153-BB	0.30	3.5	1.14	0.88	1.6	0.003	0.021
90153-CC	0.34	4.4	1.15	0.57	2.7	0.003	0.012
90153-DD	1.06	4.9	2.75	1.56	8.4	0.010	0.077
90153-EE	0.35	3.4	0.14	0.30	0.5	0.023	0.250
90153-FF	0.28	2.0	0.14	0.41	0.8	0.019	0.124
90153-GG	0.50	4.0	0.66	0.54	1.2	0.002	0.016
90153-HH	0.49	4.4	0.94	0.75	1.2	0.006	0.022

Sample ID	Assay						
	Cu %	Fe %	S %	Au g/t	Ag g/t	CuOx %	CuCN %
Oxide 1	0.39	4.0	0.12	1.19	0.9	0.206	0.101
Oxide 2	0.21	2.9	0.09	0.69	0.6	0.103	0.047
Oxide 3	0.40	2.7	0.12	1.06	0.6	0.173	0.102
Oxide 4	0.22	2.2	0.02	0.23	0.2	0.055	0.029
Oxide 5	0.18	3.5	0.09	0.32	0.1	0.085	0.019
Oxide 6	0.21	2.9	0.07	0.50	0.3	0.113	0.015
Oxide 7	0.27	3.5	0.03	0.85	0.7	0.128	0.038
SUL A	1.36	5.1	1.38	7.38	4.9	0.052	0.325
SUL B	0.21	2.9	0.39	1.15	1.3	0.003	0.022
SUL C2	0.38	4.0	0.61	0.44	0.7	0.005	0.027
SUL D	0.34	3.4	0.46	2.12	5.3	0.013	0.050
SUL E	0.10	3.2	0.15	0.31	0.2	0.016	0.019
SUL F	0.08	3.2	0.09	0.36	0.1	0.006	0.028
SUL G	1.12	5.6	3.26	1.60	9.2	0.018	0.100
SUL H	0.35	3.9	0.28	0.99	1.4	0.027	0.102
SUL I	0.17	3.4	0.81	0.35	0.9	0.004	0.006
SUL J	0.21	3.3	0.19	0.62	0.9	0.017	0.087
SUL K	0.29	3.3	0.34	0.34	0.6	0.007	0.021
CK20-01C	0.10	2.4	0.44	0.34	0.2	0.002	0.013
CK20-03C	0.58	3.4	0.83	2.27	1.7	0.001	0.015
CK20-04CB	0.60	2.5	0.72	2.65	1.6	0.007	0.015
Mixed 1	0.15	3.5	0.10	0.41	1.0	0.026	0.027
Mixed 2	0.21	3.4	0.57	0.69	0.8	0.008	0.015
Mixed 3	0.36	3.7	0.09	1.90	1.0	0.196	0.074

This sample set is sufficiently variable in grade with copper ranging between 0.02 and 1.41%, gold between 0.23 and 6.94 gpt and silver between 0.4 and 8.4 gpt. Sulfur assays ranged between 0.03 and 2.75% indicating that in common with previous programs, sulfide gangue mineral (pyrite) content is a relatively minor constituent.

The oxide (CuOx)and cyanide soluble (CuCN) copper analyses provide a good general indication of copper deportment. The deportment of copper between oxide, sulfide, and cyanide soluble minerals is also quite variable, with relatively high oxide content noted in certain samples. Examination of the ratios of CuOx and CuCN to total copper content indicates that:

- 9 of the 58 samples had more than 20% oxide copper and are assumed to be influenced by the high oxide copper content.
- 28 of the 58 samples had greater than 10% cyanide soluble copper and are assumed to be influenced by the secondary enriched copper sulfide content.

- 21 of the 58 samples had less than 10% combined oxide + cyanide soluble copper and are assumed to be "primary sulfide" samples. These samples should perform well in a sulfide flotation environment.

The distribution of CuOx and CuCN content is illustrated for all 58 samples in Figure 10.2 below.

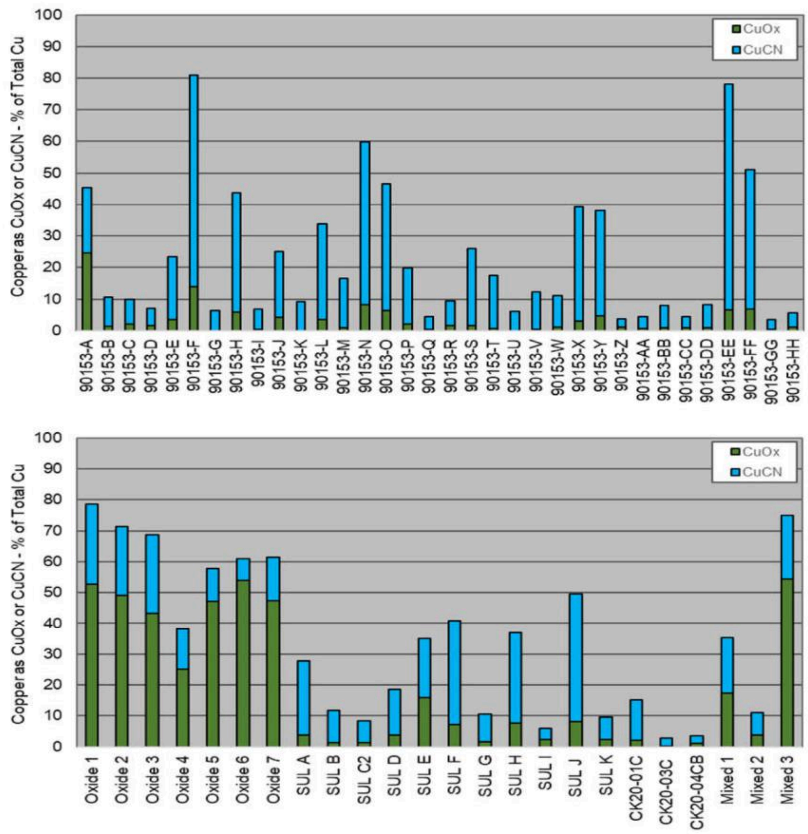


Figure 10.2 - Variability Program Copper Deportment

In addition to the variability work, two sulfide composites were prepared for testing based on sulfide type (primary or secondary/enriched). These two composites are summarized in Table 10.20 below.

Table 10.20: BL-0835 Main Composite Head Assays

Composite	Cu %	Fe %	S %	Au gpt	Ag gpt	Cu_{Ox}	Cu_{CN}
Primary Sulphide	0.36	3.4	0.65	0.96	1.3	0.006	0.024
Enriched Sulphide	0.35	3.4	0.39	1.44	1.9	0.018	0.087

BL-0882

The BL-0882 program focused on the characterization of four Master "Ore Type" composites, namely Shallow Sulfide (C1), Deep Sulfide (C2), Oxide (C3), and Mixed (C4). These master composites were prepared from a variety of BL-0835 variability composites as described in Table 10.21 to Table 10.24.

Table 10.21: Mixed (C4) Composite Construction

Variablitlty Comp	Au ppm	Cu (%)	Mass (kg)
MIXED 1	0.40	0.14	22.6
MIXED 2	0.74	0.22	8.8
MIXED 3	0.58	0.12	22.9
Total	0.53	0.14	54.3

Table 10.22: Shallow Sulphide (C1) Composite Construction

Variablitlty Comp or DC Interval	Au ppm	Cu (%)	Mass (kg)
SUL A	0.36	0.13	5.0
SUL B	0.41	0.14	4.0
SUL C2	0.53	0.19	2.3
SUL E	2.03	0.23	3.4
SUL F	1.15	0.27	7.1
SUL H	1.26	0.31	2.8
SUL J	0.87	0.20	22.6
Total	0.91	0.21	47.2

Table 10.23: Deep Sulphide (C2) Composite Construction

Variablitlty Comp	Au ppm	Cu (%)	Mass (kg)
CK20-01C	0.36	0.13	6.3
CK20-04CB	0.41	0.14	10.8
SUL D	0.53	0.19	2.7
SUL G	2.03	0.23	3.5
SUL I	1.15	0.27	11.2
SUL K	1.26	0.31	5.7
CK 20-14 RC 820-825			5.2
CK 20-019 C 638-743			5.4
CK 20-04 CB 796-801			6.0
CK 20-14 890-895			1.0
CK 20-12 RC 710-715			10.5
CK 20-12 RC 655-660			9.5
Total	NA	NA	77.8

Table 10.24: Oxide (C3) Composite Construction

Variablitlty Comp	Au ppm	Cu (%)	Mass (kg)
OX 1	1.20	0.40	7.9
OX 2	0.70	0.26	8.6
OX 3	1.60	0.39	7.8
OX 4	0.25	0.20	5.2
OX 5	0.34	0.15	3.3
OX 6	0.60	0.21	5.2
OX 7	1.30	0.28	6.0
Total	0.94	0.29	44.0

The resultant master composites were submitted for head assay, with results summarized in Table 10.25 below.

Table 10.25: Master Composite Head Assays

Composite	Cu %	Fe %	S %	Au gpt	Ag gpt	Cu_{Ox}	Cu_{CN}
C1-SS	0.35	3.4	0.35	1.08	1.1	0.014	0.090
C2-DS	0.2	3.5	0.59	0.78	1.5	0.005	0.010
C3-OX	0.31	3.5	0.05	0.71	0.4	0.107	0.067
C4-Mix	0.25	3.3	0.16	0.82	0.6	0.082	0.047

10.4.2 Mineralogy

BL-0882 master composite samples were subjected to a QEMScan PMA analysis, giving quantitative bulk modal mineralogy and liberation data. The data, summarized in Table 10.26 includes information regarding copper deportment and silicate gangue distribution and helps to explain some of the differences in flotation response (copper recovery and concentrate grade).

Table 10.26: BL-0882 Modal Mineralogy

Mineral	Mineral Assays (Wt. percent)					Notes
	DS	SS	MIX	OX	LG	
Chalcopyrite	0.81	0.62	0.24	0.05	0.47	
Bornite	0.00	0.08	0.00	0.00	0.01	Tends to slime during milling and slow to recover
Chalcocite/Covellite	0.02	0.07	0.08	0.11	0.01	Tends to slime during milling and slow to recover
Cuprite	0.00	0.00	0.01	0.01	0.00	Not recoverable by flotation
Cu Metal	0.00	0.01	0.00	0.00	0.00	Limited floatability
Chrysocolla/Cu Chlorite	0.00	0.02	0.36	0.54	0.00	Not recoverable by flotation
Sphalerite	0.05	0.04	0.02	0.03	0.06	Minor contaminant in copper conc.
Pyrite	0.42	0.20	0.15	0.13	0.55	Main sulphide gangue component
Iron Oxides	1.27	1.70	2.47	3.33	2.43	
Quartz	**20.3**	19.8	19.4	24.3	**17.9**	Easy to reject during flotation
Plagioclase Feldspar	**39.1**	38.8	36.9	36.9	**45.8**	Major silicate gangue mineral, easy to reject
Biotite/Phlogopite	4.85	3.83	1.17	1.85	**11.1**	Minor gangue component
K-Feldspars	**15.7**	14.1	13.5	14.9	9.01	Roughly 10-15% w/w - minor silicate mineral, easy to reject
Muscovite	4.62	3.84	6.10	6.56	2.44	
Amphibole (Actinolite)	2.11	3.64	4.62	1.05	2.45	
Epidote	2.93	3.01	3.57	2.25	2.53	
Chlorite	6.07	7.85	8.73	6.35	2.55	Can be problematic in flotation & can recover to conc.
Kaolinite	0.26	0.31	0.37	0.40	0.19	V low in all cases - good for flotation
Calcite	0.03	0.49	0.29	0.04	0.53	
Rutile/Anatase	0.95	0.95	1.23	0.76	0.00	
Apatite	0.41	0.38	0.43	0.29	0.53	
Zircon	0.04	0.04	0.03	0.04	0.04	
Fluorite	0.000	0.000	0.000	0.000	0.005	smallest traces of Fl in all samples
Others	0.14	0.22	0.25	0.13	1.30	
Total	100.0	100.0	100.0	100.0	100.0	

Mineral liberation data from this program is also instructive: overall, the liberation at a nominal 90 µm P_{80} was not outstanding. The two sulfide composites (C1 and C2) with approximately 50% copper sulfide liberation would be expected to perform reasonably well in a sulfide flotation system, albeit with a somewhat high proportion of middling particles that might require longer residence times and/or higher rougher mass pull. The Mixed and Oxide composites (C3 and C4) had lower liberation levels (40% and 38%, respectively), suggesting a finer copper distribution and more challenging metallurgy in general.

The data points to the requirement for a relatively fine concentrate regrind target in the 10-15 µm range. Coarser grinds than this will tend to impact the copper concentrate grade negatively.

10.4.3 Comminution

A Hardness Index Testing (HIT) program was completed on a subset of ten samples from the variability program to improve the overall comminution data set. The HIT tests were carried out on particles in the 19 mm – 22.4 mm size range and are designed to estimate the A x b parameter defined by the JK Tech SMC test. HIT test results are summarized in Table 10.27 below.

Table 10.27: Variability Samples, Comminution Results			
Sample ID	**ECs (kWh/t)**	**t_{10}(%)**	**HIT-Axb Full DWT (est)**
Oxide 1	0.17	6.1	45.6
Oxide 2	0.15	3.9	31.4
Oxide 3	0.17	4.7	35.2
Oxide 4	0.14	3.7	32.1
Oxide 5	0.17	4.8	34.4
Oxide 6	0.15	3.3	27.5
Oxide 7	0.14	2.8	24.6
CK20-04cb Lot B	0.16	3.5	26.3
CK20-04cb Lot A	0.16	3.4	26.4
CK20-03c Lot B	0.18	3.9	27.7
CK20-03c Lot A	0.19	5.0	32.6
CK20-01c Lot B	0.19	4.9	31.8
CK20-01c Lot A	0.18	3.9	27.6

The range of the A x b parameter is reasonable, from the least impact-resistant sample (Oxide 1, measuring 45.6) to the most impact-resistant sample (Oxide 7, measuring 24.6). Samples in this resistance range indicate mineralization amenable to SAG milling, albeit tending towards the more competent side.

Comminution tests on the master composites were limited to Bond BWi tests, and these are summarized in Table 10.28 below.

Table 10.28: Master Composites, Comminution Results				
	Shallow Sulphide (C1)	**Deep Sulphide (C2)**	**Oxide (C3)**	**Mixed (C4)**
Bond Ball Wi, kWh/mt (CSS=106µm)	15.5	16.7	16.4	15.1

These values are slightly higher than earlier work (KCA, 2020), and the samples listed here are considered moderately hard for ball milling.

10.4.4 Rougher Flotation

Rougher flotation testing of the four master composites was limited to a short primary grind confirmation work program. Grind P_{80} sizes of 75 µm and 125 µm were tested against the baseline grind of 90 µm.

For copper, no appreciable performance improvement was seen at the fine grind increment, but a decrease in performance was noted for the coarse setting. In almost every case, the finer grind setting gave a higher, rougher concentrate mass recovery (and subsequently a lower grade). The results are supported by mineralogical data, which indicates a very fine distribution of copper sulfides, fine enough to remain poorly liberated at a P_{80} of 75 µm. Very fine primary grinds are costly (capex and opex) and would also negatively impact the tailing filtration process.

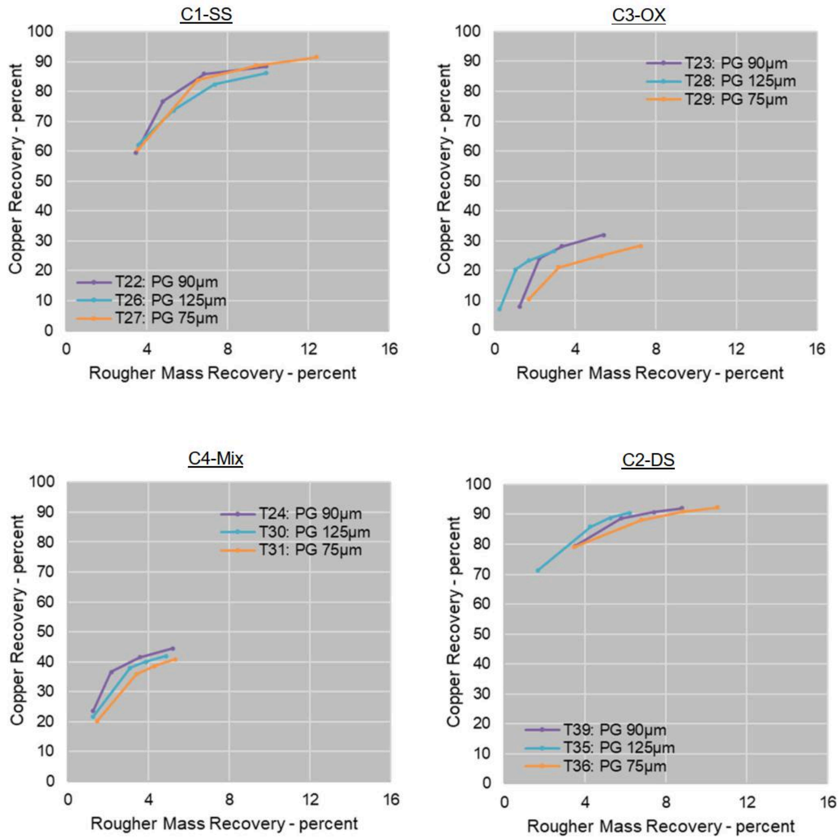


Figure 10.3: Grind Analysis – Rougher Flotation Results

A slight improvement in gold recovery was noted at 75 µm with almost a 5% difference compared to the 125 µm result. This was achieved at a higher mass pull and lower grade. The results support the conclusion SGS drew in 2009 that grinds finer than 80-90 µm are likely not economically beneficial.

BML concluded that the base case 90 µm primary grind was suitable for subsequent cleaner tests and LCTs.

10.4.5 Cleaner Flotation

Variability Samples

The BL-0835 work program tested 8 of the 58 variability samples through the standard flotation flowsheet (90 µm primary grind, pH of 9.5 using lime, a 26-54 µm regrind and previously tested collectors) and the BL-0882 program tested another 21 samples. Results were variable, again demonstrating the impact of copper mineralogy on grade and recovery. Copper recovery of 0.7% to 92.9% and concentrate copper grades of between 9.4 and 42.5% clearly represent a wide range of feed mineralogy, although the metallurgical response can be loosely linked to the ratio of %CuOx to %CuT. The results of this work are summarized in Table 10.29.

Table 10.29: Variability Cleaner Test Work

Composite	Test	Mass	Assay - percent					Distribution - percent				
		%	Cu	Fe	S	Ag	Au	Cu	Fe	S	Ag	Au
90153D	1*	2.5	9.40	33.8	41.3	42	20.5	82.1	34.0	75.8	63.4	68.0
90153F	2*	0.7	27.1	10.1	14.4	76	84.3	71.4	2.2	68.2	57.3	54.4
90153H	3*	0.9	31.2	20.1	25.9	128	120	76.3	4.9	73.5	63.1	60.8
90153J	4*	1.1	15.7	16.2	19.2	60	55.5	78.1	5.1	77.8	56.6	62.0
90153N	5*	1.3	42.5	18.5	26.4	102	142	73.5	5.2	82.4	52.0	56.8
90153Q	6*	2.0	26.1	29.9	34.9	64	71.6	90.9	24.0	81.1	68.6	67.8
90153R	7*	0.7	19.6	26.9	31.1	54	55.5	71.0	6.6	63.5	47.3	51.9
90153Z	8*	1.0	27.7	25.5	29.9	60	55.2	78.0	5.6	76.4	45.7	44.2
Oxide 1	1	0.3	32.4	9.7	18.5	138	224	29.0	0.8	48.2	62.3	57.5
Oxide 2	2	0.3	17.7	8.3	10.9	112	136	19.7	0.7	41.2	39.8	51.5
Oxide 3	3	0.3	35.0	10.9	18.9	124	175	26.5	1.1	60.0	59.5	54.5
Oxide 4	4	0.1	22.2	7.6	20.0	4	169	10.1	0.3	41.6	2.7	54.0
Oxide 5	5	0.1	3.74	10.0	8.82	1	220	1.7	0.2	13.8	0.7	48.5
Oxide 6	6	0.1	1.38	6.2	2.41	2	282	0.7	0.2	5.6	1.1	59.2
Oxide 7	7	0.3	33.5	4.8	6.38	162	244	33.0	0.4	44.1	68.1	62.9
SUL A	8	3.7	30.1	24.4	30.2	79	113	81.5	17.7	84.2	66.0	67.3
SUL B	9	1.2	14.6	20.1	24.9	51	45.3	82.2	7.9	77.1	65.4	68.3
SUL C	10	1.4	21.7	25.0	30.4	44	39.5	88.6	9.1	82.1	78.8	75.7
SUL D	11	1.4	18.9	23.0	27.6	202	58.5	85.0	8.9	80.9	66.0	57.9
SUL E	12	0.4	16.4	24.7	30.1	64	95.7	62.9	3.7	69.1	63.7	75.2
SUL F	13	0.3	23.2	19.4	23.5	68	56.9	80.8	1.6	68.8	67.3	54.4
SUL G	14	7.3	13.4	29.6	37.5	84	14.4	92.9	40.8	88.6	77.1	79.0
Mixed 1	15	0.3	13.9	14.1	16.3	63	66.5	29.3	1.2	50.8	20.3	53.8
Mixed 2	16	1.3	13.0	27.7	32.1	36	37.0	84.8	10.2	77.9	48.6	71.5
Mixed 3	17	0.3	10.5	14.3	8.0	57	102	10.9	1.2	33.6	18.6	32.7
SUL H	18	0.9	26.2	17.7	22.1	100	53.2	77.0	4.2	64.9	61.0	50.9
SUL I	19	1.2	9.30	31.5	41.0	39	16.3	72.4	11.9	68.1	48.3	55.5
SUL J	20	3.4	3.90	8.0	3.4	15	6.67	60.5	8.1	51.7	54.3	45.5
SUL K	21	0.9	21.2	19.4	22.7	26	16.5	74.7	5.4	67.1	28.9	37.5
Average		**1.2**	**20.1**	**18.5**	**22.7**	**71**	**95.7**	**59.5**	**7.7**	**62.7**	**50.1**	**57.9**

Plotting copper and gold recovery as a function of "Oxidation Ratio" (i.e., CuOx/CuT), a tentative trend is apparent for copper (Figure 10.5), but not for gold (Figure 10.4). The copper response seems intuitive, based on the mineralogical results obtained and our knowledge of flotation rates for the different copper minerals. It should be noted, however, that the copper and gold recoveries plotted in these charts are obtained at quite different final concentrate grades, meaning that results are not strictly like for like. For example, test 17 and test 18 achieved 10.5% Cu grade and 26.2% Cu grade, respectively. As recovery is also generally related to concentrate mass pull, the true recovery vs oxidation state relationship is likely not represented correctly in these charts. Adjustments to these charts are discussed further in the metallurgical part of the Metallurgical Discussion (section 10.6).

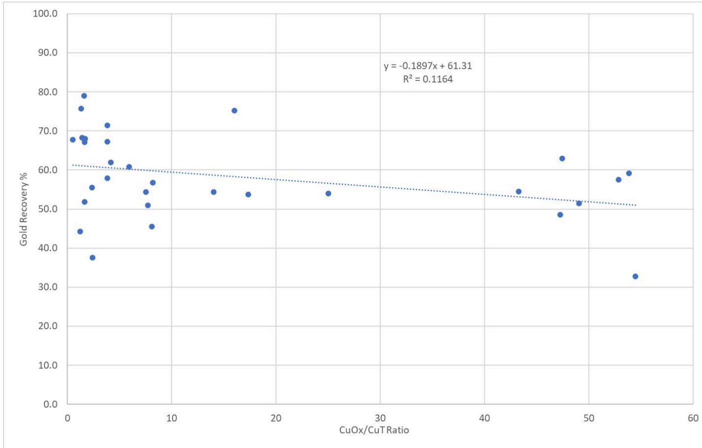


Figure 10.4: Variability Samples, Au Recovery v CuOx/CuT Ratio

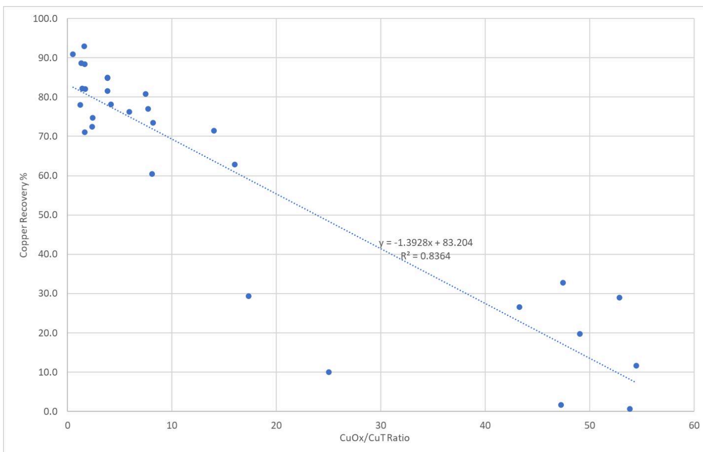


Figure 10.5: Variability Samples, Copper Recovery v CuOx/CuT

Master Composites

Batch cleaner tests were carried out on the BL-0882 master composites in preparation for locked cycle testing. Extra testing of the SS and DS composites allowed for the optimization of concentrate copper grade whilst maximizing gold recovery. The results are summarized in Table 10.30 below.

Table 10.30: Batch Cleaner Test Results						
Composite	**Final Concentrate Grade**			**Final Concentrate Recovery**		
	% Cu	**gpt Au**	**gpt Ag**	**Cu %**	**Au %**	**Ag %**
C1-SS	18.0	59	60	76.2	63.6	61.0
	15.5	46	44	73.7	55.6	51.6
	19.7	48	57	37.2	25.8	31.9
	23.2	59	63	73.0	54.8	56.3
C2-DS	19.7	39	97	82.9	60.2	72.6
	18.9	38	77	87.3	65.7	71.9
	22.2	43	87	83.2	62.0	46.3
C3-OX	32.0	315	207	13.8	46.0	62.6
C4-MIX	19.9	129	88	33.3	54.1	61.2

10.4.6 Locked Cycle Testing

A total of 11 locked cycle tests were completed on the main composites as part of the PFS/FS characterization program. Single tests were completed on the BL-0835 composites (Primary Sul and Enriched Sul), while the BL-0882 composites had two or three tests completed. A summary of the various LCT conditions is given in Table 10.31 below. All tests were completed using a primary grind of 80% -90 µm.

Most of these tests used 2 kg test charges, with the last two using 4 kg charges to boost metal units in the cleaner circuit for improved grade control.

Table 10.31: Locked Cycle Test Conditions							
Composite	**Regrind p_{80}**	**Lime gpt**	**CMC**	**PFSDB**	**7150**	**MIBC**	**H57**
Primary Sul	32µm	275	-	3.5	3.5	21	50
Enriched Sul	25µm	315	-	3.5	3.5	63	80
C1-SS	24µm	200	60	10	10	49	-
	31µm	200	80	10	10	21	-
C2-DS	35µm	380	30	10.5	10.5	175	-
	26µm	410	35	10.5	10.5	147	40
	26µm	230	-	10.5	10.5	40	75
C3-OX	19µm	200		10.5	10.5	63	-
	26µm	200		10.5	10.5	14	80
C4-MIX	26µm	200		10.5	10.5	77	-
	18µm	200	50	10.5	10.5	56	-

Table 10.32: Locked Cycle Test Results						
Composite	Final Concentrate Grade			Final Concentrate Recovery		
	% Cu	gpt Au	gpt Ag	Cu %	Au %	Ag %
Primary Sul	15.9	31	65	88.4	67.2	86.9
Enriched Sul	25.8	93	145	85.7	69.6	69.2
C1-SS	18.3	51	52	81.9	66.1	76.1
	18.9	56	58	81.9	65.7	52.8
C2-DS	22.3	49	84	84.2	67.6	54.6
	17.4	44	76	87.5	74.4	60.7
	19.9	50	76	88.5	73.8	83.6
C3-OX	28.0	292	144	19.1	62.2	54.8
	28.7	203	138	21.6	54.2	55.1
C4-MIX	16.8	121	82	33.4	59.2	53.9
	22.7	156	103	34.8	59.8	48.7

The results of this LCT work showed good consistency within the different composite types and above-average performance considering the head grades. Copper recoveries were once more heavily dependent upon the ratio of copper oxide to total copper content.

10.4.7 LCT Final Concentrate Analysis

Samples of final concentrate from each of the BL-0882 LCT's were submitted for minor element analysis. Results are summarized in the below Table 10.33. These results generally indicate that a relatively clean copper concentrate will be produced, and commercial penalties from smelters will be very rare.

Table 10.33: Locked Cycle Test Minor Element Analysis

Analyte	Unit	Detection Limit	Method	C1-SS	C2-DS	C3-OX	C4-Mix
Al	%	0.01	FUS-Na2O2	2.82	1.82	3.11	2.52
As	ppm	5	FUS-MS-Na2O2	68	151	318	191
Ba	ppm	3	FUS-MS-Na2O2	393	247	604	349
Bi	ppm	2	FUS-MS-Na2O2	28	40	29	30
Ca	%	0.01	FUS-Na2O2	6.66	0.69	0.72	1.64
Cd	ppm	2	FUS-MS-Na2O2	11	50	49	21
Ce	ppm	0.8	FUS-MS-Na2O2	57.3	17.6	38.4	35.5
Co	ppm	0.2	FUS-MS-Na2O2	104	168	259	280
Cr	ppm	30	FUS-MS-Na2O2	220	70	460	140
Cs	ppm	0.1	FUS-MS-Na2O2	0.6	0.3	1.8	0.6
Dy	ppm	0.3	FUS-MS-Na2O2	2.3	0.3	1	1.3
Eu	ppm	0.1	FUS-MS-Na2O2	1.4	0.1	0.7	0.7
Ga	ppm	0.2	FUS-MS-Na2O2	8.1	5.2	9.1	7.2
Gd	ppm	0.1	FUS-MS-Na2O2	3.8	1	1.7	1.8
Ge	ppm	0.7	FUS-MS-Na2O2	1.6	0.9	1	1.4
Hg	ppm	1	AR-ICP	7	6	12	10
In	ppm	0.2	FUS-MS-Na2O2	0.9	1.5	2	1
K	%	0.1	FUS-Na2O2	0.6	0.5	0.8	0.6
La	ppm	0.4	FUS-MS-Na2O2	25.8	8.2	18.7	16.6
Mg	%	0.01	FUS-Na2O2	0.34	0.11	0.21	0.25
Mn	ppm	3	FUS-MS-Na2O2	489	97	264	257
Mo	ppm	1	FUS-MS-Na2O2	73	191	270	109
Na	%	0.001	AR-ICP	0.023	0.008	0.013	0.019
Nb	ppm	2.4	FUS-MS-Na2O2	11.5	< 2.4	4.6	3.4
Nd	ppm	0.4	FUS-MS-Na2O2	34.4	7.8	16.3	14.8
Ni	ppm	10	FUS-MS-Na2O2	150	170	330	240
P	%	0.001	AR-ICP	0.103	0.05	0.085	0.065
Pb	ppm	0.8	FUS-MS-Na2O2	739	845	1520	4160
Pr	ppm	0.1	FUS-MS-Na2O2	7.4	1.8	5.4	4
Rb	ppm	0.4	FUS-MS-Na2O2	10.3	17.5	15.7	15
Sb	ppm	2	AR-ICP	17	15	26	33
Se	ppm	8	FUS-MS-Na2O2	106	158	285	81
Si	%	0.01	FUS-Na2O2	9.09	5.56	13.1	8.21
Sm	ppm	0.1	FUS-MS-Na2O2	7.8	1.5	1.8	4.6
Sn	ppm	0.5	FUS-MS-Na2O2	3.7	5.2	18.2	2.1
Sr	ppm	3	FUS-MS-Na2O2	402	196	272	279
Tb	ppm	0.1	FUS-MS-Na2O2	0.7	< 0.1	0.3	0.3
Te	ppm	1	AR-ICP	19	31	16	23
Te	ppm	6	FUS-MS-Na2O2	14	21	< 6	7
Th	ppm	0.1	FUS-MS-Na2O2	6.3	4.1	10.6	6.3
Ti	%	0.01	FUS-Na2O2	0.33	0.04	0.1	0.08
Tl	ppm	0.1	FUS-MS-Na2O2	1.9	1.8	1.9	2.3
U	ppm	0.1	FUS-MS-Na2O2	5.3	2.4	2.8	4.5
V	ppm	5	FUS-MS-Na2O2	43	15	37	25
W	ppm	0.7	FUS-MS-Na2O2	3.1	1.2	12.5	2.6
Y	ppm	0.1	FUS-MS-Na2O2	14.3	2	4.7	4.8
Yb	ppm	0.1	FUS-MS-Na2O2	1.1	0.2	0.8	0.7
Zn	ppm	30	FUS-MS-Na2O2	2210	> 10000	2320	3380
Zr	ppm	1	AR-ICP	16	7	10	9

10.4.8 LCT Tailings Dewatering

The final tailing slurry from a selection of the main composite LCTs was used as feed for a settling and filtration testwork program at BML. The work included flocculant scoping and static settling tests, with subsequent pressure filtration testing of thickened slurries.

The scoping tests considered several well-known flocculant products and tested different addition rates and pH adjustments. The work demonstrated that a very high molecular weight, slightly anionic polyacrylamide flocculant (Magnafloc 10), was effective and that adding lime helped improve the supernatant clarity.

Therefore, the static settling test series was completed using the MF10 flocculant and a pH adjustment to 11.0 with lime. Different flocculant dosages give a variety of settling rates and final underflow densities. Underflow density of between 55% and 63% solids was achievable, although rheology tests were not conducted to determine pumping characteristics at these densities.

In general, a 20-40 gpt flocculant addition was deemed sufficient to obtain good settling rates, and the addition of lime to thickener feed helped to give superior overflow clarities.

Table 10.34: Static Settling Test Results

Sample	Test	pH	Floc	Dosage g/t	Initial Density % solids	Final Density (% solids)	Settling Rate mm/s
T43, Final Tl C,D,E	S1	11.0	MF10	20	14.0	63.0	2.8
	S2	11.0	MF10	40	14.0	61.9	2.8
	S3	11.0	MF10	60	14.0	60.6	3.5
T44, Final Tl C,D,E	S4	11.0	MF10	20	14.0	61.6	3.4
	S5	11.0	MF10	40	14.0	60.2	2.2
	S6	11.0	MF10	60	14.0	61.4	4.1
T47, Final Tl C,D,E	S7	11.0	MF10	20	12.4	59.4	2.3
	S8	11.0	MF10	40	12.4	59.5	8.0
	S9	11.0	MF10	60	12.5	57.1	4.6
T48, Final Tl C,D,E	S10	11.0	MF10	20	13.6	59.5	1.5
	S11	11.0	MF10	40	13.6	58.5	1.9
	S12	11.0	MF10	60	13.5	58.0	3.0

Batches of tailing slurry from 3 of the BL-0882 LCTs were thickened to 60% solids then presented to a lab scale pressure filtration unit equipped with membrane squeeze and air-blow. The results of this work are plotted on a single chart (Figure 10.6), showing the filtration rate vs cake moisture trends for each composite.

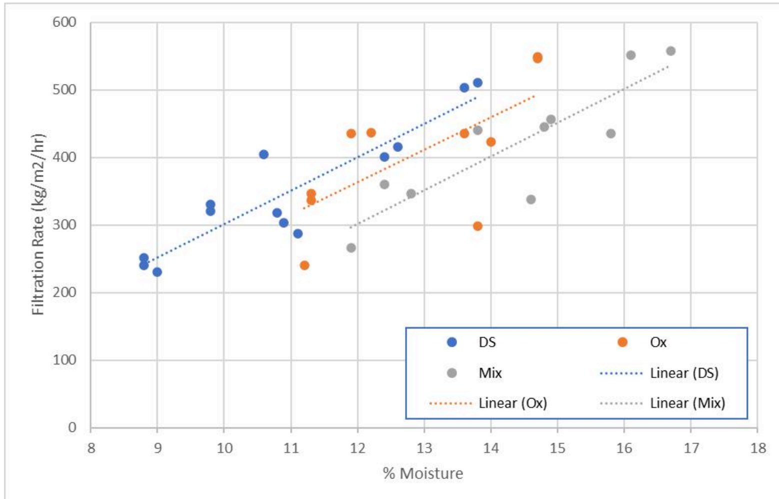


Figure 10.6: Pressure Filtration Testwork Results

Each sample gives a slightly different response, with the Deep Sulfide (DS) sample providing the highest filtration rates at the target moisture of 14% (w/w). As this composite represents mineralization that will dominate the reserve tonnage, the DS data is suitable for design purposes, but with the understanding that occasional periods of additional mixed or oxide mineralization might de-rate the filtration process.

10.5 BASE METALLURGICAL LABS (BL-0980 & 1066, 2022)

As the PFS-level geometallurgical studies continued at BML, it became apparent that the average reserve grade for the Project was somewhat lower than composite grades in earlier programs. BL-0980 and 1066 were therefore appended to address head grade-related concerns. Lower-grade drill core intervals were targeted as part of the sample selection algorithm, and composite grades reflect this. BL-980 and BL-1066 sample sets were shipped on separate dates.

10.5.1 Sampling

For the BL-980 program, 21 samples of ½-core and two samples of reject material totaling 100 kg were selected from six holes within the PFS pit outline. For BL-1066, 22 samples of ½-core and four samples of RC material (6 mesh) totaling 91 kg were selected from eight holes within the PFS pit outline.

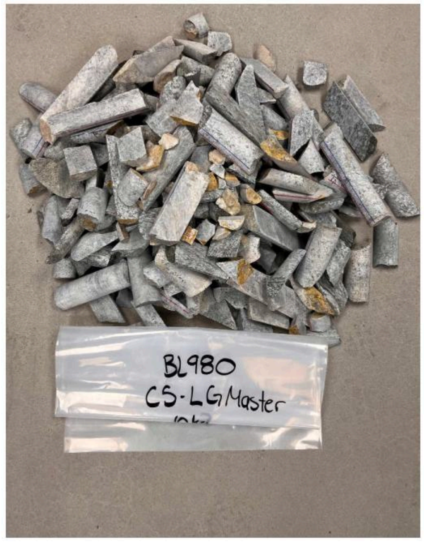


Figure 10.7: BL-980 Typical Sample

Replicate cuts were removed from each composite sample as part of the blending, crushing, and subsampling process. Average head assays for each pair are summarized in Table 10.35.

Table 10.35: BL-0980 Head Assay								
Ref	**Description**	**% Cu_T**	**% Cu_{Ox}**	**%Cu_{CN}**	**%Fe**	**%S**	**gpt Ag**	**gpt Au**
LG Comp	**BL-980 Master Comp**	0.18	0.002	0.012	3.8	0.45	0.9	0.45
LG Comp 2	**BL-1066 Master Comp**	0.16	0.01	0.03	3.2	0.38	0.9	0.35

Of note, the most significant pay metals, gold, and copper, are in close agreement with the life of mine PFS reserve grades. Secondary sulfide and oxide copper species were slightly higher in the second composite (LG Comp 2) but still represent minor fractions. As such, these composites are useful reference points for performance predictions.

10.5.2 Mineralogy

The first LG composite was subjected to a QEMScan PMA analysis, giving quantitative bulk modal mineralogy and liberation data. The modal data is detailed in the BML report but is very similar to previous studies, albeit with a lower sulfide content (1.1% in this sample). The copper deportment data shows Chalcopyrite as the dominant copper mineral (92.2%), with Bornite and Chalcocite/Covellite as minor species (3.5% and 4.2% respectively). Only traces of oxide copper minerals were noted, making this composite sample a deep sulfide equivalent with limited copper recovery downside.

10.5.3 Comminution

Sub-samples of half-core were selected from the LG Comp crushing/blending process and tested in-house at BML. Results are summarized in Table 10.36 below.

Table 10.36: Comminution Test work Results							
Ref	**Description**	SG	BWi (kWh/t)*	DWi (kWh/m^3)	A x b	t_a	SCSE (kWh/t)*
LG Comp	**BL-980 Master Comp**	2.72	14.8	8.4	32.4	0.31	10.96

* Note: data is based on metric tonnes.

The A x b results give an indication of resistance to impact breakage and, in this case, show that the LG Sulfide composite is on the competent side of average, similar to the results of previous work.

10.5.4 Cleaner Flotation

10 kg cleaner tests were conducted on the LG composite to calibrate equipment and fine-tune locked cycle test conditions. A 10 kg charge size was used in this work, as the larger, rougher concentrate mass tends to help with cleaner circuit grade control. A primary grind of 90 µm was used in all tests.

Table 10.37: Batch Cleaner Tests on LG Composites						
Composite	**Final Concentrate Grade**			**Final Concentrate Recovery**		
	% Cu	**gpt Au**	**gpt Ag**	**Cu %**	**Au %**	**Ag %**
LG COMP	18.3	42.8	97	87.2	63.4	89.7
	25.1	59.7	114	74.4	53.4	60.2
LG COMP 2	16.8	34.7	81	85.6	67.8	59.6
	24.4	45.0	118	82.0	58.4	51.3

Copper concentrate grades were reasonable in most tests, with the first LG COMP 2 test being a little off-spec. Due to the lower head grade in these samples, copper and gold recoveries tended to be slightly lower than past performance.

10.5.5 Locked Cycle Testing

2 x 10-kg locked cycle tests were completed on the LG composites.

A summary of the various LCT conditions is given in Table 10.38. All tests were completed using a primary grind of 80% -90 µm, and pH was controlled to 9.5 using lime. 10-kg test charges were used for these tests, allowing far greater control over mass pull in the cleaner circuit. These larger LCTs utilized a 40-litre rougher flotation cell, and the normal 4-lire D12 was used for cleaner flotation.

Table 10.38: LG Composites, LCT Conditions						
Composite	**Regrind p_{80}**	**Lime gpt**	**PFSDP**	**7150**	**MIBC**	**H57**
LG COMP	28 µm	245	10	10	80	-
LG COMP 2	17 µm	265	8.5	7.5	90	8

Table 10.39: LG Composites, LCT Results							
Composite	**Mass Pull %**	**Final Concentrate Grade**			**Final Concentrate Recovery**		
		% Cu	**gpt Au**	**gpt Ag**	**Cu %**	**Au %**	**Ag %**
LG COMP	0.9	17.6	40.4	91	86.5	65.1	75.8
LG COMP 2	0.6	24.9	47.9	116	86.6	67.0	70.7

The results of the LCT work showed that, as expected, the lower head grade samples tend to give rise to slightly lower recoveries. The BL-980 LCT gave a slightly disappointing result, with similar recoveries despite the higher mass pull (and lower copper concentrate grade). Throughout this test, the 40-litre rougher flotation cell was challenged by inferior froth characteristics, whereas in the LG COMP 2 test, this issue was addressed by adding a stronger frother in addition to the MIBC. This helped froth stability and improved performance. The LG COMP 2 LCT is judged to be a better representation of flotation circuit performance for the bulk of the deposit (i.e., primary sulfide material) at the life of mine average head grades. Based on the low-grade LCT result, Wells suggests that an 18% Cu concentrate grade and a 69% gold recovery should be used in the financial analysis for low-grade material. Two new low-grade samples must be prepared to confirm this result and investigate a finer regrind.

10.5.6 LCT Final Concentrate Analysis

Samples of final concentrate from the BL-0980 and BL-1066 LCTs were submitted for minor element analysis. Results are summarized in Table 10.40 and Table 10.41 below. As with previous results, this data indicates that a clean copper concentrate can be produced, and commercial penalties from smelters will be very rare.

Table 10.40: Locked Cycle Test Minor Element Analysis BL-0980

Element	Unit Symbol	Detecton Limit	Analysis Method	Test 4 Cu Con D+E	Element	Unit Symbol	Detecton Limit	Analysis Method	Test 4 Cu Con D+E
Ag	ppm	0.2	AR-ICP	87.8	Mo	ppm	1	AR-ICP	4
Al	%	0.01	FUS-Na_2O_2	0.85	Na	%	0.001	AR-ICP	0.01
As	ppm	5	FUS-MS-Na_2O_2	174	Nb	ppm	2.4	FUS-MS-Na_2O_2	< 2.4
B	ppm	10	FUS-MS-Na_2O_2	160	Nd	ppm	0.4	FUS-MS-Na_2O_2	3.9
Ba	ppm	3	FUS-MS-Na_2O_2	243	Ni	ppm	1	AR-ICP	148
Be	ppm	3	FUS-MS-Na_2O_2	< 3	P	%	0.001	AR-ICP	0.048
Bi	ppm	2	AR-ICP	< 2	Pb	ppm	0.8	FUS-MS-Na_2O_2	2100
Ca	%	0.01	FUS-Na_2O_2	0.22	Pr	ppm	0.1	FUS-MS-Na_2O_2	1
Cd	ppm	2.00	FUS-MS-Na_2O_2	77	Rb	ppm	0.4	FUS-MS-Na_2O_2	3.6
Ce	ppm	0.8	FUS-MS-Na_2O_2	8.3	S	%	0.01	AR-ICP	14
Cl	%	0.01	INAA	< 0.01	Sb	ppm	2	AR-ICP	18
Co	ppm	1	AR-ICP	229	Sc	ppm	1	AR-ICP	< 1
Cr	ppm	1	AR-ICP	72	Se	ppm	8	FUS-MS-Na_2O_2	89
Cs	ppm	0.1	FUS-MS-Na_2O_2	0.4	Si	%	0.01	FUS-Na_2O_2	2.48
Dy	ppm	0.3	FUS-MS-Na_2O_2	< 0.3	Sm	ppm	0.1	FUS-MS-Na_2O_2	0.5
Er	ppm	0.1	FUS-MS-Na_2O_2	< 0.1	Sn	ppm	0.5	FUS-MS-Na_2O_2	2.4
Eu	ppm	0.1	FUS-MS-Na_2O_2	0.2	Sr	ppm	3	FUS-MS-Na_2O_2	110
F	%	0.01	FUS-ISE	< 0.01	Ta	ppm	0.2	FUS-MS-Na_2O_2	< 0.2
Fe	%	0.01	AR-ICP	23.1	Tb	ppm	0.1	FUS-MS-Na_2O_2	< 0.1
Ga	ppm	0.2	FUS-MS-Na_2O_2	2.4	Te	ppm	6	FUS-MS-Na_2O_2	15
Gd	ppm	0.10	FUS-MS-Na_2O_2	0.50	Th	ppm	20	AR-ICP	< 20
Ge	ppm	0.7	FUS-MS-Na_2O_2	0.8	Ti	%	0.01	FUS-Na_2O_2	0.03
Ho	ppm	0.2	FUS-MS-Na_2O_2	< 0.2	Tl	ppm	2	AR-ICP	< 2
Hf	ppm	10	FUS-MS-Na_2O_2	< 10	Tm	ppm	0.1	FUS-MS-Na_2O_2	< 0.1
Hg	ppm	1	AR-ICP	18	U	ppm	0.1	FUS-MS-Na_2O_2	2.5
In	ppm	0.2	FUS-MS-Na_2O_2	2.8	V	ppm	5	FUS-MS-Na_2O_2	6
K	%	0.1	FUS-Na2O2	0.1	W	ppm	0.7	FUS-MS-Na_2O_2	1.4
La	ppm	0.4	FUS-MS-Na_2O_2	4.9	Y	ppm	0.1	FUS-MS-Na_2O_2	1.2
Li	ppm	15	FUS-Na2O2	< 15	Yb	ppm	0.1	FUS-MS-Na_2O_2	< 0.1
Mg	%	0.01	FUS-Na2O2	0.05	Zn	%	2	FAAS	2.36
Mn	ppm	5	AR-ICP	79	Zr	ppm	1	AR-ICP	8

Table 10.41: Locked Cycle Test Minor Element Analysis BL-1066

Analyte Symbol	Unit Symbol	DL	Analysis Method	Cu Con D+E	Analyte Symbol	Unit Symbol	DL	Analysis Method	Cu Con D+E
Ag	ppm		AR-AAS	116	Li	ppm	15	FUS-Na2O2	< 15
Al	%	0.01	FUS-Na2O2	0.74	Mg	%	0.01	FUS-Na2O2	0.09
As	ppm	5	FUS-MS-Na2O2	77	Mn	ppm	3	FUS-MS-Na2O2	71
Au	ppm	0.05	Fire-AAS	47.9	Mo	ppm	1	FUS-MS-Na2O2	286
B	ppm	10	FUS-MS-Na2O2	< 10	Nb	ppm	2.4	FUS-MS-Na2O2	< 2.4
Ba	ppm	3	FUS-MS-Na2O2	< 10	Nd	ppm	0.4	FUS-MS-Na2O2	6
Be	ppm	3	FUS-MS-Na2O2	< 3	Ni	ppm	10	FUS-MS-Na2O2	230
Bi	ppm	2	FUS-MS-Na2O2	36	Pb	ppm	0.8	FUS-MS-Na2O2	598
Ca	%	0.01	FUS-Na2O2	0.27	Pr	ppm	0.1	FUS-MS-Na2O2	1.5
Cd	ppm	2	FUS-MS-Na2O2	40	Rb	ppm	0.4	FUS-MS-Na2O2	3.5
Ce	ppm	0.8	FUS-MS-Na2O2	12.4	S	%	0.01	LECO	36.9
Cl	%	0.01	INAA	0.02	Sb	ppm	2	FUS-MS-Na2O2	6
Co	ppm	0.2	FUS-MS-Na2O2	228	Se	ppm	8	FUS-MS-Na2O2	117
Cr	ppm	30	FUS-MS-Na2O2	200	Si	%	0.01	FUS-Na2O2	2.29
Cs	ppm	0.1	FUS-MS-Na2O2	< 0.1	Sm	ppm	0.1	FUS-MS-Na2O2	0.8
Cu	%	0.01	AR-AAS	24.9	Sn	ppm	0.5	FUS-MS-Na2O2	2.4
Dy	ppm	0.3	FUS-MS-Na2O2	< 0.3	Sr	ppm	3	FUS-MS-Na2O2	94
Er	ppm	0.1	FUS-MS-Na2O2	< 0.1	Ta	ppm	0.2	FUS-MS-Na2O2	0.5
Eu	ppm	0.1	FUS-MS-Na2O2	0.2	Tb	ppm	0.1	FUS-MS-Na2O2	< 0.1
F	%	0.01	FUSE-ISE	<0.01	Te	ppm	6	FUS-MS-Na2O2	12
Fe	%	0.05	AR-AAS	29.0	Th	ppm	0.1	FUS-MS-Na2O2	3
Ga	ppm	0.2	FUS-MS-Na2O2	1.6	Ti	%	0.01	FUS-Na2O2	0.03
Gd	ppm	0.1	FUS-MS-Na2O2	0.5	Tl	ppm	0.1	FUS-MS-Na2O2	0.9
Ge	ppm	0.7	FUS-MS-Na2O2	< 0.7	Tm	ppm	0.1	FUS-MS-Na2O2	< 0.1
Hg	ppm	1	AR-ICP	6	U	ppm	0.1	FUS-MS-Na2O2	3.5
Ho	ppm	0.2	FUS-MS-Na2O2	< 0.2	V	ppm	5	FUS-MS-Na2O2	12
Hf	ppm	10	FUS-MS-Na2O2	< 10	W	ppm	0.7	FUS-MS-Na2O2	2.4
In	ppm	0.2	FUS-MS-Na2O2	1.6	Y	ppm	0.1	FUS-MS-Na2O2	1.2
K	%	0.1	FUS-Na2O2	0.1	Yb	ppm	0.1	FUS-MS-Na2O2	0.2
La	ppm	0.4	FUS-MS-Na2O2	7.2	Zn	ppm	30	FUS-MS-Na2O2	7160

10.6 METALLURGICAL DISCUSSION

10.6.1 General

Various companies carried out metallurgical test programs between 1985 and 2022. The work helped identify froth flotation as the preferred processing technology for the Project, and the QP (Wells) concurs with that decision.

A good deal of effort was put into identifying and understanding factors affecting the flowsheet performance, with reagent recipes and grind targets modified and progressively optimized. The selection of samples for composites and the style of mineralization within the deposit were also given considerable focus, resulting in a series of tests between 2020 and 2022. A brief chronology of the work and sample selection criteria is given below.

The 2009-2010 work at SGS Lakefield characterized composite samples of sulfide and oxide mineralization and indicated gold and copper recovery of 68% and 77%, respectively, for sulfide and 55% gold and 10% copper recovery for oxide. Copper concentrate grade was 25-26% Cu for sulfide and 15% Cu for the Oxide composite. Whilst lower in copper, this oxide concentrate had an exceptionally high gold content of 380 gpt Au. By industry standards, these grades and recoveries would be considered reasonable, considering the relatively low copper head grade and the mineralogy.

The subsequent 2020 test program at KCA scheduled work on drill core samples from seven dedicated metallurgical holes, which were used to produce three main composites, namely high-grade oxide, oxide, and sulfide. The work program commenced with the high-grade oxide test, and these tests produced results that were better than anticipated, considering the high proportion of non-sulfide copper minerals present. Concentrate grades of 25% copper, with gold and copper recoveries of 69% and 56%, respectively, were achieved from open circuit cleaner tests. A 3% overall increase in copper recovery (included in the 56%) was achieved by adding a gravity circuit, which recovered coarse, liberated native copper from flotation tailings. It was later discovered that the High-Grade Oxide zone represents only a small part of the deposit (less than 2% by weight), so the motivation for this additional processing step disappeared, and the step was dropped from future programs.

Following the encouraging results on the high-grade oxide composite, test work then commenced in parallel on the Oxide and Sulfide composites, which represent about 6-8% and 90%, respectively, of the deposit. The initial rougher tests, using conditions established for the High-Grade Oxide, gave high recoveries of copper (90%), gold (74%), and silver (61%), albeit with a higher mass recovery and low copper concentrate grade. These results encouraged the commencement of cleaner tests, which consistently encountered difficulty upgrading the concentrates to a commercially acceptable grade. Attempts to increase the final cleaner concentrate grade above 20% Cu resulted in unacceptable copper, gold, and silver losses to the tailings streams.

A single locked cycle test was attempted on the high-grade oxide composite, and this, too, resulted in a low final concentrate grade (around 10% Cu).

At this point (April 2021), U.S. Gold decided to transfer samples to BML in Kamloops, Canada. Test work at BML commenced shortly thereafter, and after adjusting the reagent recipe, they could replicate the recoveries and concentrate grades achieved by SGS in 2010. A full test program was then initiated at BML to test all three composites, including rougher optimization tests, rougher-cleaner open cycle tests, and finally, a series of LCTs.

This work was successful and confirmed the favorable mineralogy reported earlier by FLSmidth. The principal difference between BML and KCA was the much lower collector addition at BML, which essentially used “starvation” quantities of these chemicals. As confirmation, KCA was able to improve its results in line with SGS and BML after switching to the revised reagent schedule.

As the flowsheet development continued at BML, it was noted that the Sulfide composite contained 10-15% non-sulfide copper minerals that originated from a mixed ore zone within the deposit. This likely accounted for some of the difficulties experienced with copper recoveries. A second Sulfide composite (Sulfide Comp 2) was prepared, avoiding the mixed zone, which immediately realized improved recoveries.

A variability program included comminution testing of 10 samples and open circuit flotation testing of 29 samples. The variability work forms a basis for geometallurgical modelling discussed elsewhere in this document.

A program of locked cycle tests was completed on composites of oxide, mixed shallow sulfide, and deep sulfide mineralization. These results are incorporated into the variability data set and highlighted within the scatter plots as higher-confidence data points.

Finally, several tests were run on lower-grade sulfide samples to ensure the data set covered the grades noted in PFS mine plans. Two locked-cycle tests were run on low-grade composites.

The results of the variability batch cleaner test and locked cycle test have been used to prepare metallurgical models for incorporation into mine plans and project cashflows. The PFS recovery model is summarized in Section 14.0.

10.6.2 Sampling

A significant sample mass has been shipped from the site to the various metallurgical laboratories:

- 2008 – roughly 500 kg of oxide, mixed and sulfide ½-core from an unknown number of drill holes.
- 2020 – roughly 800 kg of oxide, mixed and sulfide ½-core from seven drillholes.
- 2021 – roughly 100 kg of oxide and sulfide from multiple drillholes.
- 2022 - roughly 200 kg of lower-grade sulfide from multiple drillholes.

Sample mass is considered sufficient, and sample location is sufficiently diverse spatially to achieve good coverage. Sampled holes are generally within or near the Pre-Feasibility study pit shell.

No relationships linking sample location to metallurgical recovery have been discovered. As geometallurgical programs are developed, this aspect could be examined, although all indications are that samples are metallurgically consistent within the different oxidation zones.

Sample grades have tended to be high, and most composites have had grades significantly higher than average reserve grades. This fact drove the selection of samples for the two final LG composites, tested during the summer of 2022 at BML.

10.6.3 Mineralogy

Quantitative mineralogical work by SGS, FLSmidth, and the BML has improved understanding of copper deportment throughout the deposit, and currently represents a good foundation for more comprehensive geometallurgical modelling that would ideally be carried through into operations. The work by FLSmidth in 2021 provides some additional insight into gold deportment and liberation.

Copper is primarily found in chalcopyrite, with lesser amounts in secondary sulfides such as bornite or chalcocite, or in oxides such as chrysocolla, cuprite or clays/micas. In a small, centrally located high grade oxide zone, native copper is common, although this is only noted occasionally through the bulk of the deposit. The metallurgical response of samples from different locations within the deposit depends upon the specific blend of oxide and sulfide copper minerals.

The main sulfide gangue component within the deposit is pyrite, although this generally occurs in lower concentrations (relative to copper) than in many porphyry deposits. Base metal sulfides such as sphalerite and galena are noted in trace concentrations. Although these recover to the flotation concentrate, they do so in amounts that avoid smelter penalties.

The host rocks are mainly feldspar (about 45%), quartz (about 25%), and mica (about 14%).

Liberation data is variable but suggests that copper sulfides are not well liberated at a P_{80} of 100-125 µm, and therefore, rougher concentrate regrinding is an essential aspect of the flowsheet. In general, the very small quantities of rougher concentrate generated in laboratory scale tests make concentrate regrind optimization studies difficult, but the program has settled on a regrind P80 target in the 20-25 µm size range for the PFS. Whilst this is a workable range, the opportunity to run the cleaner flotation circuit with a finer grind is significant.

By the grades involved, the statistical significance of gold deportment data is less than that for the sulfides. However, the measurements made by FLSmidth suggest that most gold/silver particles are less than 10 µm in size and locked or associated with various sulfides, silicates and oxides. Again, much finer (likely uneconomic) grinds are necessary to realize significant performance gains.

10.6.4 Primary Grind

The primary grind used in most flotation tests at SGS, KCA, and BML has been in the P_{80} range of 75 to 125 µm. Early work by SGS concluded that the optimum (cost vs benefit) primary grind appeared to be between 90 µm and 100 µm.

As discussed above, the measurements provided by mineralogical studies indicate that both the copper sulfides and the gold/silver/electrum are fine-grained, which points to a relatively fine primary grind. More recent results from studies by BML indicated the optimum grind to be between 75 µm and 90 µm, although there is some variability. Metallurgical performance appears to deteriorate rather quickly at grinds coarser than 100 µm.

The PFS included an order of magnitude trade-off study for five P_{80} grind sizes that evaluated the recovery of copper and gold and capital and operating costs. The fundamentals of this analysis have not changed, and the results of this 2021 trade-off study are summarized in Table 10.42 below.

Table 10.42: Evaluation of the Primary Grind					
P_{80} (µm)	differential capex US$M	differential opex, US$/t	Au recovery (%)	Cu recovery (%)	differential NPV, US$M
80	+1.1	+0.07	73.5	86.0	+21M
86	+0.8	+0.05	72.5	85.5	+6M
90	Base Case	0	72.0	85.0	0
106	-0.5	-0.06	69.0	84.0	-37M
120	-1.9	-0.12	67.5	83.0	-58M

Copper recovery is relatively insensitive to grind size. However, gold recovery drops more rapidly at coarser sizes, which is a major revenue driver. The NPV is similar for P_{80}s between 80 and 90 µm. At coarser grind sizes, the revenue drops quite rapidly due to reduced gold recovery.

A P_{80} of 90 µm was selected for the PFS process plant design and subsequent work in 2022 has not deviated from this assertion. A finer grind would surely improve the copper and gold grade vs. recovery relationship, but grinding costs, together with the negative impact of fines on the tailing filtration process effectively cancel these gains.

Further optimization of the grind can form part of more detailed post-PFS economic optimizations.

10.6.5 Rougher Concentrate Regrind

Much of the recent metallurgical test work has targeted a regrind P_{80} of between 20 µm and 25 µm, although rougher concentrate mass pull variability has resulted in regrind P_{80} of up to 40 µm on occasion. Mineralogical studies by BML suggest that a regrind P_{80} of 15-20 µm might provide performance gains.

Although somewhat limited by sample mass, implementing larger-scale flotation tests (10 kg or larger feed mass) will allow better control of the laboratory regrinding process, and this data will allow a more accurate assessment of regrinding requirements.

10.6.6 Gravity Concentration

Although laboratory-scale gravity tests were unsuccessful for the KCA Oxide composite and the Sulfide composite, a reasonable gravity concentrate was achieved for the Hole 4 high-grade oxide composite due to of higher gold grades and significant concentrations of native copper. Adding gravity to a standard flotation circuit results in an overall copper recovery increase of 3% for the Hole 4 sample.

Despite the reasonable Hole 4 performance, comparisons of overall flowsheet performance with and without the gravity stage show similar gold and copper performance, suggesting that a simple “no gravity” approach will be most effective. Gravity has not been included in the Pre-Feasibility Study flowsheet design.

10.6.7 Flotation

Certain elements of the flotation circuit, such as the flowsheet configuration and the primary grind, have remained consistent since the SGS work in 2008/9. Reagent recipes and dosages have changed, with the most notable change being the reduction in collector dosage to starvation levels ahead of cleaning. This change helped improve selectivity within the cleaner circuit and removed the need for excessive depressant addition, which in turn helped to improve recovery.

pH adjustment to levels >8.5 helps to depress the flotation of pyrite, although, for many tests, the sulfur recovery suggests that pyrite has been well-recovered to concentrate. The lime added to raise pH also helps with froth stability, and this, in turn, can assist flotation performance. The calcium ions in solution from lime addition can also help with settling and filtration.

CMC addition in certain areas of the deposit (where chlorite and/or talc may be present in minor concentrations) may be required, although it is presently assumed that any areas containing active gangue minerals will be diluted down by surrounding material. The addition of CMC can be harmful to gold recovery.

Flotation kinetics are quite reasonable, so flotation residence times are not excessive, and the primary flotation circuit need not be enormous to achieve low tailing grades. High sulfide recoveries have been achieved at very high upgrade ratios for several composites, which speaks to the host rock's non-floatable nature (feldspar and quartz).

The regrind target is discussed above, but ultra-fine regrinds followed by modern fine particle cleaner flotation technology (Jameson cell, column flotation, etc.) should be viewed as an opportunity to shift the grade vs. recovery curve into positive territory. Cavitation reactor technology such as the MACH reactor might also be considered.

10.6.8 Tailings and Concentrate Dewatering

The project flowsheet includes a dewatering circuit designed to filter tailings and deposit the plant residues as a 14% moisture filter cake. This eliminates a wet tailings dam and reduces the demand for fresh make-up water.

Vacuum and pressure filtration testwork was carried out on samples of LCT tailing at Pocock in 2021 as part of the KCA metallurgical program and again in 2022 at BML using a laboratory-scale pressure filtration unit to dewater samples of LCT Tailings. The somewhat fine primary grind used in these tests reduces tailing filtration rates, and vacuum filtration tests all produced cake with high moisture levels. Pressure filtration is therefore judged to be the only suitable process for dewatering tailings to the specified 14% moisture levels, deemed necessary to aid site water balances.

At a primary P_{80} of 90-100 µm, the tailings filter well uses pressure filtration technology. At finer grinds, performance is expected to deteriorate somewhat.

Flotation concentrate masses generated in laboratory scale tests are particularly small when lower-grade mineralization is tested. For this reason, no specific pressure filtration testwork has been completed for the Project to date. However, equipment vendors carry significant databases for this type of material and are comfortable sizing equipment based on regrind P_{80}, mineralogical composition, and pH alone. The PFS has assumed that a final concentrate product can be dewatered to 8 or 9% moisture before bagging in FIBC containers.

10.6.9 Jameson Flotation Cell Testwork

In 2024, U.S. Gold and Glencore Technologies discussed the possible application of Jameson Flotation Cells in the plant flowsheet. Glencore claimed that the Jameson technology improved metallurgical performance (concentrate grade and/or recovery) and potentially reduced installed capital cost and operating costs.

A proposal was requested from Base Metals Laboratory (BML) in Kamloops, Canada, with input from Glencore, to evaluate Jameson flotation. Two composites were prepared, "Sulfide Composite" (0.30%Cu, 0.9 g/t Au) and "Sulfide Composite 2" (0.36%Cu, 0.9 g/t Au), using available material in the Wyoming core shed. These composites were prepared using the same or similar core to the previous test programs at BML in 2021-22.

The test work commenced in September 2024 and was completed at the end of November 2024, with the final report issued in January 2025. This was too late for thorough evaluation and inclusion in this PFS.
The testwork comprised five stages of work, which were:

- Conventional rougher flotation tests,
- Conventional three-stage cleaner tests,
- Jameson dilution tests,
- Locked Cycle tests,
- Finally, rougher tests were performed with the Jameson L150 Pilot Unit.

The results are provided in the January 2025 BML Report, Project 1702, and are summarized here.

Conventional Rougher Tests were carried out to ensure that the samples responded similarly to previous sulfide composites. Results summarized in Table 10.43 below.

Table 10.43: Conventional Rougher Summarized Test Results

	%Wt	%Cu	g/t Ag	g/t Au	Recovery %Cu	Recovery %Ag	Recovery %Au
Sulfide Comp, Ro Conc	5.0	5.27	18.4	13.5	89.0	76.2	75.5
Sulfide Comp2, Ro Conc	6.6	4.80	12.2	9.4	89.5	74.3	74.2

These recoveries are in line with previous test work. Collectors were PF4782 and PF7150, with H57 and MIBC frothers at pH9.1, and nominal grinds of 90u and 25u. These conditions were then applied to all tests.

Table 10.44: Conventional Cleaner Tests

	%Wt	%Cu	g/t Ag	g/t Au	Recovery %Cu	Recovery %Ag	Recovery %Au
Sulfide Comp, Cleaner Conc	1.0	23.4	84.6	91.6	79.5	64.7	72.3
Sulfide Comp 2, Cleaner Conc	1.2	22.6	60.2	50.8	84.5	74.8	68.5

Again, these results confirm previous work. They suggest gold recovery of 70-75% should be possible at the lower target concentrate grade of 18%Cu.

Glencore specifically developed this test procedure to predict performance in Jameson cells.

Table 10.45: Jameson Dilution Tests

	%Wt	%Cu	g/t Ag	g/t Au	Recovery %Cu	Recovery %Ag	Recovery %Au
Sulfide Comp, Ro Conc	4.4	5.5	21.0	14.3	87.2	76.5	72.7
Sulfide Comp 2, Ro Conc	5.6	5.2	14.6	10.4	87.5	74.0	73.7

These results are confirmation of and similar to the results of the conventional rougher tests.

Table 10.46: Locked Cyce Tests

	%Wt	%Cu	g/t Ag	g/t Au	Recovery %Cu	Recovery %Ag	Recovery %Au
Sulfide Comp, Cleaner Conc	0.9	25.1	90.0	63.7	83.0	81.0	64.8
Sulfide Comp2, Cleaner Conc	1.0	27.1	68.0	55.8	83.9	74.1	65.5

Note: the high copper concentrate grade. Both results suggest a 68-70% gold recovery with an 18%Cu concentrate grade could be possible. The regrind p80 was +/- 30u, now considered too coarse.

Glencore provided a new L150 pilot unit for rougher tests. Only a limited program was possible as approximately 18-20 kg of material is required for each test. After two trials, L150 tests were completed on both samples.

Table 10.47: L150 Pilot Unit

	%Wt	%Cu	g/t Au	Recovery %Cu	Recovery %Au
Sulfide Comp, Ro Conc	8.4	2.93	8.24	85.2	71.0
Sulfide Comp2, Ro Conc	6.0	4.35	9.07	88.0	67.4

These results might be disappointing after the previous rougher, cleaner, and dilution tests. Mitigating this were the relative inexperience of operating staff, limited material to optimize, and some issues associated with commissioning new equipment.

Based on these results, Wells considers that Jameson might not offer improved metallurgy for Copper King. A study commissioned by Glencore with an independent engineer has shown that capital and operating cost advantages exist with a Jameson circuit. Also, Glencore has offered to do a more comprehensive test work program at their own laboratory. This would, however, require the preparation of new, larger composites (preferably with more representative copper and gold LOM grades). Application of Jameson technology may, therefore, be worthy of further study if time allows.

11.0 MINERAL RESOURCE ESTIMATES

This Section has been updated from the "S-K 1300 Technical Report Summary CK Gold Project," dated Dec. 1, 2021, to include new drillhole datasets.

11.1 INTRODUCTION

The most recent CK resource estimate for gold, copper, and silver was last updated by Gustavson Associates in the report, "S-K 1300 Technical Report Summary CK Gold Project," dated December 1, 2021. The 2021 Mineral Resource Estimate (MRE) incorporated data from 35 infill drillholes, including 12 reverse circulation holes totaling 12,340 ft (3,760 m) and 23 core holes totaling 19,057 ft (5,810 m) completed by U.S. Gold, as well as five additional reverse circulation holes totaling 2,370 ft (830 m) and two rotary holes totaling 380 ft (115 m) intended as monitoring wells, drilled between 2017 and 2020. Drilling completed by U.S. Gold in 2021, which totaled 29,562 ft (9,010 m) in 24 reverse circulation and core holes, was ongoing and incomplete during the development of the 2021 MRE but is now included in this update.

Mark Shutty, CPG, Principal of Drift Geo LLC, used Leapfrog Geo/Edge software (2024.1) to construct the geologic models of the CK Gold deposit for the updated MRE. The constraining pit shell and in-pit resource reporting were completed using MinePlan (v16.2.1). This updated MRE includes all available drillhole datasets, including the previously unavailable 2021 drilling data completed by U.S. Gold.

A three-dimensional (3D) block model was constructed using the following standard procedures:

- Import topographic data to create a digital terrain model of the current surface topography.
- Import and review drillhole interval datasets using Leapfrog Geo tools.
- Construct implicit geological and mineralized domain models using Leapfrog, interpret oxidation state based on visual logging, and assign specific gravity values.
- Evaluate and model experimental variograms aligned with observed mineral trends, establishing ranges of sample influence for grade estimation.
- Estimate and validate gold, copper, and silver grades within the 3D block model.
- Classify mineral resources into confidence categories: Measured, Indicated, and Inferred.
- Apply economic constraints for resource reporting within an optimized pit shell.

11.2 GEOLOGIC MODELS

Beginning in 2020, U.S. Gold facilitated the relogging of all available legacy drill core to ensure consistent interpretation of rock types across the 2020 and 2021 drilling programs. U.S. Gold's geologic datasets were used to evaluate samples and construct 3D geological models in Leapfrog. The lithologic model predominantly consists of granodiorite (GD), with discrete occurrences of mylonite (MYL) and potassic-altered granodiorite (GDK). Mafic dykes, pegmatites and veins are relatively small bodies, and the drilling density is insufficient to model these as throughgoing features. Therefore, mafic dyke bodies were constructed in Leapfrog as discrete volumes and pegmatites were not modeled and assigned the host rock type.

The primary lithologic model for the CK Gold Project includes Proterozoic granodiorite (GD), with varying intensities of potassic alteration (GDK) and mylonitic fabrics (MYL). Mafic dikes (MD), younger pegmatites

(PEG), and undifferentiated veins (VN) represent a smaller volume within the mineralized granodiorite domain. Unmineralized domains were also modeled, including a metasediment unit (MSED) east of the Copper King Fault and overlying Quaternary cover (QC).

Leapfrog software was used to aggregate and model the GDK, MYL, and MD intrusive sub-units within the GD domain. The CK deposit trends NW-SE (290°-110°), and the general orientation for all modeled intrusive domains is at -70°, 020° (dip, dip direction). An anisotropy of 3:3:1 (ratio of maximum, intermediate, minimum) was applied for the GD domain, while an anisotropy of 5:5:1 was used for the internal GDK and MYL lithologies.

The geological model was employed to assign varying rock densities throughout the block model and establish an eligible volume for grade estimation. Longitudinal and cross-sectional reviews of the deposit show that mineralization generally follows the anisotropies of the lithologies, with most mineralization occurring within the central portion of the deposit. Figure 11.1.

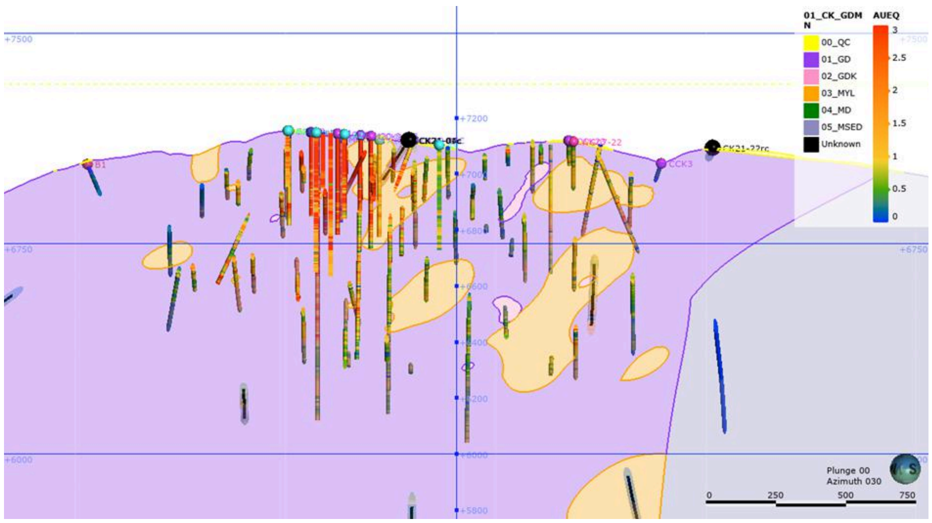


Figure 11.1: Vertical Section Looking 030deg Showing Lithologic Boundaries and Drillhole Grades (AUEQ gpt). 2021 drillholes are displayed with black collar points and downhole traces

An oxidation model was created using drillhole data in Leapfrog. Surfaces were generated to produce oxide, mixed, and sulfide solids based on logging in the database, Figure 11.2. A global isometric trend was applied for all surfaces. The oxidation methodology is discussed in Section 11.3.

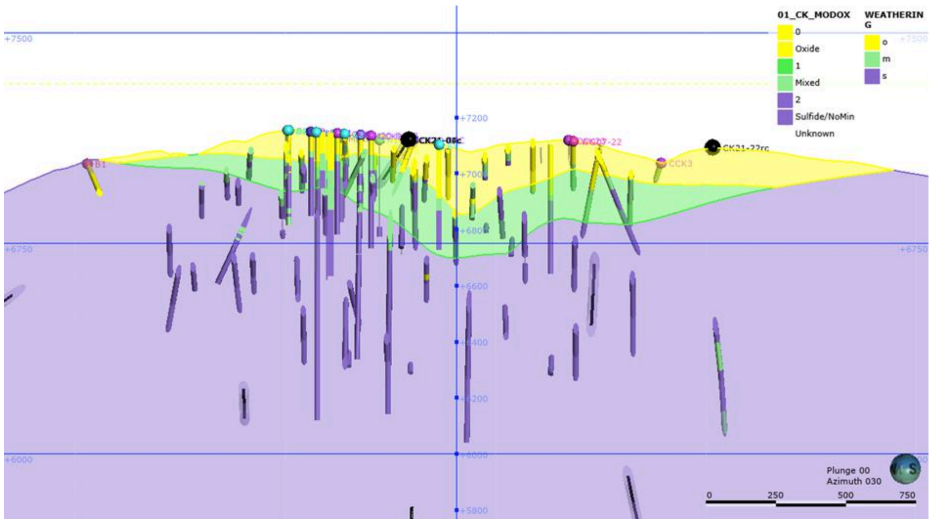


Figure 11.2: Vertical Section Looking 030° Showing Oxidation Boundaries and Drillhole Weathering. 2021 drillholes are displayed with black collar points and downhole traces

U.S. Gold geologists modeled fault surfaces in Leapfrog using surface exposure, geophysical survey data, and downhole televiewer data. Structure orientation data from the televiewer reconciliation work, interpreted by Piteau Associates, facilitated U.S. Gold's interpretation of additional faults for evaluation within the model space Figure 11.3. Mineralized drill samples within the fault-bound blocks were reviewed visually and statistically.

CK mineralization is bounded by a hard structural/lithologic boundary to the east by the Copper King Fault and constrained to the north, northwest, and west by more ambiguous NW Fault, NE 1 Fault, and West Block Faults, respectively. While the NE 2 Fault is projected to intersect the CK deposit, it remains a poorly defined feature, characterized by drillhole data as a broad zone of deeper oxidation and lower-grade mineralization Figure 11.4 and Figure 11.5. Therefore, bounding structures were used to constrain a single mineralized domain that accommodates the influence of the internal NE 2 Fault on mineralization and oxidation for use in the resource model.

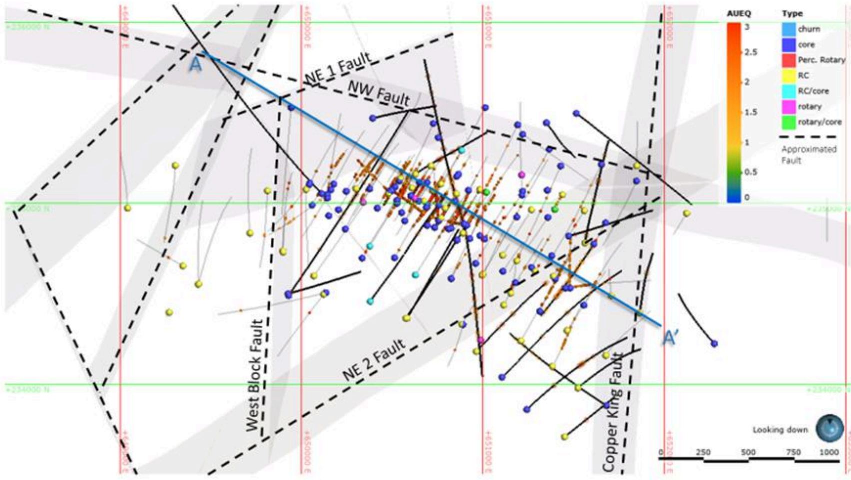


Figure 11.3: Fault Map with Drillhole Grades (≥ 1.5 gpt AUEQ). 2021 drillholes are displayed with thick black downhole traces

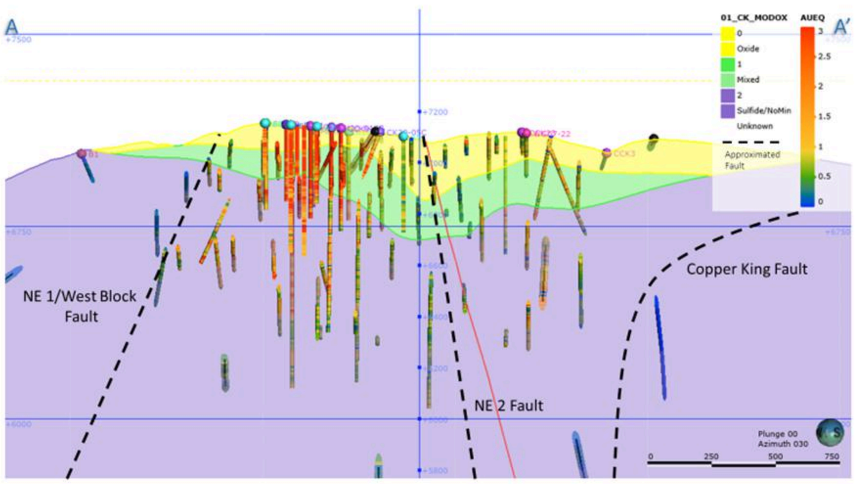


Figure 11.4: Vertical Section A-A' Looking 030° Showing Location of Interpreted NE 2 Fault Zone, Oxidation Boundaries and Drillhole Grades (AUEQ gpt). 2021 drillholes are displayed with black collar points and downhole traces

A hybrid numerical indicator model was developed using a calculated gold equivalent (AUEQ) variable at a 0.2 gpt cut-off. The model incorporates a varying structural trend that aligns with bounding faults, observed mineralization trends, and modeled intrusive anisotropies. This mineralization model represents a single domain that constrains estimated mineral resources within the modeled intrusive rock complex.

The AUEQ variable was calculated in Leapfrog using capped assay values for Au (AUCAP), Cu (CUCAP), and Ag (AGCAP), weighted by their respective price ratios relative to gold. The calculation employed the following metal prices: $1,900.00 USD/oz (Au), $23.00 USD/oz (Ag), and $4.00 USD/lb (Cu). The formula is as follows:

- AUEQ=[AUCAP]+([AGCAP]×0.01)+([CUCAP]×1.67)

The capping methodology is discussed in detail in Section 11.8.

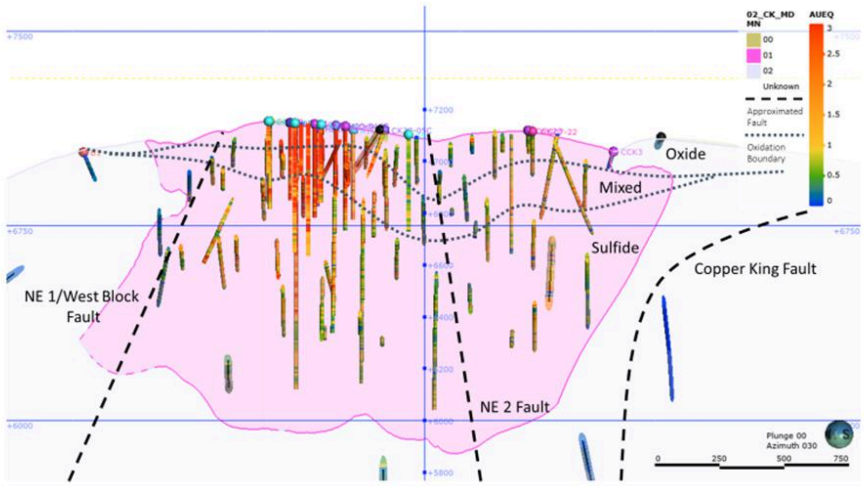


Figure 11.5: Vertical Section Looking 030° Showing Mineralized Domain, Modeled Oxidation, Structures and Drillhole Grades (AUEQ gpt). 2021 drillholes are displayed with black collar points and downhole traces

11.3 OXIDATION ASSIGNMENT

Metallurgical testing of mineralized rock indicates that sulfide recovery is a function of oxidation state. During core logging, geologists visually estimated the oxidation state and categorized it as either oxide, mixed, or sulfide. The oxidation boundary contacts were modeled in Leapfrog to encompass logged oxidation intervals and modeled structures, resulting in a series of surfaces used to code the block model.

11.4 BLOCK MODEL ORIENTATION AND DIMENSIONS

A 3D model with 20 ft x 20 ft x 30 ft block dimensions was defined to accommodate the CK deposit and optimization pit shell while facilitating the use of a 30' bench height mining unit. All work was conducted using the NAD83 Wyoming State Plane East coordinate reference system, using imperial units of feet. The block model maintains a north-south and east-west orientation with no rotation and is not sub-blocked. The block model dimensions, and model limits are shown in Table 11.1.

Table 11.1: Block Model Dimensions

Parameter	Minimum	Maximum	Unit Block Size	Number of Blocks
Northing	233,2000	237,000	20	250
Easting	648,810	653,810	20	200
Elevation	5,090	7,400	30	977

11.5 COMPOSITING

Nominal sample lengths vary by drill program, but drillholes used in the resource model have a global mean sample length of 5.1 ft. Capped assay intervals were composited to 10-foot fixed-length intervals within the mineralized domain for use through Ordinary Kriging estimation, described in Section 11.10, with the model's block size (20'x20'x30'). This method computes 10-foot composite intervals down each drillhole, and length-weight averages the portions of assay intervals that fall within the 10-foot interval. Composites were broken at the mineralized domain boundary using a 50% threshold, with specified handling of residual lengths of less than 5 ft to be added to the previous interval. Descriptive statistics of lengths and metal grades for the raw (original) and composited samples were compared and reviewed in 3D as a means of validation, Table 11.2.

Table 11.2: Block Model Dimensions

	Au		Cu		Ag	
	Composited	**Original**	**Composited**	**Original**	**Composited**	**Original**
Count	8,099	15,819	8,099	15,819	6,015	12,393
Length	80,926	80,910	80,926	80,910	60,141	59,948
Mean	0.58	0.58	0.19	0.19	1.48	1.48
SD	0.79	0.85	0.15	0.17	1.59	1.75
CV	1.37	1.46	0.83	0.92	1.07	1.18
Variance	0.63	0.71	0.02	0.03	2.53	3.08
Minimum	0.00	0.00	0.00	0.00	0.05	0.05
Maximum	9.94	11.00	3.00	3.00	20.00	20.00

11.6 EXPLORATORY DATA ANALYSIS

Raw drillhole sample data and logged lithology data were reviewed visually within Leapfrog's 3D environment and statistically using a merged assay-lithology dataset. Attributes such as drillhole program, type, operator, and location were evaluated against the primary Au and Cu variables, as well as an Au equivalency variable (AUEQ), to identify a subset of drillholes suitable for resource estimation.

In the mineralized resource area, Caledonia's 1987-era drilling—comprising 25 vertical percussions rotary drillholes totaling 9,980 ft—was deemed unsuitable for inclusion in the resource model. This decision was based on potential sample contamination associated with the drilling method, the vertical orientation of the drillholes, missing Cu assays, and the use of composited sample intervals. Drillholes far outside the mineralized resource area were also excluded from the resource drillhole database.

Datasets from 24 RC and core drillholes completed by U.S. Gold in 2021, totaling 29,562 ft (9,010 m), were integrated into the drillhole database. These datasets were evaluated for compliance and utilized to refine the modeled geology, oxidation, and mineralization. All other drillholes meeting the required criteria were included in the resource drillhole database Table 11.3.

Table 11.3: Drillhole Database Summary

Operator & Program	Drillhole Count	Sum of Drilling (ft)
U.S. Gold	**59**	**60,132**
2021	24	29,562
2020	25	20,449
2018	8	8,090
2017	2	2030
Saratoga Gold	**35**	**25,462**
2008	8	7,167
2007	27	18,295
Mountain Lake 1997	**4**	**1,880**
Compass 1994	**25**	**9,202**
Henrietta 1973	**9**	**3,073**
ASARCO/Henrietta 1973	**1**	**700**
ASARCO	**12**	**3,963**
1970	7	2,563
1938	5	1,400
USBM	**3**	**2,630**
Copper King	**6**	**2,630**
Grand Total	**154**	**109,673**

1. Table of drillholes used in the resource model.

Furthermore, metal grades were evaluated against logged and modeled lithologic, structural, and oxidation domains in combination with surface geology and interpretive geophysical overlays to delineate mineralized trends and define domains for geostatistical analysis.

A series of contact plots and box plots for the principal metals (Au and Cu) were generated to evaluate the distribution of these variables within the CK's major mineralized host rock types (GD, GDK, and MYL). Statistical box plots, Figure 11.6 and Figure 11.7, for the intrusive host rocks reveal similarly elevated metal grades and contact plots.

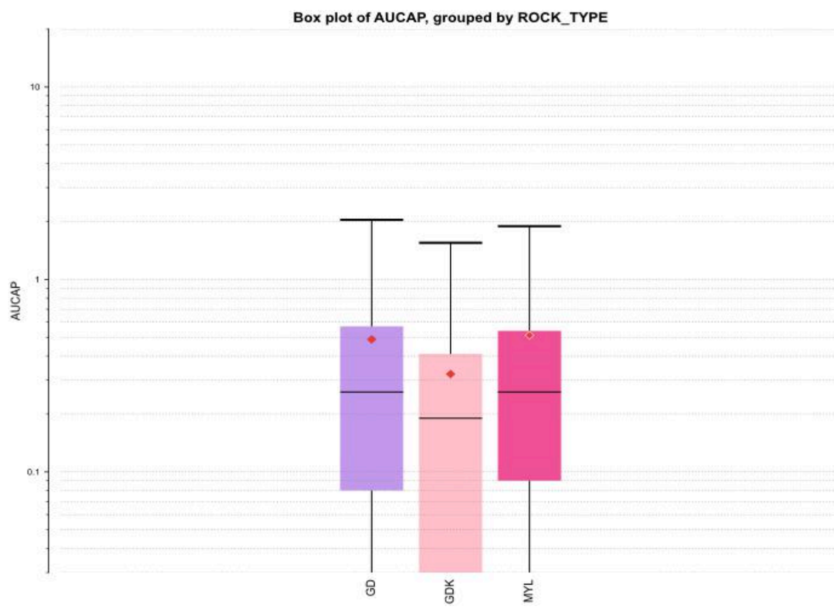


Figure 11.6: Log box Plot for AUCAP (gpt) Variable by Host Rock

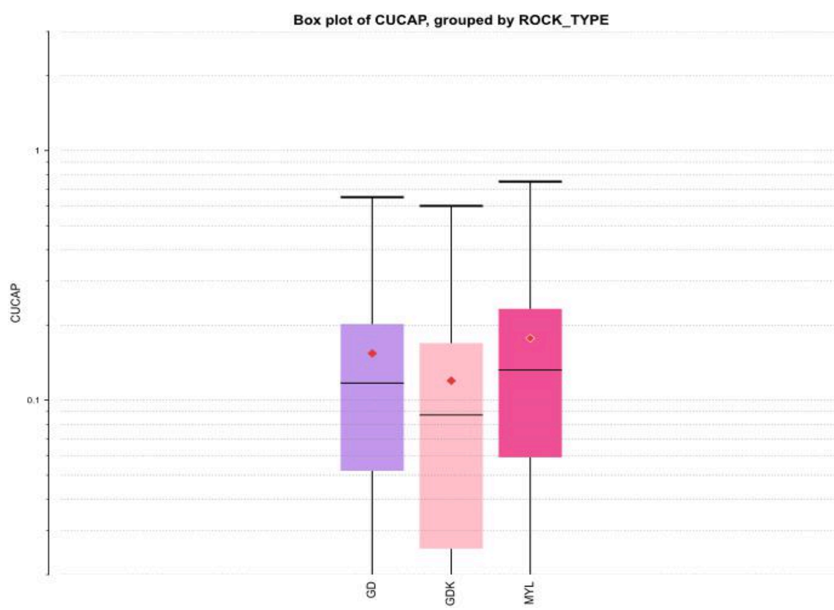


Figure 11.7: Log Box Plot for CUCAP (%) Variable by Host Rock

Figure 11.8 demonstrates gradational Au and Cu grade changes between the logged lithologies. The GD and MYL hosts generally have nearly identical Au and Cu sample populations, while metal grades in the altered GDK host are lower.

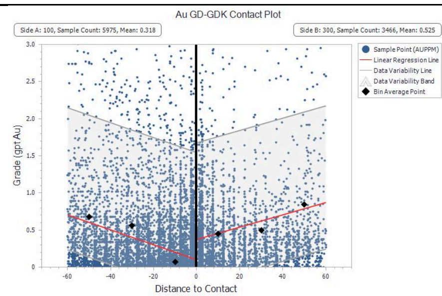
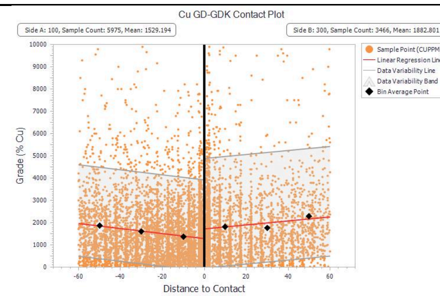
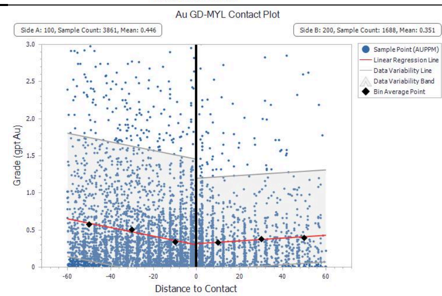
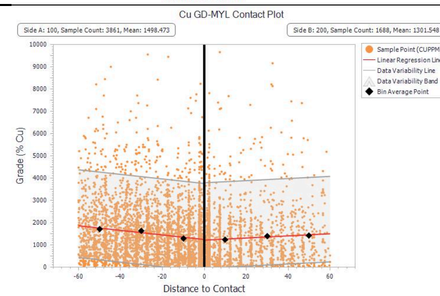
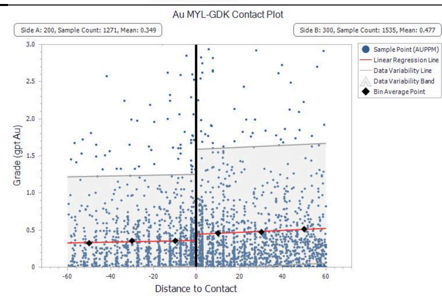
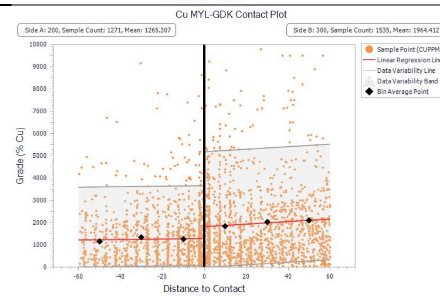
<table>
<tr>
<td>GD (l) GDK (r)
Contact Plots</td>
<td>
</td>
<td>
</td>
</tr>
<tr>
<td>GD (l) MYL (r)
Contact Plots</td>
<td>
</td>
<td>
</td>
</tr>
<tr>
<td>MYL (l) GDK (r)
Contact Plots</td>
<td>
</td>
<td>
</td>
</tr>
<tr>
<td></td>
<td>Au (g/t) Contact Plots – 60’ range</td>
<td>Cu (%) Contact Plots – 60’ range</td>
</tr>
</table>

Figure 11.8: Contact plot showing binned mean sample grades for the Au (blue) and Cu (orange) variables within a 60 ft distance

For the resource model, the major mineralized rock types were grouped based on sample population similarities and shared lithologic genesis (Figure 11.9).

Granodiorite (GD), potassic-altered granodiorite (GDK), and mylonite (MYL) are interpreted to originate from the same granodiorite protolith. MYL exhibits superimposed mylonitic textures, while GDK displays gradational potassic alteration.

Approximately 94% of the deposit's total contained gold (Au) and copper (Cu) is hosted within samples logged as GD or MYL, while the remaining ~6% is associated with GDK. Potassic-altered granodiorite (GDK) is primarily located at the periphery of the deposit's higher-grade GD-MYL core.

Modeled sediments to the east of the Copper King Fault are unmineralized and sparsely drilled.

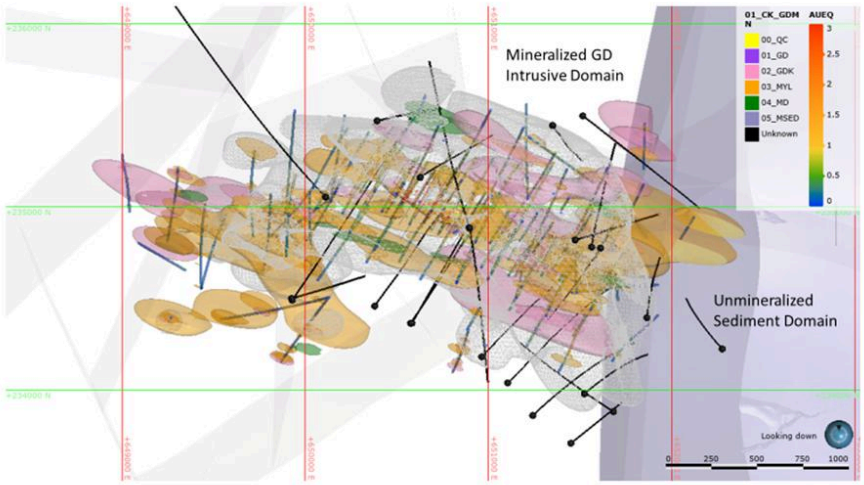


Figure 11.9: Geology and Mineralization (transparent gray wireframe) with Drillhole Grades (gpt AUEQ). 2021 drillholes are displayed with black collar points and downhole traces.

11.7 BULK DENSITY DETERMINATION

There are no records of bulk density measurements before 2007-2008, during which Saratoga performed 1,336 drill core sample density tests. U.S. Gold later added 80 density measurements through their drilling programs, bringing the current bulk density database to 1,416 determinations.

Approximately 47% of the samples are from the primary mineralization host, granodiorite. The results reveal minimal variation in specific gravity with depth and a small standard deviation for each rock type, indicating consistent bulk density characteristics across the deposit.

A comparison of bulk density relative to depth for granodiorite is presented in Figure 11.10, the other rock types exhibit a similar uniformity with depth.

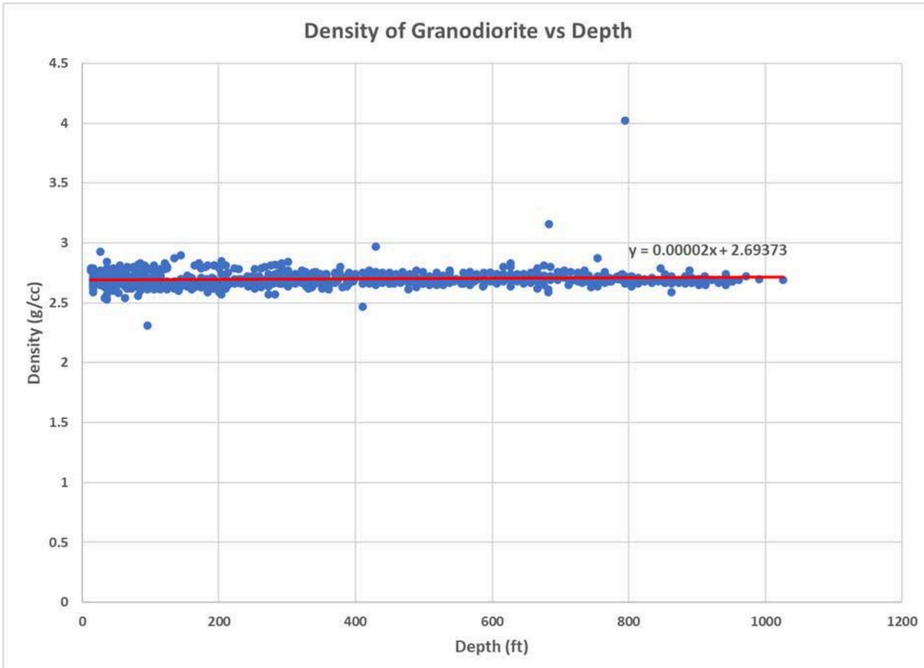


Figure 11.10: Density of Granodiorite vs Depth

The bulk density values were converted to tonnage factor (st/ft^3) and assigned to the block model by rock type, Table 11.4. The core is generally whole "stick rock" with infrequent broken zones. Therefore, no deduction from density measurements to account for fracture zones is warranted at this time and should continue to be monitored.

Table 11.4: Bulk Density Values by Rock Type

Rock Type	# of Determinations	Density Average (g/cm3)	Std Dev of Density	Tonnage Factor (st/ft3)
Granodiorite	665	2.70	0.08	0.0843
Potassic-Altered Granodiorite	273	2.68	0.06	0.0837
Mafic Dike	55	2.81	0.10	0.088
Mylonite	372	2.70	0.07	0.0843
Not Logged	13	2.69	0.10	0.0843
Pegmatite	33	2.94	0.06	0.0821
Unknown	5	2.70	0.10	0.0843
Total	1,416	2.70	0.08	0.0843

There is no density data available for overburden. An SG value of 1.8 g/cm^3 (0.0562 st/ft^3 was assigned to blocks coded as quaternary cover.

11.8 GRADE CAPPING/OUTLIER RESTRICTIONS

Raw gold, copper, and silver assays were evaluated within the Resource drillhole database with histogram and probability plots to identify statistical outliers. These data are generally reflective of a single sample population with few outliers. Outliers were examined to ensure they were not the result of a database transcription error and were geologically reasonable; the location of high-grade samples with respect to nearby samples, lithology, and oxidation was reviewed ahead of establishing capping thresholds, which generally occur at distribution changes noted in the individual metal probability plots Figure 11.11.

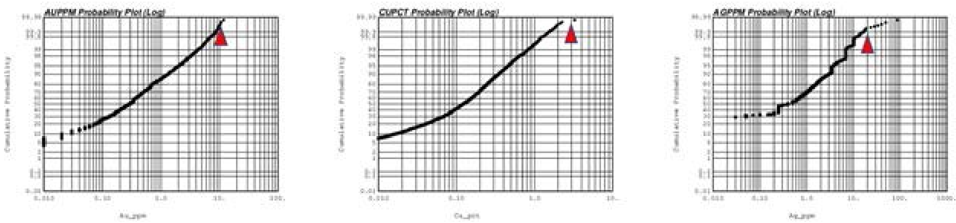


Figure 11.11: Sample Distribution

Capping was applied using a calculation within the database, with capped results stored in newly defined fields (AUCAP, CUCAP, and AGCAP), which were used for sample compositing and resource estimation.

Gold (Au) is capped at 11.0 gpt, Cu is capped at 3.0 % and Ag is capped at 20.0 gpt. The impact of capping is presented in the table below, which summarizes the number of samples affected by capping and the total metal reduction Table 11.5.

Table 11.5: Capping Thresholds and Metal Loss Table

Grade Item	Capping Threshold	Capped Samples	Metal Loss (%)
Au	11.0 gpt	4	0.28%
Cu	3.00%	5	0.36%
Ag	20.0 gpt	8	1.54%

11.9 VARIOGRAPHY

Experimental pairwise relative variograms for the AUCAP, CUCAP, and AGCAP variables were generated to evaluate sample variance, establish search ellipse parameters, and model variograms for grade estimation via ordinary kriging within Leapfrog's Edge module. All variography was completed using 10.0 ft fixed-length composite samples from resource drillholes falling within the mineralized wireframe domain, with a -74.0° (dip), 26.0° (dip dir.), 100.0° (pitch) orientation, Figure 11.12 and Figure 11.13. This geometry accommodates the apparent steep, NNE-dipping Au-Cu core and shallow SSW-dipping mineralization observed outside of the mineralized core.

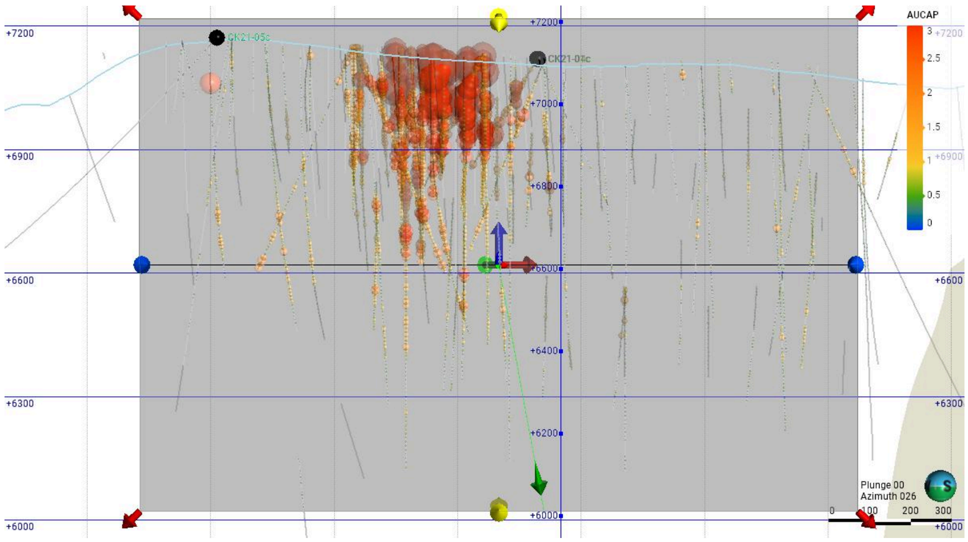


Figure 11.12: Au Composite Points for Resource Drillholes, looking 026° at Plane of Best-fit Mineralization (green arrow indicating 100° pitch) used for Spatial Modeling (Variography)

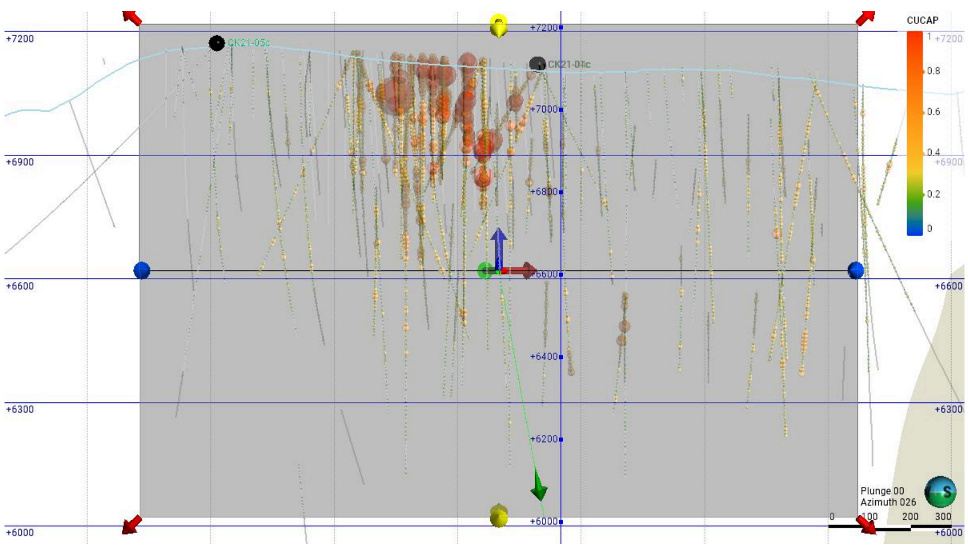


Figure 11.13: Cu Composite Points for Resource Drillholes, looking 026° at Plane of Best-fit Mineralization (green arrow indicating 100° pitch) used for Spatial Modeling (Variography)

Variograms were modeled for the AUCAP, CUCAP, and AGCAP variables using a nugget component and two additional structures, Figure 11.14. The best-fit orientation of the major, intermediate, and minor axis (-74°, 026°, 100°) for the primary AUCAP and CUCAP variables was applied to AGCAP variable, Table 11.6.

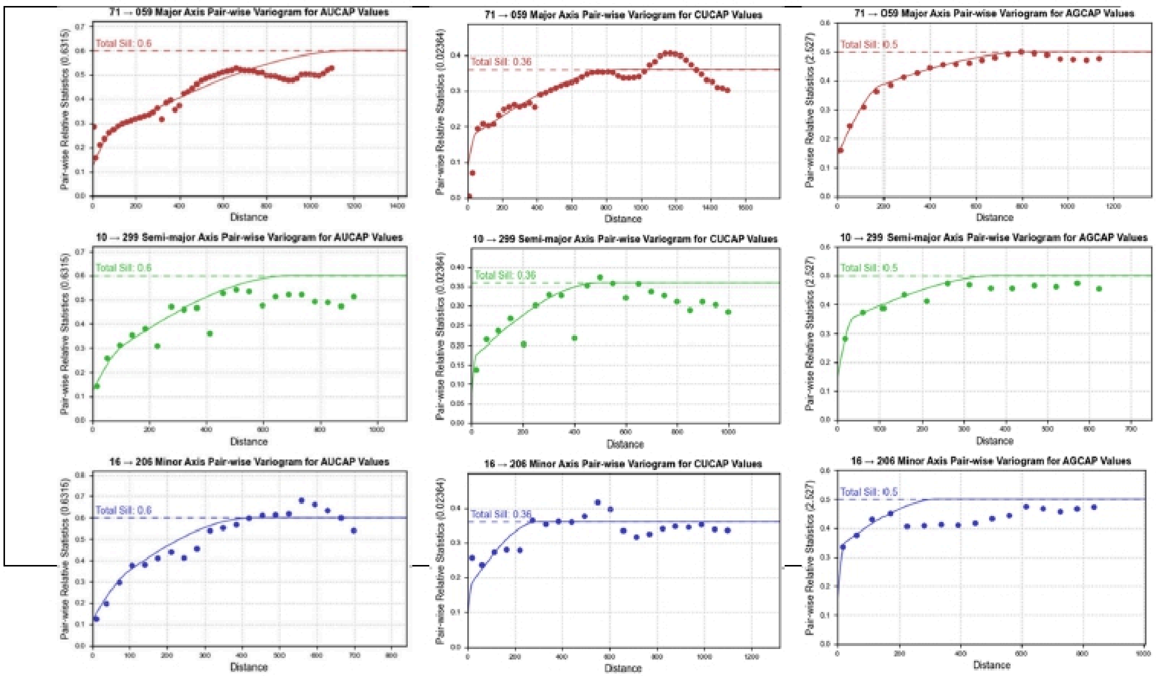


Figure 11.14: Pairwise relative variograms and modeled structures for Major (top), Intermediate (middle) and Minor axis (bottom) for AUCAP (left), CUCAP (center), and AGCAP (right)

Table 11.6: Variogram Parameter Table

General	Direction (deg.)				Structure 1					Structure 2				
Variogram	**Dip**	**Dir.**	**Pitch**	**Nugget**	**Sill**	**Structure**	**Major (ft)**	**Semi-major (ft)**	**Minor (ft)**	**Sill**	**Structure**	**Major (ft)**	**Semi major (ft)**	**Minor (ft)**
AUCAP	74	26	100	0.12	0.07	Spherical	100	110	99	0.38	Spherical	1200	700	431
CUCAP	74	26	100	0.14	0.48	Spherical	200	40	25	0.17	Spherical	850	380	325
AGCAP	74	26	100	0.08	0.00	Spherical	50	20	20	0.20	Spherical	900	500	300

11.10 ESTIMATION/INTERPOLATION METHODS

The behavior of metal-grade populations within the modeled mineralization domain was analyzed to establish appropriate estimation procedures for the Au, Cu, and Ag variables. Hard boundaries were applied to restrict the influence of composites within the mineralized domain, ensuring that only composites inside the domain contributed to grade estimation for blocks within the same domain. For estimation, original sample grades, capped as necessary, were composited to fixed 10-foot lengths within the mineralized domain. A two-pass Ordinary Kriging (OK) strategy was employed to estimate metal grades throughout the mineralized domain within the 3D block model. This approach utilized metal-specific variogram models for the primary AUCAP (gold) and CUCAP (copper) variables, while AGCAP (silver) was estimated using a single OK pass. Estimation search parameters and sample criteria for each OK pass for Au, Cu, and Ag are summarized in Table 11.7.

A hierarchical approach was applied for the Au and Cu estimators, with high-confidence estimates requiring composites from multiple drillholes over shorter ranges superseding lower-confidence estimates based on composites sourced from greater distances. Nearest Neighbor (NN) estimators were also defined and used to validate the estimated resource models.

Table 11.7: Estimation Search and Sample Parameters

	Ellipsoid Ranges (ft)			Ellipsoid Directions (deg.)			Number of Samples		
Interpolant	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Min	Max	Max per Hole
AUCAP_OK1	400	220	140	74	26	100	4	30	2
AUCAP_OK2	200	110	70	74	26	100	4	30	2
CUCAP_OK1	400	220	160	74	26	100	4	30	2
CUCAP_OK2	200	110	80	74	26	100	4	30	2
AGCAP_OK1	400	200	160	74	26	100	4	12	2

11.11 CLASSIFICATION OF MINERAL RESOURCES

The estimated block grades were classified into Measured, Indicated, and Inferred resource categories based on a combination of estimator attributes and composite sample parameters to ensure cohesive resource block assignment.

- Measured Classification: Blocks were assigned a Measured classification if their metal grades were estimated during the high confidence pass for the primary metal (AUCAP_OK2), using composites from two or more drillholes and an Ordinary Kriging (OK) variance ≤ 0.20.
- Indicated Classification: Blocks were assigned an Indicated classification if they were estimated with the same interpolant (AUCAP_OK2) using composites from two or more drillholes and an OK variance ≤ 0.225.
- Inferred Classification: All remaining estimated blocks within the constraining mineralized domain were classified as Inferred.

The Kriging variance parameter is an additional distance-correlation metric derived from the more restrictive Au spatial model. This approach ensures that resource classification reflects the confidence in grade estimation and spatial continuity of sample locations.

Figure 11.15 and Figure 11.16, display a longitudinal section and a cross-section, respectively, of the classified estimated blocks.

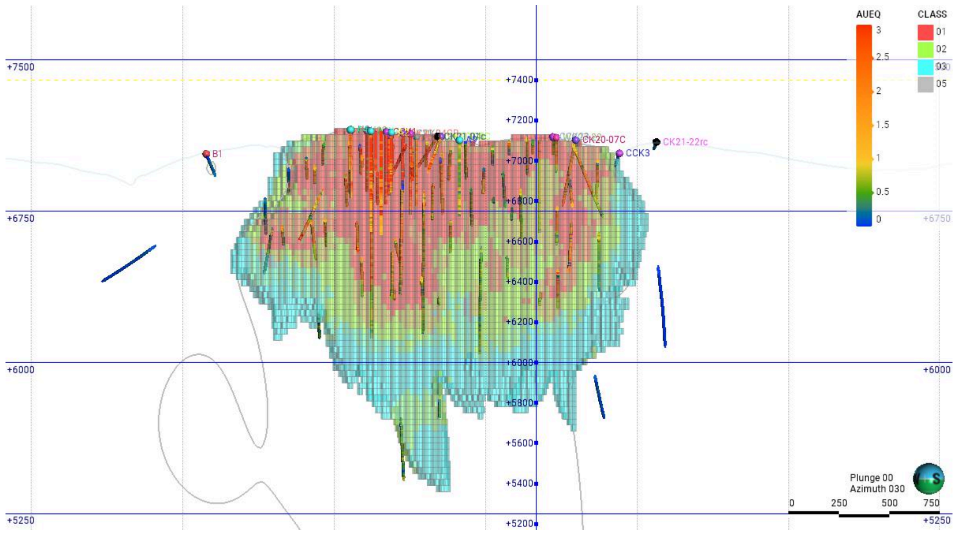


Figure 11.15: Longitudinal (100 ft field of view), looking 030° through the 3D block model, showing Measured (red), Indicated (green) and Inferred (blue) class resources with 2021 drillholes displayed with black collar points.

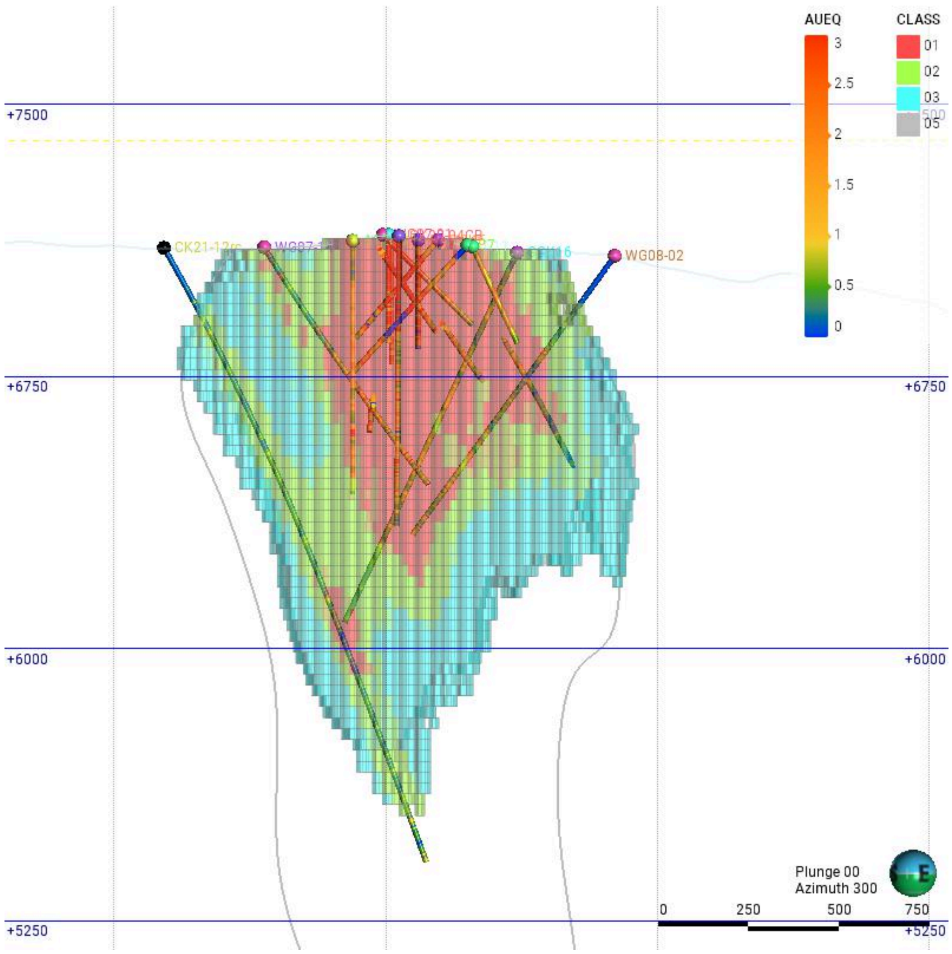


Figure 11.16: Cross-section slice (100 ft field of view), looking 300° through the 3D block model, showing Measured (red), Indicated (green) and Inferred (blue) class resources with 2021 drillholes displayed with black collar points.

11.12 GRADE MODEL VALIDATION

The estimated Ordinary Kriging (OK) grades and the extent of interpolated mineralization were reviewed visually against drillhole composites using bench-level and section slices in Leapfrog's 3D environment and validated through statistical methods Figure 11.17. A strong correlation between drillhole composite grades and estimated block grades was observed.

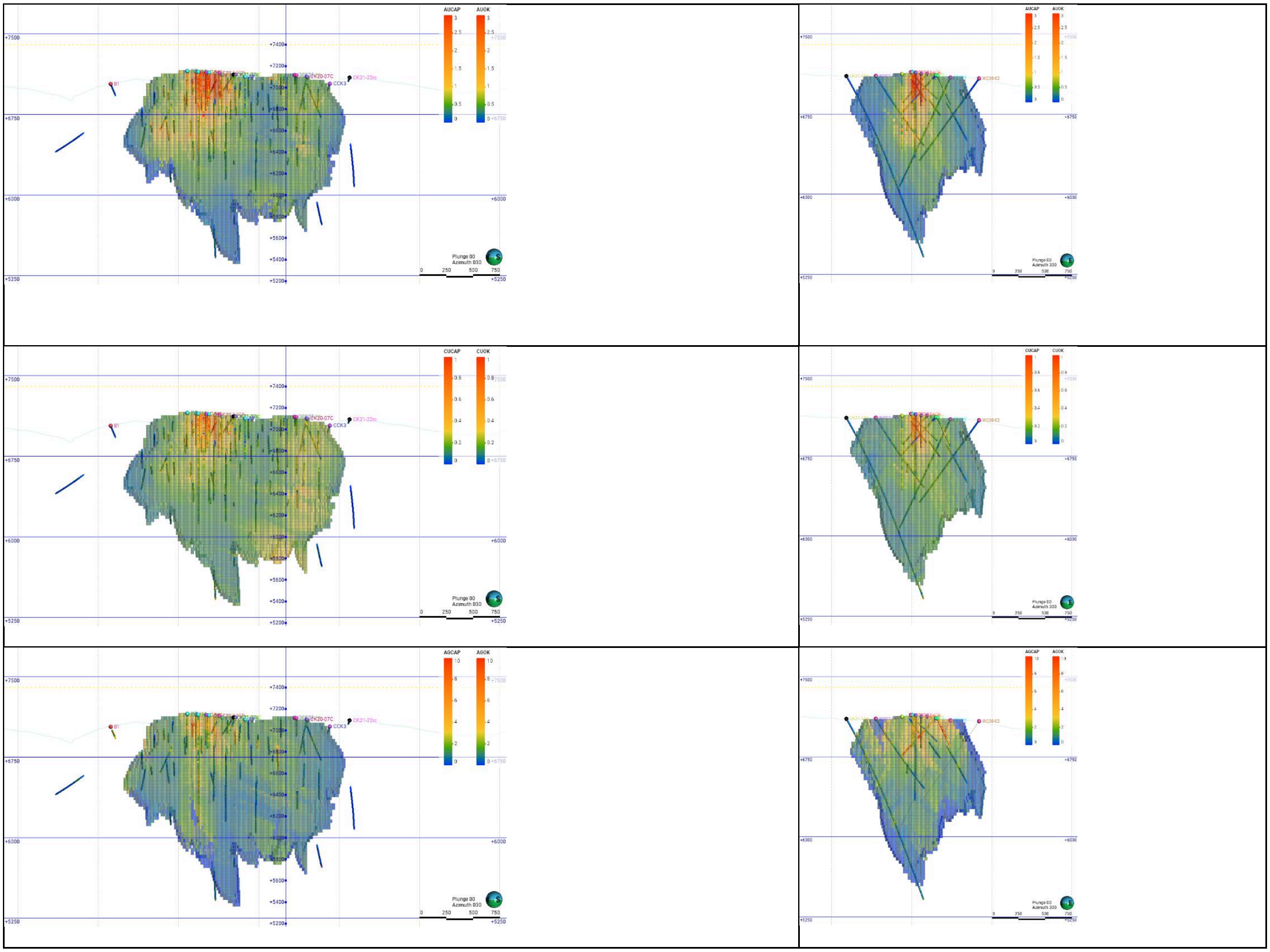


Figure 11.17: Model validation slices (longitudinal and cross-section), with 100 ft field of view looking 030° and 300° respectively, through the Au (top), Cu (center) and Ag (bottom), 2021 drillholes are displayed with black collar points.

These model validation slices (longitudinal and cross-section), have a 100 ft field of view looking 030° and 300° respectively, through the Au (top), Cu (center), and Ag (bottom) showing estimated resource block models with 10 ft composites displayed along drillhole traces. Analytical results for the 2021 drillholes display black collar points and downhole traces showing the grade and distribution of Au, Cu, and Ag sample intervals against estimated block grades within the constraining mineralized domain.

Global estimated OK metal grades were compared to global estimated Nearest Neighbor (NN) grades at a 0.0 AuEq cut-off for all classified resources within the modeled mineralized domain as a means of identifying global bias (Table 11.8). The estimated metal grades between the OK and NN models for Au, Cu, and Ag were found to be within acceptable tolerances (±1.5%):

- **Au (OK vs. NN):** OK grades are 0.39% lower than NN grades.
- **Cu (OK vs. NN):** OK grades are 1.06% higher than NN grades.
- **Ag (OK vs. NN):** OK grades are 0.31% higher than NN grades.

Table 11.8: Global Estimation Comparison

Domain	Cut-off (AUEQ)	Density	Mass	AUOK	AUNN	AGOK	AGNN	CUOK	CUNN
	gpt	ft^3/sh. Ton	kt	gpt	gpt	gpt	gpt	%	%
MDMN	0.00	11.81	162,854	0.333	0.334	1.25	1.24	0.147	0.145

Local bias was evaluated using directional swath plots (Figure 11.18) to compare mean grades and volumes of OK and NN estimations for Au, Cu, and Ag. Differences in mean grades observed between the two models existed in areas outside the pit-constrained resources, representing relatively small volumes of Inferred Class fringe mineralization along the margins of the modeled deposit.

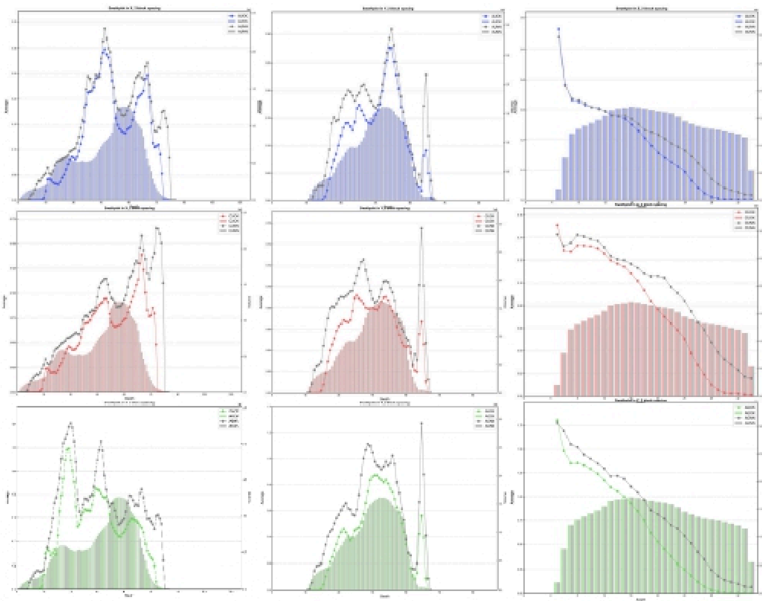


Figure 11.18: X (left), Y (center) and Z (right) swath plots showing mean grades and volume histograms

These X (left), Y (center) and Z (right) swath plots show mean grades and volume histograms for the AUOK/AUNN models (blue/gray, top), the CUOK/CUNN models (red/gray, middle), and the AGOK/AGNN models (green/gray, bottom)

An additional validation step was completed to evaluate the introduction of any litho-metal bias, particularly within the lower-grade GDK lithologic domain. Estimated OK resources within the modeled GDK domain contained 6% (±2%) of the deposit's combined Au and Cu, while the more similar GD and MYL lithologic domains contained the remaining 94% (±2%). While no matching lithologic/block coding between blocks and composites was used during estimation, drill density was sufficient to yield resources that retain identical original logged coding to raw assay litho-metal ratios.

11.13 REASONABLE PROSPECTS OF EVENTUAL ECONOMIC EXTRACTION

The Mineral Resources presented are confined within a pit optimization excavation limit, with a breakeven cut-off grade applied. The Lerchs–Grossmann (LG) pit optimization method establishes an economic excavation limit based on project parameters, including metal prices, recovery rates, operating costs, 50° slope and a 150ft (45.7m) drainage buffer. The metal prices used for pit optimization and the determination of the gold-equivalent cut-off grade are based on historical data and are intended to reflect long-term estimates. Metal prices and recoveries used in the LG pit optimization process are shown in Table 11.11 and Table 11.12, respectively.

A cut-off grade is applied to differentiate resource material from waste within the excavation limit. The internal cut-off grade is based on the gold-equivalent grade (AuEq) and is calculated as follows: Cut-off Grade = Cost / (Metal Value * Metallurgical Recovery %)

The AuEq grade item simplifies the representation of secondary metals (Cu and Ag) by expressing them as an equivalent grade of the primary metal (Au). This conversion allows the mass of copper and silver in each resource block to be expressed as an equivalent mass of gold, which is then added to the gold content of the block. The AuEq ratio for secondary metals is calculated on a recovery-weighted basis for each ore type (oxide, mixed, and sulfide). The following example illustrates the calculation of the cut-off grade for sulfide material:

1. Calculate the AuEq ratio for copper (Cu): Cu AuEq ratio = (Realized Cu price × Cu recovery) / (Realized Au price × Au recovery)
2. Calculate the AuEq ratio for silver (Ag): Ag AuEq ratio = (Realized Ag price × Ag recovery) / (Realized Au price × Au recovery)
3. Determine the sulfide AuEq grade: Sulfide AuEq grade = Au grade + (Cu × Cu AuEq ratio) + (Ag × Ag AuEq ratio)

This process ensures that all metal grades are expressed consistently, facilitating the accurate application of economic and technical criteria in resource estimation and reporting.

Table 11.9: AuEq Definitions

Value	Equation
Realized gold price	Au Market Price * (1-Royalty %)
Realized copper price	Cu Market Price * (1-Royalty %)
Realized silver price	Ag Market Price * (1-Royalty %)
Au recovery	Varying Avg. (55% Oxide/Mixed, 64% Sulfide)
Cu recovery	Varying Avg. (30% Oxide, 78% Mixed, 87% Sulfide)
Ag recovery	Varying Avg. (61% Oxide/Mixed, 70% Sulfide)

Table 11.10 contains the AuEq cut-off grades used in the Mineral Resource statement. Table 11.11 shows the metals pricing used in the LG cut-off grade calculation, and Table 11.12 indicates the LG recovery parameters for metals assigned oxide, mixed and sulfide material types.

Table 11.10: AuEq Cut-off Grades

Material Type	Imperial		Metric	
Oxide	0.011	oz/ton	0.39	g/tonne
Mixed	0.011	oz/ton	0.39	g/tonne
Sulfide	0.010	oz/ton	0.34	g/tonne

Table 11.11: Metal Prices (LG and AuEq Cut-off)		
Royalty*	2.1	%
Gold Market Price	1900	$/oz
Gold Realized Price	1860.10	$/oz
Copper Market Price	4.00	$/lb.
Copper Realized Price	3.92	$/lb.
Silver Market Price	23.00	$/oz
Silver Realized Price	22.52	$/oz

*Net royalty value is sourced from table 12.1.1

Table 11.12: Varying Metal Recoveries by Material Type (LG)				
Metal	**Material by Grade Bin**	**Oxide**	**Mixed**	**Sulfide**
Au	Oxide (<0.30 gpt)	50%		
	Oxide (0.30gpt - 1.30 gpt)	60%		
	Oxide (>1.3 gpt)	84%		
	Mixed (<0.27 gpt)		50%	
	Mixed (0.27gpt - 1.00 gpt)		60%	
	Mixed (>0.65 gpt)		87%	
	Sulfide (<0.35 gpt)			60%
	Sulfide (0.35gpt - 0.65 gpt)			67%
	Sulfide (>1.3 gpt)			88%
Cu	Oxide/Mixed (<0.10%)	20%	50%	
	Oxide/Mixed (0.10% - 0.40%)	30%	78%	
	Sulfide (<0.15%)			84%
	Sulfide (0.15% - 0.40%)			78%
	Oxide/Mixed/Sulfide (>0.40%)	35%	80%	88%
Ag	Oxide/Mixed/Sulfide (<0.5 gpt)	50%	58%	0%
	Oxide/Mixed/Sulfide (>0.5 gpt)	61%	61%	70%

11.14 MINERAL RESOURCE STATEMENT

Mark Shutty, CPG is the Qualified Person (QP) responsible for the mineral resource estimation in Table 11.13 and Table 11.14. The QP believes that the presented resources reasonably represent the in-situ resources for the CK Gold Project using all available data as of the effective date. The resources are reported with an applied AuEq cut-off grade and inside an optimized pit shell, ensuring reasonable prospects for eventual economic extraction.

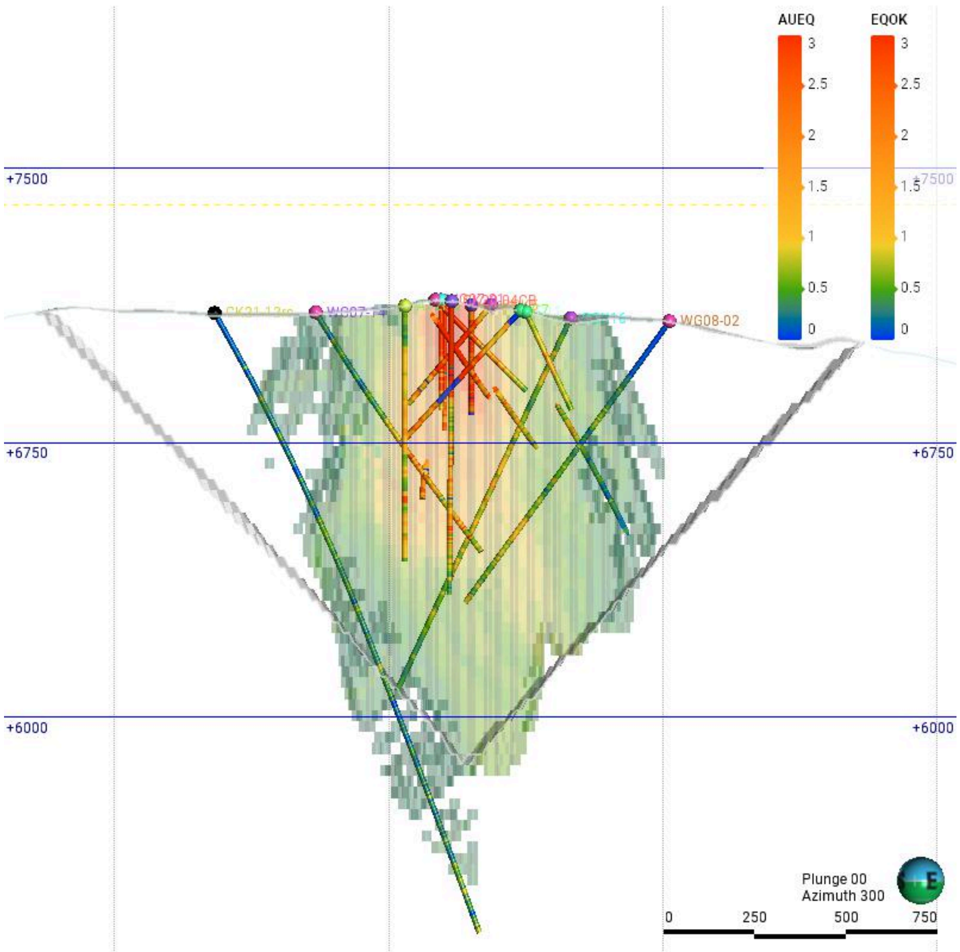


Figure 11.19 Cross section showing AuEq resources (>0.3 gpt cutoff) and constraining LG pit shell.

Table 11.13: Mineral Resource Statement									
	Mass	Gold (Au)		Copper (Cu)		Sliver (Ag)		Au Equivalent (AuEq)	
	Tons (000's)	Oz (000's)	oz/st	lbs (millions)	%	Oz (000's)	oz/st	Oz (000's)	oz/st
Measured (M)	36,400	608	0.0167	138	0.189	1,703	0.047	975	0.0268
Indicated (I)	51,200	544	0.0106	163	0.159	1,901	0.037	1,001	0.0195
M + I	**87,600**	**1,152**	**0.0131**	**301**	**0.172**	**3,604**	**0.041**	**1,976**	**0.0226**
Inferred	34,900	334	0.0096	112	0.161	1,073	0.031	653	0.0187

1. Mineral resources are estimated using Ordinary Kriging, constrained by geological domains based on lithology and mineralization controls. The underlying datasets supporting the resource estimate have been reviewed, validated, and verified by the Qualified Person (QP).

2. Mineral resources are reported in short tons within an optimized pit shell, using a breakeven gold equivalent (AuEq) cut-off grade of 0.011 oz/st for Oxide and Mixed material and 0.010 oz/st for Sulfide material. The overall average AuEq cut-off grade for all reported resources is 0.010 oz/st. No dilution or mining recovery factors have been applied.

3. The AuEq cut-off grade is calculated using realized metal prices of $1,860.10/oz Au, $3.92/lb Cu, and $22.52/oz Ag, with average metallurgical recoveries by oxidation type as follows:

   Gold (Au): 55% (Oxide/Mixed), 64% (Sulfide)
   Copper (Cu): 30% (Oxide), 78% (Mixed), 87% (Sulfide)
   Silver (Ag): 61% (Oxide/Mixed), 70% (Sulfide)

4. The optimized pit shell was generated using the Lerchs-Grossman method, incorporating all classified resources, realized metal prices, $2.50/ton mining costs, $9.20/ton processing costs, a 50° slope angle, and varying metallurgical recoveries as detailed in Table 11.12.

5. No dilution or mining recovery factors have been applied to the resource estimate.

6. There are no known legal, environmental, or permitting issues that impact the reported resources.

7. Resources are reported within the company's permitted land tenure/exploration license boundaries.

8. Mineral resources are classified in accordance with S-K 1300 definitions and are reported inclusive of mineral reserves.

9. Rounding may result in minor discrepancies in tonnage, grade, and contained metal totals.

10. There is no guarantee that mineral resources will be converted to mineral reserves.

11. The mineral resource estimates were prepared, reviewed, and validated by Mark Shutty, CPG, the independent Qualified Person (QP) for these estimates, in accordance with S-K 1300 Definition Standards adopted December 26, 2018.

12. The effective date of the mineral resource estimate is January 6, 2025.

Table 11.14: Mineral Resource Statement (Metric)									
	Mass	Gold (Au)		Copper (Cu)		Sliver (Ag)		Au Equivalent (AuEq)	
	Tonnes (000's)	Oz (000's)	gpt	Tonnes (000's)	%	Oz (000's)	gpt	Oz (000's)	gpt
Measured (M)	33,000	608	0.57	62.4	0.189	1,703	1.60	975	0.92
Indicated (I)	46,500	544	0.36	74.0	0.159	1,901	1.27	1,001	0.67
M + I	**79,500**	**1,152**	**0.45**	**136.4**	**0.172**	**3,604**	**1.41**	**1,976**	**0.77**
Inferred	31,600	334	0.33	50.9	0.161	1,073	1.06	653	0.64

1. Mineral resources are estimated using Ordinary Kriging, constrained by geological domains based on lithology and mineralization controls. The underlying datasets supporting the resource estimate have been reviewed, validated, and verified by the Qualified Person (QP).

2. Mineral resources are reported in short tons within an optimized pit shell, using a breakeven gold equivalent (AuEq) cut-off grade of 0.39 g/t for Oxide and Mixed material and 0.34 g/t for Sulfide material. The overall average AuEq cut-off grade for all reported resources is 0.35 g/t. No dilution or mining recovery factors have been applied.

3. The AuEq cut-off grade is calculated using realized metal prices of $1,860.10/oz Au, $3.92/lb Cu, and $22.52/oz Ag, with average metallurgical recoveries by oxidation type as follows:

   Gold (Au): 55% (Oxide/Mixed), 64% (Sulfide)
   Copper (Cu): 30% (Oxide), 78% (Mixed), 87% (Sulfide)
   Silver (Ag): 61% (Oxide/Mixed), 70% (Sulfide)

4. The optimized pit shell was generated using the Lerchs-Grossman method, incorporating all classified resources, realized metal prices, $2.50/ton mining costs, $9.20/ton processing costs, a 50° slope angle, and varying metallurgical recoveries as detailed in Table 11.12.

5. No dilution or mining recovery factors have been applied to the resource estimate.

6. There are no known legal, environmental, or permitting issues that impact the reported resources.

7. Resources are reported within the company's permitted land tenure/exploration license boundaries.

8. Mineral resources are classified in accordance with S-K 1300 definitions and are reported inclusive of mineral reserves.

9. Rounding may result in minor discrepancies in tonnage, grade, and contained metal totals.

10. There is no guarantee that mineral resources will be converted to mineral reserves.

11. The mineral resource estimates were prepared, reviewed, and validated by Mark Shutty, CPG, the independent Qualified Person (QP) for these estimates, in accordance with S-K 1300 Definition Standards adopted December 26, 2018.

12. The effective date of the mineral resource estimate is January 6, 2025.

All Inferred Resources, along with Measured and Indicated Resources exclusive of Reserves, primarily located in area between the Reserves pit and the Resource pit are presented in Table 11.15 and Table 11.16.

The relatively small difference between Measured and Indicated Mineral Resources (exclusive of reserves) and Mineral Reserves is primarily due to the presence of modeled mineralization located outside the current Resource pit shell, particularly at depth and to the southeast (Figure 11.20). These areas are defined by limited and wide-spaced drilling, which, combined with prevailing metal prices and economic considerations, precludes their inclusion within the current Resource pit shell.

At present, drilling density is insufficient to support the classification of additional mineral resources that could materially impact the geometry of the constraining Resource pit shell. While higher metal prices could potentially justify a larger Resource pit shell, current economic, metallurgical, and drilling constraints limit the conversion of additional resources. The primary constraint is the lack of sufficient drill data rather than a lack of in-situ mineralization.

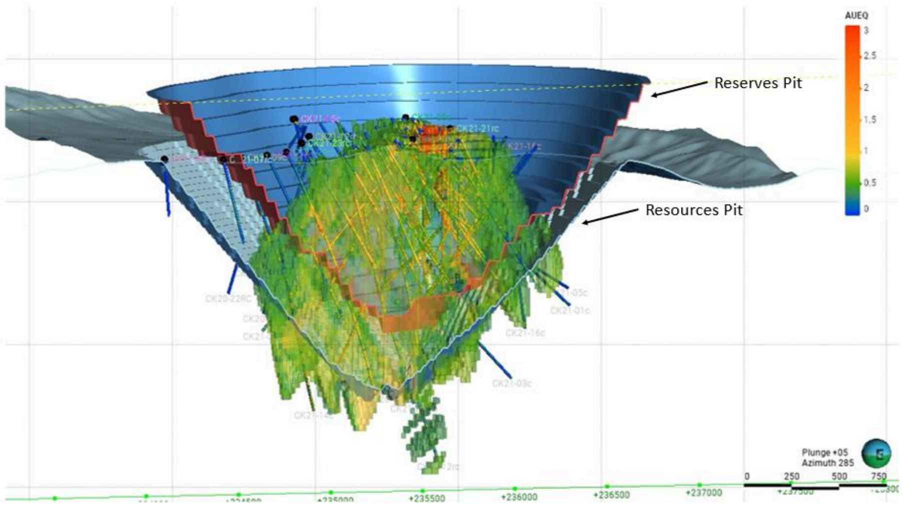


Figure 11.20: Cross section showing above cutoff AuEq Resource with nested Resource and Reserves pit shells. Note excluded mineralization located outside of the resource pit at depth and to the southeast.

Table 11.15: Mineral Resource Statement – Exclusive of Reserves									
	Mass	Gold (Au)		Copper (Cu)		Sliver (Ag)		Au Equivalent (AuEq)	
	Tons (000's)	Oz (000's)	oz/st	Lbs (millions)	%	Oz (000's)	oz/st	Oz (000's)	oz/st
Measured (M)	1,900	13	0.011	5	0.135	112	0.065	66	0.041
Indicated (I)	12,400	118	0.009	36	0.143	484	0.037	238	0.018
M + I	14,400	131	0.009	41	0.147	596	0.041	304	0.021
Inferred	34,900	334	0.010	112	0.161	1,073	0.031	653	0.019

1. Mineral resources are estimated using Ordinary Kriging, constrained by geological domains based on lithology and mineralization controls. The underlying datasets supporting the resource estimate have been reviewed, validated, and verified by the Qualified Person (QP).

2. Mineral resources are reported in short tons within an optimized pit shell, using a breakeven gold equivalent (AuEq) cut-off grade of 0.011 oz/st for Oxide and Mixed material and 0.010 oz/st for Sulfide material. The overall average AuEq cut-off grade for all reported resources is 0.010 oz/st. No dilution or mining recovery factors have been applied.

3. The AuEq cut-off grade is calculated using realized metal prices of $1,860.10/oz Au, $3.92/lb Cu, and $22.52/oz Ag, with average metallurgical recoveries by oxidation type as follows:

   Gold (Au): 55% (Oxide/Mixed), 64% (Sulfide)
   Copper (Cu): 30% (Oxide), 78% (Mixed), 87% (Sulfide)
   Silver (Ag): 61% (Oxide/Mixed), 70% (Sulfide)

4. The optimized pit shell was generated using the Lerchs-Grossman method, incorporating all classified resources, realized metal prices, $2.50/ton mining costs, $9.20/ton processing costs, a 50° slope angle, and varying metallurgical recoveries as detailed in Table 11.12.

5. No dilution or mining recovery factors have been applied to the resource estimate.

6. There are no known legal, environmental, or permitting issues that impact the reported resources.

7. Resources are reported within the company's permitted land tenure/exploration license boundaries.

8. Mineral resources are classified in accordance with S-K 1300 definitions and are reported exclusive of mineral reserves.

9. Rounding may result in minor discrepancies in tonnage, grade, and contained metal totals.

10. There is no guarantee that mineral resources will be converted to mineral reserves.

11. The mineral resource estimates were prepared, reviewed, and validated by Mark Shutty, CPG, the independent Qualified Person (QP) for these estimates, in accordance with S-K 1300 Definition Standards adopted December 26, 2018.

12. The effective date of the mineral resource estimate is January 6, 2025.

Table 11.16: Mineral Resource Statement (Metric) – Exclusive of Reserves									
	Mass	Gold (Au)		Copper (Cu)		Sliver (Ag)		Au Equivalent (AuEq)	
	Tonnes (000's)	Oz (000's)	gpt	Tonnes (000's)	%	Oz (000's)	gpt	Oz (000's)	gpt
Measured (M)	1700	13	0.202	2	0.171	112	1.968	66	1.288
Indicated (I)	11300	118	0.298	16	0.156	484	1.332	238	0.670
M + I	13100	131	0.298	18	0.131	596	1.410	304	0.719
Inferred	31,600	334	0.33	50.9	0.161	1,073	1.06	653	0.64

1. Mineral resources are estimated using Ordinary Kriging, constrained by geological domains based on lithology and mineralization controls. The underlying datasets supporting the resource estimate have been reviewed, validated, and verified by the Qualified Person (QP).

2. Mineral resources are reported in short tons within an optimized pit shell, using a breakeven gold equivalent (AuEq) cut-off grade of 0.39 g/t for Oxide and Mixed material and 0.34 g/t for Sulfide material. The overall average AuEq cut-off grade for all reported resources is 0.35 g/t. No dilution or mining recovery factors have been applied.

3. The AuEq cut-off grade is calculated using realized metal prices of $1,860.10/oz Au, $3.92/lb Cu, and $22.52/oz Ag, with average metallurgical recoveries by oxidation type as follows:

   Gold (Au): 55% (Oxide/Mixed), 64% (Sulfide)
   Copper (Cu): 30% (Oxide), 78% (Mixed), 87% (Sulfide)
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4. The optimized pit shell was generated using the Lerchs-Grossman method, incorporating all classified resources, realized metal prices, $2.50/ton mining costs, $9.20/ton processing costs, a 50° slope angle, and varying metallurgical recoveries as detailed in Table 11.12.

5. No dilution or mining recovery factors have been applied to the resource estimate.

6. There are no known legal, environmental, or permitting issues that impact the reported resources.

7. Resources are reported within the company's permitted land tenure/exploration license boundaries.

8. Mineral resources are classified in accordance with S-K 1300 definitions and are reported exclusive of mineral reserves.

9. Rounding may result in minor discrepancies in tonnage, grade, and contained metal totals.

10. There is no guarantee that mineral resources will be converted to mineral reserves.

11. The mineral resource estimates were prepared, reviewed, and validated by Mark Shutty, CPG, the independent Qualified Person (QP) for these estimates, in accordance with S-K 1300 Definition Standards adopted December 26, 2018.

12. The effective date of the mineral resource estimate is January 6, 2025.

11.15 RELEVANT FACTORS THAT MAY AFFECT THE MINERAL RESOURCE ESTIMATE

The CK Gold Project is subject to factors that may affect this Mineral Resource estimate:

- **Metal Prices**: Fluctuations in metal prices can influence the cut-off grade, affecting the estimated quantity of resources.
- **Operating Costs**: Variations in assumed operating costs may alter the cut-off grade and the quantity of estimated resources.

- **Tonnage and Grade Estimates**: Additional drilling, new assay data, and updated tonnage factor information may change tonnage and grade estimates.
- **Recovery Assumptions**: Modifications to recovery rates can affect the quantity of estimated resources.
- **Regulatory and Operational Assumptions**: The ability to maintain mining claims and surface rights, secure site access, obtain environmental and other regulatory permits, and achieve a social operating license may also influence the resource estimate.

11.16 RESPONSIBLE PERSON OPINION

The mineral resource estimate is well-constrained by three-dimensional wireframes representing geologically realistic volumes of mineralization within the granodiorite intrusive host rocks. Exploratory data analysis conducted on assays and composites shows that the wireframes define appropriate domains for mineral resource estimation. Grade estimation was performed using an interpolation strategy designed to minimize bias in the resulting grade models.

Mineral resources are constrained and reported using economic and technical criteria to ensure a reasonable prospect for economic extraction. The resources are presented at a cut-off grade and further constrained within a pit optimization shell. The application of a pit shell constraint prevents the projection of discontinuous resources to uneconomic depths, even at elevated concentrate prices. Together, these constraints form the basis for establishing reasonable prospects for economic extraction.

U.S. Gold's 2021 drilling program datasets were integrated with the existing resource drillhole database, modeled mineralization, geological interpretations, oxidation domains, and estimated grades of gold (Au), copper (Cu), and silver (Ag), as well as block classifications. This drilling validated previously modeled resources and methodologies, particularly in developing the constraining mineralized domain and within the pit shell envelope.

Drillhole samples from the 2021 program passing through Measured and Indicated resource blocks within the constraining pit intersected the same host lithologies and exhibited mean Au and Cu grades consistent with the global mean grades of the modeled resource (Table 11.17). For samples intersecting Inferred resource blocks within the constraining pit (Figure 11.21), metal grades were generally in line locally but slightly lower globally than previously modeled. In contrast, drillholes passing through Inferred resource blocks on the margins of modeled mineralization showed decreasing grades with distance from the deposit's higher-grade core. Notably, mean Ag grades in the 2021 drilling are slightly elevated relative to the modeled resource.

Table 11.17: Global Mean Grades of Estimated Metals (Model Mean) vs. 2021 Drillhole Grades					
Class (Domain)	**Metal**	**Sample Count**	**Length (ft)**	**Sample Mean**	**Model Mean**
Measured	AUCAP (gpt)	394	1,657	0.68	0.62
	CUCAP (%)	394	1,657	0.23	0.19
	AGCAP (gpt)	394	1,657	1.27	1.70
Indicated	AUCAP (gpt)	732	3,135	0.37	0.35
	CUCAP (%)	732	3,135	0.17	0.16
	AGCAP (gpt)	732	3,135	1.00	1.11
Inferred	AUCAP (gpt)	741	3,290	0.27	0.35
	CUCAP (%)	741	3,290	0.12	0.15
	AGCAP (gpt)	741	3,290	0.81	0.50

1. Table of mean grades of estimated metals (Model Mean) vs. 2021 drillhole grades (Sample Mean) by Class (Domain) within the constraining pit shell.

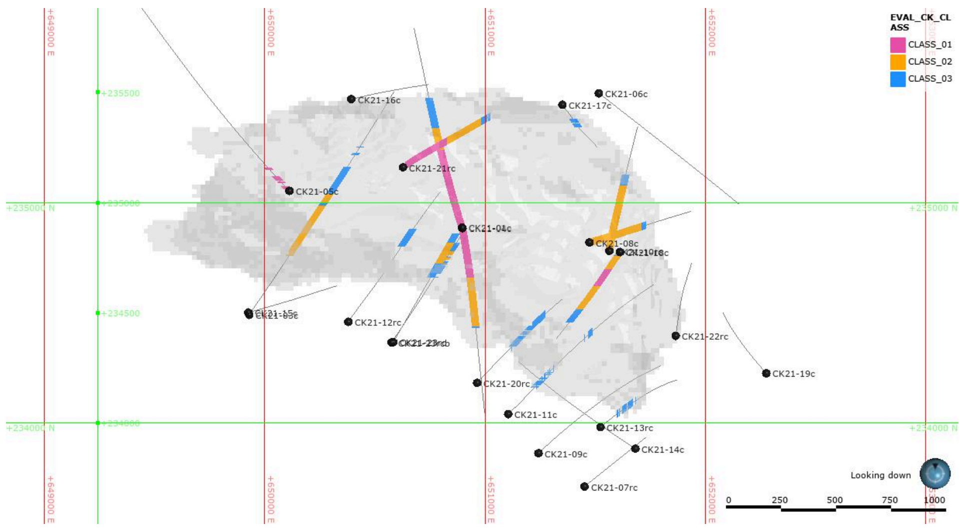

Figure 11.21: Plan Map of 2021 RC and core drillholes coded by material class

Mark Shutty, CPG, is the QP responsible for resource estimation and resource tabulation. The QP believes that this mineral resource estimate for the CK Gold Project is an accurate estimation of the in-situ resources based on the available data and that the available data and the mineral resource model are sufficient for mine design and planning.

12.0 MINERAL RESERVE ESTIMATES

The Mineral Resources described in Section 11 are the primary basis for the estimate of Mineral Reserves described in this report section. The parameters discussed in Section 12.1 are part of the qualifiers that allow the conversion of Mineral Resources to Mineral Reserves. The Mineral Resources described in Section 11 are the primary basis for the Mineral Reserve estimate described in this section. The parameters discussed in Section 12.1 are part of the qualifiers that allow the conversion of Mineral Resources to Mineral Reserves. The Mineral Resource refers to the inventory of mineralization that can reasonably be expected to become economic under stated parameters, while the Mineral Reserves identified report a subset of the Mineral Resource that is economic under more rigorous parameters that conform to industry standards and practice, principally metal prices.

The CK Gold Project Mineral Reserve estimate lies inside an open pit design. The pit sits inside a larger, potentially economic resource shell for the property. The pit design is guided by an economic pit limit analysis based on the economic parameters described in this section. The designed pit is then scheduled in a mine plan spanning the project life, and a discounted cash-flow model to assess the Project's economic viability.

12.1 BASIS, ASSUMPTIONS, PARAMETERS, AND METHODS

12.1.1 Pit Optimization

AKF Mining Services Inc. (AKF) performed economic pit-limit analysis using Vulcan's Pit Optimizer software, which uses the Lerchs–Grossmann (LG) algorithm to determine an economic excavation limit based on input optimization parameters shown in Table 12.1.

Table 12.1: Pit Optimization Parameters

Item	Unit	Value
Gold Price	$/oz	1,755.00
Copper Price	$/lb	3.77
Silver Price	$/oz	23.00
NSR Royalty*	%	2.1
Concentrate Smelting & Transport — Oxide	$/lb Cu recovered	0.29
Concentrate Smelting & Transport — Mixed	$/lb Cu recovered	0.32
Concentrate Smelting & Transport — Sulfide	$/lb Cu recovered	0.37
Cu Refining Charge	$/lb Cu	0.07
Au Refining Charge	$/oz	5.00
Ag Refining Charge	$/oz	0.45
Oxide—Cu Recovery (>0.1% & <0.4%)	%	30
Oxide—Au Recovery (>0.3gpt & <1.3 gpt)	%	60
Oxide—Ag Recovery (>0.5 gpt)	%	61
Mixed—Cu Recovery (>0.1% & <0.4%)	%	78
Mixed—Au Recovery (>0.27 gpt & <1.0 gpt)	%	60
Mixed—Ag Recovery (>0.5 gpt)	%	61
Sulfide—Cu Recovery (>0.15% & <0.4%)	%	87
Sulfide—Au Recovery (>0.3 5gpt & <0.65 gpt)	%	67

Table 12.1: Pit Optimization Parameters		
Item	**Unit**	**Value**
Sulfide—Ag Recovery (>0.5 gpt)	%	70
Smelter Payable — %Cu	%	97
Smelter Payable —Au oz/st	%	98
Smelter Payable — Ag oz/st	%	95
Concentrate Grade %Cu — Oxide	%	23
Concentrate Grade %Cu — Mixed	%	21
Concentrate Grade %Cu — Sulphide	%	18
Mining Cost	$/st	2.50
Process Cost	$/st processed processed	7.00
Tailings Cost	$/st processed	1.65
Site-Wide General & Administrative Cost	$/st processed	1.50
Pit Slope	Degrees	48

* Note: See definition of Royalty for Wyoming State Land Lease, Section 3.4.

The pit optimization used for guiding the final pit design considers only Measured and Indicated Resources; metal classified as Inferred Resource is ignored. The metal pricing used in the optimization parameters is weighted long-term forecast comprising a three-year trailing average. The QP believes that this is a reasonable assumption; additional information is provided in Section 16.

The economic excavation boundary (pit shell) indicated by the pit optimization is used to guide the final pit design, which becomes a hard boundary in the conversion of Mineral Resources to Mineral Reserves. Mineral Resources in the Measured and Indicated categories inside the final pit design can convert, or classify, as Mineral Reserves, subject to resource classification and cut-off grade. Section 13 contains additional details about the mine design.

12.1.2 Value Per Ton Cut-off Grade Calculation

The value per ton (VPT) "milling cut-off value" calculation for all areas was completed as follows:

- VPT = (Block Revenue – Milling Cost – G&A Cost)/Resource Tons
- Where:
  - Block Revenue = Resource tons x Grades x Recovery x Net Price for each metal
  - Milling Cost = Resource tons x Milling Cost per ton
  - General & Administrative (G&A) Cost = Resource tons x G&A Cost per ton

This calculation is sometimes called the "milling cut-off value" because the mining cost is not considered. The mining cut-off uses a similar calculation but includes the mining cost. The mining cut-off is used to determine the boundary of an economic pit shell, and the milling cut-off has been used in this case to determine the reserves contained within that same shell. For the reserves, the block was considered mill feed if the VPT milling cut-off was equal to or greater than a value of $0.01/st. If the value was less than this, the block was considered waste.

12.1.3 Dilution

Due to the disseminated nature of mineralization, dilution is not expected to be an issue during mining, and a dilution factor is not used in the Mineral Reserves determination. A grade-control program would likely be sufficient to prevent excessive dilution or ore losses.

12.2 MINERAL RESERVES

CK Gold Mineral Reserves are given in Table 12.2. Antonio Loschiavo, P. Eng., is the QP responsible for the Mineral Reserves statement. Mineral Reserves are reported inside a detailed pit design using suitable parameters for the site, which was guided by pit optimization.

Table 12.2: Mineral Reserve Statement

	Mass	Gold (Au)		Copper (Cu)		Sliver (Ag)		Au Equivalent (AuEq)	
	Tons (000s)	Oz (000s)	oz/st	M lb	%	Oz (000s)	oz/st	Oz (000s)	oz/st
Proven (P1)	34,500	595	0.017	133	0.192	1,591	0.046	909	0.026
Probable (P2)	38,800	426	0.011	127	0.164	1,417	0.037	763	0.020
P1 + P2	**73,200**	**1,022**	**0.014**	**260**	**0.177**	**3,008**	**0.041**	**1,672**	**0.023**

1. Reserves tabulated above a "milling cut-off value" per ton (see text).
2. Note only 3 significant figures shown, may not sum due to rounding.

12.3 CLASSIFICATION AND CRITERIA

Section 11.11 discusses resource classification. Measured and Indicated Resources inside the designed pit are classified as Proven and Probable Mineral Reserves, respectively. Mineral Reserves use the same cut-off grade definitions as Mineral Resources. This reserve classification does not affect the Mineral Resource statement.

12.4 RELEVANT FACTORS

The CK Gold Project is subject to factors that may impact the Mineral Reserve statement:

- Economic factors such as changes in metals prices, operating costs, or capital expenditures.
- Changes to the estimated Mineral Resources.
- Metallurgical factors affecting recovery.
- Maintenance of social and environmental license to operate.

12.5 2025 PFS VS 2021 PFS RESERVES

AKF completed the 2021 PFS design pits based on the 2021 PFS metal prices and operating costs. During the 2025 PFS update, metal prices and operating costs increased, which triggered a review of the Mineral Reserves by rerunning the LG optimizations based on the latest metal prices and operating costs. As a result, the comparison between the 2021 PFS and 2024 PFS Mineral Reserves shows a 3% increase in ore tons and a 5% increase in waste tons.

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13.0 MINING METHODS

13.1 INTRODUCTION

Open pit, surface mining is the selected mining method for the CK Gold Project. This mining method is selected based on the size, shape, location, and value of the mineralization on the property. The Project's disseminated type mineralization has a large extent and is located near to or outcropping at surface. Additionally, open pit optimizations attempting to maximize the recovery of the in-situ resource show economic excavation results using current project parameters and base case metal prices.

Surface mining is a cyclical process where the four main tasks including drilling, blasting, loading, and haulage are occurring concurrently at different areas of the property. In areas to be excavated vertical blast holes are drilled in a regular pattern and charged with blasting agents. The material will be blasted, loaded into 100 st class rigid frame haul trucks, and transported based on material type to one of four different locations, run-of-mine (ROM) Crusher Stockpile, Co-Disposal Tailings Facility, Ore Stockpile or Waste Rock Facility. Wherever possible Crusher Stockpile ore will be directly dumped into the primary crusher at the process plant.

Owner operator mining has been selected as the preferred method for the purposes of this PFS. This decision, in large part, is due to the location of the Project, local mining and the availability of potential labor within 30 miles of the site (Laramie and Cheyenne, Wyoming). As part of the studies, mining costs were estimated from first principles with equipment depreciation. Contractor mining is only considered for the blasting crew but is not eliminated as an option for mine development, pending further review.

13.2 GEOTECHNICAL PARAMETERS

Piteau Associates (Piteau) conducted a geotechnical investigation for the project. Piteau issued a technical memorandum dated September 6th, 2022, titled "Recommended Feasibility-Level Geotechnical Slope Designs for the Copper King Open Pit." This section contains a summary of the report. Following the September 6th report, an updated May 1st, 2024 report was completed due to the change in bench height from 20ft to 30ft.

The following list summarizes the scope of work that Piteau performed as part of the geotechnical investigation:

- Full geotechnical logging of 5 core holes, detailed structure logging.
- Rock mass strength assessments, laboratory testing and analysis.
- Structure assessment, Kinematic analysis.
- Recommended end of life slope design.
- An assessment of the effects of ground water and pore pressure on slope stability.

Table 13.1 and Figure 13.1 outline the latest slope design recommendations and pit design sectors based on the 30ft bench design.

Table 13.1: Recommended Slope Designs for Presplit Blasted Benches				
Design Sector	Max Interramp Slope Angle	Max Interramp Slope Height	Catch Bench Width	Face Angle
I	52	410	38.8	75
II	54	380	36.7	75
III	54	370	34.6	75
IV	54	480	41.1	75
V	53	460	36.7	75
VI	54	480	41.1	75
VII	53	470	36.7	75
VIII	52	510	41.1	75
IX	53	500	38.8	75
X	54	490	36.7	75
XI	53	460	38.8	75

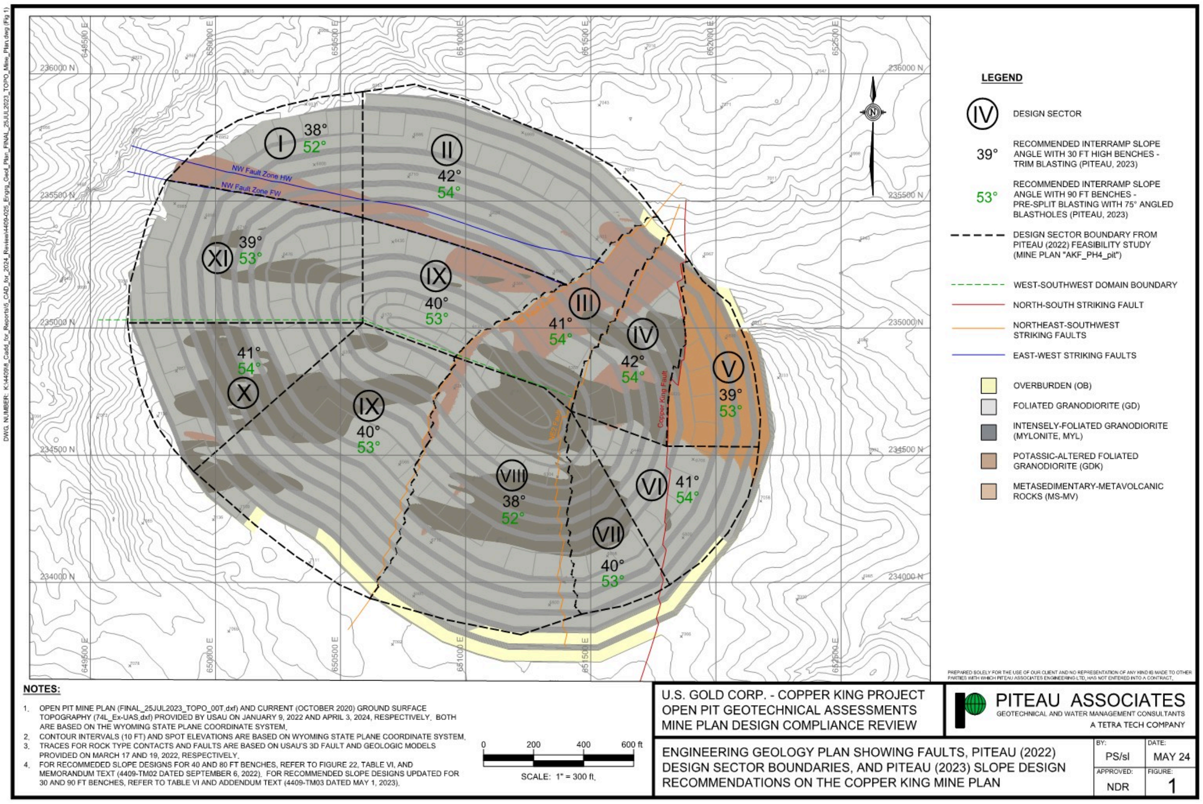


Figure 13.1: Pit Sectors and Recommended Slopes

13.2.1 Geotechnical General Recommendations

Below is a summary of the General recommendations:

13.2.1.1 Blending and Finalizing Designs

Where a range of interramp angle (IRA) is indicated between adjacent design sectors, blending should occur within the design sector with the steeper (greater) design IRA. Similarly, blending from weaker to stronger materials should occur in the stronger (better quality) rock mass materials.

13.2.1.2 Benching Trials

In the early stages of mining below the overburden and weathered bedrock horizon, benching trials for 80 ft high benches should be considered in areas where bench performance is expected to have the least impact on the stability of haul roads or other critical slope areas to confirm that structural continuity of adversely oriented joint sets is limited and therefore has limited impacts on the bench designs. Bench designs should be updated based on ongoing evaluation of bench performance.

13.2.1.3 Transitioning from Single to Double Benches

At the transition from single- to triple-benches, the triple-bench catch bench width should be implemented at the crest level of the first triple-bench to avoid steepening the design IRA.

13.2.1.4 Controlled Blasting

The following recommendations are made with respect to the potential for benches to be excavated up to 90 ft high:

1. Optimizing slope designs to maximize IRA while maintaining safe working conditions requires controlled blasting on final walls to minimize the damage to intact rock bridges and preserve cohesion on discontinuity surfaces.

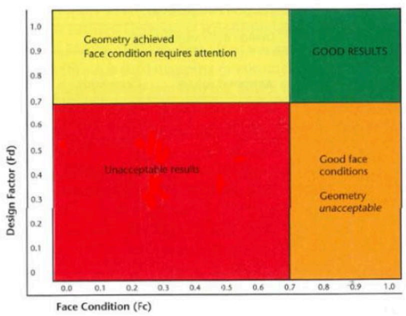
    a. Pre-split blasting (with trims) should be considered to improve (increase) effective bench face angles (BFAs) and catch bench widths.
    b. Pre-split blasting could provide increased success for the proposed double benching below the overburden and weathered bedrock zone near the slope crest.
    c. Blast monitoring and pre/post blast inspection should be conducted to continually assess potential blast damage and improve blast performance.

2. Once a revised CK Gold Project mine plan is developed with the enclosed feasibility-level slope design recommendations, ongoing evaluation of potential hazards and risks should be carried out through the implementation of standard operating procedures (SOPs), a ground control management plan (GCMP), and regular geotechnical inspection.
3. An inspection and sign-off system should be used to confirm that the bench crests throughout the pit are adequately scaled, significant breakback is not occurring, and bench face conditions are acceptable. An evaluation of bench design achievement should be carried out to verify that face and crest conditions are adequate for safe development of multiple-lift (double) benches. A qualitative bench design achievement system presented by Read and Stacey (2009) can be modified for specific site conditions as shown in Figure 13.2, and includes evaluation of:

a. Design face achievement (Df) for the bench configuration; and
b. Face condition (Fc).
c. The components of the system are summarized in the following ratings tables and chart shown below. For consideration of double benching, bench design achievement results should fall under the "Good Results" category.

Design achievement (Df)

Component (weighting)	Assigned values		Comments
Bench face angle (50%)	≥ Design	50	Achieved overall bench face angle relative to design
	Design –3°	25	
	Design –5°	10	
	Design –10°	0	
Bench width (40%)	≥ Design	40	Achieved average bench width relative to design
	Design –1 m	35	
	Design –2 m	25	
	Design –3 m	15	
	Design –5 m	0	
Toe position (10%)	On design	10	Is design toe is being achieved?
	Design –1 m	8	
	Design – m	5	
	Design –3 m	0	



Face condition (Fc)

Component (weighting)	Assigned values		Comments
Half-barrels visible (20%)	≥80%	20	If half-barrels only visible in lower part of bench reduce by 5–10 points
	70–80%	15	
	60–70%	12	
	50–60%	8	
	30–50%	5	
	10–30%	2	
	<10%	0	
Intact rock breakage (15%)	<1/m3	15	Subjective evaluation, interpolate between 0–15
	>5/ m3	0	
Open joints (10%)	All closed – 10	10	Subjective evaluation, interpolate 0–10
	Many open – 0	0	
Loose material on face (20%)	No blocks	20	Assess in terms of rockfall hazard
	Few small blocks	15	
	Large blocks	10	
	Many blocks	0	
Face profile (20%)	Straight	20	Shape of face and basis for variations
	Hard toe	10	
	Overhang crest	5	
	Irregular face	0	
Crest condition(15%)	Achieved	15	For loose rock on crest deduct 0–5 points more
	<1 m loss	12	
	1–2 m loss	10	
	2–3 m loss	5	
	>3 m loss	0	

Figure 13.2: Design Face (Df) versus Face Condition (Fc) Chart

4. To minimize rockfall potential in bedrock, careful bench scaling should be carried out with the shovel bucket during bench excavation. Depending on bench performance, the following additional items may be required:

a. Daylight-only mining with a spotter; and regular geotechnical inspection.
b. Construction of rockfall impact berms or other rockfall control measures (e.g., wire mesh, rockfall attenuation fences, etc.) that are appropriately sized to contain rockfall hazards using rockfall modeling.
c. Local step-outs to gain adequate bench catchment width.
d. Scaling of the bench crest and face using chain pulled behind a dozer (provided adequate bench width is available).
e. Scaling with a long-reach backhoe to remove potential rockfall hazards.
f. Crest trenching with a backhoe in advance of excavation in areas of weaker or highly fractured rock (e.g., weathered bedrock, exposed fault zones or dykes).
g. Implementing angled pre-split blasting with small diameter blastholes; and/or h. Manual scaling using a scaling contractor with ropes.

13.2.1.5 Changes to the Slope Design

As a general comment for future advancement of the CK Gold Project, it is recommended that any new pit designs or significant revisions to the mine plan (for example to the bench, interramp or overall slope angles or heights) be forwarded to Piteau for review, conformance check, and comment. Additional geotechnical evaluations and analyses may be necessary to check stability.

13.2.1.6 Bench Scaling and Cleaning Catch Benches

Bench scaling should be carried out with the shovel bucket during bench excavation. Depending on bench performance, additional scaling may be required with scaling chain. Careful pull-back procedures should be carried out to minimize filling of subsequent benches with spilled material.

13.2.1.7 Slope Monitoring

All slopes should be visually inspected at regular intervals for signs of distress and overall slope movement. Also, a slope displacement monitoring system consisting of survey prisms should be established during early stages of mining and maintained throughout the mine life and operation. Current practice is to measure prisms with automated systems such as robotic total stations (RTS) that include data acquisition and management tools for processing, interpretation, and reporting so that results can be evaluated regularly to assess slope behavior. This can also be supplemented with radar monitoring equipment (if needed) that can provide near real-time monitoring of slope deformations should the need arise. Both prism and radar monitoring provide advanced warning of possible large-scale instability and allow time for appropriate remedial measures to be implemented or mining plans to be modified, to accommodate the instability. Manual or automated wireline extensometers could be used to augment prism or radar monitoring in areas of observed surface deformation and cracking.

If slope movements are measured, monitoring (velocity) thresholds and trigger action response plans (TARPs) should be developed based on the observed slope performance and adjusted as required, to account for the effects of error and noise and to verify and maintain their effectiveness.

Other monitoring slope monitoring techniques such as inclinometers and time domain reflectometers (TDR) (for monitoring subsurface ground movements), or satellite-based surface surveying with global positioning systems (GPS) or InSAR (Interferometric Synthetic Aperture Radar) may need to be incorporated into the slope monitoring system if the need arises.

13.2.1.8 Visual Inspection Monitoring

Regular inspection of the crest and exposed benches on the mine plan should be carried out to identify any signs of tension cracking, increased raveling/rockfall, or other signs of instability. The locations of observed tension cracks should be surveyed and added to geotechnical plans to allow assessment of slope deformation with respect to slope monitoring data and any potential mechanism(s) of instability. Any unusual signs of slope raveling or distress should be communicated to the Mine Geotechnical Team and assessed accordingly.

13.2.1.9 Ongoing Data Acquisition, Verification and Updating Design Criteria

Systematic documentation of bench performance (achieved BFA) and structural mapping (by manual or photogrammetric methods) is recommended to be carried out while mining the Copper King pit. If ongoing bench or slope performance is unfavorable and/or structural mapping indicates adverse conditions as new geology is exposed, local revisions to the mine plan may be required.

Documentation of the as-built bench performance of mined slopes is recommended using reliable methods such as photogrammetry models, high-resolution laser scan digital terrain models (DTM), or manual bench documentation mapping. This information can be used to calibrate breakback angles calculated from the kinematic CFA assessments and support potential optimization of the IRAs during mining.

In addition, rockfall field testing and modelling is recommended to calibrate rockfall model input parameters and develop a "site-specific" design catch berm width for rockfall protection instead of the Modified Richie Criteria (Equation 3) which was adopted for this study. Such rockfall calibration could also support potential optimization of the IRAs.

It is recommended that future drilling in bedrock should include geotechnical logging of all parameters comprising RMR (according to Bieniawski, 1976) and consistent PLI testing as described in Section 2.5 in the final (Piteau) report. This geomechanical information should be incorporated into the current geomechanical and rock strength databases developed for the feasibility study and would support future geotechnical evaluations of the CK Gold Project.

13.2.1.10 Slope Depressurization Measures

Deep-seated stability analysis of the slopes indicated that the east and southeast walls (Design Sectors V and VI near Section E1 and Design Sectors VI and VII near Section SE1) require slope depressurization to meet the design acceptance criteria of a minimum FOS of 1.20 for overall, interramp, and compound slopes. Depressurization targets at these two sections are defined in terms of Hu and are based on the EOM groundwater surface provided by Neirbo. To achieve acceptable stability, it is required that pore pressures in the east wall slopes (Section E1, west of the Copper King fault) be reduced to levels equivalent to a 1.0 Hu (hydrostatic conditions) (from a 1.4 Hu defined by Neirbo). In the southeast slope (Section SE1, north of N 234,025) it is required that pore pressures be reduced to levels equivalent to a 1.2 Hu (from a 1.4 Hu). Both Hu targets are for the lower slopes and assume that a 0.8 Hu will be present in the MS-MV unit in the upper slope east of the Copper King fault (at Section E1) and that a 1.0 Hu will be present in the granodiorite rock mass in the upper slope south of N 234,025 (at Section SE1).

Based on these Hu targets, it is recommended that additional 3D hydrogeological modeling be performed that includes simulation of active depressurization measures (such as pumping wells and/or horizontal or inclined drains) in the east and southeast slopes to determine what measures are needed to achieve the depressurization targets. This hydrogeologic modeling should also incorporate a mine plan that uses the feasibility slope design recommendations and that has been checked by Piteau for conformance to the design. After hydrogeologic modeling of active depressurization is complete, it is recommended that the calculated pore pressures be provided to Piteau (for example, as a "grid" defined by x, y, z coordinates and pore pressure, u) to perform new 2D anisotropic stability analyses of the east and southeast slopes to check if the depressurization targets have been achieved and confirm that the FOS of the overall, interramp, and compound slopes meet the 1.20 design acceptance criteria.

13.2.1.11 Hydrogeologic Monitoring

Stability of the east and southeast slopes is dependent on achieving specific depressurization targets and these areas will likely require some form of active depressurization (i.e., pumping wells and/or drains) which can be defined through additional hydrogeological modeling as described in the Piteau report. As integral part of active depressurization, it is also recommended that hydrogeological monitoring (such as multi-level vibrating-wire piezometers or VWPs) be installed to monitor pore pressures and verify that the required targets in the critical areas of the slope are being achieved in advance of mining and during the life of mine.

13.2.1.12 Surface Water Control

To assist achieving and maintaining depressurization targets as well as avoid development of erosional gullies and slope instabilities within the mine plan, the following surface water controls are recommended:

1. Use perimeter ditches behind the pit crest to capture and divert surface water away from the pit.
2. Grade haul roads inwards to divert surface water away from the outside edge of the haul roads and create a ditch along the inside lane to capture the water.
3. Collect surface water in appropriately placed sumps (for example, pit bottom or intermediate locations along haul roads) and pump to proper discharge points outside the pit.

13.2.1.13 Contingency Planning

The mine plan has only one main haul road providing access to the pit. Single haul road access could pose potential risks to the mining sequence and ore delivery if instabilities develop above or below the haul roads. Ongoing slope monitoring and visual stability inspections should be carried out to prevent the loss of this single main access point into and out of the mine plan.

13.3 HYDROGEOLOGICAL PARAMETERS

A hydrogeology investigation for the Project was conducted by Neirbo Hydrogeology (Neirbo). Neirbo issued a technical report in December 2023 titled “Hydrogeological Characterization and Groundwater Flow Model.” This section contains a summary of the report.

The CK Gold Project property is located in the Silver Crown mining district of southeast Wyoming, approximately 20 miles west of the city of Cheyenne. The property comprises about 1,120 acres (2 square miles) on the southeastern margin of the Laramie Mountains. The Project is fully-owned by U.S. Gold. The Project facilities include an open pit, tailings management facility, two waste rock facilities, plant site, and an ore stockpile area.

The highest elevation in the open-pit area is about 7,100 ft and the pit will be excavated to 6,120 ft. The mine plan has eight years of mining and passive dewatering as the open pit is advanced. The post-mining phase includes pit backfilling with tailings and waste rock. The first two years after mining ends will be dedicated to site reclamation.

The orebody is hosted in granitic rocks that have limited permeability and limited water-storage capacity. Groundwater wells completed in the granite rocks have typically yielded 0 to 5 gpm. The Project has completed an extensive hydrogeologic site characterization to support development of a regional groundwater flow model (Flow Model). Aquifer testing has included pumping tests and discrete depth-interval packer testing. Hydraulic conductivity and specific storage properties were estimated from these tests. Groundwater levels and pore pressures were obtained from wells and Vibrating Wire Piezometers.

A calibrated Flow Model was developed to represent the hydrogeologic system. The Flow Model simulates pre-mining conditions and hydrologic changes during the mining and post-mining phases. The Flow Model predicts groundwater system changes due to passive pit dewatering, natural recharge changes due to facility construction, and pit backfill during the post-mining phase.

Predictions during the mining and post-mining periods included groundwater-level, pit inflow, streamflow, and evapotranspiration changes. Predicted mine-induced drawdown was greatest near the pit and it decreased rapidly away from the pit. Predicted drawdown was 10 ft or less outside the Operating Permit Boundary at the end of mining. After 150-years, the discernable predicted drawdown extended 180 ft outside the Permit operating boundary in a small area shown on Figure 13.3. The nearest domestic wells were 2,000 ft from the predicted 10-feet drawdown area. At this distance, any mine induced drawdown would likely not be discernable from natural variation and groundwater-level changes induced by the domestic wells themselves.

The Middle Fork of Crow Creek is the nearest stream, and its flow was predicted to decrease 0.02 cubic feet per second ten years after mine-ending. The other stream segments had zero to 0.01 cubic feet per second changes in flow.

Average annual groundwater pit inflow was expected to be less than 15 gpm. This low pit inflow would be manageable using passive, in-pit sumps. Dewatering wells are not expected to be necessary. Cumulative pit inflows during mining were predicted to be 130 acre-feet.

After mining ends, the pit will be backfilled with tailings and waste rock. Groundwater and precipitation will flow into the backfill material and water-levels will slowly rise until they stabilize at 6,717 ft after about 130 years. A pit lake is not expected to form since evaporation losses will keep the groundwater level below the top of backfill. This will result in the pit being a hydraulic sink with no groundwater outflows.

Quarterly background groundwater quality data have been obtained in seven project area wells from 2020 Quarter 4 to 2022 Quarter 1. The background water samples indicate the water quality is generally below regulatory standard concentrations. However, a few constituents in select wells have exceeded the standards for domestic, agriculture, and livestock uses. The domestic water-quality standard for fluoride and pH was consistently exceeded in four of the seven wells. Each well has exceeded standards for iron, manganese, mercury, adjusted gross alpha, or sodium adsorption ratio on at least one occasion. Well MW-7, in the middle of the proposed pit, has consistently exceeded the standard for uranium and gross alpha. The adjusted gross alpha standard was exceeded in three of the six samples in MW-7.

Figure 13.3, Figure 13.4 and Figure 13.5 show the predicted drawdown at the end of mining and 150 years post mining, groundwater monitoring locations and predicted open pit groundwater inflows, respectively.

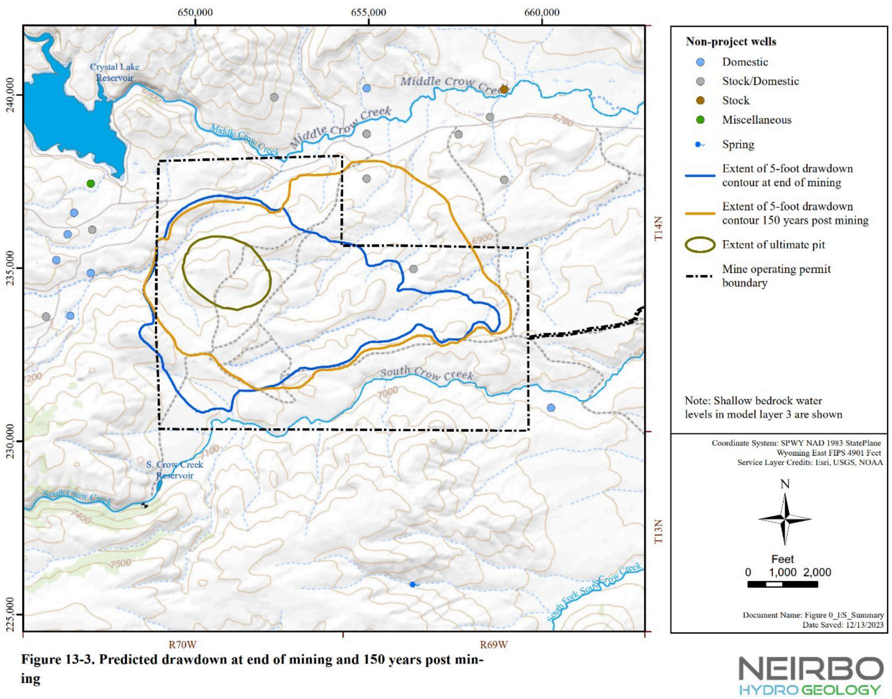


Figure 13.3: Predicted drawdown at the end of mining and post-mining year 150

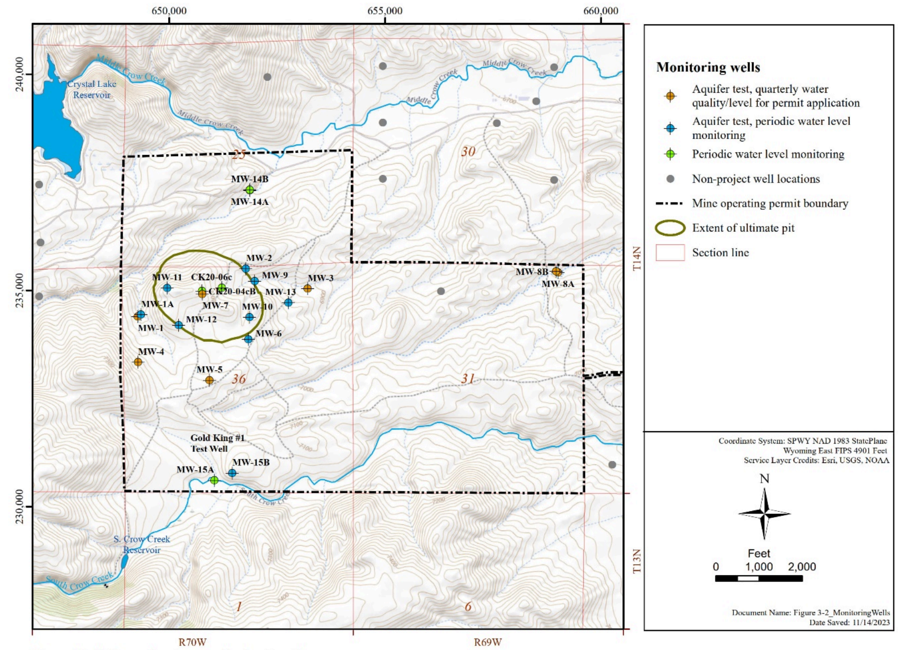


Figure 13-4. Groundwater monitoring locations

Figure 13.4: Groundwater Monitoring Locations

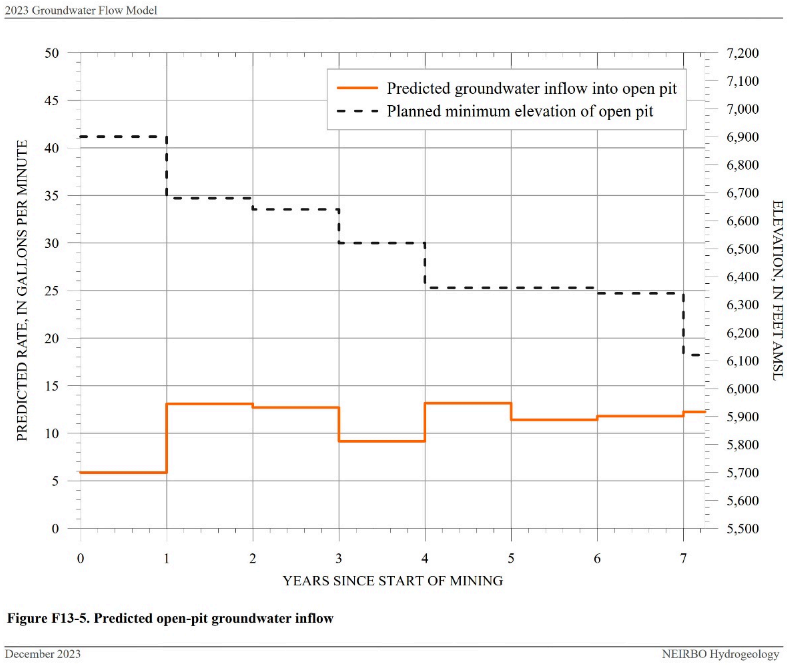


Figure 13.5: Predicted Open Pit Groundwater Inflows

13.4 MINE DESIGN

U.S. Gold contracted AFK Mining to develop a mine design and schedule for the Project. The final mine design is guided by the pit optimization described in Section 12.1.1. The final mine design is comprised of four phases to divide and schedule the excavation. Design parameters are suitable for the mining equipment selected and the geotechnical parameters provided in Section 13.2.

13.4.1 Mine Design Parameters

A summary of the mine design parameters is shown in Table 13.2.

Table 13.2: Mine Design Parameters	
Parameter	**Value**
Road Width (Dual/Single)	90 ft / 70 ft
Road Gradient	10%
Bench Height (Single/Quad)	30 ft / 90 ft
Catch Bench	Every 3 benches
Catch Bench Width	31 ft – 41 ft
Face Angle	75 degrees
Inter-Ramp Angle	52-55 degrees

13.5 MINE SCHEDULE

The primary driver of the mine schedule is the production of sufficient ore, which drives the excavation of waste and other materials to ensure sufficient ore is exposed for mining. The nominal ore production rate was set at 20,000 stpd or 7.3 Mstpy (18,100 t/calendar day, or 6.6 Mt/year) of ore delivered to the crusher. In the first year, ore production is 90% of full capacity to account for commissioning of the concentrator. Mine life is approximately eight years with almost another two years of ore stockpile processing.

Pre-production stripping is scheduled for the year before production begins (Year -2 Q1) which consists of 700,000 st of material. There are no other development requirements to achieve the mine schedule.

Table 13.3: Mine Schedule

	Ore Mined	Waste Mined	Total Material Mined		Ore to Stockpile	Stkple Mined		Mill Total	Au (oz/st)	Cu (%)	Ag (oz/st)	Au (000's Ounces)	Cu (Mlbs)	Ag (000's Ounces)
Total	73,200	68,100	141,300		24,300	24,300		73,200	0.0140	0.177	0.0411	1,022	260	3,008
Year-2	-	700	700		-	-		-					-	
Year-1	-	-	-		-	-		-	-	-	-	-	-	-
Year 1	10,400	9,100	19,500		4,000	200		6,570	0.0264	0.231	0.0685	173	30	450
Year 2	13,000	6,500	19,500		6,200	500		7,300	0.0201	0.205	0.0535	147	30	391
Year 3	9,500	10,000	19,500		2,900	700		7,300	0.0146	0.187	0.0392	107	27	286
Year 4	6,400	13,100	19,500		2,000	2,900		7,300	0.0151	0.185	0.0455	111	27	332
Year 5	8,200	11,300	19,500		1,600	700		7,300	0.0146	0.183	0.0354	107	27	259
Year 6	7,600	11,900	19,500		1,700	1,400		7,300	0.0124	0.185	0.0345	91	27	252
Year 7	12,400	4,400	16,800		5,100	-		7,300	0.0126	0.192	0.0324	92	28	237
Year 8	5,800	1,000	6,800		800	2,300		7,300	0.0120	0.182	0.0325	87	27	237
Year 9	-	-	-		-	7,300		7,300	0.0064	0.113	0.0340	47	17	248
Year10	-	-	-		-	7,300		7,300	0.0074	0.123	0.0383	54	18	280
Year 11	-	-	-		-	945		945	0.0065	0.125	0.0388	6	2	37

13.6 MINING FLEET REQUIREMENTS

The basis for the calculation of mining fleet is the mining schedule and the haulage model. The amount and type of material moved, and the destination of that material determines the total number operational hours that is needed for each category of mining equipment. The total operational hours required then determine the number of units needed and costs associated with operation.

Owner operator mining has been selected as the preferred method for the purposes of this PFS. The owner will also operate the mine planning, ore control, process plant and general site administration (G&A). This decision is due to the location of the Project, local mining and the availability of potential labor within 30 miles of the site (Laramie and Cheyenne, Wyoming). Hybrid owner/contractor operations are still being evaluated to leverage the regional mine contractor expertise and possible reduction in project capital costs.

13.6.1 Equipment Productivity and Usage

For major pieces of mining equipment, the productivity of each unit is estimated based on manufacturer specifications, job site parameters and observed parameters from similar surface mines. Mining equipment has either a variable annual usage basis on the mining schedule or a fixed annual usage. Variable usage equipment has a maximum number of annual hours available for work and a productivity associated with it, shown in Table 13.4. The annual available hours for each piece of equipment are based on the expected availability and utilization. 6,300 hours per year equates to an availability and utilization of approximately 85% each except for drills and dozers at 80%. Table 13.5 shows the annual fleet hours and unit requirements.

Table 13.4: Variable Usage Equipment

Equipment	Annual Hours Available	Productivity	Units
Excavator	6,300	1,950	stph
Loader	6,300	1,500	stph
Haul Truck	6,300	430 - 250	stph
Dozer	5,600	1,000	stph
Drill	5,600	1,000	stph

Table 13.5: Annual Schedule of Variable Usage Equipment

Year	1	2	3	4	5	6	7	8	9	10	11	Total
Excavator Hours (000s)	6.2	6.2	6.2	6.2	6.2	6.2	6.2	3.5				46.9
Excavator Units	1	1	1	1	1	1	1	1				1
Loader Hours (000s)	9.4	10.1	10.3	11.8	10.3	9.8	8	6.4	9.7	9.7	1.3	96
Loader Units	2	2	2	2	2	2	2	2	2	2	1	2
Truck Productivity (stph)	499	445	582	372	409	361	381	250	226	223		375
Truck Hours Req'd 000s)	52.2	60.2	46.1	72	65.5	72.4	56.7	47.2	32.3	28.5	3.2	536
Truck Units	10	10	10	10	10	10	10	8	6	6	2	10
Dozer Hours (000s)	18.5	23.3	22.6	21.2	20.3	19.5	20.1	17.9	17	17	7.5	205
Dozer Units	4	4	4	4	4	4	4	4	4	4	3	4

Haul truck productivity is variable and is based on a haulage model that calculates cycle times based on the location of the material mined and the destination. Cycle times and the mine schedule are used to estimate the truck hours needed to meet the schedule. The annual available hours are based on the distance and the average speed for the haulage segment, with allowances for loading, dumping, and waiting. For excavators and wheel loaders the estimated productivity is based on the calculated loading times to position and fill the selected haul trucks. Dozer productivity is based on manufacturer nomographs. Blasthole drill productivity is based on average penetration rates and blast spacing to break the scheduled rock. Other minor and support equipment does not have a calculated productivity, but a fixed annual usage is assigned based on similar surface mining operations. Table 13.6 shows the fleet size and scheduled hours for the fixed usage equipment.

Table 13.6: Fixed Usage Equipment

Equipment	Hours Scheduled per Unit	Fleet Size
Water Truck	3,000	1
Motor Grader	3,000	1
Service/Fuel Truck	6,000	1
Crane Truck	1,000	1
Excavator	3,000	1

13.7 MINE PERSONNEL REQUIREMENTS

Hourly mine personnel requirements for equipment operators and mechanic labor are based on the annual equipment hourly usage. Salaried based employees are specified at typical staffing levels. All hourly mine employees and supervision of all mine employees are by the mine owner. The owner also provides Site General and Administrative (Site G&A) labor, mine planning and engineering, and environmental compliance. Table 13.7 shows the total project employment over the life of the Project and subsequent tables provide mine employment, Table 13.8; Mine Employment, Table 13.9; Tailings Disposal Employment, and Table 13.10 Site G&A Employment.

Table 13.7: Project Employment

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	Max
Total Project Employment	14	15	173	174	174	174	174	174	174	141	112	105	63	174
Mine Employment	5	2	111	108	108	108	108	108	108	79	58	51	22	112
Tailings Employment	0	0	40	44	44	44	44	44	44	40	36	36	24	44
Site G&A	9	13	22	22	22	22	22	22	22	22	18	18	17	22

Table 13.8: Mine Employment

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	Max
Mine Employment	5	2	112	108	108	108	108	108	108	79	58	51	22	112
Loading and Hauling	-	-	48	44	44	44	44	44	44	34	26	26	8	48
Excavator/Loader Orators	-	-	8	8	8	8	8	8	8	8	4	4	2	8

Table 13.8: Mine Employment

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	Max
Mine Employment	5	2	112	108	108	108	108	108	108	79	58	51	22	112
Truck Operators	-	-	24	20	20	20	20	20	20	16	12	12	4	24
Mechanics/Welders	-	-	16	16	16	16	16	16	16	10	10	10	2	16
Drill and Blast (CONTR)	5	0	26	26	26	26	26	26	26	11	0	0	0	26
Lead Blaster (Contractor)	1	-	1	1	1	1	1	1	1	1	0	0	0	1
Equipment Operators	1	-	16	16	16	16	16	16	16	6	0	0	0	16
Labor	3	-	3	3	3	3	3	3	3	3	0	0	0	3
Mechanics/Welders	-	-	6	6	6	6	6	6	6	1	0	0	0	6
Mine Support	-	-	33	33	33	33	33	33	33	29	27	22	11	33
Equipment Operators	-	-	21	21	21	21	21	21	21	17	17	14	8	21
Labor	-	-	8	8	8	8	8	8	8	8	8	6	1	8
Mechanics/Welders	-	-	4	4	4	4	4	4	4	4	2	2	2	4
Mine G&A	-	2	5	5	5	5	5	5	5	5	5	3	3	5
Mine Manager	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Foreman	-	1	4	4	4	4	4	4	4	4	4	2	2	4

Table 13.9: Tailings Disposal Employment

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	Max
Tailings Disposal Employment	0	0	40	44	44	44	44	44	44	40	36	36	24	44
Tailings Foreman	0	0	2	2	2	2	2	2	2	2	2	2	2	2
Technician	0	0	2	2	2	2	2	2	2	2	2	2	2	2
Loader Operators	0	0	4	4	4	4	4	4	4	4	4	4	2	4
Truck Operators	0	0	16	20	20	20	20	20	20	16	12	12	2	20
Equipment Operators	0	0	16	16	16	16	16	16	16	16	16	16	16	16

Table 13.10: Site G&A Employment

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	Max
Site G&A Total	9	13	22	22	22	22	22	22	22	22	18	18	17	22
Accountant	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Admin/Contract	1	2	2	2	2	2	2	2	2	2	2	2	2	2
Safety/Trainer	0	3	3	3	3	3	3	3	3	3	3	3	3	3
Warehouse Clerk	0	1	1	1	1	1	1	1	1	1	1	1	1	1
Payroll	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Engineer	0	0	3	3	3	3	3	3	3	3	2	2	2	3
Geologist	0	0	2	2	2	2	2	2	2	2	0	0	0	2
Technician	1	0	4	4	4	4	4	4	4	4	3	3	2	4
Security	4	4	4	4	4	4	4	4	4	4	4	4	4	4

13.8 MINE END OF YEAR MAPS

End of year topographic maps showing the excavation progression are shown for Year 1, Figure 13.6; Year 3, Figure 13.7; Year 5, Figure 13.8; and Year 8 final pit limits, Figure 13.9.

13.9 2025 PFS VS 2021 PFS MINING METHODS

AKF completed the 2021 PFS final design pits and phases based on the 2021 PFS geotechnical report. In May 2024 Pre-Feasibility Study Geotechnical report was to update the bench height parameters from 20ft to 30ft.

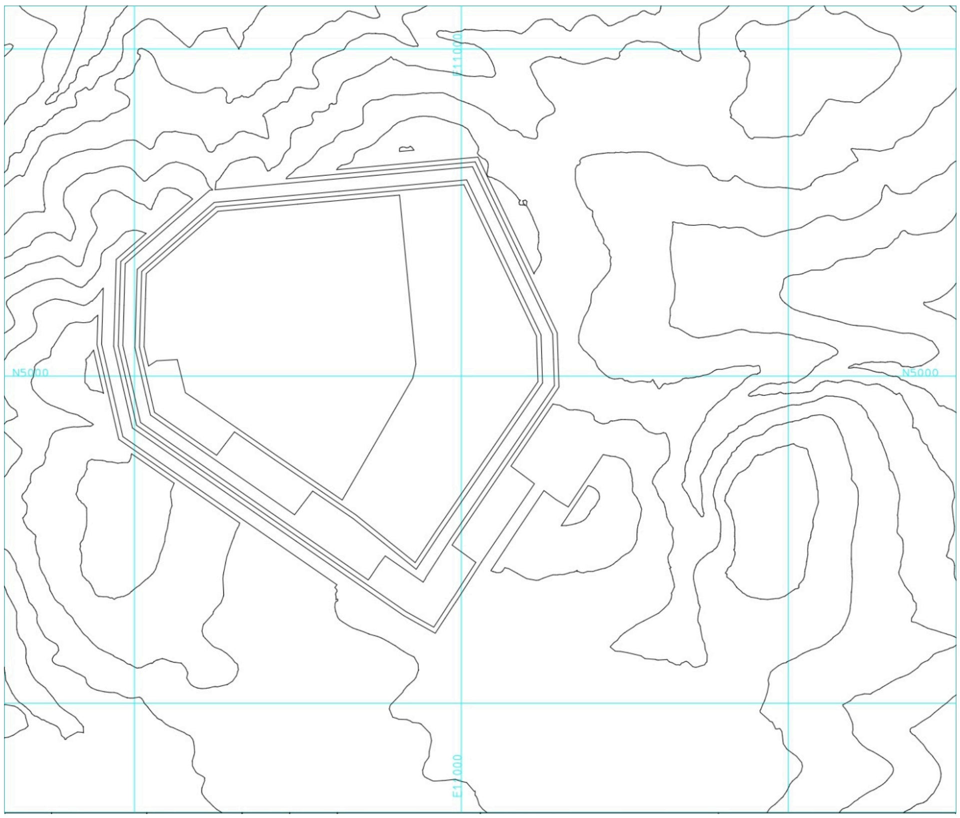


Figure 13.6: Mine Map End of Year 1

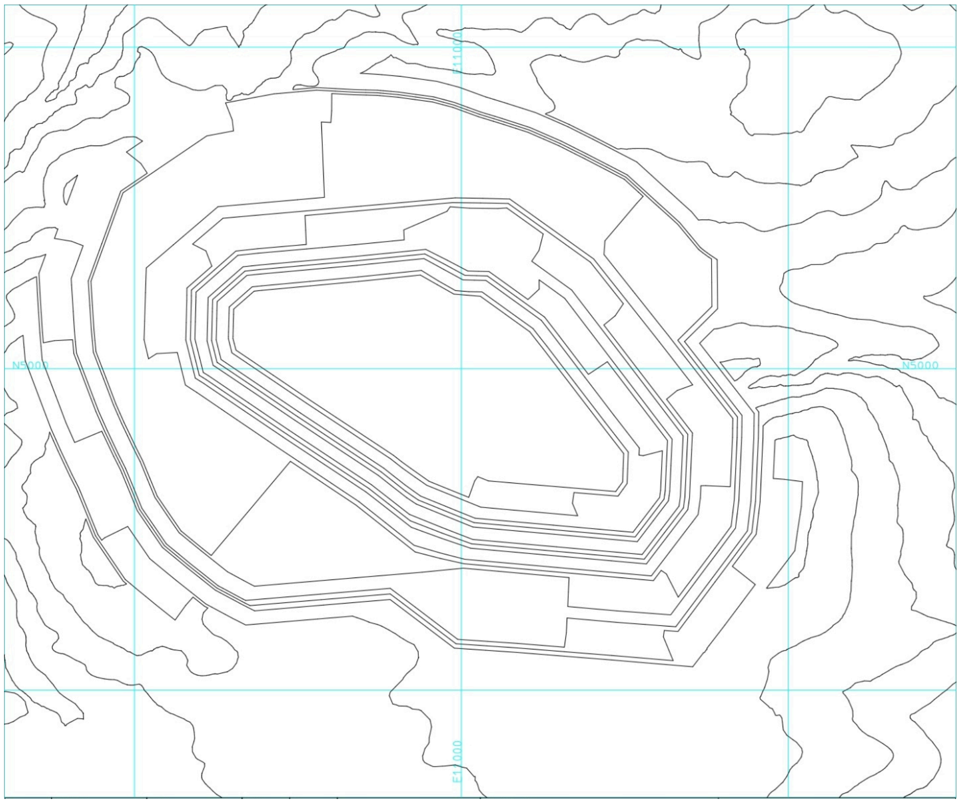


Figure 13.7: Mine Map End of Year 3

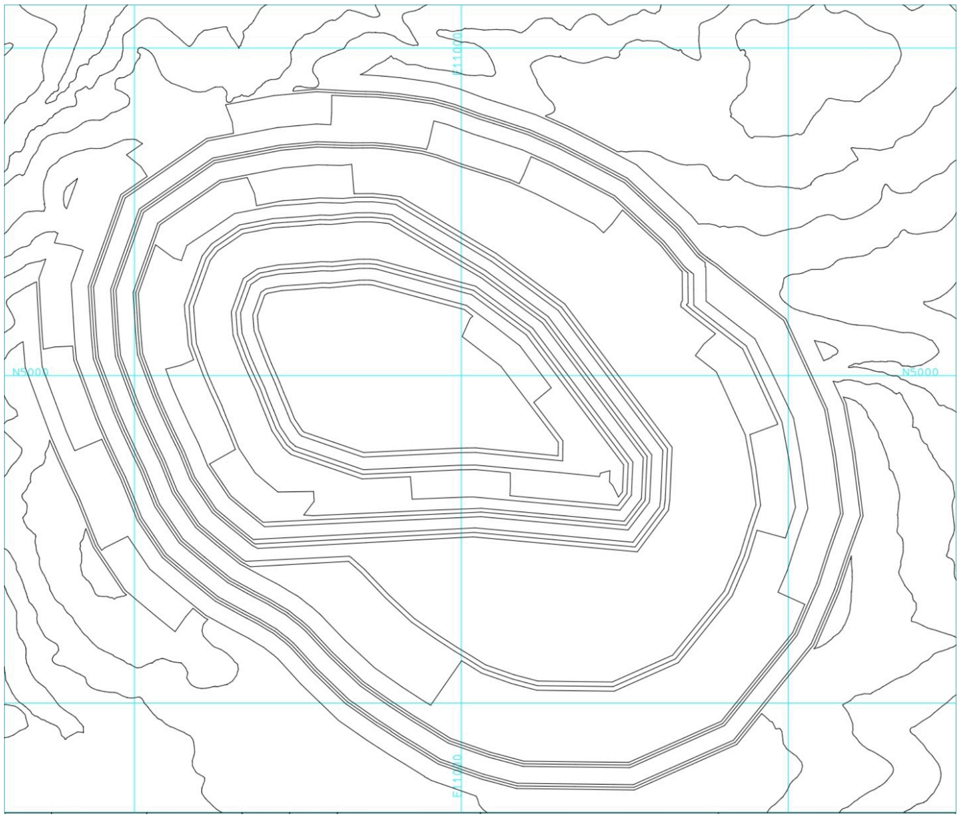


Figure 13.8: Mine Map End of Year 5

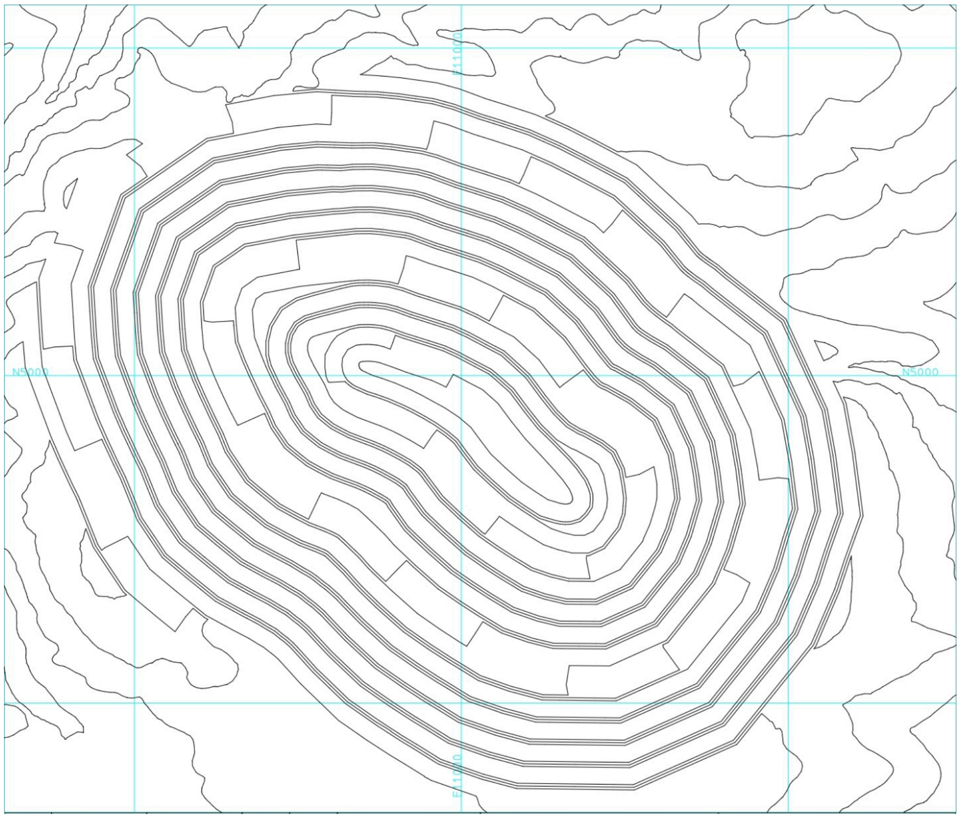


Figure 13.9: Mine Map End of Mine Life Year 8

14.0 PROCESSING AND RECOVERY METHODS

14.1 INTRODUCTION

he CK Gold processing facility has been designed to process 20,000 stpd of gold/copper sulfide ore. The processing facility and the unit operations therein are designed to produce a concentrate at 17.0% Cu or greater, with an average gold grade of 41 g/st.

The processing facility will consist of a ROM crushing circuit, crushed ore storage, a semi-autogenous grinding (SAG) mill/ball mill comminution circuit, rougher flotation, regrind circuit, and cleaner flotation to liberate, recover, and upgrade the copper and gold from the ROM ores. Flotation concentrate will be thickened, filtered, sent to a concentrate load-out bin, and bagged for subsequent shipping.

Tailings from the process will be filtered and conveyed to a tailings bin, where the dry-filtered cake will be loaded into a haul truck for transportation to the dry-stack tailings facility.

The process plant will consist of the following unit operations and facilities:

- Coarse ore receiving and storage area from the open pit mine.
- Jaw crushing system, crushed ore stockpile, and stockpile reclaim system to convey crushed ore to the process.
- SAG/Ball mill circuit incorporating cyclones for classification.
- SAG mill pebble crushing circuit.
- Rougher flotation circuit.
- Rougher concentrate regrinding circuit.
- Cleaner flotation circuit incorporating three flotation stages and cleaner scavenger flotation cells.
- Concentrate thickening and filtration circuit, including a concentrate bin and bagging station.
- Tailings thickening and filtration circuits.
- Tailings disposal at a dry-stack storage facility.
- Reagent handling, utilities, process water, and fresh-water systems

The block flow diagram for the processing facility is shown below in Figure 14.1

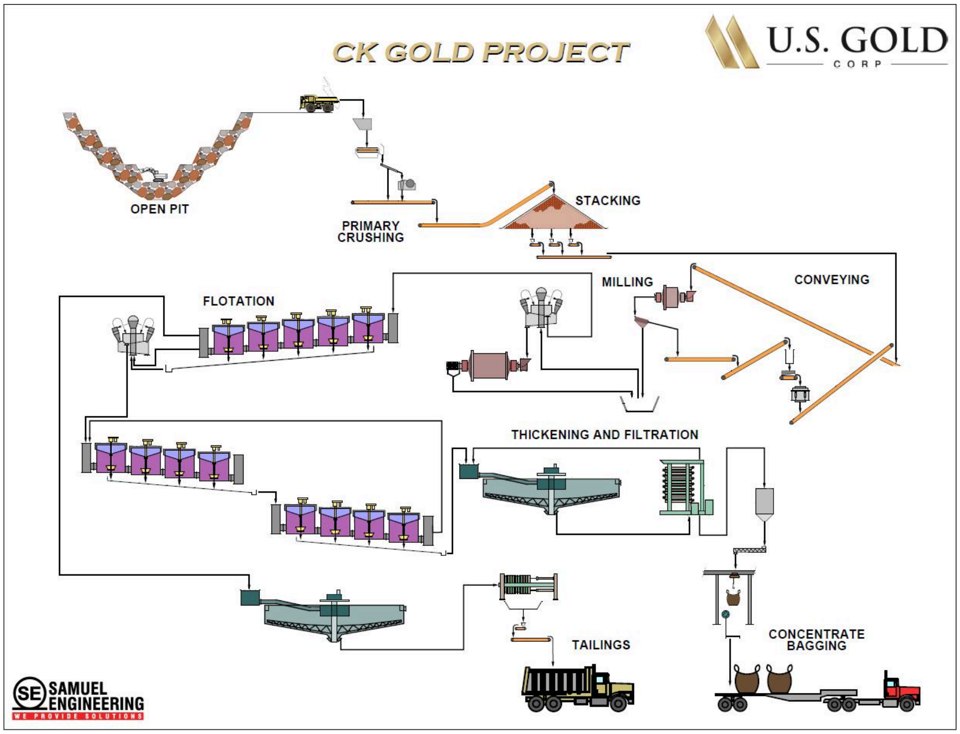


Figure 14.1: Block Flow Diagram – Processing Facility

14.2 PROCESS PLANT DESIGN

14.2.1 Major Design Criteria

The processing facilities were designed to process 20,000 stpd, equivalent to 7,300,000 stpy. The major design criteria used in the design are outlined in Table 14.1.

Table 14.1: Major Design Criteria

Criteria	Unit	Value
Operating Days per Year	day/y	365
Plant Availability (Crushing)	%	65.0
Plant Availability (Concentrate)	%	92.0
Mine Life	y	10
Daily Dry ROM Feed	stpd	20,000

Annual Dry ROM Feed	stpy	7,300,000
Copper Feed Assay	%	0.176
Gold Feed Assay	g/mt	0.48
Gold Feed Assay	ozt/st	0.014
Annual Dry Concentrate Production	stpy	61,063
Annual Dry Concentrate Production	mtpy	55,396
Copper Concentrate Assay	%	17.0
Gold Concentrate Assay	g/mt	45.2
Gold Concentrate Assay	ozt/st	1.32
Copper Recovery	%	72.4
Gold Recovery	%	68.1

14.2.2 Operating Schedule and Availability

The processing plant will be designed to operate in two 12-hour shifts per day, 365 days per year.

The crushing circuit availability is expected to be 65% throughout the LOM, and the comminution and flotation circuit availability is expected to be 92% throughout the LOM. This will allow sufficient downtime for scheduled and unscheduled maintenance of process plant equipment.

Major scheduled maintenance commonly requires five consecutive days and occurs twice yearly (10 days total). The remaining 19.2 operating days per year allocated to maintenance reflect minor scheduled and unscheduled maintenance.

14.3 PROCESS PLANT DESCRIPTION

14.3.1 Primary Crushing

Ore from the open pit will be delivered by haul trucks (or loader) to a dump hopper, where an apron feeder will feed the ore across a vibrating grizzly. The fine material will fall through to a discharge conveyor, and the oversize will report to the jaw crusher. The discharge from the jaw crusher will report to the same discharge conveyor and be conveyed to the crushed ore stockpile.

The crushing circuit will be equipped with a fog dust suppression system to control the fugitive dust generated during ore dumping and crushing.

14.3.2 Crushed Ore Stockpile and Reclaim

The crushed ore stockpile will have a live ore capacity of 13,333 st (16 hours) and a total capacity of 30,000 st (36 hours). Ore from the crushed ore stockpile will be reclaimed using apron feeders under controlled feed rate conditions. These feeders will discharge the reclaimed ore onto a conveyor belt feeding the semi-autogenous mill (SAG mill).

A belt scale will control the feed to the SAG mill by controlling the rate at which the apron feeders operate.

14.3.3 Comminution

The grinding circuit will be a comminution circuit with a SAG mill in series with a ball mill. It will be a two-stage operation with the SAG mill in a closed circuit with a pebble crusher and the ball mills in a closed circuit with the classifying hydrocyclones. The SAG mill will be equipped with pebble ports to remove coarse pebbles. Grinding will be conducted as a wet process at a nominal rate of 906 stph of material (dry basis).

The grinding circuit will include:

- SAG mill feed conveyor.
- Pebble crusher feed and discharge belts.
- Conveyor belts.
- Conveyor belt weigh scales and metal detectors.
- SAG mill, 34 ft diameter x 15 ft, 2 x 6,000 hp motors.
- Ball mill, 22 ft diameter x 35.5 ft , 2 x 6,000 hp motors.
- Pebble crusher, 500 hp.
- SAG mill discharge vibrating screens.
- Cyclone feed slurry pumps.
- Hydrocyclone cluster with 14 (11 operating, three spare) hydrocyclones.

Crushed ore reclaimed from the stockpiles will be fed to the SAG mill at a controlled rate. Water will be added to the SAG mill feed for wet ore grinding. The SAG mill will generally operate at 78% of its theoretical critical speed.

The SAG mill discharge will be equipped with pebble ports to remove critical-size material. Oversize material removed at the SAG mill discharge will be conveyed via transfer conveyors to the pebble crusher. A cone crusher will crush the pebbles to a P_{80} of 0.5 inch. The crushed material will be returned to the conveyor belt feeding the SAG mill for further grinding. The SAG mill discharge screen underflow will be discharged into the cyclone feed pumpbox.

The ball mill, subsequent to the SAG mill, will operate in closed-circuit with classification hydrocyclones mounted in a cluster. The product from the ball mill will be discharged into the cyclone feed pumpbox combining with the SAG mill discharge to become the cyclone feed. The classification size for the cyclones will be a P_{80} of 90 µm, and the circulating load to the ball mills will be targeted at 250% with the cyclone underflow returning to the ball mill as feed material. Dilution water will be added to the grinding circuit as required.

Cyclone overflow from the classification circuit will discharge into the feed of the rougher flotation circuit at the head of the flotation process. The pulp density of the cyclone overflow slurry will be approximately 37% solids.

Grinding media will regularly be added to the SAG and ball mills to maintain charge level and grinding efficiency. An automatic ball charging system will add steel balls to each mill.

14.3.4 Flotation and Regrind Circuits

Milled pulp will be processed using rougher flotation to recover the targeted minerals. The regrinding of rougher concentrate and cleaner flotation processes will be used to upgrade the rougher concentrate further into a high-grade copper/gold concentrate. Tank style flotation cells will be used in the rougher flotation, 1st and 2nd cleaner flotation, and the cleaner scavenger flotation. The 3rd cleaner flotation will be performed in a column flotation cell.

The flotation circuit will include the following equipment:

- Flotation reagent addition facilities.
- Rougher flotation tank cells, 5 x 7,063 ft^3 each.
- Regrind tower mill feed distribution box.
- One concentrate regrind tower mill, 2,000 hp.
- Regrind cyclone feed pumpbox.
- Regrind circuit classification cyclone cluster.
- 1st cleaner flotation tank cells – 4 x 706 ft^3 each.
- Cleaner scavenger flotation tank cells – 5 x 353 ft^3 each.
- 2nd cleaner flotation tank cells – 4 x 353 ft^3 each.
- 3rd cleaner column flotation cell, 10 ft diameter x 26 ft tall.
- Pumpboxes and standpipes.
- Slurry and concentrate pumps.
- Sampling system.

The cyclone overflow from the grinding circuit will feed the flotation circuit by gravity flow from the ball mill grinding circuit cyclone cluster. The slurry will be monitored for P_{80} particle size, and flotation feed samples will be taken periodically for process control and metallurgical accounting.

Cyclone overflow from the ball mill will discharge into the feed end of the five rougher flotation cells operating at a design solids total feed rate of 906 stph. Flotation reagents will be added to the flotation circuit as defined through testing. The flotation reagents added will be the collectors, PFSPD as well as PF7150 and the frother, MIBC. Provision will be made for supplementary reagent addition to the cleaner stages of the flotation circuit.

The sulfide minerals will be selectively floated into a rougher concentrate away from the other minerals-gangue components present in the ore slurry. The rougher concentrate will constitute approximately 10.4% mass of the plant feed. The rougher tailings will be sampled automatically before being discharged into the final tailings pumpbox for process control and metallurgical accounting purposes. The tailings thickener pumpbox will also receive the cleaner scavenger tailings. This combined stream will constitute the final tailings leaving the plant.

Regrinding and upgrading via cleaner flotation will be incorporated to more fully liberate the fine grains of sulfide minerals from the gangue constituents and enhance the copper/gold concentrate grade. A single stage of regrinding, three stages of cleaner flotation, and a stage of cleaner scavenger flotation are the selected methods for producing a final concentrate of acceptable grade and recovery.

Rougher concentrate enters the cleaner flotation section and is combined with regrind mill discharge at the regrind cyclone feed pumpbox. The regrind circuit cyclone cluster separates reground flotation concentrate into a fine cyclone overflow product and a coarse cyclone underflow product according to a target design particle size P_{80} 22 µm. The regrind mill will be a single vertical stirred tower mill. The regrind mill will discharge finely milled material into a cyclone feed pumpbox. This will be combined with rougher flotation concentrate and the cleaner scavenger concentrate, constituting the feed for classification by the cyclones.

The regrind cyclone overflow will become feed to the 1st cleaner flotation stage. Tailings from the 1st cleaner stage will report directly to the cleaner scavenger flotation stage. Tailings from the cleaner scavenger flotation stage will report to the final tailings pumpbox. The 1st cleaner scavenger concentrate will report to the regrind cyclone feed pumpbox for re-classification.

The 1st cleaner concentrate will feed the 2nd cleaner flotation stage. The 2nd cleaner concentrate will be fed to the 3rd cleaner column flotation cell. Tailings from the 3rd cleaner stage will be recycled back to the feed of the second cleaner stage. The concentrate of the 3rd cleaner stage will be the final concentrate with a design copper concentrate grade of 17.0% and a design gold grade of 41 g/st.

The concentrate will feed directly to the concentrate thickener for dewatering. Provision will be made for the copper concentrate thickener overflow water to be re-used in the grinding and flotation circuit as process water, providing this does not have a deleterious effect on the flotation of the sulfide minerals.

14.3.5 Concentrate Handling

Cleaner flotation concentrate will be thickened, filtered, and stored before shipment. The concentrate handling circuit will have the following equipment:

- Concentrate thickener of 29-foot diameter.
- Concentrate thickener overflow pumps.
- Concentrate thickener underflow slurry pumps.
- Concentrate filter feed tank.
- Concentrate filter press feed pumps.
- Concentrate filter press.
- Filter press washing and filtrate handling equipment.

The copper concentrate produced will be pumped from the 3rd cleaner flotation stage to the concentrate thickener feed well. Flocculant will be added to the thickener feed to aid the settling process. Thickened concentrate will be pumped to the concentrate filter feed tank using thickener underflow slurry pumps. The underflow density will be approximately 62% solids. The concentrate filter feed tank will be agitated. The concentrate filter will be a vertical filter press. Since filtration will be a batch process, the concentrate filter feed tank will also act as a surge tank for the filtration operation. The filter press will dewater the concentrate, producing a final concentrate with a moisture content of approximately 10%. Filtrate will be returned to the concentrate thickener. Filter press solids will be discharged directly onto the concentrate filter cake bin. Dewatered concentrate will be stored in the concentrate filter cake bin to be loaded into supersack bags. Concentrate thickener overflow will be collected in the process water tank for recycling within the mill circuit.

14.3.6 Tailings Handling

The final tailings for the Open Pit Process Facility will be thickened, filtered, and dry-stacked in the tailings compound.
The following process equipment will be required in the tailings handling area:

- Tailings thickener at 138-foot diameter.
- Tailings thickener overflow pumps.
- Tailings thickener underflow slurry pumps.
- Tailings filter feed tank (with agitator).
- Tailings filter feed pumps.
- Three tailings filter presses.
- Filter press washing and filtrate handling equipment.
- Filter press belt feeders.
- Filter press transfer conveyor.
- Tailings filter cake bin.

The rougher flotation tailings, together with the cleaner scavenger tailings, will be the final plant tailings. This combined stream will be pumped to the tailings filtration area, where it will be thickened and filtered, producing dry stack tailings as part of the tailings handling process. Once filtered, these tailings will be loaded into trucks and hauled to the tailings impoundment area for dry stacking.

The final plant tailings will initially be thickened in the tailings thickener to an underflow density of 60% solids. Flocculant will be used to facilitate the settling of the solids and aid in supernatant clarity.

Thickened tailings will be pumped to the tailings filter feed tank using thickener underflow slurry pumps. The tailings filter feed tank will be agitated. Tailings filtration will be done in multiple filter press units. Since filtration will be a batch process, the tailings filter feed tanks will also act as a surge tank for the filtration operation. There will be six presses, and each filter press will dewater the tailings to produce a “dry” cake with a moisture content of about 14%. The filtrate will be returned to the process water pond. The filter press solids will be discharged onto belt feeders, which in turn feed the transfer conveyor, which will feed the tailings filter cake bin.

Thickening and filtration of the tailings will facilitate the recovery of process water required for re-use in the plant before the final deposition of the plant tailings. Reclaim process water will be recovered as overflow from the tailings thickener and as filtrate from the tailings filters.

14.3.7 Reagent Handling and Storage

Various chemical reagents will be added to the process slurry streams to facilitate the recovery of the copper and gold minerals during the flotation process. Preparation of the various reagents will require:

- A bulk handling system.
- Mix and holding tanks.
- Metering pumps.
- A flocculant preparation facility.

- A lime mixing and distribution facility.
- Eye-wash stations and safety showers.
- Applicable safety equipment.

Various chemical reagents will be added to the grinding and flotation circuit to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the concentrate product. Fresh water will be used to prepare the various reagents, which will be supplied in powder/solids form or as solutions, which require dilution prior to addition to the slurry. These reagent solutions will be added at the addition points of the various flotation circuits and streams using metering pumps.

The PFSPD and PF7150 collector reagents will arrive at the plant as a solution in reagent totes. The solution will be pumped from the reagent totes directly to additional points in the circuit. The frother reagent, MIBC, will be delivered in bulk, transferred to a holding tank, and pumped to the appropriate addition points using metering pumps.

Flocculant will be prepared in a flocculant mix system to produce a dilute solution with a 0.40% weight solution strength. This solution will be further diluted using in-line mixers. Two flocculant make-up facilities will be required, one for the concentrate area and one for the dry stack tailings area.

Hydrated lime will be delivered in bulk and will be off-loaded pneumatically into a silo. The lime slurry will then be prepared as a 20% weight concentration slurry in a lime mixing tank. This lime slurry will be pumped to the points of addition. Discharge valves on the closed loop will be controlled by pH monitors that will regulate the amount of lime added.

Grinding media will be added to the various mills used throughout the process as required. Mill charging will be conducted using automatic ball charging systems.

The estimated consumption rate for grinding media is based on historical data from similar projects, using the average abrasion index of the deposit and the estimated equipment power consumption. To ensure spill containment, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest tank. In addition, each reagent will be prepared in its own bounded area to limit spillage and facilitate its return to its respective mixing tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire, safety protection, emergency shower and eye wash stations, and Material Safety Data Sheet stations will be provided at the facility. Each reagent line and addition point will be labeled following the Mine Safety and Health Administration (MSHA) standards. All operational personnel will receive MSHA training and additional training for safely handling and using the reagents.

14.3.8 Water Supply

The process plant will have individual fresh and process water distribution systems. The freshwater supply will come from mine dewatering and purchased water from local sources.

14.3.9 Fresh-Water Supply System

Fresh water from the combined fresh and fire water tank will be supplied to each area. Fresh water will be used primarily for the following:

- Fire water for emergency use.
- Dust suppression in the crushing system.
- Gland service for slurry pumps.
- Reagent preparation water.
- Concentrate and tailings filter wash water.
- Make up water for the main process facility.

The fresh/fire water tank will be equipped with a fresh-water standpipe to ensure that at least half of the 1,227,000 US gallon tank is available for fire water supply.

14.3.10 Process Water Supply System

Process water recovered from the concentrate and tailings thickener overflows will be re-used in the plant's process circuit via the facility's process water pond.

Reclaimed water from the concentrate and tailings filters will be recycled back to the process water pond for distribution to the usage points. As process water demand in the flotation and grinding circuit is expected to be greater than the amount reported to the process water pond, including the portion returning from the tailings facility, fresh water make-up will be required for the main process.

14.3.11 Air Supply

Process air service system will supply air to the following areas:

- Low-pressure air for flotation cells.
- Drying air and pressing air for the concentrate filter press operation.
- Drying air and pressing air for tailings filter press operations.
- Air compressors are also supplied for general plant distribution.
- Instrument air will be prepared from the plant air compressors, dried, and stored in a dedicated air receiver.

14.3.12 Process Plant Manpower

Process plant salaried personnel estimates were developed to provide adequate supervision and technical support for the daily operation of the process facility. The required salaried personnel for the process facility is estimated at 13 persons, as detailed in Table 14.2.

Table 14.2: CK Gold Salaried Personnel		
Area	**Position**	**Count**
Management		**4**
	Plant Manager	1
	Maintenance Manager	1
	Assistant Mill Manager	1
	Operations Clerk	1
Technical		**3**
	Chief Metallurgist	1
	Senior Metallurgist	1
	Junior Metallurgist	1
Operations		**4**
	Shift Supervisor	4
Maintenance		**2**
	Maintenance Foreman (day)	1
	Maintenance Planner (day)	1
Total Salaried Manpower		**13**

Salaried personnel will supervise a total of 65 hourly employees, as detailed in Table 14.3. Process positions, both salaried and hourly, that require 24-hour coverage per day will be staffed by rotating 12-hour shifts.

Table 14.3: CK Gold Hourly Personnel		
Area	**Position**	**Count**
Operations		**41**
	Control Room Operator (shift)	4
	Crusher/Conveying Area Lead Operator (day)	4
	Crusher/Conveying Area Laborer (day)	4
	Shift Operators Grinding	4
	Shift Operators Flotation	4
	Shift Operators Concentrate Handling/Bagging	8
	Shift Operators Tailings Thickening/Filtration	4
	Reagent Area Operators (day)	1
	Utility Operator (day)	2
	Non-Specialty Operator (Roaming)	4
	Concentrator General Laborer (day)	2

Maintenance		**24**
	Electrician (shift)	4
	Mechanical Fitter (shift)	4
	Mechanical Fitter (day)	4
	Boilermaker (day)	2
	Electrician (day)	3
	Instrument Technician (day)	2
	Trades Assistant (day)	5
Total Hourly Manpower		**65**

15.0 INFRASTRUCTURE

15.1 ROADS

15.1.1 Project Access Road

As shown in Figure 15.1, the CK Gold Project (Project) access road is a gravel road that initiates at County Road 210 (also named Crystal Lake Road), heads south and then west to the Project site boundary. The access road is approximately 4.2 miles long and 26 ft wide, generally centered along a 60-foot-wide right-of-way. The Project site boundary extends to County Road 210 following the access road right-of-way. A typical cross section of the access road is shown on Figure 15.2.

Fencing will be installed along the right-of-way boundary. The access road does not cross any streams. The material for sub-base, base, and gravel surfacing will be sourced from borrow areas within the Project site.

Access road construction will be one of the first tasks performed in the construction phase, following stripping and stockpiling of topsoil. The Project will obtain a permit for the access road connection to County Road 210 from Laramie County.

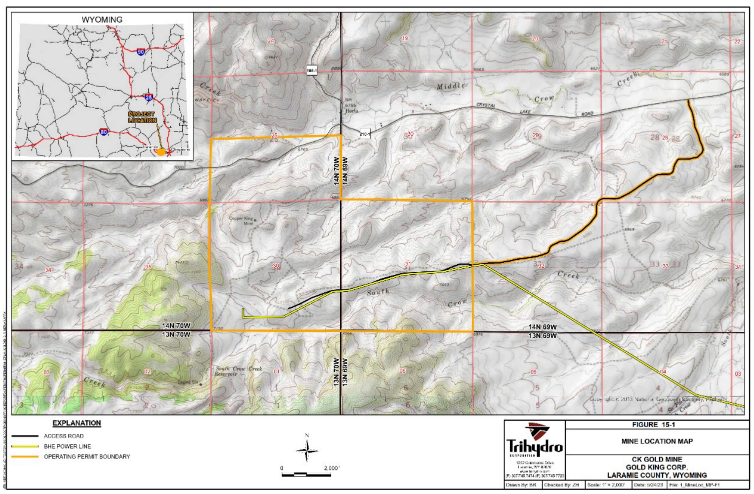


Figure 15.1: Project Access Road

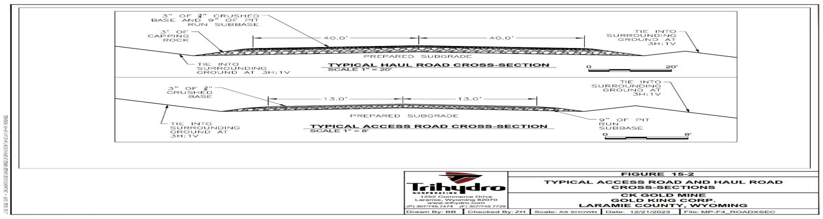


Figure 15.2: Typical Cross Section of the Access Road

15.1.2 Ex-Pit Haul Roads

Ex-pit haul roads are designed to be 80 ft in width to accommodate 100 st haul trucks and will generally follow the existing two-track roads within the Project area where possible. Construction of the internal haul roads assumes that material for the haul roads can be sourced from within the Project area from the pit.

15.2 ORE STOCKPILE AND WASTE ROCK FACILITIES

The Project will utilize an Ore Stockpile storage facility for storing mineralized material and the West and East Waste Rock (storage) Facilities (WWRF/EWRF) for storing non-mineralized material from the pit. Figure 15.3 (in the next section) shows the location of each storage area in proximity to the pit, mill area, truck area, and Tailings Management Facility (TMF). Each storage facility will have the topsoil stripped and stockpiled in designated areas prior to placing rock material.

15.2.1 Ore Stockpile

The Ore Stockpile is located entirely within Section 36 in the valley to the south and west of the pit. It will be used to store up to 20 Mst of mineralized material for future processing. The Ore Stockpile will have a composite liner system (CLS) consisting of geomembrane overlying a prepared subgrade of compacted Project area clays and silts. The CLS will have an effective permeability of 10^{-7} cm/s or lower, as required by the DEQ – WQD. The Ore Stockpile CLS will be covered with a gravel protection layer to allow for Project equipment to place mineralized material. An underdrain will be installed prior to installing the CLS and a collection drain will be installed prior to receiving mineralized material. The Ore Stockpile is designed to have a stack height of up to approximately 214 ft for a top elevation of 7,230 ft above mean sea level (amsl). Erosion control measures will also be implemented to control stormwater runoff. Future studies should develop a stacking plan for the ore stockpile and evaluate slope stability including the liner interface.

As a final note, the Ore Stockpile facility is a temporary facility as the Ore Stockpile will be completely depleted over the life-of-mine. Once the Ore Stockpile facility has been depleted, the CLS will be removed, topsoil replaced, and revegetation performed for closure purposes.

Table 15.1 summarizes estimated cut and fill volumes associated with haul road construction for the Project. The road construction costs are included in the mining costs.

Table 15.1: North and South Haul Road Quantities			
	North	South	Total
Haul Rd Const. / Cut (BCY)	12,850	132,200	145,050
Haul Rd Const. / Fill (LCY)	358,450	779,100	1,137,550

15.2.2 West and East Waste Rock Facilities

The WWRF and EWRF are located mostly in Section 36 and partially in Section 31 in the valley in the ephemeral Middle tributary to Middle Crow Creek (to the southeast of the pit). The WWRF will accommodate approximately 17 Mst of waste rock and the EWRF will accommodate approximately 15 Mst of waste rock. The WWRF and EWRF will both be 7,130 ft amsl at full height. The slopes of the WWRF and EWRF are designed to 3H:1V. At the end of the mine life, replacement of topsoil and revegetation is planned for closure purposes.

15.3 TAILINGS DISPOSAL

Tailings generated in the flotation process will be filtered to an optimum low moisture content to produce “dry stack” tailings, thereby maximizing water conservation and structural strength and avoiding the need for a tailings dam and the associated environmental and safety risks. The tailings slurry produced by flotation initially containing about 65% water (by weight) will first be thickened for initial water recovery. The water content of the thickened underflow slurry will be reduced to about 45%, while the thickener overflow water will be returned to the process for reuse. The thickened slurry will be pumped to storage tanks ahead of a large pressure filtration plant comprising multiple large pressure filters that further reduce the water content to less than 15% (typically 14% metallurgical[1]). This leaves the solids as a compressed “cake” material that will be dropped from the press onto a conveyor for transportation to the TMF.

Approximately 2,400 stph of slurry will be sent to the tailings thickener, with approximately 1,057 stph of tailings produced on average. Processed tailings will be hauled to and placed in the TMF until Year 8.25. After that, the remaining tailings produced will be hauled to and placed in the open pit (as described in Section 15.3.4).

15.3.1 Chemical Characteristics

Geochemical testing of mine rock and tailings using industry standard methods on representative samples (Geochemical Solutions 2023) indicates limited probability to produce acid rock drainage (ARD) and/or metal release to water. Static geochemical testing on tailings samples produced by locked cycle laboratory testing indicates that the tailings are not acid generating. Static geochemical testing of waste rock samples indicates only a small percentage of waste rock is potentially acid generating (PAG). Confirmatory kinetic and leach test results show no, or low, production of acidic water or metal release for the tested samples. Section 17.1.4 presents additional details on the geochemical characterization of tailings and pit rock.

[1] Metallurgical water content is tailings moisture by total weight. Geotechnical water content is measured by dry weight. A 14% tailings moisture metallurgical is equivalent to 16.3% geotechnical.

15.3.2 TMF Design and Construction

The TMF is sited east of the process plant within a valley formed by the ephemeral South tributary to Middle Crow Creek (Figure 15.3). The TMF begins near the northeast corner of Section 31. The basin’s topography contains and directs the placement of tailings down-valley to the east of the South Crow Creek water transmission pipeline.

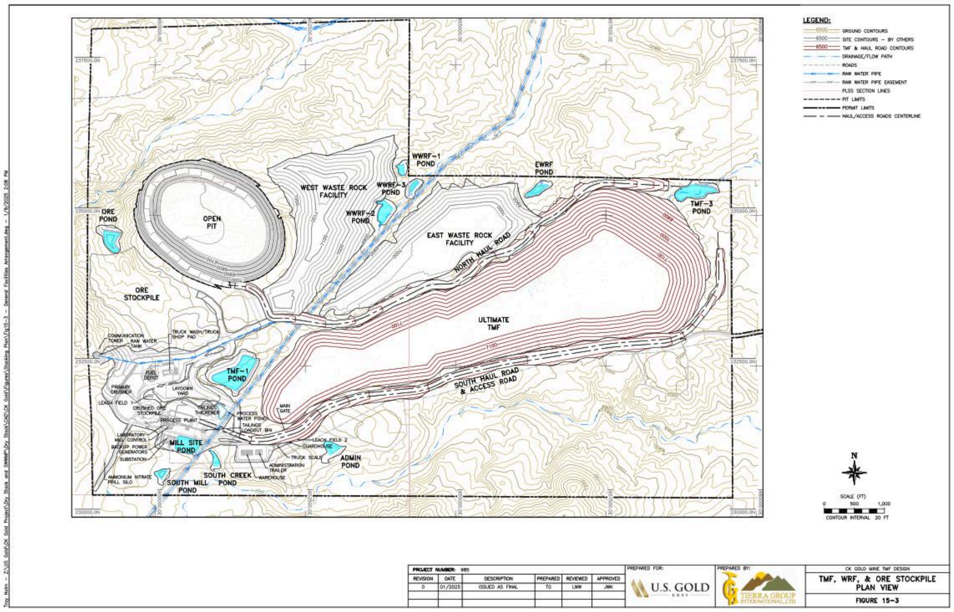


Figure 15.3: TMF, WRF, & Ore Stockpile Plan View

Tailings filtration produces tailings near their optimum moisture content for compaction, maximizing their geotechnical strength and stability. The risk of spills and the magnitude of seepage to groundwater are thereby significantly reduced. The filtered tailings will be co-deposited with waste rock to provide structural buttresses for stability and a cover to protect against weathering and wind erosion. Tierra Group (2025a) has performed limited equilibrium stability analyses of the TMF under static, pseudo-static and post-peak loading conditions, including liquefaction assessment, to verify that acceptable factors of safety are obtained for all cases.

The TMF will be developed in three phases, as shown on Figure 15.4. The TMF will ultimately store 52 Mst of tailings over the facility's life. Each phase of the TMF will consist of a prepared subgrade, underdrain collection system, CLS, seepage collection system (overdrain), tailings, and waste rock. The tailings will be placed in the TMF in 10- to 20-foot lifts and the waste rock buttress and shell will be installed in 10- to-20-foot lifts as the tailings increase in elevation. The waste rock starter berm built prior to Phase 1 of the TMF will be constructed in 2- to 3-foot lifts. The TMF underdrain collection system is shown on Figure 15.4 and Figure 15.5 and illustrates the TMF cross-sections. The tailings are contained by a waste rock retention shell functioning as a buttress. The top of the TMF is capped with 3 ft of waste rock. The ultimate configuration will have tailings built out at 1.8H:1V slopes and waste rock at 3H:1V slopes around the TMF's perimeter. A vegetated soil cover will be placed over the closed TMF to promote the conveyance of stormwater, prevent surface water ponding, disperse runoff, limit erosion, and promote native vegetation.

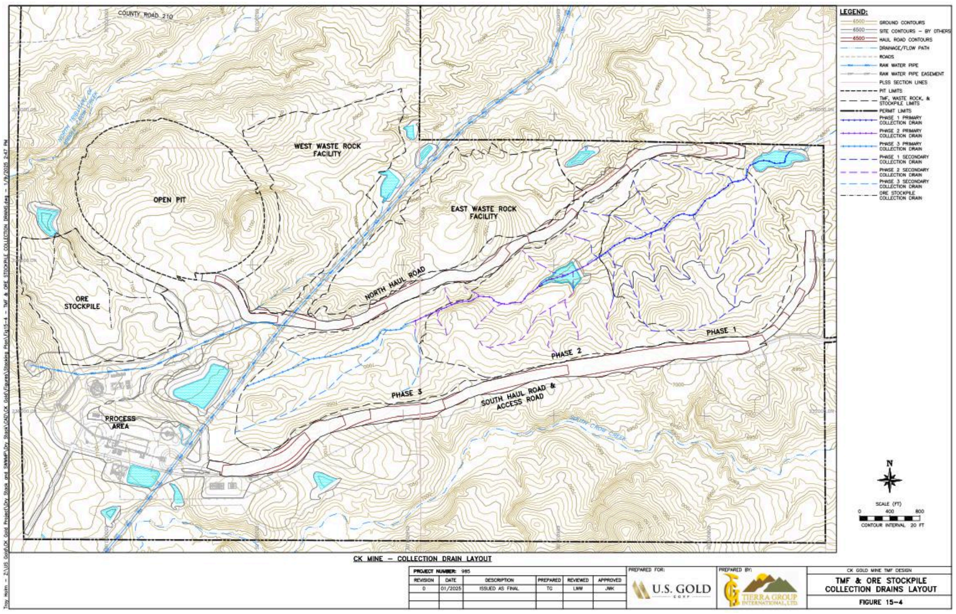


Figure 15.4: TMF& Ore Stockpile Collection Drain Layout

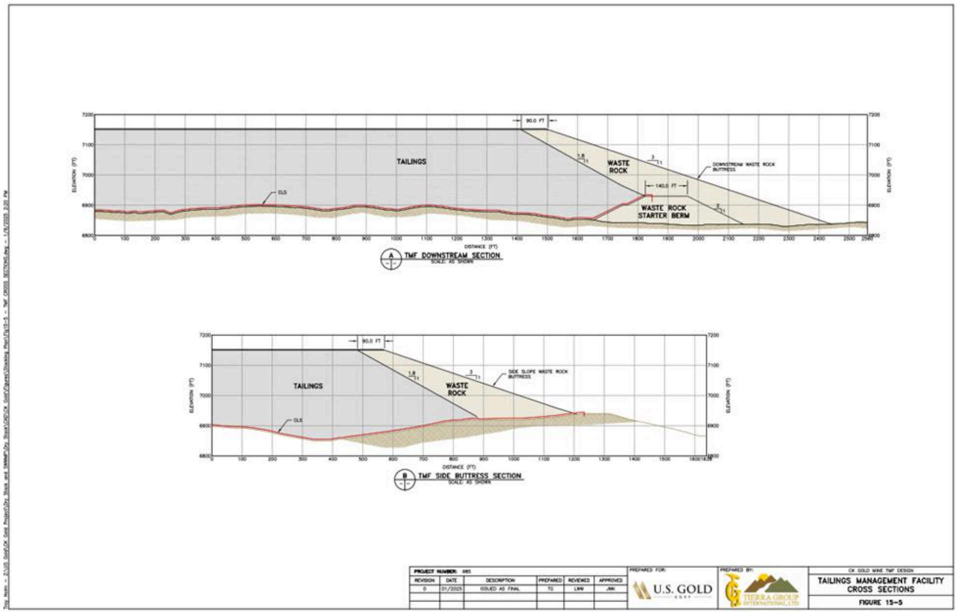


Figure 15.5: TMF Downstream and Side Buttress Typical Cross Sections

TMF foundation preparation will include clearing, grubbing, and stripping of topsoil. Unsuitable overburden material will also be removed, including soils that are unable to be compacted and used in the CLS subgrade, such as saturated soils or soils that are not clay or silt. Saturated soils may be reworked and dried for later use. Unsuitable soils will be segregated depending on type and either used in other mine operations (e.g., pond embankment construction, road maintenance, waste rock facility pad development, etc.) or stockpiled south of the access road segregated from the topsoil piles. The portions of the soil within the White River Formation will be ripped and worked to remove the light cementing structure in the material and make it suitable for compaction and use in the CLS subgrade. The metasediments rock outcrop where the valley narrows in the Phase 2 of the TMF will be drilled, blasted, and dozed to a slope of approximately 2.5H:1V. The slope will be dressed and covered with at least 6 inches of compacted soil prior to liner construction. The excess rock will be used in applications that require crushed rock if the rock quality is deemed suitable. The remaining subgrade will be compacted to a minimum of 90% Standard Proctor Maximum Dry Density (SPMDD) to provide a firm surface for underdrain and CLS construction. Figure 15.6 shows the typical cross sections of the primary and secondary underdrain and overdrain (seepage collection drain system).

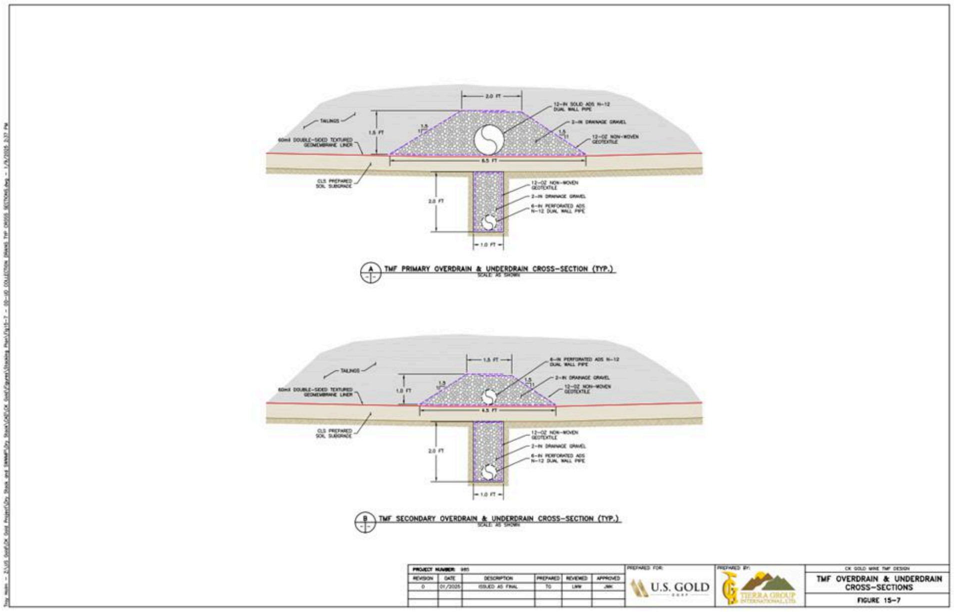


Figure 15.6: Overdrain & Underdrain Collection System Cross Sections

An underdrain will be installed below the liner to segregate the tailings from groundwater and convey incident groundwater seeps beneath the TMF down the thalweg (line of lowest elevation) of the TMF valley. The underdrain will consist of a central artery in the thalweg fed by secondary drains set in the minor valley bottom topography as shown on Figure 15.6. This underdrain system is comprised of perforated HDPE drainpipes wrapped in non-woven geotextile and placed in a gravel-lined trench. It will discharge to TMF-3, downstream of the TMF. Underdrain flows were estimated based on the spring observed in the TMF area. Each secondary drain was assumed to carry a flow equivalent to the observed spring flow. Flows will be cumulative in the underdrains, with the maximum flow in the primary drainpipe aligned at the bottom of the valley's low point. The primary and secondary drainpipes will be a minimum 6-inch diameter, designed for inspection and maintenance, and capable of conveying the maximum design flow. Figure 15.6 shows the typical primary and secondary underdrain cross sections.

Above the underdrain, the TMF will have a CLS consisting of geomembrane overlying a prepared subgrade of compacted Project area clays and silts. The CLS will have an effective permeability of 10^{-7} cm/s or lower, as required by the DEQ – WQD.

A seepage collection drainage system will be installed on the liner and prior to the placement of tailings. The purpose of the seepage collection system is to maintain low hydraulic head in the bottom of the tailings mass, to promote free drainage of the tailings, and minimize the possibility for the tailings to become saturated. Like the underdrain, the above-liner seepage collection system consists of a primary drain that is constructed to follow the valley thalweg and secondary drains that are set in the minor valley bottom topography as shown on Figure 15.6. The primary drain will be solid HDPE pipe wrapped in non-woven geotextile surrounded by drainage gravel. The secondary drains will be perforated HDPE pipe wrapped in non-woven geotextile surrounded by drainage gravel. The primary seepage collection drainpipe will be 12 inches in diameter. The secondary seepage collection drainpipe will be 6 inches in diameter.

Phase 1 will consist of approximately 3,230 ft of primary underdrains, 2,740 ft of primary seepage collection drains., 11,460 ft of secondary underdrains, and 10,100 ft of seepage collection drains. Phase 2 will include approximately 2,700 ft of primary underdrains, 2,700 ft of seepage collection drains, 6,295 ft of secondary underdrains, and 5,470 ft of seepage collection drains. Phase 3 will include approximately 2,400 ft of primary underdrains, 1,940 ft of seepage collection drains, 3,820 ft of secondary underdrains, and 3,450 ft of seepage collection drains.

Tailings will be placed and compacted adjacent to and above the seepage collection drains. The tailings will be hauled by trucks from the tailings loadout bin at the mill along the south haul road to the TMF, where they will be end-dumped in 10- to 20-foot-thick lifts and spread with low-ground-pressure dozers. Tailings consist primarily of silt-sized grains with lesser fine sand and clay that will be unlikely to damage the seepage collection drains or the CLS. Tailings will be placed and spread in a manner that prevents damage to the drains or CLS. Placement of waste rock on the tailings surface will be necessary to form access roads to support the haul trucks. The waste rock roads will also serve as drains and connect to the buttress to prevent build-up of pore pressure within the tailings. Haul road development will precede the placement of tailings in each lift. The crest of the tailings will be graded to prevent standing water from pooling on top of the tailings. Surface water run-on will be controlled with temporary ditches around the perimeter to divert water around the TMF. The top lift in areas where tailings are not being actively placed will be rolled with a smooth drum compactor to 90% SPMDD to reduce infiltration and prevent fugitive dust from being generated on the tailings.

Waste rock will be placed concurrently with the tailings to form a retention shell once the tailings are at the design levels and to form a buttress providing structural support and protecting the tailings from erosion due to precipitation and wind. The minimum width of the waste rock buttress is 90 ft as determined by geotechnical analyses and modeling by Tierra Group (2025b). Additional waste rock buttress width is required when exceeding the slope heights determined in Tierra Group's geotechnical stability analyses. The waste rock will be dumped in 10- to 20-foot-thick lifts, spread with dozers, and compacted by compactors and haul truck traffic. The tailings will be placed with a maximum outer slope angle of 1.8H:1V, with the outer structural rock retention shell at a maximum outer slope angle of 3H:1V. The TMF is stable when the waste rock retention shell is built out to the maximum section of 3H:1V with a minimum waste rock width of 90 ft at the crest. The slopes and the final crest of the facility will be covered with waste rock (retention shell and cap) to protect against wind and water erosion.

15.3.3 TMF Environmental Management

The following specific environmental management aspects will be incorporated into the TMF operation and maintenance plan:

- Erosion and sediment control
- Water management and seepage control
- Dust control

- Off-specification tailings management
- PAG waste rock disposal
- Monitoring and inspection
- Reclamation

These environmental management controls are further described in Section 17.2.1.

15.3.4 Pit Backfilling

The pit is planned to be excavated for approximately 8.25 years and will generate an ore stockpile to be fed to the process plant. The stockpiled ore will be depleted during the last two years of post-mining mineral processing and the associated tailings will be transported to the pit bottom for backfilling up to an elevation of 6,630 ft amsl (assuming this plan is consistent with other possible closure plans for the pit concerning its potential alternative use as a water storage reservoir). Then, with a combination of blasting and earthmoving, the pit rim will be dozed into the pit to create a 3H:1V final pit wall slope. The final backfilled pit elevation will be approximately 6,720 ft amsl, as shown on Figure 15.7 The associated long-term ARD implications and effects on groundwater are described in Sections 17.1.2 and 17.2.1.

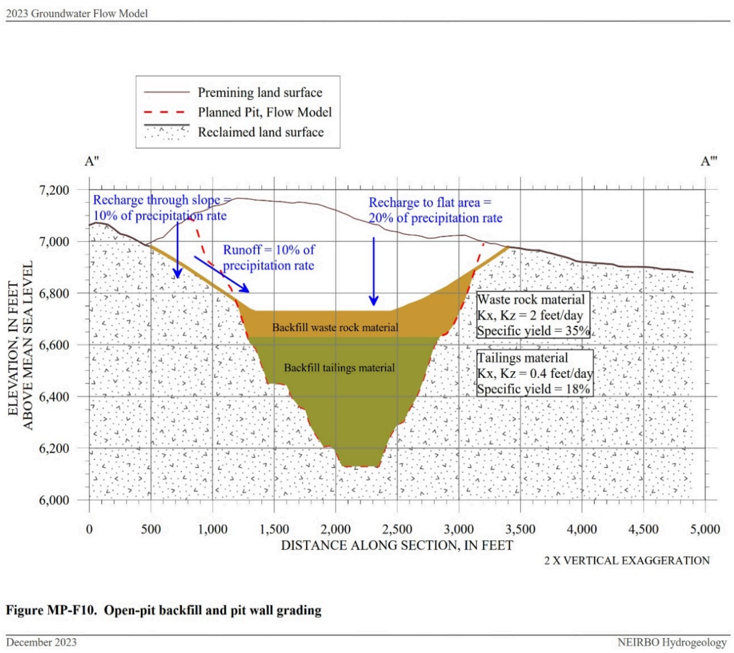


Figure 15.7: Open-Pit Backfill and Pit Wall Grading

15.4 PLANT FACILITY EARTHWORK

The Project's mill and infrastructure facilities are located south of the pit in Section 36 and are shown in Figure 15.8. Figure 15.9 shows the layout of the mill facilities in more detail. Proposed ground grading designs were prepared for the mill area, mining equipment maintenance area, substation, administration building and warehouse area, primary crusher, and coarse ore stockpile, and supporting facilities. The site grading design looked to balance the cut and fill volumes as well as possible, address stormwater runoff and reduce erosion. Generally, each pad was designed with a slight slope to facilitate drainage. Table 15.2 summarizes the bank cut and loose fill volumes and overall grading area.

Table 15.2: Plant Area Quantities

Grading Area	Value
Grading Cut Volume	500,000 yd^3
Grading Fill Volume	610,000 yd^3
Total Cut + Fill Volume	1,100,00 yd^3
Net Cut + Fill Volume	110,000 yd^3
Grading Area (Plant Area)	60 Acres

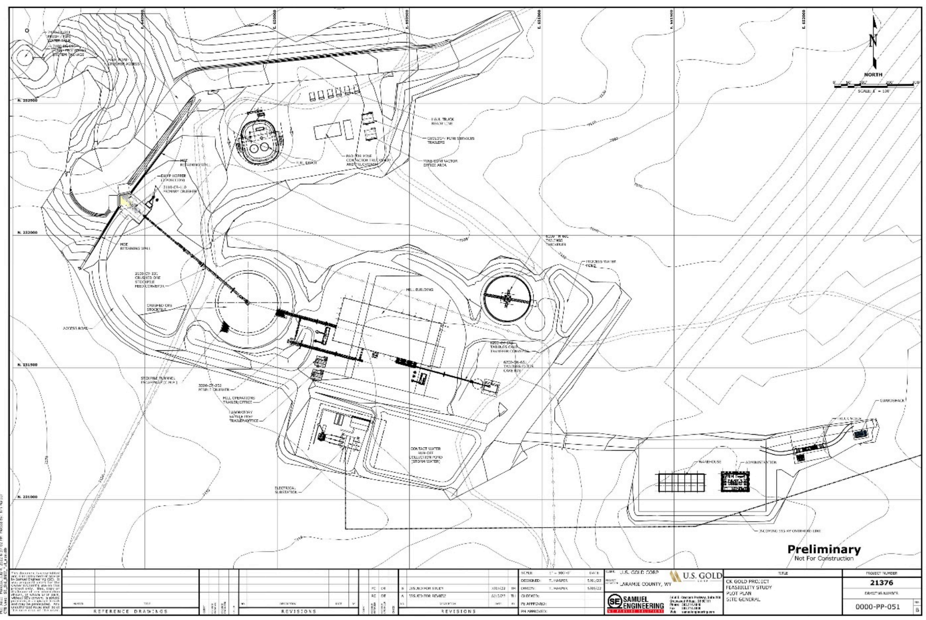


Figure 15.8: Mill and Truck Area

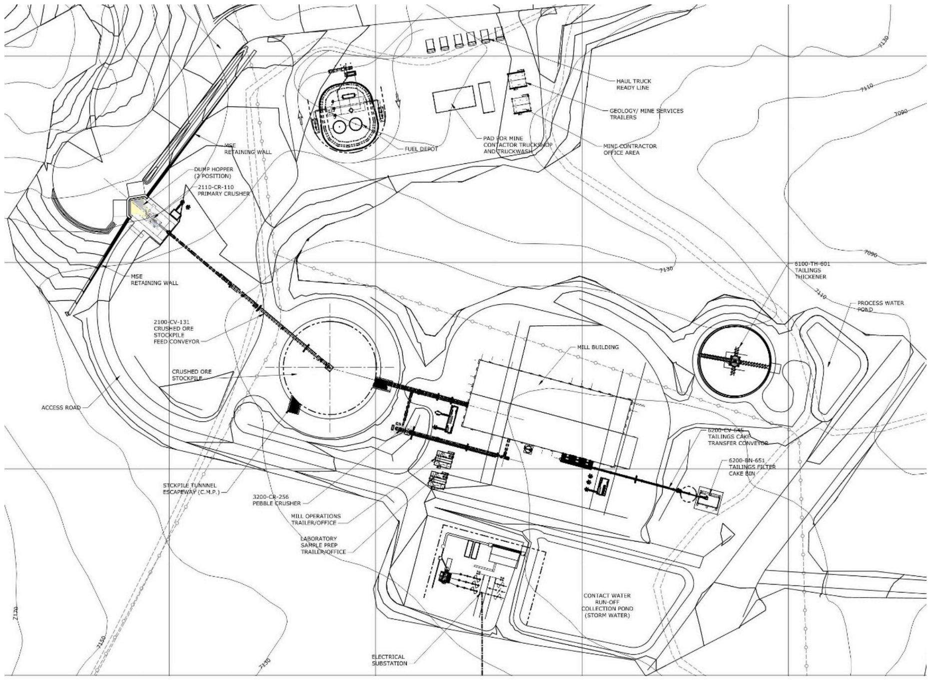


Figure 15.9: Mill Area Plan View

15.5 POWER AND WATER

15.5.1 Power Supply

Electrical power for the CK Gold Project will be supplied by a local utility company, Black Hills Energy, under an Industrial Contract Service Agreement. The anticipated Connected Power Load for the Project is approximately 40.4 megawatts (MW) with a Demand Power Load of 27.2 MW. The power demand for the Project requires that a new 115 kV power line be constructed for the Project by Black Hills Energy. The power line would be constructed from Black Hills Energy's West Cheyenne substation, located approximately 16 miles east of the Project to a new Black Hills Energy owned, built and operated 115/13.8 kV (50 MVA) distribution substation (including transformer) near the mine. The powerline alignment would take advantage of existing easements and planned county roads in the vicinity of the CK Gold Project. The alignment would require easements from the City of Cheyenne, State of Wyoming, and two local ranches.

The mine electrical facilities would be required to provide sufficient reactive support for the mine's electrical system to maintain reliability and voltage levels on the Black Hills Energy system. Black Hills Energy performed a load addition report to determine the impact of the CK Gold proposed mining operation.

Unit costs for construction of the infrastructure for the power line and unit rates for the delivered power under an Industrial Contract were provided by Black Hills Energy in August 2022. A cost estimate for the right-of-way easement was also provided by Black Hills Energy. The estimated construction costs for the proposed power line, easement cost and substation have an option to be amortized in an addition to the base power unit rate charged. The estimated construction cost for the proposed power line is $17 to $18 million and the easement costs are $140,000 per mile. The power unit rate is 7.5₵/kWh inclusive of amortized power supply construction costs the unit rate is estimated to be 7.9. c/kWh.

15.5.2 Water Supply

The Project will operate in a net water deficit situation, given that the mean annual evapotranspiration exceeds the mean annual precipitation. The project's total average water consumption is 562 gpm. This number is the estimated total consumption, excluding reductions in demand for water from off-site sources associated with planned water saving measures. Water consumption includes use for mineral processing, general operations and dust control. Tierra Group developed a site-wide water management plan to maximize water reuse and minimize freshwater make-up. Details of the site-wide water management plan can be found in Section 17.2.3.

The Project has a water supply agreement with the Cheyenne Board of Public Utilities (BOPU). Sunrise Engineering was retained to provide a hydraulic analysis for a transmission main and a service line from the Lone Tree Wellfield, located south of Interstate 80 in section 17 of Township 13N and Range 69W, to the proposed CK Gold water storage tank, located on the CK Gold site in section 36 of Township 14N and Range 70W. After discussion with the BOPU in Cheyenne Wyoming, a twelve-inch minimum pipe was analyzed to determine the required head to lift raw water from the Lone Tree Wellfield to a proposed BOPU diversion structure located East of the CK Gold Site. A bore under Interstate 80 and the Union Pacific Railroad was determined to be feasible after a geotechnical report was completed for the site. At the diversion structure, a sixteen-inch pipeline will be used to convey water to and from South Crow Creek Reservoir. An eight-inch line was determined to be sufficient to convey the needed water from the proposed sixteen-inch waterline to the CK Gold water storage tank, although it does require a booster pump to provide the needed head pressure. A construction and final engineering cost estimate was provided at $17,259,500 and $930,000 respectively.

Potential sources of water for the Project include the BOPU water supply system, on-site existing surface water rights and potential new on-site wells. The BOPU has sufficient capacity currently to supply the Project’s needs. The Cheyenne City Council has approved an outside water agreement allowing BOPU to contract with the Project to provide water. Water generated from pit dewatering, surface runoff, and waste rock and tailings seepage will be recycled for use in mineral processing and/or dust suppression.

U.S. Gold, operating under a water agreement with Ferguson Ranch, is conducting a water exploration program on land immediately north of the Project area. The Casper Formation, a significant water-bearing rock, has been intercepted twice in the Red Canyon area one mile north of the proposed project water tank. Significant sandy intervals have been logged and the geophysical log indicates high resistivity consistent with sand-bearing intervals. U.S. Gold is conducting draw down tests and will model the hydrology and estimate the water well pumping capacity. The potential ease of construction, operation and cost savings make the Red Canyon water well the primary source of water and the BOPU agreement and pumping from the Lone Tree well field. The costs to construct and operate the Red Canyon well field are estimated and included in the cash flow model.

Water will be transported to site via an HDPE pipeline from the well field to the project water tank. Pumps, SCADA control equipment, pumphouses, and well field infrastructure have been engineered and costed.

16.0 MARKET STUDIES

16.1 FLOTATION CONCENTRATES

16.1.1 Flotation Concentrates

This section considers the smelting and refining terms available for the flotation concentrate generated over approximately ten years of project life. This filter cake product will include marketable concentrations of copper, gold, and silver, will be largely free of deleterious elements, and is expected to be of significant interest to domestic and overseas smelters alike. The concentrate will contain 8 to 9% moisture and be stored and transported in Flexible Intermediate Bulk Containers (FIBCs), also known as Bulk Bags.

Increased demand for copper in the Asian markets has stimulated the expansion of processing capacities for copper raw materials in the East, and this has driven a reduction and/or elimination of similar processing capacities elsewhere in the international market. The balancing of supply and demand is expected to continue, where the newly created processing capacity should absorb much of the new copper concentrate production capacity that will be realized.

The quantity, quality, and estimated value of the Project's flotation concentrate product allow shipment to a wide range of geographic regions; therefore, as the Project advances, those regions and consumers providing the least commercial risk and the optimum return to the Project should be considered. Of note, the concentrate's relatively high gold and silver content suggests that those locations providing high accountability for these precious metals must be considered.

16.1.2 General Considerations

Based on the results of recent metallurgical testwork, the flotation concentrate will be a clean product that will be in demand for its contained gold and lack of deleterious elements. While the planned average copper content is slightly lower than many copper concentrates with an 18 to 19% copper grade, the higher gold grade of 40 to 90 gpt and a sulfur-to-copper ratio of approximately 1:1 will make it attractive to several domestic smelter facilities.

As the Project develops toward production, it is recommended that focus is maintained on selective smelting and refining complexes that currently process copper concentrates in North America. Compared to overseas markets, transportation logistics and timelines should be more streamlined, resulting in more attractive payment terms.

As a result of the total contained metal value of this product, a flotation concentrate bagging facility has been included in the process plant design. Whilst logistics considerations will vary slightly from those of bulk concentrate shipments, shipment of this product in 4,500lb FIBCs will improve overall metals accountability by minimizing the losses of gold during transportation.

Normal deviations in moisture content and the methods established to sample and determine the settlement dry weight must be closely examined and controlled to ensure appropriate confidence in the metallurgical balance. It is recommended that moisture samples be taken when the filter cake product batches are weighed and sampled for assay. Care must be taken to immediately seal the moisture sample and follow the established procedures for drying and determination of dry weight. Sampling for assay determination should be carefully monitored but is expected to follow normal procedures. Samples will be taken from the bags via representative "spearing" when departing the mine area, and this sampling process can be automated if required.

Note that final settlement results will be determined from samples taken at the receiving smelter, at which the Seller may be present and/or represented.

Assaying, exchange of assay results, and the splitting limits for determining settlement results must be professionally managed. The use of bagged products should avoid unnecessary precious metals loss due to handling and transportation.

16.1.3 Metal Pricing

Gold, copper, and silver each contribute to the project revenue stream and so future price predictions are necessary for this Pre-feasibility Study. The metal price assumption outlined below for purposes of the economic analysis in this study differ from the metal prices used to establish the resource and reserve inventories which are cast at lower levels, see relevant sections. A conservative approach was adopted in outlining resource and reserve inventories. Commodity Price Forecasts use a combination of analysis of three-year rolling averages, long-term consensus pricing, and benchmarks to pricing used by industry peers over the past year, when considering long-term commodity price forecasts. Higher metal prices are used for the mineral resource estimates to ensure the mineral reserves are a subset of, and not constrained by, the mineral resources, in accordance with industry-accepted practice. The base-case metal prices used in the Project's economic evaluation within this Pre-Feasibility Study shown in Table 16.1**.**

Table 16. 1: Pre-Feasibility Study Base Case Metal Prices

Metal	Unit	Base Case Price for PFS
Gold	$US/oz	2100
Copper	$US/lb	4.10
Silver	$US/oz	27

16.1.4 Accountable Metals

The flotation concentrate shipped from the Project will contain accountable copper, gold, and silver levels.

16.1.5 Smelting and Refining Charges

The smelting and refining terms used within the Prefeasibility Study economic models are consistent with current market trends. No forward-looking adjustments are made to these terms in later years.

Discussions with several concentrate offtake companies have progressed and two indicative term sheets have been received for Project concentrate. No definitive smelter agreements have been obtained for the concentrate, although it is apparent that it will not be difficult to market under normal market conditions. This is partly due to the higher gold grade in the copper concentrate and the lack of deleterious elements Metallurgical testwork results indicate that deleterious element penalties need not be applied in the terms for the concentrate.

Table 16.2 summarizes the indicative smelter terms utilized for PFS economic analysis received from a confidential 3rd party. These terms have been applied quarterly for the first three years and so account for short-term variability in payable metal grades as the mix of plant feed ore type changes. The determination of concentrate grade is discussed within Section 10 of this Report.

Table 16. 2: Smelting and Refining Terms – LOM Average

Term	Unit	Copper	Gold	Silver
Cu Minimum Deduction	%	1.2		
Au Minimum Deduction	g/dmt		0	-
Base Smelting Charge	$US/dmt	80	-	-
Cu Refining Charge	$US/lb payable	0.080	-	-
Payable Metal	%	96.5	97.5	90.0
Au/Ag Refining Charge	$US/oz	-	5.00	0.50
Concentrate Moisture	%	10	-	

Transportation costs of $187.00 per wet short ton have been assumed, based on estimated north American destination.

16.2 MINING CONTRACT

A mining contract will be negotiated for the pre-production mining of approximately 1.200,000 st of waste in Year -1 Q1. Waste material mined as part of these operations will be used in project construction activities. At the end of Year -1 Q1 this mining contract will be closed.

It is contemplated that one of a number of contractors will be selected to conduct topsoil stripping, ground preparation and pre-production mining to satisfy initial construction needs. Run-of-mine rock will be crushed and screened to provide various crushed product sizes to serve as aggregate, material for drainage infrastructure, top-dressing for roads and around the site, and over-liner material.

16.3 OTHER CONTRACTS

Besides power supply, negotiation will be held for major consumer item supplies encompassing fuels oils and grease, reagent supply. Proximal to the project is a major prill manufacturer for ANFO explosives and contractors for the downhole supply of explosives for blasting on site. Additionally, contracts for several non-core activities such as employee bussing, security and waste disposal will be established. Where possible contract services for administrative functions will be sought in nearby Cheyenne.

17.0 ENVIRONMENTAL, SOCIAL, AND PERMITTING

This chapter summarizes the status of environmental compliance, permitting and community engagement, including the following specific topics:

- Results of environmental studies (Section 17.1): Environmental studies began in October 2020 to establish the pre-mining site conditions and fulfill the information requirements for Project permitting. The scope and results of these studies, which include environmental baseline characterization, groundwater and seepage modeling, and geochemical characterization of tailings and mine rock, are summarized herein.

- Requirements and plans for waste and tailings disposal, site monitoring, and water management during operations and after mine closure (Section 17.2): Based on the Project mine plan and results of the environmental studies, specific requirements, and plans have been identified and summarized for management of waste rock and tailings, site monitoring and water management, to avoid or mitigate environmental impacts throughout the Project life cycle.

- Project permitting requirements, the status of permit applications, and requirements to post a reclamation bond (Section 17.3): Permitting is primarily at the state and local level; no major federal permits are required. The principal state permits have been obtained and are described herein. Additional required state and local level permitting is also identified. Bonding is in place for the reclamation of areas to be disturbed during the first year of construction and mining operations, and additional reclamation bonding will be required annually for subsequent operations.

- Plans, negotiations, and agreements with local individuals and groups (Section 17.4): Other than permitting, various agreements with local stakeholders needed for the construction and operation of the Project are described.

- Mine closure plan, including remediation and reclamation, and associated costs (Section 17.5): The state has approved a reclamation plan covering the full extent of the project, which is summarized herein. The state also developed a reclamation cost estimate, which it accepted as part of the reclamation bonding process.

- The qualified person's opinion on the adequacy of current plans to address issues related to environmental compliance, permitting, and local individuals or groups (Section 17.6).

- Commitment to local procurement and hiring (Section 17.7).

17.1 ENVIRONMENTAL STUDIES

17.1.1 Baseline Characterization

Baseline characterization studies began in October 2020 to establish the pre-mining site conditions and fulfill the information requirements for Project permitting. The baseline studies have been concluded and the associated reports submitted to the state as part of the various permit applications required by the Wyoming Department of Environmental Quality (Section 17.3).

17.1.1.1 Land Use

The Project site is located on land owned by the State of Wyoming (Section 36) and the Ferguson Ranch (south half of Section 25 and Section 31), as shown on Figure 17.1. In Section 36, the surface and minerals are owned by the State of Wyoming and the surface is leased for grazing to the Ferguson Ranch, Inc. The Project site has been used as rangeland for cattle grazing and mineral exploration.

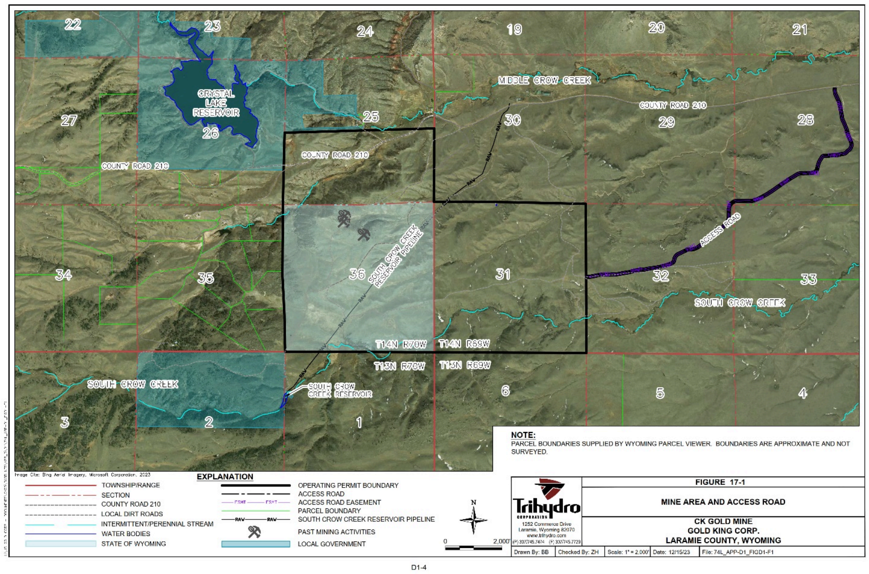
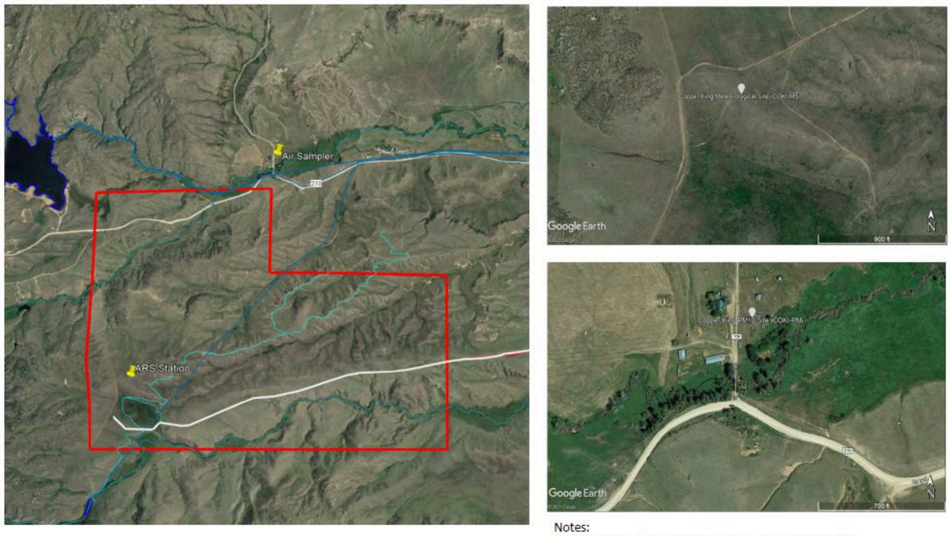


Figure 17.1: Project Site and Access Road Location

Past mining activity occurred on the site and in the surrounding historical Silver Crown Mining District since the district was established in 1879, including prospecting, exploration drilling, surface mining, and expansive underground excavation. The Project is centrally located in, and the focus of, past exploration and mining activities associated with the historic Copper King Mine. The mine is considered one of the top five gold deposits in the state of Wyoming (Hausel 2019). The deposit was first discovered by James Adams of the Adams Copper Mining and Reduction Company in 1881. The deposit was primarily developed as an underground copper mine. But despite several mining campaigns spanning several generations and transfers of ownership, much of the deposit is still intact. At least 13 exploratory drilling programs with over 173 drillholes have been developed on the site since 1930 for metallurgical, technical, hydrological, and resource expansion purposes.

17.1.1.2 Climate

The CK Gold Project has operated a weather station on the Project site since November 2020. Figure 17.2 shows the location of the Project weather station. Additionally, more than 20 weather stations are located between Laramie and Cheyenne and provide temperature, precipitation, wind speed, and wind direction measurements.



Notes:
"ARS Station" is the Copper King Meteorological Site (COKI-MS)
"Air Sampler" is the Copper King PM10 Site (COKI-PM)
The red line delineates the Project site boundary

Figure 17.2: Locations of the Meteorological Station & PM10 Monitoring Station (from Air Resource Specialists)

Based on data compiled from the site weather station and other surrounding stations (the latter over at least a ten-year period), the daily average temperature ranges from about 25° F in February to about 70° F in July. The average low temperature is -11° F in February and the average high is 90° F in July.

The Project site is in a net water deficit. The average annual precipitation is about 17 inches, while the annual evaporation is about 53 inches, as determined by the on-site meteorological station. May is the wettest month, with an average of about 3 inches; January is the driest, with an average of about 0.6 inches. Snowfall typically occurs from September to May.

The site experiences relatively strong winds, with an average monthly wind speed ranging from about 8 mph in July to about 17 mph in December. For those same months, the average maximum wind speeds are 43 and 63 mph, respectively, with peak wind speeds of 55 and 75 mph (86 mph for January). The predominant wind direction is westerly.

17.1.1.3 Air Quality

The Project has monitored baseline air quality since November 2020 to collect ambient air quality data and establish the pre-mining air quality. The air quality monitoring station is located approximately 0.2 miles north of the Project site on the Ferguson Ranch along County Road 210, as shown on Figure 17.2. The location was selected in general accordance with 40 CFR Part 58 Ambient Air Quality Surveillance. The station collects integrated particulate matter data sized less than 10 microns (PM10) once every six days over 24 hours using two collocated BGI PQ200 particulate air samplers. The samplers collect integrated 24-hour samples in accordance with EPA protocols (Quality Assurance Guidance Document 2.11, Reference Method for the Determination of Particulate Matter as PM10 in the Atmosphere).

To date, the background air quality has met the National Ambient Air Quality Standard 24-hour PM10 level of 150 micrograms per cubic meter ($\mu g/m^3$), with PM10 measurements ranging from 0 to 45 $\mu g/m^3$.

17.1.1.4 Surface Water and Wetlands

An Aquatic Resources Inventory (ARI) was performed in September 2020 (Trihydro 2020) to identify jurisdictional Waters of the United States in and around the CK Gold Project site. The United States Army Corps of Engineers (USACE) regulates jurisdictional Waters of the US, which are defined and regulated by Section 404 of the Clean Water Act (CWA) 33 CFR Part 328.3 and Section 10 of the Rivers and Harbors Act (RHA) 33 USC 1344, including streams and wetlands. The jurisdictional waters and wetlands were identified to facilitate Project infrastructure planning to prevent impacts on the Waters of the US.

The surface water features investigated under the ARI are shown on Figure 17.3. They include the intermittent South Crow Creek, the ephemeral South and Middle tributaries of Middle Crow Creek, and the perennial/intermittent North tributary of Middle Crow Creek. Based on the findings of the ARI, on 5 February 2021 the USACE issued an Approved Jurisdictional Determination (AJD) for the drainages and wetlands within the CK Gold Project area. The AJD is the official determination from the USACE on the Waters of the US that are present in the Project area. The jurisdictional Waters of the US identified in the AJD include South Crow Creek and the North tributary to Middle Crow Creek. The AJD concluded that the drainages and wetlands associated with the South and Middle tributaries to Middle Crow Creek are not jurisdictional Waters of the US. The Project mine facilities have been designed to avoid and will not impact jurisdictional Waters of the US.

In November 2023, Western EcoSystems Technology (WEST) prepared an additional ARI report (WEST 2023a) for the proposed Project access road and vicinity. This ARI identified one dry drainage with a defined channel that may be jurisdictional. The access road will not cross the drainage, and mine activities will not affect it.

A surface water baseline monitoring program was initiated in October 2020 and completed in April 2022. The program included a collection of surface water quality samples, field water quality parameters, and stream flow measurements monthly at up to six monitoring locations within the Project site, as shown in Figure 17.3. The monitoring locations are located along the primary surface water features within the Project and include the intermittent South Crow Creek, the South and North tributaries to Middle Crow Creek, and one spring in the South tributary of Middle Crow Creek.

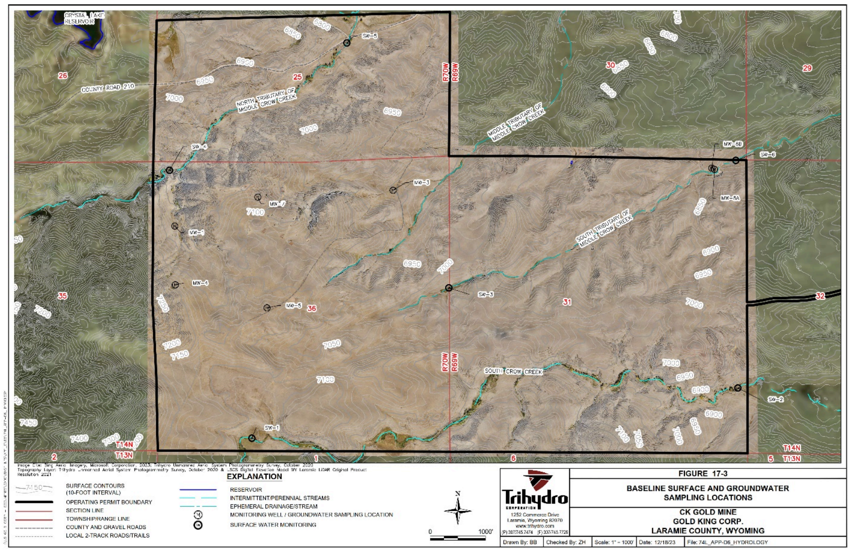


Figure 17.3: Surface and Groundwater Sampling Locations

Surface water flow in the drainages is typically derived from snowmelt runoff, rainfall-runoff following precipitation events, and contributions from groundwater (springs). The six monitoring points were established at the upgradient and downgradient locations along the primary drainages as they cross the Project boundary in general accordance with the Wyoming Department of Environmental Quality - Land Quality Division (DEQ-LQD) Guideline 8 baseline hydrology recommendations.

Surface water flow has been observed to be inconsistent due to the intermittent and ephemeral nature of the drainages. Measured flow rates at the surface water monitoring points between October 2020 and April 2022 ranged from zero to approximately six cubic feet per second, with the highest flow rates observed from April to June. Three of the monitoring points, including two points within the ephemeral South tributary of Middle Crow Creek (site of the proposed TMF) and one point in the north tributary of Middle Crow Creek, had no observed flow during the entire 19-month monitoring program.

Surface water samples have been analyzed for the recommended constituents in DEQ-LQD's Guideline 8, including additional trace metals. The baseline surface water quality is relatively good and meets the Wyoming DEQ - Water Quality Division's (DEQ-WQD's) Surface Water Quality Standards for livestock, irrigation water, and drinking water.

17.1.1.5 Groundwater

Groundwater monitoring at the Project site began in 2020 to characterize the potentiometric surface, groundwater flow, and groundwater quality (Neirbo Hydrogeology 2023). Data has been collected over a period of approximately 18 months using monitoring wells, standpipe wells, vibrating wire piezometers (VWP), HQ core holes, and reverse-circulation boreholes. This data formed the basis for development of a groundwater flow model, as described in Section 17.1.2.

Quarterly groundwater monitoring has been performed at seven monitoring wells within the Project site (MW-1, MW-3, MW-4, MW-5 MW-7, MW-8a, MW-8b), as shown on Figure 17.3. Groundwater sampling started in the fourth quarter of 2020. Results from six quarterly sampling events were included in the Mine Operating Permit application submitted to the DEQ-LQD in January 2024.

Results indicate that the groundwater is generally a bicarbonate type. Wells MW-7 and MW-8a are drilled in granodiorite and alluvium, respectively, and have calcium-bicarbonate type water, whereas MW-1, 3, 4, and 5, drilled in granodiorite and metasediments, have sodium bicarbonate water. Monitoring well MW-8b has a mixed calcium-sodium bicarbonate water and is the only well screened in the White River Formation (Neirbo Hydrogeology 2023).

Water quality has mostly met standards, although some measurements have been above DEQ limits, variably between domestic, agricultural, and livestock standards. Table 17.1 summarizes the baseline groundwater quality that exceeds DEQ standards in each monitoring well for each quarter, as reported by Neirbo Hydrogeology (2023).

Table 17.1: Baseline Monitoring Wells with Constituent Concentrations Exceeding Water Quality Standards

Constituent	2020 Quarter 4	2021 Quarter 1	2021 Quarter 2	2021 Quarter 3	2021 Quarter 4	2022 Quarter 1
Fluoride	1, 3, 4, 5	1, 3, 4, 5	1, 3, 4, 5	1, 3, 4, 5	1, 3, 4, 5	1, 3, 4, 5
pH	1, 3, 4, 5	1, 3, 4, 5	1, 3, 4	1, 3, 4, 5	1, 3, 4, 5	1, 3, 4
Dissolved Iron	3, 5	5	--	--	7	--
Total Iron	1, 3, 5, 7	3, 5	3, 4, 5	--	5, 7	4, 7
Mercury	--	7	7	--	--	--
Manganese	3, 7, 8a, 8b	7, 8a, 8b	7, 8a, 8b	7, 8a, 8b	7, 8b	4, 7
Sodium Adsorption Ratio (SAR)	1, 4	4	4	1, 4	4	4
Dissolved Uranium	7	7	7	7	7	7
Total Uranium	*	*	*	*	7	7
Gross Alpha	7	3, 7	7	3, 7	7	3, 7
Adjusted Gross Alpha	--	3	7	--	7	7

From Neirbo Hydrogeology 2023
Notes:
Well names are preceded by "MW-"
Wells listed for each constituent exceeded at least one of the DEQ water-quality standards for Class 1 Domestic, Class 2 Agriculture, or Class 3 Livestock uses
-- No wells exceeded standards
* Not measured

17.1.1.6 Soils

The Project site is located on the eastern flank of the Laramie Range between the Rocky Mountains and High Plains sections of the Great Plains physiographic province. The Laramie Range is an approximately 130-mile-long mountain range between Laramie and Cheyenne, WY, that trends north from the Colorado-Wyoming border towards Casper, WY. The Laramie Range consists of granite/granodiorite peaks and rolling hills bound to the east non-conformably by shallow eastward dipping sedimentary rocks of the White River Formation. East of the CK Gold Project area, towards Cheyenne, WY, the topography transitions to flatter plains along the western margin of the Great Plains physiographic province. The Project site geology is further described in Section 6.

The Natural Resource Conservation Service (NRCS) database of mapped soil units was reviewed. The nine soil units described by the NRCS soil database at the Project site were identified and field verified in July 2021. Preselected sample locations and respective field survey soil profile descriptions were used to confirm or modify the coverage of the nine soil map units. For soil map units that were modified, the acreages were revised (Figure 17.4).

A test pitting subsurface exploration program was implemented around the same time to evaluate the soils in the proposed development areas (Trihydro 2022). The ore body is exposed at the hilltop and is generally surrounded by granite. Weathered soil is located around the base of the slopes. The north and western faces of the hill are the steepest portions of the Project site and have the least amount of soil cover. The northeast and southern saddle areas have gentler slopes and generally contain more soil.

Topsoil was generally encountered throughout the Project site at the ground surface with localized areas of outcropping bedrock. The topsoil consists of brown to dark brown silt with trace sand and gravel and decomposing organic matter. The topsoil typically ranges from approximately 0.25 to 4.25 feet in thickness with an average thickness of 1.1 feet. Generally, topsoil was found to be thickest in the drainages and valley bottoms and thinner along slopes and ridges.

Subsoils are typically aeolian or colluvial soils or were derived from the lightly cemented White River Formation, which is composed primarily of lightly cemented alternating layers of siltstone, sandstone, and claystone.

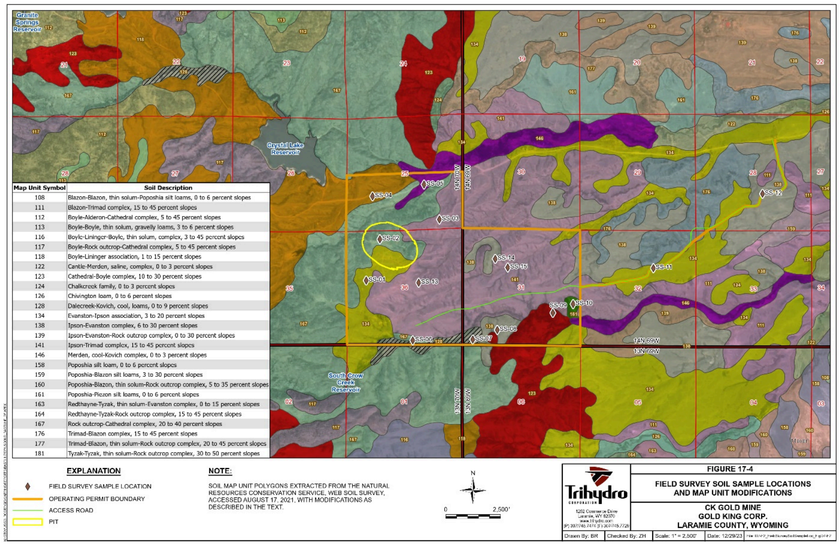


Figure 17.4: Field Survey Soil Sample Locations and Map Unit Modifications

Silty soil is common, as the White River Formation has a primarily silty matrix. Silty soil tends to be low plasticity and lie above massive siltstone beds. The silt is predominantly dark brown and either dry or slightly moist and contains sand. Silts observed in test pits were predominantly under 5 feet thick, with some reaching up 10 feet thick. Silty sand layers were also encountered and generally found overlying sandstone beds. They tend to be olive brown in color, lean, dry to slightly moist and up to a few feet thick.

The clayey soil encountered is primarily lean clay with brown to gray color and tends to have noticeable sand content. The lean clays, as classified by the Unified Soil Classification System, are primarily associated with the B soil horizon where fine grained particles migrate down from the topsoil into the subsoil regions and create the silty clay layer. Lean clays can be found primarily in the Mill Area, the Ore Facility, and the TMF. Fat clays were encountered primarily to the southeast of the Mill Area and portions of the TMF. The fat clays also have noticeable sand and gravel content. Fat clays are likely to have a shrink or swell potential in response to moisture changes; they shrink as the soil dries and swell as more water is added.

Loose sand and gravel are commonly found overlying sandstone or claystone with significant sand or gravel content. These loose soils are typically light gray or brown with significant silt content. Gradation ranges from poor to well graded.

17.1.1.7 Vegetation

The Project area consists primarily of rolling grassland/herbaceous habitat with forested and shrub/scrub-covered drainages. Most of the Project site consists of prairie grasslands, with some areas of xeric forest and sparse areas of foothill, sagebrush shrublands, and riparian vegetation. Habitat to be disturbed by Project development consists almost entirely of the grassland/herbaceous type.

Trihydro performed a desktop review of national and state vegetation databases in 2021 as part of the Mine Operating Permit application to identify vegetation types in the Project area and potential special status plant species. Figure 17.5 shows the different vegetation types at the Project site according to the US Geological Survey's National Land Cover Database.

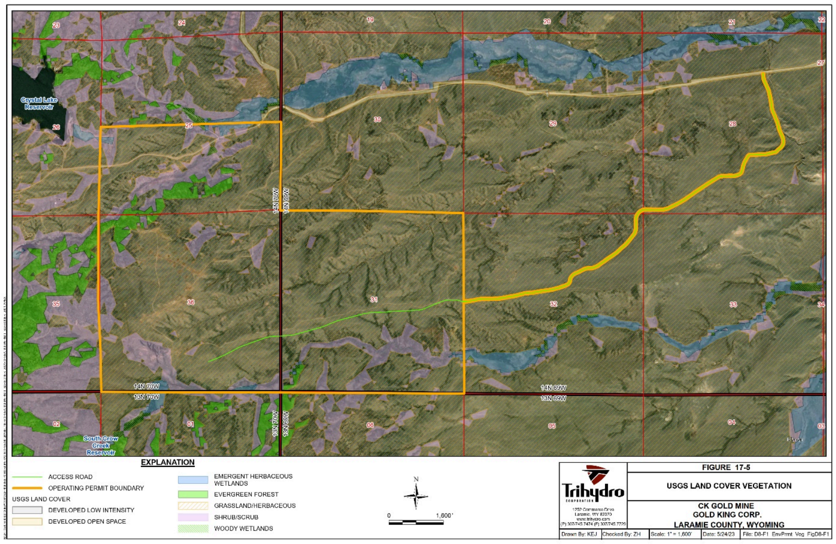


Figure 17.5: USGS Land Cover Vegetation

Based on field surveys conducted in July 2021 by Trihydro and June 2023 (WEST 2023b), it was concluded that the Project site does not contain suitable habitat associated with special status plant species, and no such species were observed. The most common native species identified during the field survey were in the grassland/herbaceous habitat and include needle and thread (Hesperostipa comata), western wheatgrass (Pascopyrum smithii), blue grama (Bouteloua gracilis), prairie junegrass (Koeleria macrantha), and Sandberg bluegrass (Poa secunda). Notably, cheatgrass (Bromus tectorum), a non-native, aggressively invasive weed species in Laramie County, was the sixth-most common species found.

17.1.1.8 Wildlife

A desktop study reviewed national and state data sources to determine the potential for listed wildlife species within the Project site. The US Bureau of Land Management (BLM) Wyoming Sensitive Species List includes 16 species potentially occurring at the Project site, including four mammals, 11 birds, and one amphibian. The US Fish and Wildlife Service (USFWS) Planning and Conservation website further identified four federally listed species potentially present at the site, including the Preble's meadow jumping mouse (Zapus hudsonius preblei), piping plover (Charadrius melodus), whooping crane (Grus Americana), and pallid sturgeon (Scaphirhynchus albus). No critical habitat was identified within the Project site. However, a portion of the Project site falls within the pronghorn antelope (Antilocapra americana) crucial winter range, and the whole Project site and surrounding area is within the mule deer (Odocoileus hemionus) crucial winter range. In consultation with the WGFD, mitigation action will be taken for the disturbance of mule deer crucial winter range during Project construction and mining operations, including minimization of vehicular traffic by worker busing, installation of wildlife-friendly fencing, and a $300,000 payment to the WGFD.

A field wildlife survey was conducted in and around the Project site by Trihydro in June 2021 as part of the Mine Operating Permit application, focused primarily on the BLM sensitive species and the federally listed species. WEST conducted additional field surveys from May to July 2023 (WEST 2023c, d, e), focused on raptors, fish, and species designated by WGFD as Species of Greatest Conservation Need (SGCN), including upland sandpiper (Bartramia longicauda), swift fox (Vulpes velox), smooth greensnake (Opheodrys vernalis), western tiger salamander (Ambystoma mavortium), and northern leopard frog (Lithobates pipiens).

Two BLM sensitive bird species were observed: the northern goshawk and the Brewer's sparrow. The Project site was determined to contain potentially suitable habitat only for one of the four USFWS federally listed species, the Preble's meadow jumping mouse, although this species was not found and its associated potential habitat along the creeks is degraded from cattle grazing.

No raptor nests were found within planned areas of Project disturbance, and no golden eagle nests were observed within the Project site. None of the SGCN species were seen or heard on the Project site, though the site is within the upland sandpiper and swift fox predicted distribution areas. Project development will avoid the flowing streams that offer potential habitats for amphibians and fish. Almost all the wildlife field observations occurred in the riparian corridors along South Crow Creek and the North tributary of Middle Crow Creek, both outside of the planned Project disturbance areas.

In a concurrence letter for the Mine Operating Permit, WGFD recommended ongoing consultation with the agency regarding raptors and monitoring for swift fox (Vulpes velox) before disturbing the ground within the Project area between April 1 and September 30 each year.

17.1.1.9 Archeology and Paleontology

A Class I cultural resource data review was completed in June 2021 (Western Archaeological Services 2021). The review examined the State Historic Preservation Office (SHPO) records for documented cultural resources within the Project boundary. Two sites were identified near or within the Project boundary: the Fort D. A. Russell to Fort Sanders Wagon Road, which is eligible for nomination to the National Register of Historic Places (NRHP); and the historic Copper King Mine, which is ineligible for nomination to the NRHP.

The wagon road passes north of County Road 210 in the northeast portion of the south half of Section 25 within the Project site boundary. It is a previously documented cultural site and is eligible for nomination to the NRHP with SHPO concurrence. No Project activity is proposed north of County Road 210. Therefore, this site will not be disturbed by the Project.

The historic Copper King Mine, located within the Project site, had two mine shafts, three adits, nine exploratory pits, and an excavation. The Class I data review found that the Copper King Mine is not eligible for NRHP nomination. The DEQ reclaimed the mine features - Abandoned Mine Lands Division (AML) in July 2017. Before the reclamation, the DEQ-AML performed a National Environmental Policy Act (NEPA) determination and verified that the reclamation conformed with exclusion criteria and was exempted from further NEPA compliance.

A Class III cultural resources field survey was conducted on the Project site in September 2024 (Centennial 2024) to identify potential additional cultural sites. No identified sites were recommended for National Register of Historic Places classification. Management measures will be implemented to protect additional cultural sites during Project construction, mining, and reclamation operations.

Most of the construction and mining-related excavation will take place within the Pre-Cambrian age granite formation, an igneous intrusive rock that does not contain fossils. According to the USGS, some activity will occur in the sedimentary White River formation, which could host paleontological resources but is considered unlikely to contain preserved fossils (Bartos et al. 2014). Project activities will be subject to "chance finds" protocol, requiring notification of state agencies in the event of a cultural or paleontological find and a work stoppage at the affected location.

The Project is not located adjacent to indigenous, Native American, or Bureau of Indian Affairs lands.

17.1.2 Groundwater Modeling

The orebody is hosted in Precambrian granitic rock with limited permeability and water-storage capacity. Groundwater wells completed in the granite typically yield 0 to 5 gallons per minute (gpm). The granite groundwater flows from the higher elevation areas of the Laramie Range, west of the project area, to the east. The White River formation is underlain by Cretaceous formations east of the mine. Figure 17.6 shows the hydrogeologic units, groundwater level and flow direction.

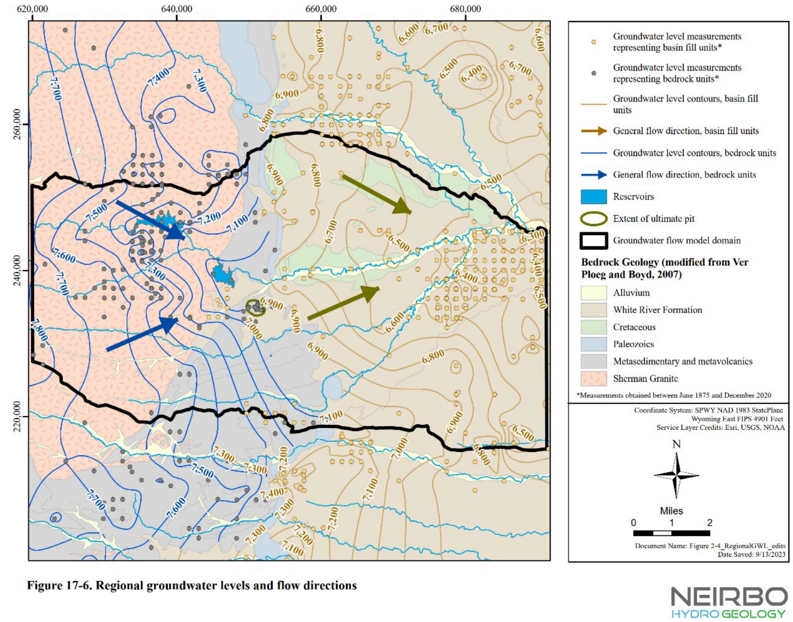


Figure 17.6: Hydrogeologic Units, Groundwater Level, and Flow Direction

The Project has completed extensive hydrogeologic site characterization to support the development of a regional groundwater flow model. Aquifer testing has included pumping tests and discrete depth-interval packer testing. These tests estimated hydraulic conductivity and specific storage properties. Groundwater levels and pore pressures were obtained from wells and vibrating wire piezometers.

Neirbo Hydrogeology (2023) developed a calibrated groundwater flow model to represent the hydrogeologic system and assess the interactions between the proposed mine and the groundwater system. The model incorporates hydrogeologic features, including streams, reservoirs, irrigated land, and wells in the project area, as well as aquifers, faults, stream-aquifer interactions, recharge, evapotranspiration, and external boundary conditions.

The model simulates pre-mining conditions and hydrologic changes during the mining and post-mining phases. The model predicts groundwater system changes due to passive pit dewatering, natural recharge changes due to facility construction, and pit backfill during the post-mining phase.

Model predictions during the mining and post-mining periods include groundwater level, pit inflow, streamflow, and evapotranspiration changes. The predicted mine-induced drawdown is greatest near the pit and decreases rapidly away from the pit (Figure 17.7). Predicted drawdown is generally 5-feet or less outside the Project site at the end of mining. After 150 years the discernable predicted drawdown is at its maximum, extending about 180 feet outside the Project site boundary (Figure 17.8). The nearest domestic wells are 2,000 feet from the predicted 5-feet drawdown area. At this distance, mine induced drawdown would likely not be discernable from natural variation and groundwater level changes induced by the domestic wells themselves.

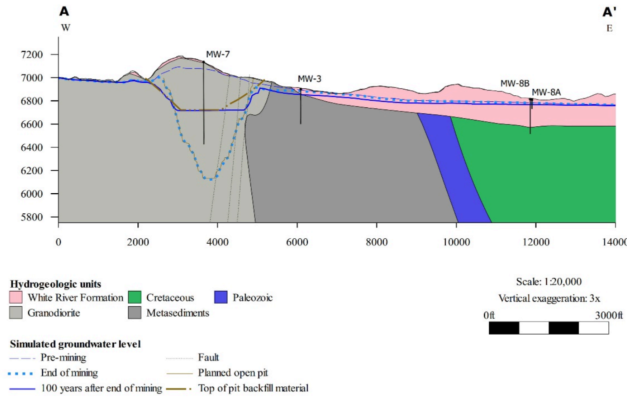


Figure 17-7. Predicted groundwater levels West to East Across the Pit

Figure 17.7: Cross-section of groundwater levels

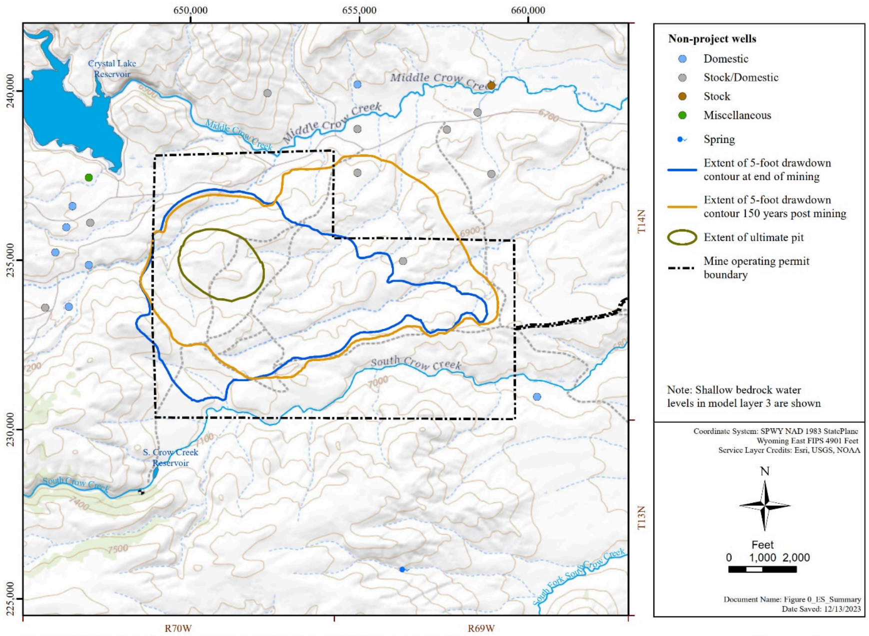


Figure 17-8. Predicted drawdown at end of mining and 150 years post mining

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Figure 17.8: Predicted drawdown at the end of mining and 150 years post-mining

The Middle Crow Creek is the nearest stream, and its flow is predicted to decrease by 0.03 cubic feet per second 10 years after mining. The other stream segments have zero to 0.02 cubic feet per second changes in flow.

The average annual groundwater pit inflow is expected to be less than 15 gallons per minute. This low pit inflow would be manageable using passive, in-pit sumps, and dewatering wells are not expected to be necessary.

After mining, the pit will be backfilled with tailings and waste rock. Groundwater and precipitation will flow into the backfill material, and water levels will slowly rise until they stabilize at 6,717 feet after about 130 years. A pit lake is not expected to form since evaporation losses will keep the groundwater level below the top of the backfill. This will result in the pit being a hydraulic sink with no groundwater outflows.

The groundwater modeling conducted to date precedes the recent development of Project water supply wells in the vicinity of the Project site approximately 1.25-miles northwest of the pit (see Figure 17.13). The Project supply wells will extract groundwater from the Casper Formation, which underlies the formations previously investigated and modeled. The new wells are not expected to induce significant drawdown in the overlying units hosting the neighboring domestic water supply wells; however, this is pending confirmation through additional hydrogeologic assessment.

17.1.3 Tailings Seepage and Stability Analysis

Tailings stability was analyzed by Tierra Group (2025b). The tailings were modeled overlying the Tailings Management Facility's (TMF's) composite liner system (CLS), which in turn overlies a prepared foundation consisting of native soils that are underlain by weathered bedrock.

17.1.3.1 Seepage

The DEQ-LQD review of the MOP application required a rework of the liner system, and the Project will now use a CLS. The CLS will consist of a geomembrane overlying a prepared subgrade composed of compacted Project area clays and silts. As required by WQD R&R, the CLS will have an effective permeability of 10-7 cm/s or lower. The inclusion of the CLS means that tailings seepage modeling was not required by WDEQ-LQD for the TMF.

17.1.3.2 Stability

Limit equilibrium stability analyses were performed on the TMF for static (long-term) conditions, seismic loading conditions using pseudo-static method, and post-peak (post-liquefaction) conditions. The slope stability models assumed a phreatic surface at the interface between the upper and lower foundation soils (approximately 20 feet below the ground surface). The model also assumes a phreatic surface along the CLS and tailings interface, as a phreatic surface is not likely to develop within the tailings mass. Slope stability analyses were completed for the downstream and side buttress sections. Slope stability for the downstream section was modeled as the TMF advanced construction to its full height in Year 3. The side buttress section was selected at the greatest embankment height, where the starter berm had not been constructed.

The requisite factors of safety are met for the stability analyses completed for the two sections when the ultimate waste rock retention shell is constructed. Additional analyses were completed to analyze the TMF during construction and allow for operating flexibility. The TMF stability results are detailed in the TMF Stability Analyses Technical Memo (Tierra Group, 2025b).

17.1.4 Geochemical Characterization of Mine Rock and Tailings

Geochemical Solutions (2023) evaluated the potential to generate acid rock drainage (ARD) and metal leaching from the mine rock and tailings storage. Fifty-six representative rock samples and four tailings samples were collected for geochemical characterization. The 56 rock samples represent in-place mine rock at the projected surface of the proposed pit shell and the rock to be mined. The rock samples are distributed widely both horizontally and vertically across the proposed pit and surrounding the ore body, as shown on Figure 17.9.

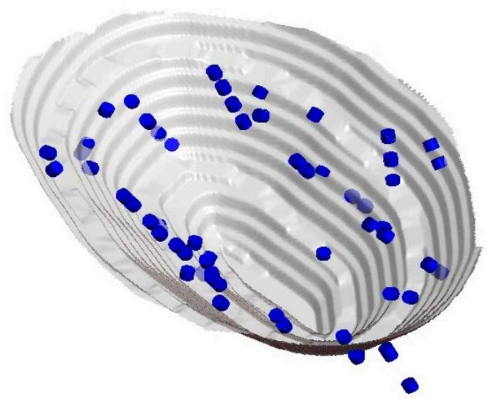

Figure 17.9: Mine rock sample spatial distribution (from Geochemical Solutions 2023)

The four tailings samples were derived from bench-scale metallurgical (locked cycle) testing of representative ore samples. Bench scale process water samples from the locked cycle testing were also submitted to an analytical laboratory for analysis.

Geochemical analyses included:

- *Whole rock characterization* assesses the bulk geochemical composition of the waste rock, tailings, and low-grade ore materials.
- *Acid-base accounting (ABA) determines the balance of acid-generating sulfide minerals and acid-neutralizing* minerals in the samples.
- *Net acid generation (NAG)*: This method uses hydrogen peroxide to oxidize the exposed sulfide minerals in the samples. The oxidation provides a high-end estimate of the acidity that may be produced through oxidative weathering of any exposed materials. It also allows the identification of potential elemental release through oxidative weathering of mine materials.
- *Meteoric Water Mobility Procedure (MWMP):* a single-pass column leach test used for non-acid generating mine rock to assess the chemical quality of contact water.
- *Humidity cell testing (HCT) is a multi-week column weathering test that provides the test sample with excess water and oxygen to facilitate rapid oxidation of sulfide minerals. Weekly column rinses are analyzed for various parameters (such as pH, alkalinity, iron, and sulfate),* and a monthly rinse sample is analyzed for a range of regulated metals and metalloids.

Geochemical Solutions (2023) also evaluated the mineralogy and petrography of mine rock to better understand the controls on acid-generation potential (AP) and neutralization potential (NP). Mineralogic analyses included:

- *Quantitative mineralogy by x-ray diffraction (XRD).*
- *Optical microscopic examination.*
- *Scanning electron microscopy (SEM), using backscattered electron imaging.*

The ABA and NAG tests are considered static test procedures, while the subsequent MWMP and HCT tests are considered kinetic tests. Water samples are obtained weekly from the testing apparatus to evaluate whether leaching is occurring and when it may be expected to start. The HCTs were conducted over a 108-week period. Figure 17.10 summarizes the ABA results, and Figure 17.11 summarizes the HCT results.

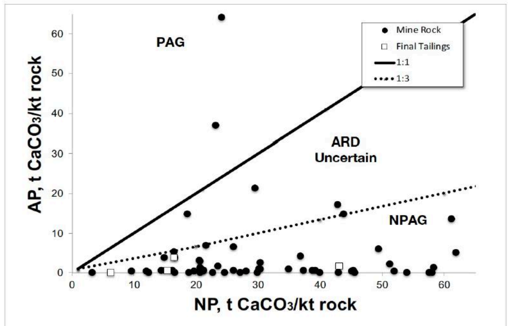

Figure 17.10: Results of ABA Tests (from Geochemical Solutions 2023)

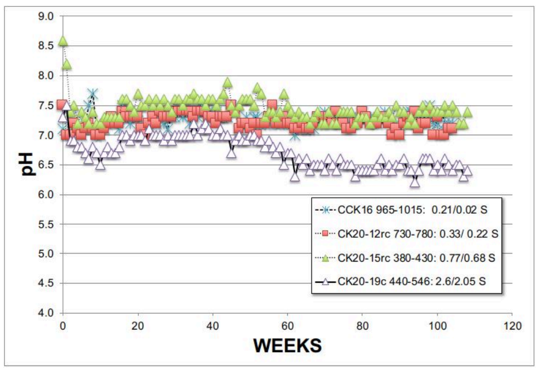

Figure 17.11: Results of Humidity Cell Tests (from Geochemical Solutions 2023)

Using industry-standard methods, the characterization of the geochemical properties of Project mine rock and representative tailings indicates the limited probability of the rocks and tailings producing ARD in contact water. ABA and NAG static testing results indicated the presence of potentially acid producing mine rock and release of metals in 5 of the 56 samples, two located approximately halfway up the west side of the projected pit surface and three with excavated waste rock. Some higher sulfur-containing samples indicate the limited and local presence of PAG mine rock. However, little mine rock is mapped as having elevated sulfur and increased ARD potential. Most of mine rock is characterized as NPAG, with an overall median Net Neutralization Potential (NNP) of 24.5 short tons of CaCO3 per 1,000 tons of rock (tCaCO3/1,000t rock) and Neutralization Potential Ratio (NPR) of 33.3; rock with either NNP greater than 20 tCaCO3/1,000t rock or NPR greater than three is considered NPAG. The median NAG pH was 6.2; samples with NAG pH greater than 4.5 were classified as NPAG. Results from the NAG metal analysis showed that arsenic, cadmium, copper, lead, and zinc were observed in five samples. However, HCT and MWMP results show no low pH (acidic) water or metal release production, which resulted in NPAG classification, regardless of sulfur content.

The mineralogy of the mine rock affects the potential for the formation of ARD. Sulfide minerals appear primarily as small pyrite and chalcopyrite, with trace percentages of other sulfides disseminated in the silicate matrix. Silicate minerals provide the bulk of NP. Based on the extended HCT results, it appears that the rate of NP from silicate minerals is able to keep pace with the limited rate of acid production.

MWMP leach testing of NPAG demonstrates low to no leaching of dissolved regulated metals. Leaching of total iron and manganese was observed to produce concentrations that exceeded domestic use criteria but were consistent with ambient background groundwater concentrations. In one instance domestic use criteria was exceeded for dissolved arsenic. One sample exceeded the agricultural use criteria for total iron. MWMP results for representative tailings samples indicated that leached water from tailings were consistently below domestic use criteria. HCT testing of rock samples, which spanned the range of sulfur concentrations results from the ABA data, resulted in neutral to slightly alkaline pH conditions for up to 108 weeks of testing with metal release observed to be negligible and low sulfate release rates. The four metallurgical testing tailings samples analyzed contain limited sulfide sulfur; therefore, the representative tailings produced NPAG results.

Four samples representative of process water were submitted for analysis. Arsenic concentrations in these samples routinely exceeded domestic and agricultural use criteria, but not livestock use criteria. The remaining constituents were below regulatory criteria.

17.2 REQUIREMENTS AND PLANS FOR WASTE AND TAILINGS DISPOSAL, SITE MONITORING, AND WATER MANAGEMENT

This section is divided into three subsections as follows:

- Waste Rock and Tailings Management (Section 17.2.1)
- Site Monitoring (Section 17.2.2)
- Water Management (Section 17.2.3)

This section summarizes design and operational requirements during construction, mining, mineral processing, closure, and post-closure.

17.2.1 Waste Rock and Tailings Management

Waste rock and tailings generated during mining and mineral processing will be deposited in engineered facilities on the Project site.

17.2.1.1 Waste Rock

The waste rock consists of excavated overburden and rejected material from the pit containing insufficient concentrations of copper or gold for economic mineral processing. Waste rock will have various on-site uses/destinations, including construction and capping of haul roads and erosion control features, deposition in the West and East Waste Rock facilities (WWRF and EWRF, respectively), and use as the TMF’s outer retention shell and buttress. The waste rock facility design and construction are described in Section 15. This section focuses on the associated environmental management controls.

The following environmental management controls will be incorporated:

- *Stability:* The WWRF and EWRF are designed to have a slope angle of 3H:1V, substantially flatter than the rock's angle of repose, inherently providing an acceptable safety factor for geotechnical stability. These facilities will be constructed using 20 to 30-foot-thick lifts. Construction will start from the lower ground surface elevations, moving upward and outward a lift at a time, stepping back such that the final angle of the entire facility is 3H:1V.

- *Water management and seepage control:* Each lift will have a running surface that drains precipitation away from the dumping fronts for stability and to minimize percolation. The driving surface will be compacted by the haul trucks. Runoff and seepage will be collected in detention ponds constructed at the downstream toe of the two waste rock facilities. Overflow spillways will be provided to prevent the overtopping of detention ponds during runoff events exceeding the design storm event (Section 17.2.3). The water in the detention ponds will be pumped out for use in dust control on-site or other production-related uses. Accumulated sediments will be periodically removed from the ponds and disposed of in the TMF.

- *ARD control:* Kinetic testing on waste rock resulted in non-potentially acid rock drainage (ARD)/metal leaching (Section 17.1.4). The Project will implement a Material Testing Program (MTP) to test blast hole cuttings to quantify Au, Cu, and Ag grades to differentiate between ore and waste rock. Additionally, the waste rock blast hole cuttings will be subjected to Net Acid Generation (NAG) pH testing to delineate non-potentially acid generating (NPAG) and potentially acid generating (PAG) waste rock polygons. Waste rock will be considered non-PAG (NPAG) if the NAG pH is greater than or equal to 4.5, per the Global Acid Rock Drainage (GARD) Guide (INAP 2023). PAG waste rock will be routed to either the CLS lined TMF or temporarily to the CLS lined Ore Stockpile. PAG waste rock that is placed within the Ore Stockpile will be relocated to the pit after Year 8 of operations. NPAG waste rock will be placed in the WWRF, EWRF, or the TMF rock buttresses or shell. A mine-bench scale 3-D database comprised of NAG pH grades and coordinates will be maintained and used for short term and life of mine planning. Results of the NAG pH analyses will be made available within 24 hours, transmitted electronically to the ore control engineer to delineate NPAG and PAG waste rock polygons. In the event of delayed assay results, the waste rock would either remain in the pit until assays are received or handled as PAG.

- *Reclamation:* The WWRF and EWRF will be reclaimed by topsoil covering and revegetation. The soil growth medium component of the cover will limit infiltration, promoting vegetation growth, runoff, and evapotranspiration. The soil growth medium layer thickness will be 12 inches. Geotechnical site investigations indicate there is sufficient material located on the Project site suitable for a soil cover that meets these requirements. The waste rock is expected to be suitable for a base for the soil cover. Some waste rock processing will be required to produce a transition zone between the rock and the soil growth medium cover material. Preliminary design of the transition zone indicates a minimum two-foot-thick layer of well graded (coefficient of uniformity greater than four) material with a maximum particle size of three inches.

17.2.1.2 Tailings

The tailings will be filtered to extract as much moisture as feasible prior to their deposition, maximizing their structural strength and geotechnical stability, thereby avoiding the need for a tailings dam and the associated stability and seepage risks. Filtered tailings also maximize the amount of water that can be recycled to mineral processing, reducing make-up water requirements and minimizing overall water consumption (Section 17.2.3). The processed tailings will be hauled to and placed in the TMF until Year 8.25. After that, the remaining tailings produced will be hauled to and placed in the open pit.

The following environmental management controls will be incorporated into the TMF operation and maintenance plan:

- *Stability:* Tailings filtration produces tailings near their optimum moisture content for compaction, maximizing their geotechnical strength and stability. Thus, the risk of slope failures and spills is significantly reduced. The filtered tailings will be co-deposited with waste rock. The waste rock retention shell will function as a buttress to stabilize the TMF. The TMF outer surfaces will be monitored for movement, and piezometric pore pressure will be monitored within the tailings mass for signs of potential decreased stability.

- *Erosion and sediment control:* Grading of the TMF will be controlled to maintain the active crest surface of the TMF with a gradient that slopes downhill to avoid pooling and infiltrating water into the placed tailings. The general design of the TMF includes zonation, such that a waste rock retention shell will be constructed concurrently with tailings placement. During wet conditions, placement of tailings will be in the interior of the TMF, away from the perimeter. Compaction will be performed as quickly as feasible following initial tailings deposition using a smooth roller compactor to seal the surface, prevent fugitive dust, and promote runoff.

- *Water management and seepage control:* Runoff and seepage from the TMF will be collected in detention ponds. Overflow spillways will be provided to prevent overtopping of detention ponds during runoff events exceeding the design storm event. A pond will be constructed upstream of the TMF to capture runoff from the watershed to the west of the TMF. Overflow from this pond will be conveyed through the TMF underdrain, overdrain, or both, depending on the stage of the project. The water in the detention ponds will be pumped out for use in the process plant and dust control on site. Accumulated sediments will be periodically removed from the ponds and disposed of in the TMF. Seepage control of the TMF is provided by the seepage collection drain installed above the TMF liner as discussed in Section 15.3.2. The drain will maintain a low hydraulic head in the bottom of the tailings mass, to promote free drainage of the tailings, and minimize tailings saturation.

- *Dust control:* To minimize fugitive dust emissions from the TMF, compaction of the top of the tailings surfaces will be performed as quickly as feasible following tailings deposition and spreading by dozers using a smooth roller compactor to seal the surface. The waste rock retention shells will be placed over the exposed tailings slopes once the final tailings slope and elevation have been achieved. Speed limits will be imposed and enforced throughout the Project site. Water will be sprayed on active surfaces to control fugitive dust emissions as required. Use of soil binders and tackifiers or other approved dust suppressants may be considered, depending on the effectiveness of the above measures. Erection of wind breaks may also be considered as a backup solution, if required.

- *Off-specification tailings management:* The process plant will use a batch filtration process for drying the tailings. Bench-scale testing has been performed on tailings samples to determine the type and size of filter press needed for the Mine to achieve the design moisture content criteria of at or below 14% metallurgical [2]. It is expected that commercial-scale tailings filtration equipment will generally meet the moisture content criteria. However, there may be variations in the ore feed (e.g., clay content) that could affect the performance of the filters, requiring adjustments to be made. During the adjustment period, off-specification tailings may result. In addition, as the plant transitions from one filtration unit to the other there may be upset conditions. The plant has been designed to cater to these conditions, but for limited periods the moisture content specifications may not be achieved until adjustments are made to the filtration units. Off-specification tailings may also occur during the initial commissioning of the filter presses as the equipment is adjusted to field conditions. Off-specifications tailings delivered to the TMF will be air dried at the placement site prior to roller compaction. Air drying will be enhanced by blading and/or discing the tailings surface into windrows on a regular basis until a lower, workable moisture content is achieved. Monitoring and adjustments will be made, as necessary, to the filter presses to regularly meet the specifications to allow hauling, placement, and surface rolling of the tailings. The moisture content of the delivered tailings will be monitored and no tailings with moisture content exceeding the criteria will be disposed at the TMF. If wet conditions cause excess moisture in the tailings, then placement may need to stop until suitable conditions can be restored. Monitoring and adjustments will be made, as necessary, to the filter process to regularly meet the specifications.

- *PAG waste rock deposition in the TMF:* PAG waste rock identified during the operational life of the TMF will be placed on top of the CLS and within the interior of the waste rock retention shell on the south side of the TMF, to isolate it from weathering effects and prevent it from acting as a potential source of ARD and metal leaching. The PAG waste rock will be spread to limit vertical accumulation in concentrated areas, which will limit contact with the limited amount of infiltrating water migrating vertically through the waste rock. The CLS will prevent seepage that may have come into contact with PAG materials from infiltrating into the groundwater.

- *Monitoring and inspection:* An Operations, Maintenance and Surveillance (OMS) Plan will be prepared and implemented for the TMF addressing requirements for the operation, safety, and environmental performance of the facility, including a framework for identifying, evaluating, and reporting significant observations. Specific monitoring and inspections related to the TMF will include:

    - Structural stability assessment of the TMF and related water control structures,

    - Water quality sampling at designated monitoring points, and

    - Piezometric monitoring of water levels in the tailings mass.

- *Reclamation:* A vegetated soil cover will be placed over the closed TMF to achieve a stable hydrologic configuration and minimize infiltration. The cover will promote conveyance of stormwater; prevent surface water ponding; disperse rather than concentrate runoff; limit erosion and channel scour; provide long-term erosional stability; and promote establishment of perennial, self-sustaining, native vegetation. The soil growth medium component of the cover will limit infiltration, promoting vegetation growth, runoff, and evapotranspiration. The soil growth medium layer thickness will generally be 12 inches. Geotechnical site investigations indicate there is sufficient material located on the Project site suitable for a soil cover that meets these requirements. The waste rock shell is expected to be suitable for a base for the soil cover. Some waste rock processing will be required to produce a transition zone between the rock structural shell and the soil growth medium cover material. Preliminary design of the transition zone indicates a minimum two-foot-thick layer of well graded (coefficient of uniformity greater than four) material with a maximum particle size of three inches. Micro-topographical undulations will be created in the TMF slope for wildlife habitat. The TMF will receive shrub-specific vegetation for wildlife on the south face. Rock outcroppings will also be constructed to enhance wildlife habitat. Post-mining, the TMF landforms will provide long vegetated south-facing slopes with shrubbery to support local wildlife.

[2] Metallurgical water content is tailings moisture measured by total weight. Geotechnical water content is measured by dry weight. A 14% tailings moisture metallurgical is equivalent to 16.3% geotechnical.

As described in Section 15, after the pit is fully excavated during Year 8, the pit will be backfilled with tailings produced during the last two years of post-mining mineral processing up to an elevation of 6,630 feet amsl. Then, with a combination of blasting and earthmoving, the pit rim will be dozed into the pit to create a 3H:1V final pit wall slope and final backfilled pit elevation of approximately 6,720 feet amsl.

Groundwater and precipitation will flow into the pit backfill material and the groundwater level will slowly rise within the pit until it stabilizes at about 6,717 feet elevation about 130 years after mining (Neirbo Hydrogeology 2023). As described above, geochemical testing of mine rock and tailings indicates limited potential to produce ARD and/or metal release, therefore water contacting the pit wall rock and backfill is not expected to result in detectable metal leaching. A pit lake is not expected to form because evaporation losses will keep the groundwater level below the surface of the backfill. The pit is predicted to act as a hydraulic sink with no groundwater outflows.

17.2.2 Site Monitoring

The scope of site monitoring activities during construction, mining, mineral processing, reclamation, and closure is derived from impact and risk assessment, permit conditions of approval, and commitments made in the permit applications (Section 17.3). The following site monitoring activities will be performed:

- *Meteorology:* The current meteorological monitoring program (Section 17.1) will continue through the construction and operating phases of the mine.
- *Air quality:* Continued ambient air quality monitoring will be conducted for PM10 emissions. Opacity monitoring will be conducted at the crusher, screens, conveyor transfer points, and other points of fugitive emissions. Water and chemical dust suppression use will be recorded, including quantities and water truck operating hours. Emergency generator usage will be recorded.
- *Surface water:* Monitoring of flow and water quality in streams, post-storm seeps, and at detention ponds and associated channels and other engineered flow paths will be conducted per WYPDES permit conditions (Section 17.3).
- *Groundwater:* Monitoring of groundwater level and quality will be conducted. Additional groundwater monitoring wells will be installed and periodically sampled. Some existing and planned monitoring wells will be lost to mine development. Open pit dewatering water quality and flow rates will be monitored during operations.
- *Waste rock ARD potential:* Blast hole cuttings will be geochemically tested to classify the rock as either PAG or NPAG for handling accordingly (Section 17.2.1).
- *TMF Operations, Maintenance and Surveillance (OMS) Plan:* TMF performance monitoring and inspections will be conducted, including structural stability, water quality sampling, and piezometric monitoring of water levels in the tailings mass (Section 17.2.1).

- *Pit wall stability:* Survey monuments will be placed around the pit excavation to monitor for movement. Ongoing geotechnical mapping and monitoring of the pit slope faces will be conducted. Movement beyond that which would be expected from rock mass dilation and unloading will trigger redesign or remedial measures. Piezometric water levels in the pit wall rock will be monitored for signs of potential decreased stability.

- *Noise and vibrations:* Ground vibration, air overpressure, flyrock distances, and dust and gas emissions from blasting will be measured.

- *Topsoil stockpiles:* Monitoring of wind and water erosion of stockpiles will be ongoing during operations.

- *Weed growth:* Operational areas, stockpiles and reclaimed areas will be monitored to limit the spread of noxious weed species.

- *Wildlife monitoring:* Operational areas will be inspected for the presence of listed and other sensitive species (Section 17.1.1) prior to construction disturbance.

- *Cultural and paleontological finds:* A chance finds procedure will be implemented to protect unknown cultural or paleontological resources potentially encountered during initial construction disturbance.

- *Post-closure monitoring:* A post-closure monitoring plan will be implemented to verify that closure objectives are met, including water quality, the closed facilities' long-term physical and chemical stability, and establishment of post-mining land use.

17.2.3 Water Management

The CK Gold Project will operate in a net water deficit situation, given that the mean annual evapotranspiration exceeds the mean annual precipitation (Section 17.1). The Project will implement water saving measures, as summarized below. Also described below are the Project's water balance, water supply source, and groundwater/surface water management design and monitoring approach.

17.2.3.1 Water Saving Measures

The Project will implement the following water saving measures to minimize its water consumption from off-site sources:

- *Tailings filtration:* Tailings generated in the flotation process will be filtered to an optimum low moisture content to produce "dry stack" tailings, thereby minimizing water consumption and avoiding the need for a tailings dam and the associated environmental and safety risks. The tailings slurry produced by flotation initially containing about 65% water (by weight) will first be thickened for initial water recovery. The water content of the thickened underflow slurry will be reduced to about 45%, while the thickener overflow water will be returned to the process for reuse. The thickened slurry will be pumped to storage tanks ahead of a large pressure filtration plant comprising multiple large pressure filters that further reduce the water content to <15% (typically 14%). The recovered water is recycled back into the flotation process, instead of being disposed of in a tailings dam where much of it would be lost to seepage and evaporation.

- *Pit dewatering recycling:* Groundwater and precipitation inflow into the pit will be collected in a sump and used for dust control on site, lowering the overall demand for water from external sources. The Project's rights to the pit inflow water are permitted (Section 17.3).

- *Surface runoff and seepage recycling:* Surface runoff and seepage from mine facilities, including the waste rock facilities, TMF, and other facilities will be collected in detention ponds and recycled for reuse as dust control or to meet process water demand. These water rights have been permitted (Section 17.3).

- *Irrigation ditch:* Water from an existing irrigation ditch ("Simmons No. 4 Ditch") currently supplying water to a hayfield at the proposed mineral processing plant location, fed during the late spring/early summer months by the South Crow Creek Reservoir south of the Project site, will be consumed by the Project during construction and operations, and restored to its current use during the reclamation phase.

- *On-site potable water supply well:* An on-site water supply well was permitted to supply potable water for on-site staff consumption.

- *Truck wash water recycling:* Used wash water will be collected at the truck wash facility, decanted and reused for dust control on site.

- *Dust control water recycling:* The fraction of water consumed in the pit and primary crusher for dust control purposes that is left over after evaporation and infiltration will be collected and recycled for dust control on site.

17.2.3.2 Water Balance

The Project's total average water consumption is 562 gallons per minute (gpm). This number is the estimated total consumption, excluding reductions in demand for water from off-site sources associated with the water saving measures described above. Consumption for mineral processing, general operations, and dust control is as follows:

- *Process plant*: 475 gpm, based on a daily feed of 20,000 short tons of ore. The initial moisture of the incoming ore to the primary crusher is estimated to be 3%. The metallurgical test work identified the moistures of the two final products of ore processing, which are as follows: concentrate (less than 1% of the total ore feed by weight) with remnant moisture by weight of 10%; and tailings (99% of the total feed by weight) with 14% moisture.

- *Truck Wash*: 3.5 gpm, based on the design of this facility that utilizes high efficiency (low water consumption) nozzles and an average wash time of 25 minutes for each piece of equipment, 3-4 times a month for preventive and unplanned maintenance. Approximately 75% of the water may be recycled back into the system.

- *Primary Crusher*: 5.5 gpm, based on spray nozzles operating for 60 seconds each time a truck dumps in the crusher dump hopper, at a rate of 40 gpm. For a 100-ton truck there are 200 loads in a day dumped into the crusher dump hopper.

- *Dust Control*: The various consumptions below are estimated by making assumptions on the frequency and supply capacity of spraying on a daily basis:

    - Pit dust control spraying on the shot rock loading faces: 10 gpm, groundwater seepage and precipitation collected in the pit sump.

    - Waste Rock Facilities dust control spraying at the dumping locations: 5 gpm, sourced from precipitation runoff collected in the detention ponds and the water storage tank as needed.

    - Dust control spraying at the newly spread tailings on the TMF surface: 14.1 gpm, sourced from precipitation runoff collected in the detention ponds and the water storage tank as needed.

- ○ Temporary haul roads dust control spraying: 37.5 gpm, sourced from precipitation runoff collected in the sedimentation ponds and the water storage tank as needed. No water will be applied for dust suppression on the roads outside of the pit and the access road. These roads will be periodically sprayed/treated with dust suppression agents, such as magnesium chloride or other dust suppressant solution.

- *Staff*: 4.5 gpm, based on a maximum of 260 staff present each shift and an average consumption of 25 gallons per day per person.

A schematic of the Project water balance is shown on Figure 17.12.

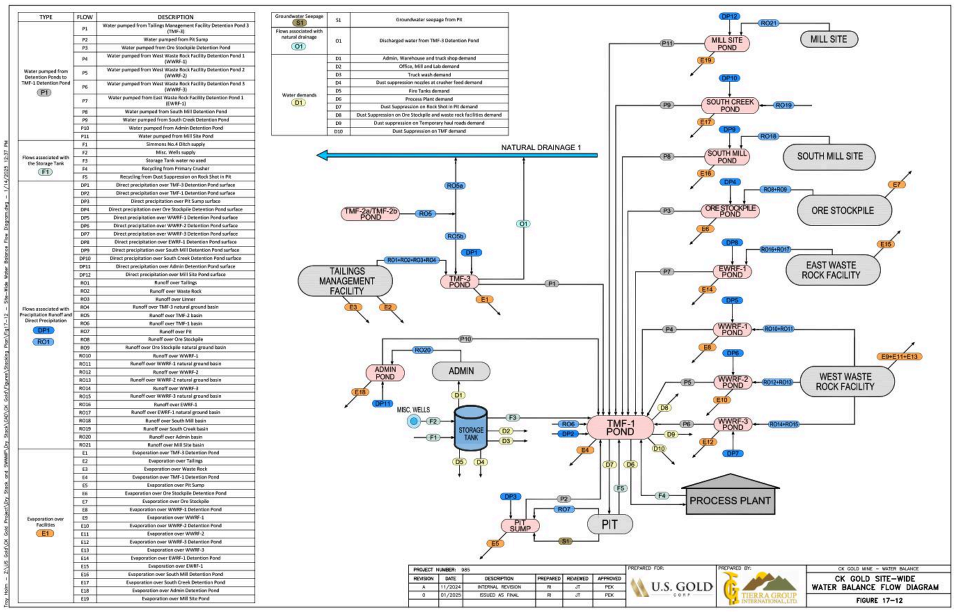

Figure 17.12: Water Balance

17.2.3.3 Recycled Water

Tierra Group developed a site-wide water balance for the CK Gold Project to maximize the reuse of contact and non-contact water within the site's watershed (Tierra Group, 2025c). The water balance assumes that the meteoric precipitation that falls on Project facilities will generally be collected by the detention ponds and pumped back to TMF-1 for reuse as dust suppression or to meet the process water demand. A system of pumps and pipelines will deliver the surface water collected in the detention ponds around site to TMF-1. The pumping system is conservatively designed to convey the design storm volume reporting to each pond within 30 days, or the maximum monthly volume calculated from the water balance, whichever is greater.

17.2.3.4 Water Supply Source

Under an agreement with the Ferguson Ranch, the surrounding landowner, a water exploration program has successfully identified a nearby source in the Red Canyon approximately 1-mile north of the project. The Red Canyon water will be less costly to develop and less costly to purchase under the agreement with the Ferguson Ranch and adjustments to the identified "source and yield" specified in the two main project permits (ISC and MOP) will be made to reflect the Red Canyon water supply once final development has been completed. Regardless of the source, water purchased will be used to make up the water deficit. Local consultants conducted preliminary engineering to confirm feasibility and costs associated with the Red Canyon supply. Following studies by TGI water generated from pit dewatering, surface runoff, and waste rock and tailings seepage will be recycled for use in mineral processing and/or dust suppression, reducing the volume of make-up water.

The Project will pump water from the Red Canyon well field to the Project water tank.

As a water supply back up, the Project negotiated a water supply agreement with the Cheyenne Board of Public Utilities (BOPU). BOPU the Lone Tree Creek (LTC) well field south of the Project and south of the I-80. BOPU also operates the South Crow Creek Reservoir and water supply pipelines that pass through and are near the Project. Should the BOPU water source be adopted, water will flow through the South Crow Creek pipeline where water will be transferred up to the water storage tank. Figure 17.13 shows the approximate alignment of the supply line.

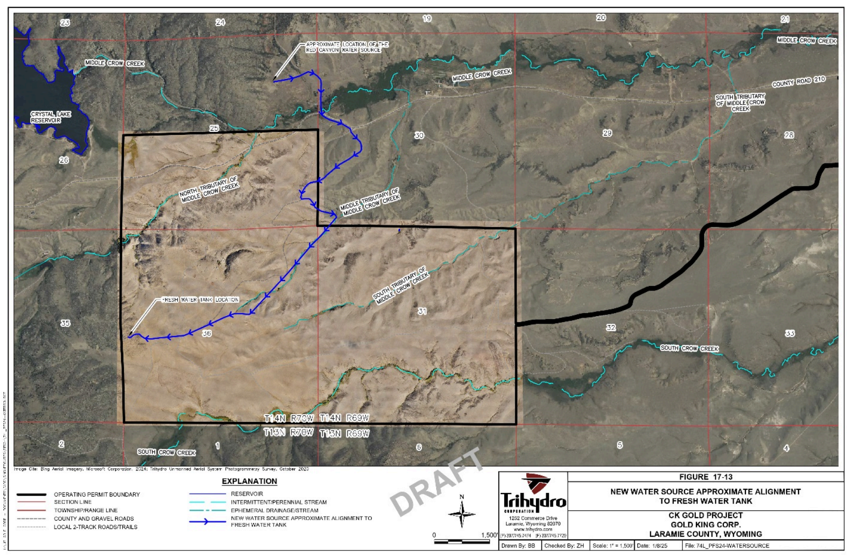

Figure 17.13: New Water Source and Approximate Alignment to Fresh Water Tank

As shown on Figure 17.14, a water line will tap into the South Crow Creek pipeline and transmit water to the Project's proposed on-site water storage tank. A pumping system may be installed at the water line tap to pump water to the Project's storage tank. The pumping system will have variable frequency drives and are required to maintain a constant water supply to the tank and to the process plant.

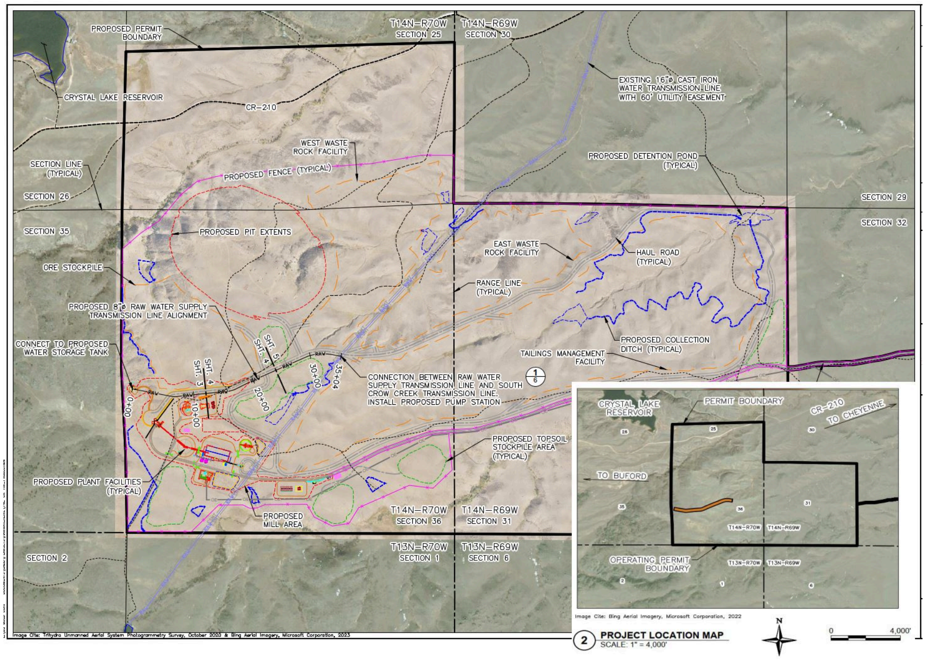

Figure 17.14: Proposed Water Transmission Infrastructure (from Trihydro 2023)

17.2.3.5 Groundwater Management

The open pit formed by mining will collect precipitation and groundwater inflow. Based on groundwater modeling performed by Neirbo Hydrogeology (2023), pit inflow is expected to be diffuse and limited due to the overall low permeability and low water storage capacity of the surrounding rock. Faults and fracture zones will yield little water and will drain rapidly due to limited spatial extents.

The annual pit bottom elevation starts at 6,900 feet AMSL in year 1 and progresses to 6,120 feet at the end of mining. Passive open pit dewatering begins when pit advancement reaches the water table. Predicted pit inflow during the first year is 6 gpm. As the pit advances pit inflows are predicted to be less than 15 gpm. This water will be recycled on site during the operations phase, as described above.

Pit dewatering during mining will result in a groundwater level decline (drawdown) relative to the pre-mining level. Drawdown will also result from changes in groundwater recharge due to changes in precipitation infiltration caused by changes in the Project site's ground surface. The groundwater model differentiates between mine-induced groundwater drawdown and groundwater level changes caused by non-Project groundwater pumping and seasonal and annual precipitation variation. Based on Project groundwater monitoring from 2020 to 2022, Project induced drawdown would need to exceed 10 feet to be distinguishable from natural and other non-Project variation.

The modeled mine-induced drawdown decreases rapidly with distance away from the pit, as shown on Figure 17.7. The 5-foot drawdown contour is predicted to remain completely within the Project boundary at except for a small jut along the western edge (Figure 17.8). The drawdown extent is limited mainly due to the low permeability of the rock.

The nearest domestic wells are approximately 2,000 feet from the predicted 5-feet drawdown area. At this distance, any mine induced drawdown would likely not be discernable from natural variation and groundwater-level changes induced by the domestic wells themselves.

After mining, the backfilled pit will slowly fill with water as precipitation and groundwater flows in. The backfill materials consist of tailings and rock dozed from the pit rim. As described in Section 17.1.4, geochemical testing of mine rock and tailings using industry standard methods on representative samples (Geochemical Solutions 2024) indicates limited probability to produce ARD and/or metal release to water. Groundwater quality is not expected to significantly deteriorate due to contact with pit wall rock, waste rock, or tailings.

The backfill surface elevation is modeled at 6,720 feet and the groundwater level is predicted to stabilize at 6,717 feet after about 130 years. The pit is roughly conical in shape, so the rate of water level rise slows as the pit volume increases with increasing elevation. Evaporation is modeled to start when the water level is within 5 feet beneath the backfill surface. Evaporation losses depress the groundwater level and prevent water from daylighting and forming a permanent pit lake. Water may temporarily pond in the pit following large precipitation events, but evaporation losses will gradually lower the water level to below the surface of the backfill. This depressed water level creates a hydraulic sink with lower groundwater levels immediately adjacent to the pit and no groundwater outflow from the pit. Therefore, any unforeseen water quality deterioration would be contained within the pit zone.

During the post-mining period, drawdown is predicted to propagate slowly and remain near the pit. Drawdown greater than 5 feet is predicted to generally extend a small distance outside the Project boundary, except for the northeast corner, at peak drawdown, 150 years after mining (Figure 17.8).

Groundwater monitoring wells around the Project site, including up- and down-gradient of mine facilities, will be sampled quarterly during the first year of mining operations. The data will be reported to the Wyoming DEQ-LQD in the Annual Reports and if the data is similar to baseline data, a request to reduce the frequency of sampling to semi-annually will be made. Actual groundwater drawdown and quality data will be recorded to confirm the modeling predictions or identify any deviations from the predictions that would trigger remedial action.

17.2.3.6 Surface Water Management

The proposed Project facilities will be limited to ephemeral drainages that are not capable of hosting aquatic life. Two water courses traversing the Mine Area (Figure 17.3) have been designated Waters of the United States, and are described in Section 17.1.1:

- South Crow Creek
- North tributary of Middle Crow Creek

Project disturbance will remain outside of these water courses and associated adjacent wetlands. The Neirbo Hydrogeology (2023) groundwater flow model predicts reductions in streamflow in these streams of only one percent or less due to mine-induced groundwater drawdown.

Mine construction, operation, and reclamation activities involving excavation and grading could potentially cause surface soil erosion and sedimentation of adjacent streams. Mitigation measures that will be implemented to avoid these potential impacts include the following:

- Phased clearing and grubbing of vegetation in areas closely preceding planned excavation and grading activities, minimizing the aerial extent and duration of surface soil exposure.
- Stockpiling of topsoil for use in covering and reseeding (reclamation) of disturbed areas.
- Implementing surface reclamation activities as soon as feasible after disturbance to minimize the duration of exposed soil surfaces, including concurrently with mining operations to the extent feasible.
- Compaction of exposed soil surfaces to minimize erosion and sediment transport.
- Deployment of erosion control materials on exposed sloped soil surfaces to minimize erosion and sediment transport.
- Directing and capturing surface runoff from Project disturbed areas via surface channels discharging into detention ponds.

Surface water flow and quality will be monitored. Water quantity and quality in the detention ponds will also be monitored as required by the WYPDES permit (Section 17.3).

Surface runoff (contact water) from the Project facilities will be collected in channels and detention ponds and recycled on-site as described above (Tierra Group, 2025c). Diversion ditches will be constructed to reduce the volume of stormwater run-on to the Project site from undisturbed areas outside of the Project boundary and to direct contact water runoff into the detention ponds. Ditches will be armored with riprap where the slope/flow velocity requires it to protect against erosion. Riprap drops and pipe drops will be constructed at the end of the diversion ditches to convey the water to a detention pond. Energy dissipators will be constructed at the end of the pipe drops to prevent erosion.

Detention ponds will be constructed to collect contact water runoff from the mine facilities. Additional ponds will be constructed in the process plant area for contact water collection and for emergency containment of process water (Figure 17.15). Generally, there is a detention pond located at the downstream end of the waste rock, Ore Stockpile, and TMF within draw bottoms to collect contact water and prevent routine discharges outside of the Project site. Water that collects in the detention ponds will be pumped to TMF-1 for use as on-site dust control and process demands. Most of the ponds are permitted by the Wyoming State Engineer's Office (SEO). TMF-1, TMF-2a, TMF-2b, and TMF-3 have been revised since the original permit submittal and the permits will need to be updated with the SEO.

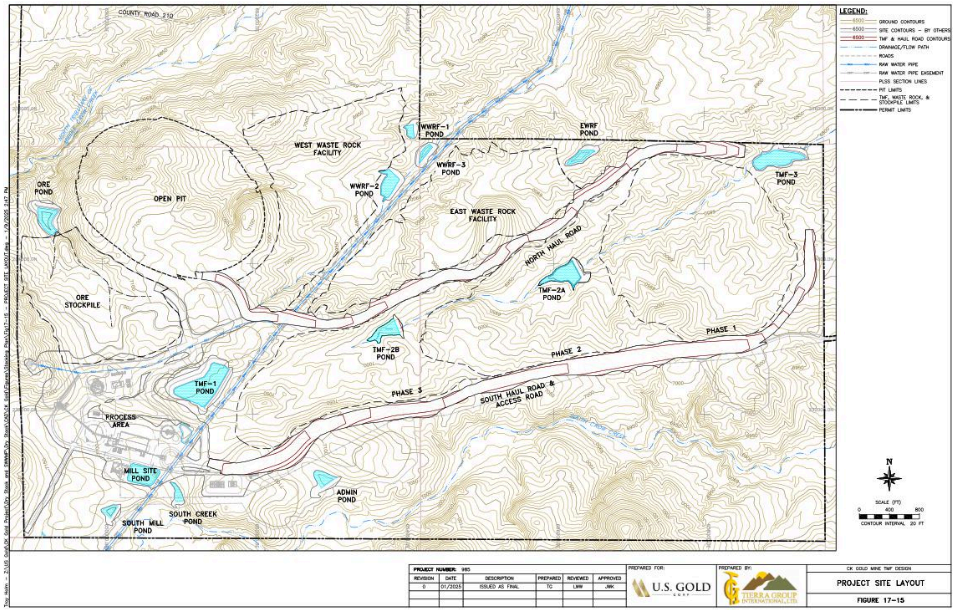

Figure 17.15: Project Site Layout

The ponds will consist of an embankment that is less than 20 feet tall and have a capacity that is less than 35 acre-feet each, maintaining a dam classification of non-jurisdictional. The embankments will be constructed from available soil at each pond's location and/or excess material from other construction operations. The embankment soil will be compacted to 90% of standard Proctor dry density. The prevalent on-site silty clays with sand and gravel are suitable for embankment construction. The embankment crest will be a minimum of 12 feet wide. The upstream slope will be no steeper than 3H:1V, and the downstream slope no steeper than 2.5H:1V. The ponds will be lined with a CLS, consisting of 60-mil HDPE liner. Overflow spillways will be provided to prevent overtopping of detention ponds during runoff events exceeding the design storm event. Ponds are designed to contain either the 10-year, 24-hour storm event (EWRF-1, WWRF-1, WWRF-2, WWRF-3, TMF-2A, TMF-2B, TMF-3) or the 100-year 24-hour storm event (TMF-1, Ore-1, Mill Site, South Mill, Admin, South Creek)while the spillways are designed to pass flow for the 100-year, 24-hour event.

17.3 REQUIRED PERMITS AND STATUS

The Project occupies state-owned and private land. Construction and operation of the mine requires various permits issued at the state and local levels. Some limited federal permitting is involved. Below is a list of the most significant agencies and associated permits. The major required permits have been obtained, as described in the sections that follow.

- *US Army Corps of Engineers*: Approved Jurisdictional Determination (Section 17.3.1)
- *US Environmental Protection Agency*: Public Water Supply Permit (Section 17.3.2)
- *Wyoming Office of State Lands and Investments*: Mining Lease (Section 17.3.3)

- *Wyoming Department of Environmental Quality*:
  - Land Quality Division
    - Exploration Permit (Section 17.3.4)
    - Mine Operating Permit (Section 17.3.5)
  - Air Quality Division: Air Quality Permit to Construct and Operate (Section 17.3.6)
  - Industrial Siting Division: Industrial Siting Permit (Section 17.3.7)
  - Water Quality Division (Section 17.3.8)
    - Wyoming Pollutant Discharge Elimination System (WYPDES) Permit
    - Stormwater Pollution Prevention Plan and Notices of Intent and Termination under the Large Construction General Permit (for Construction) and Industrial General Permit (for Operation)
    - Permit to Construct Water Supply and Wastewater Facilities
    - Operator Certification for Drinking Water System
- *State Engineer's Office:* Permits for Water Use and Water Related Facilities (Section 17.3.9)
- *State Historical Preservation Office* (Section 17.3.10)
- *State Fire Marshall* (Section 17.3.11)
- *Laramie County* (Section 17.3.12)

17.3.1 Approved Jurisdictional Determination

In February 2021 the US Army Corps of Engineers (USACE) Omaha District, Wyoming Regulatory Office, issued an Approved Jurisdictional Determination (AJD) covering the CK Gold Project site. Under this AJD, the following two surface water bodies and associated wetlands in the Project area are considered Waters of the United States and subject to USACE jurisdiction and permitting for discharging of dredged or fill materials:

- South Crow Creek
- North tributary of Middle Crow Creek

There are no plans for Project infrastructure that would lead to deposition of dredge or fill material in the above surface waters on the Project site, therefore no further USACE permitting is anticipated to be required. In April 2024 the USACE issued a confirmatory letter in this regard. The AJD is valid for five years from the date of issue. If the start of Project construction were to be delayed beyond this period (February 2026), a new AJD would need to be applied for. The legal definition of Waters of the US is subject to change in the meantime, and subsequent AJD could potentially incorporate different surface water bodies.

17.3.2 Public Water Supply Permit

USEPA Region 8 implements the Safe Drinking Water Act in Wyoming (the only state that has not taken over this responsibility itself). The Act covers public water systems with 15 or more service connections or that serve 25 or more persons for at least 60 days per year. The Project plans to supply its personnel with potable water from an on-site well and is therefore subject to this requirement. This permit has not yet been applied for. Prior to supplying potable water, an application will be filed with the USEPA Region 8. The Project will be required to monitor the quality of the supplied water and report the results to the USEPA.

17.3.3 Exploration Permit

Exploration activities conducted by the Project to date have been permitted by the Wyoming Department of Environmental Quality, Land Quality Division (DEQ-LQD), which has primary jurisdiction over mining projects in Wyoming. The Project has posted bonds to guarantee the reclamation of surface disturbance caused by the development of exploration drill pads, test pits and some roads. All such surface disturbance has been reclaimed, including revegetation. Bond release for exploration disturbance is currently still pending the reestablishment of revegetated areas.

17.3.4 Mine Operating Permit

The Project received its Mine Operating Permit (MOP) from the DEQ-LQD in May 2024. The MOP process began in October 2020 with a “Pre-Application Meeting” and a resulting Action Plan defining the information, environmental studies, and operational and closure plans required as part of the MOP application.

The MOP application package included the following main components:

1. *Adjudication File:* Signed application forms; landowner consent and list of landowners of record; tabulation of lands within the Project Permit Area; and associated maps and aerial photos. Reclamation bonding and proof of public notification are added to the Adjudication File after the public noticing and technical review.

2. *Baseline Studies*: Land use, history, archeology, paleontology, climatology, topography, geology, hydrology, soils, vegetation, wildlife, and wetlands (Section 17.1.1).

3. *Mine Plan*: General description of mining operation, mining method and schedule, mining hydrology, waste disposal, public nuisance and safety measures, and mineral processing and tailings management.

4. *Reclamation Plan*: Post-mining land use; land contouring plan; surface preparation; topsoil and/or subsoil placement; revegetation; hydrologic restoration; infrastructure and processing facility decommissioning, stabilization and reclamation; reclamation schedule; reclamation cost estimate; and public nuisance and safety measures. The reclamation cost estimate is based on the cost that would be incurred if the DEQ-LQD were to hire contractors to reclaim the mine and facilities. Reclamation bonding can take the form of an irrevocable letter of credit, self-bond, or collateral bond (including federally insured certificates of deposit, cash, government securities or real property). The bond amount is determined by the DEQ-LQD approved Reclamation Plan and associated cost estimate.

The initial MOP application package was submitted to the DEQ-LQD in September 2022. The first public notice took place in November 2022, following the issuance of DEQ-LQD's completeness review of the permit application. After the agency's subsequent technical review, the Project submitted an amended application in January 2024 addressing public and agency comments, and a second public notice was issued. In May 2024 DEQ-LQD formally approved the MOP and issued the associated License to Mine. The following conditions of approval were attached to the license, which have been fully satisfied by the Project:

1. Construction and mining may start after posting and approval of the $5,010,000 reclamation bond, covering reclamation of the first year's planned site disturbance.

2. Water discharge activities are authorized after issuance of the WYPDES permit by DEQ Water Quality Division.

3. Construction and mining may start after issuance of the Air Quality Permit by DEQ Air Quality Division.

The foregoing permit conditions are in addition to the Project commitments made in the MOP application package, namely the technical provisions in the Mine Plan and Reclamation Plan.

Additionally, the permit requires submittal to DEQ-LQD of an annual report within 30 days prior to the permit issuance anniversary date. Project requested changes to the approved MOP Mine Plan or Reclamation Plan would be highlighted in the annual report. The annual report is followed by a site inspection conducted by DEQ-LQD. A reclamation bond increase must be posted each year covering the next year's planned site disturbance, minus any credit due for completed reclamation of previous site disturbance.

17.3.5 Air Quality Permit to Construct and Operate

The Project received its Air Quality Permit to Construct from the DEQ's Air Quality Division (DEQ-AQD) in November 2024, following a public hearing held the month before during which no comments were received. The permit will expire if construction is not started by November 2026. The Project must notify DEQ-AQD of the anticipated date of mine startup between 30 and 60 days prior, and obtain the Air Quality Permit to Operate within three months after the start of mining operations (generally a simple formality, absent significant Project changes). The permit conditions of approval include specific requirements for:

- A variety of dust suppression and wind erosion control measures during construction, mining and mineral processing;

- Limiting opacity of fugitive emissions;

- Avoiding exceeding ambient air quality standards and reporting exceedances;

- Air quality and meteorological monitoring and reporting;

- Limiting use of the emergency generator (grid power will be relied upon during normal conditions);

- Limiting the size and specifications of the mobile equipment fleet as specified in the permit application; and

- Limiting blasting operations.

This permitting process consisted of a New Source Review, including the development and submittal of the Project's air emission inventory and dispersion modeling. The Project is classified as a Minor Source and falls under the DEQ-AQD's requirements for general air quality permitting to construct and minor source permitting to operate. Title V of the Clean Air Act does not apply.

17.3.6 Industrial Siting Permit

The Industrial Siting Permit (ISP) requirement is triggered by an overall project construction cost estimate amount threshold which changes each year. When the Project's ISP application was submitted to the DEQ - Industrial Siting Division (ISD) in February 2023, the construction cost estimate threshold triggering the ISP requirement was approximately $254 million. The state's intent with this requirement is to plan for and mitigate potentially significant environmental and socioeconomic community impacts arising from a temporary influx of construction workers.

The Project's ISP application was approved in June 2023 via a written Order by the Industrial Siting Council (ISC). The ISC was convened by the DEQ-ISD to review and rule on the Project's ISP application. The ISP application package included a project description, socioeconomic and environmental impact assessment, and management plan. Associated technical studies were focused primarily on Project induced noise and traffic, as well as socioeconomic impacts. Other types of environmental impacts were assessed as part of the MOP process described above. The socioeconomic impacts were generally assessed as positive in the ISP application.

The impact assessment study area covered portions of Laramie County and the adjacent Albany County to the west (the Project is wholly located within Laramie County). The Project notified and consulted with these county governments and other local government agencies. Following submittal of the permit application, public notifications were issued and public informational meetings held in the cities of Cheyenne and Laramie (the respective county seats) in December 2022. Various agencies provided written feedback to the Project and the DEQ-ISD, mainly consisting of requests, recommendations, and notification of their applicable requirements. The ISC presided over a public hearing held in May 2023, during which the Project's representative answered questions under oath.

The ISC's June 2023 permit approval Order includes a provision to award "unmitigated impact assistance funds" of approximately $408,000 to Laramie County and $726,000 to the City of Cheyenne. These awards will be funded by the state from increased state tax receipts associated with anticipated Project related procurement of materials within the state. According to the ISC's Order, "these funds are to compensate for unmitigated impacts to the affected counties, cities, and towns in the area primarily affected."

The ISP will expire if Project construction does not start by June 2026. Permit conditions of approval include:

- Obtaining and adhering to conditions of the other required state and local permits.
- Notifying the DEQ-ISD in advance of proposed Project changes in "scope, purpose, size, or schedule," and filing of an evaluation of Project changes potentially resulting in significant environmental and social impacts not evaluated in the ISP, before such changes are implemented.
- Developing a written plan and program for achieving compliance with the permit conditions and commitments made in the permit application, including identification of a compliance coordinator.

Detail procedures for local hiring in the compliance plan and file job postings with the local Workforce Center.

- Performing additional mitigation measures beyond those committed to in the ISP, if certain unforeseen adverse environmental or social impacts are caused by the Project.
- Additional notifications as follows:
  - to the DEQ-ISD of the start date of construction and when "physical components of the facility are 90 percent complete;" and
  - public notification via local newspaper ad when the "facility is nearing completion."
- Submitting annual reports through the first year of mining operations documenting:
  - efforts to comply with the permit conditions and commitments made in the permit application;
  - construction completion status relative to the approved schedule, and schedule revisions;
  - summary of construction, reclamation and other activities to be conducted the following year; and
  - demonstration of compliance with permit conditions.
- Implementing a monthly monitoring program and quarterly results reporting to DEQ-ISD of:
  - average and peak numbers of employees of the Project owner, contractors and subcontractors;
  - employee city and state residency while hired and employed;
  - number of new students enrolled by grade level and school district related to Project employees;
  - Wyoming resident vs non-resident mix; and
  - updated construction schedule.
- Notification in advance of changes in the construction workforce schedule triggering a 15% or more exceedance of the committed peak workforce number, or changes in the committed lodging plan.
- Submitting to the DEQ-ISD at least 30 days prior to the start of construction, the following documents:
  - "Spill Prevention, Control, and Countermeasure (SPCC) Plan which additionally adheres to the recommendations of the DEQ's Water Quality Division for the Fuel Depot/Truck Shop and Truck Wash Building, Standard Operating Procedures and Spill Kit, and Water Recycling;"
  - The signed Wyoming Game & Fish Department (WGFD) monitoring plan; and
  - Class III Cultural Resources Survey.

The foregoing permit conditions are in addition to the Project commitments made in the ISP application package.

17.3.7 Water Quality Division Permits

The DEQ - Water Quality Division (WQD) issues several permits applicable to the CK Gold Project as summarized below.

Wyoming Pollutant Discharge Elimination System (WYPDES) Permit

A WYPDES Permit regulating potential Project water discharges from 12 outfalls was issued by the WDEQ-WQD in May 2024. The outfalls consist of controlled discharge points from stormwater runoff and seepage detention ponds located on the Project site (Section 17.2.3). The Project's WYPDES permit number is WY0997003 and the permit expires in April 2029.

The permit imposes effluent limits in terms of concentrations of various metals (total and dissolved), pH, and total suspended solids. Daily effluent flow measurements are required, along with monthly chemical quality sampling, and quarterly reporting of results. Other requirements include:

- Notification of changes resulting in classification as a new source or changes in the nature, or increase in quantity, of pollutants discharged. Also, notification of noncompliance or potential noncompliance within 24 hours.

- Proper operation and maintenance of water treatment and control facilities. Bypasses of treatment facilities are prohibited except for essential maintenance and if effluent limits are not exceeded, or if a bypass was unavoidable to prevent loss of life, personal injury or severe property damage. Noncompliance with effluent limits may be excused during upset conditions if the water treatment and control facilities are properly designed and operated.

- Taking reasonable steps to minimize adverse impacts on receiving waters due to noncompliance.

Stormwater Pollution Prevention Plan (SWPPP) and Notices of Intent (NOI) and Termination

A Stormwater Pollution Prevention Plan (SWPPP) and Notice of Intent (NOI) must be submitted and approved by the DEQ-WQD prior to the start of construction. This is still pending. Stormwater discharges from the Project site during the construction phase are expected to be approved by the DEQ-WQD under the Large Construction General Permit (LCGP). Upon completion of the construction phase, the Project must file a Notice of Termination of the stormwater discharges approved under the LCGP. Before the start of mining operations, another SWPPP and NOI must be submitted to WQD for approval of stormwater discharges from the project site during the operations phase under the Industrial General Permit (IGP). Permit decisions by the DEQ-WQD for both the LCGP and IGP can generally be expected within 30 days of submittal of complete SWPPPs and associated notices.

Permit to Construct Water Supply and Wastewater Facilities

Construction of the Project's water supply and wastewater infrastructure will require a DEQ-WQD permit. The permit application must include plans, specifications, design data and potentially an environmental monitoring plan. This permit application is still pending. A permit decision can generally be expected in 60 days.

Operator Certification for Drinking Water System

The Project must obtain an operator certificate from the DEQ-WQD to operate the water treatment and distribution system of potable water serving Project site personnel and visitors. This is still pending. The certificate must be renewed every three years.

17.3.8 State Engineer's Office Permits for Water Use and Related Facilities

The Wyoming State Engineer's Office (SEO) issues permits to appropriate water for beneficial use, as well as permits to construct and operate water related infrastructure such as wells, mine dewatering systems and reservoirs. Between August 2022 and October 2023, the SEO issued 13 permits for water detention and storage ponds on the Project site. Three of these permits will need to be updated and one additional pond will need to be permitted with the SEO as a result of changes made to the water management plan as noted in Section 17.2.3. Additionally, in November 2022 the SEO granted permits for the planned groundwater abstraction from the pit sump and from a water supply well on the Project site.

17.3.9 State Historical Preservation Office

The Wyoming State Historical Preservation Office (SHPO) requires a Cultural Resource Clearance if cultural resources are encountered within the Project site. A Class I cultural resource review was completed in June 2021, and a Class III field survey was conducted in September 2024. In the event that cultural or paleontological resources are encountered during construction or mining operations, activities must be halted at the find location and the DEQ-LQD and SHPO must be contacted within five days of discovery. If a resource is encountered on State land (Section 36), the OSLI must also be notified. Agency approval would be required to resume work at the find location.

17.3.10 State Fire Marshal Permits

An electrical plan and above ground fuel storage tank plan must be submitted to the State Fire Marshall for approval in accordance with the National Electrical Code. This is pending.

A fire protection system plan must be submitted in accordance with the Wyoming Department of Fire Protection and Electrical Safety. The State of Wyoming has adopted the International Codes, including the International Fire Code. Additionally, the fire protection system plan must meet the Laramie County Rural Fire Protection Development Rules and the Mining Safety and Health Administration (MSHA) regulations. This is also pending.

Fire hazard in the CK Gold Project area is generally low. The pit, stockpiles, and mine facilities will be stripped of vegetation and topsoil prior to disturbance during development and mining. Mine site water trucks will be available for fire suppression. Mobile equipment must have fire extinguishers per MSHA regulations.

17.3.11 Laramie County Permits

Laramie County will require a permit for the Project access road intersection or approach to County Road 210. The County may also require a Road Use Agreement. These permits are still pending. A traffic study was conducted as part of the ISP, establishing baseline traffic volumes and modeling Project related traffic volume increases on local roads. Work on public roadways will also require coordination and review by the Wyoming Department of Transportation (WYDOT).

The County may require permits for the various buildings to be constructed on the Project site. These permits have not yet been processed. The Project may be subject to inspections by the County Building Department.

17.4 LOCAL INDIVIDUALS AND GROUPS

In addition to the permitting requirements and associated interaction with the relevant federal, state and local government agencies as summarized in the previous section, development of the CK Gold Project will require certain agreements with private local entities as follows:

- *Ferguson Ranch*: land use rights and easements for access road and power line. Irrigation ditch temporary water rights and water supply well.
- *Black Hills Energy, subsidiary of Black Hills Corporation:* power supply agreement.
- *Financing and contracting:* Subject to Project financing and satisfactory contracting arrangements.

U.S. Gold has also reached out and provided Project information to various additional local public and private entities which may be affected by and/or interested in the project, as follows:

- *Laramie County:* host county potentially affected by Project environmental and socioeconomic impacts (employment, procurement, tax revenue, worker influx, traffic, etc.).
- *City of Cheyenne:* potentially affected by Project environmental and socioeconomic impacts, and supplier of water to the Project.
- *Neighboring residents and property owners west of the Project site:* potentially affected by Project environmental impacts.
- *Wyoming State Parks:* the Project site is near Curt Gowdy State Park.
- *Wyoming Game and Fish Department:* the Project site occupies mule deer winter range.
- *US Fish and Wildlife Service:* the Project site potentially hosts federally listed species.
- *Wyoming School Boards Association:* the state-owned section of the Project site is held in trust specifically to benefit Wyoming public schools.
- *University of Wyoming:* the Geology Department has collaborated on the Project's mineral exploration activities.
- *Granite Canyon Quarry:* nearby producer of construction aggregates.
- *Sutherland and King Ranches:* neighboring cattle ranches.
- *Wyoming Mining Association:* statewide trade association representing and advocating for mining.
- *Wyoming Taxpayers Association:* trade association representing taxpayers, including large mineral taxpayers.
- *Cheyenne Area Chamber of Commerce:* local business organization.
- *Cheyenne LEADS:* economic development organization for the city of Cheynne and Laramie County, Wyoming.

The Project is not located adjacent to any indigenous, Native American, or Bureau of Indian Affairs lands.

17.5 MINE CLOSURE

The Project has submitted a Reclamation Plan as part of the MOP application (Section 17.3). The closure objective is to reclaim the site to enable the resumption of its current use of cattle grazing, mule deer winter range, and other wildlife grazing. A reclamation cost estimate has been developed and submitted to the state as part of the reclamation bonding process. The reclamation plan is summarized as follows.

Topsoil will be removed from disturbed surfaces during the mine construction and operating phases and stockpiled on site for subsequent use as cover soil and revegetation during site reclamation. Concurrent reclamation will be practiced during the life of mine to reclaim portions of the Project site as soon as feasible prior to the end of mining, securing corresponding early releases in bonding obligations. Cattle grazing will continue as feasible during mining on Project areas not directly affected by mine operations.

At the end of mineral processing operations, the mineral processing plant and support structures and facilities will be dismantled or demolished down to their foundations, with the latter left in place under a layer of revegetated cover soil. Materials and equipment will be salvaged or disposed of off-site. Process vessels and fuel and reagent tanks will be cleaned prior to salvaging or disposal, and any contents and residues will be managed and disposed of according to the applicable regulations. Certain structures or facilities may be left in place if requested by the landowners.

Quarries, borrow pits, yards, pads, drainage channels and impoundments will be regraded and revegetated. Roadways will be similarly reclaimed, except for segments to remain operational for post-closure monitoring purposes or at landowners' request. Wells will be abandoned and plugged unless the landowners wish to retain them.

The waste rock and tailings facilities' final reclaimed slopes will be 3H:1V or flatter. Micro-topographical undulations will be created on the TMF slope to promote revegetation and to support wildlife habitat. The TMF will receive shrub-specific vegetation on the south face to support mule deer and other wildlife. Rock outcroppings will also be constructed to enhance wildlife habitat.

Regraded surfaces will generally be covered with topsoil and revegetated using approved seed mixes. A transition material of crushed rock will be used to limit topsoil from being lost into TMF or waste rock facility rock voids. While the new vegetation grows, erosion control best practices will be implemented to protect against soil erosion. In certain areas of natural rock outcrop, the final exposed surface may be bare rock instead of vegetation.

Precipitation falling on the reclaimed areas will flow into natural drainages and infiltrate into the ground. Based on geochemical study results (Section 17.1.4), the waste rock and tailings are not expected to be acid generating, and seepage from these facilities is expected to meet applicable water quality standards. Seepage will be allowed to flow from the toes of the waste rock and tailings facilities into established natural drainages in a controlled manner that prevents erosion and sediment transport.

After the pit is fully excavated during Year 8, the pit will be backfilled with tailings produced during the last two years of post-mining mineral processing up to an elevation of 6,630 feet amsl. Then, with a combination of blasting and earthmoving, the pit rim will be dozed into the pit to create a 3H:1V final pit wall slope and final backfilled pit elevation of approximately 6,720 feet amsl.

Groundwater and precipitation will flow into the pit backfill material and the groundwater level will slowly rise within the pit until it stabilizes at about 6,717 feet elevation about 130 years after mining (Neirbo Hydrogeology 2023). Geochemical testing of mine rock and tailings indicates limited potential to produce ARD and/or metal release, therefore water contacting the pit wall rock and backfill is not expected to result in detectable metal leaching. A pit lake is not expected to form because evaporation losses will keep the groundwater level below the surface of the backfill. The pit is predicted to act as a hydraulic sink with no groundwater outflows.

To help increase the local area's long-term water storage capacity, discussions have begun with BOPU about the possibility of converting the post-mining open pit into a water storage reservoir. Upon completion of reclamation, water could be transferred from external sources to the new reservoir to help meet the local area's water storage needs.

A post-closure monitoring plan will be implemented to verify that closure objectives are met, including the physical and chemical stability of the closed facilities.

17.6 ADEQUACY OF PLANS

Environmental compliance to date has been applicable to mineral exploration and other site investigation pre-mining activities, including management of surface disturbance, drilling, water use and discharge, reclamation of drill pads and roads, and associated bonding. Environmental management of these activities appears to have been good. The CK Gold project has a positive, collaborative relationship with the Office of State Lands and Investments, the Department of Environmental Quality, and the affected private landowner.

Another area of current focus is community engagement, including reaching out to and negotiating with the various private and public entities with whom the Project seeks agreements to enable further Project development. Current community engagement efforts also extend to other affected and interested local groups (Section 17.4).

Prior to the start of construction of the mine facilities, a Project Environmental Management System (EMS) will be developed and implemented consisting of a series of site-specific standards, plans and procedures governing the environmental management of the specific Project activities causing potential environmental impacts during construction, operations, closure and post-closure. The plans and procedures will identify management measures designed to avoid, mitigate or compensate for such impacts. The EMS will address the physical, natural biological and human community environmental components of the Project site and surroundings, including potentially affected local individuals and groups. The final engineering design of the Project, the environmental baseline studies (Section 17.1), the environmental impact and risk assessment, and the permit conditions of approval (Section 17.3), collectively form the basis for developing the Project EMS.

17.7 COMMITMENTS TO LOCAL PROCUREMENT OR HIRING

The CK Gold Project's policy is to prioritize procurement and hiring from within the State of Wyoming to the extent feasible.

To date, the Project has found and utilized excellent local and in-state providers for the following services:

- Environmental baseline studies
- Preparation of permit applications
- Geological field work and logging
- Revegetation and reclamation
- Miscellaneous site works and preparation in support of drilling and test pit activities
- Sample transportation
- Hydrological and hydrogeological studies and engineering design

- Environmental laboratory testing of water and rock samples
- Geotechnical site investigation and laboratory testing
- Rock quality testing for aggregate
- Socioeconomic impact assessment
- Traffic study
- Site management support
- Community relations

As development of the Project moves forward, U.S. Gold will continue to prioritize local procurement of competitively available goods and services, and local hiring of qualified personnel.

18.0 CAPITAL AND OPERATING COSTS

18.1 CAPITAL COST ESTIMATE

Capital costs are categorized as either initial capital or sustaining capital. Initial capital costs are expended in the year before production begins, Year -2 and Year -1. Sustaining costs are expended starting in Year 1.

The capital cost estimate is built up by cost centers as defined by the project's WBS for Area designations as well as by prime commodity accounts. Inputs to the capital cost are derived from various sources including unit rates provided by contractors, equipment and material quotations, in-house historical data, published databases, factors and estimators' judgment (allowances). A growth allowance of 5% has been added to mechanical equipment costs. This is intended to account for incomplete criteria, the preliminary nature of specifications, and overall quotation inaccuracies.

Contingency is assessed by considering the quality of scope definition, quantification, and pricing within the estimate. Each component is assigned a percentage based on the judgment of the project team to capture uncertainty or incompleteness of costs. Specific consideration used in determining percentages include estimate allowances and factors, design maturity, project proximity relative to labor and material markets, existing infrastructure, quality of vendor and contractor quotations, regulatory environment, and comparative costs and contingencies from similar projects. The capital cost estimates have an accuracy of 20% to +25%.

Table 18.1: Initial Capital Costs

Item	Cost (Millions)
Total Initial Capital	**276.42**
Total Direct Costs	**189.44**
Aggregate Production	0.00
Site Infrastructure	4.68
Mine & Mine Facilities	19.49
Mass Earthwork	5.70
Concrete	11.51
Steel	8.24
Buildings	27.95
Mechanical	68.80
Piping	9.83
Electrical & Instrumentation	27.69
Tailings Storage Facility	5.54
Total Indirect Costs	**41.52**
Owner's Cost	**9.54**
Contingency	**35.92**

Table 18.2: Sustaining Capital Costs		
Item	**Cost (Millions)**	
	Phase 1	**Phase 2**
Total Sustaining Capital	**6.72**	**8.29**
Water Infrastructure	0.08	0.08
Ex-Pit Roads	0.02	0.02
Mineralized Material Facility	1.02	0.00
Waste Rock Tailings Co-Disposal Facility	4.30	6.64
Site Earthworks/Grading	0.19	0.19
Contingency	1.11	1.37

18.2 OPERATING COST ESTIMATE

Estimation of operating costs for the Project is performed within the economic model for the Project on an annual basis. The operating cost estimate is based on the Project and material schedule. The components of the operating cost are based on the project schedule, equipment sizing and productivity, labor estimates, and unit costs for supply items. Inputs to the operating cost are based on vendor quotes, private and commercially available cost models, and actual and factored unit costs from similar mining operations. The operating cost estimates have an accuracy of +/- 25%.

Table 18.3 shows a summary of the operating costs at the CK Gold Project categorized by general area over the duration of the Project, or life of mine (ROM). **Error! Reference source not found.** provides additional detail to the cost categories on an annual basis.

Table 18.3: Project Operating Cost Summary			
	Total LOM	**Avg Annual**	**$/ton**
	($millions)	**($millions)**	**processed**
Total Project Operating Costs	**$1,026.10**	**$101.29**	**$14.01**
Mining Cost	$277.81	$27.42	$3.79
Process Cost	$517.84	$51.12	$7.07
Tailings Haulage	$124.22	$12.26	$1.70
Site G&A	$106.22	$10.49	$1.45

Table 18.4: Annual Operating Costs												
Years	**Total**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Total Project Costs (Millions $)	**$1,026.1**	**$108.6**	**$114.1**	**$112.6**	**$115.4**	**$113.6**	**$105.8**	**$102.9**	**$90.7**	**$77.0**	**$75.6**	**$9.7**
Mining Cost	**$277.8**	**$36.9**	**$36.3**	**$35.5**	**$38.2**	**$36.6**	**$32.0**	**$28.4**	**$17.4**	**$8.0**	**$7.8**	**$0.8**
Drill & Blast	$105.2	$14.7	$14.9	$15.0	$14.9	$14.7	$13.3	$12.3	$5.4	$0.0	$0.0	$0.0
Load & Haul	$113.3	$14.5	$13.4	$12.8	$15.9	$14.3	$13.5	$10.6	$7.3	$5.1	$5.1	$0.8
Grade Control	$17.6	$2.4	$2.6	$2.3	$2.1	$2.2	$2.1	$2.4	$1.6	$0.0	$0.0	$0.0
Voids	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Soils Removal	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Pit Closure	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Mine General	$34.8	$4.6	$4.6	$4.6	$4.6	$4.6	$2.3	$2.3	$2.3	$2.3	$2.3	$0.0
Mine Maintenance	$7.0	$0.8	$0.8	$0.8	$0.8	$0.8	$0.8	$0.8	$0.8	$0.5	$0.4	$0.0
Process Cost	**$517.8**	**$48.5**	**$51.4**	**$51.4**	**$51.4**	**$51.4**	**$51.4**	**$51.4**	**$51.4**	**$51.4**	**$51.4**	**$6.7**
Labor	$95.8	$9.5	$9.5	$9.5	$9.5	$9.5	$9.5	$9.5	$9.5	$9.5	$9.5	$1.2
Fixed Power	$43.8	$4.3	$4.3	$4.3	$4.3	$4.3	$4.3	$4.3	$4.3	$4.3	$4.3	$0.6
Variable Power	$129.6	$11.6	$12.9	$12.9	$12.9	$12.9	$12.9	$12.9	$12.9	$12.9	$12.9	$1.7
Maintenance	$26.6	$2.6	$2.6	$2.6	$2.6	$2.6	$2.6	$2.6	$2.6	$2.6	$2.6	$0.3
Reagents	$48.8	$4.4	$4.9	$4.9	$4.9	$4.9	$4.9	$4.9	$4.9	$4.9	$4.9	$0.6
Grind Media	$107.2	$9.6	$10.7	$10.7	$10.7	$10.7	$10.7	$10.7	$10.7	$10.7	$10.7	$1.4
Water	$7.0	$0.6	$0.7	$0.7	$0.7	$0.7	$0.7	$0.7	$0.7	$0.7	$0.7	$0.1
Consumables	$40.5	$4.0	$4.0	$4.0	$4.0	$4.0	$4.0	$4.0	$4.0	$4.0	$4.0	$0.5
Laboratory	$18.6	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$0.2
Site G&A	**$106.2**	**$10.6**	**$10.6**	**$10.7**	**$10.8**	**$10.8**	**$10.8**	**$10.9**	**$10.9**	**$10.0**	**$9.2**	**$0.9**
G&A	$106.2	$10.6	$10.6	$10.7	$10.8	$10.8	$10.8	$10.9	$10.9	$10.0	$9.2	$0.9
Tailings Haulage	**$124.2**	**$12.6**	**$15.9**	**$15.0**	**$15.0**	**$14.8**	**$11.7**	**$12.2**	**$10.9**	**$7.6**	**$7.1**	**$1.4**
Tailings Haulage	$124.2	$12.6	$15.9	$15.0	$15.0	$14.8	$11.7	$12.2	$10.9	$7.6	$7.1	$1.4

Table 18.5 and **Error! Reference source not found.** shows additional detail on the project operating costs for mining and processing categories.

Table 18.5: Mining Costs LOM Summary				
	Total LOM	**Avg Annual**	**$/ton**	**$/ton**
	($millions)	**($millions)**	**processed**	**Mined**
Total Mining Costs	**$277.8**	**$27.4**	**$3.79**	**$1.97**
Drill & Blast	$105.2	$10.38	$1.44	$0.74
Load & Haul	$113.3	$11.18	$1.55	$0.80
Grade Control	$17.6	$1.74	$0.24	$0.12
Voids	$0.0	$0.00	$0.00	$0.00
Soils Removal	$0.0	$0.00	$0.00	$0.00
Pit Closure	$0.0	$0.00	$0.00	$0.00
Mine General	$34.8	$3.43	$0.47	$0.25
Mine Maintenance	$7.0	$0.69	$0.10	$0.05

Table 18.6: Process Operating Costs LOM Summary			
	Total LOM	**Avg Annual**	**$/ton**
	($millions)	**($millions)**	**Processed**
Total Process Costs	**$517.84**	**$51.12**	**$7.07**
Labor	$95.79	$9.46	$1.31
Fixed Power	$43.80	$4.32	$0.60
Variable Power	$129.55	$12.79	$1.77
Maintenance	$26.59	$2.62	$0.36
Reagents	$48.84	$4.82	$0.67
Grind Media	$107.22	$10.58	$1.46
Water	$6.97	$0.69	$0.10
Consumables	$40.49	$4.00	$0.55
Laboratory	$18.59	$1.84	$0.25

19.0 ECONOMIC ANALYSIS

The economic analysis of the CK Gold Project is reliant on the project schedule, mine schedule, capital, and operating costs discussed in the previous sections of this report. This economic analysis excludes Inferred Resources, and the positive economic outcome is used to delineate a Mineral Reserve for the Project. The economic parameters used are believed to be reasonable for the type of project. All figures shown represent constant Q1 2025 US Dollars.

19.1 CAUTIONARY STATEMENT

Certain information and statements contained in this section and in the Report are "forward looking" in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and study parameters of the Project; Mineral Resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and extraction recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the Project; the net present value (NPV) and internal rate of return (IRR after-tax) and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

- There being no significant disruptions affecting the development and operation of the Project.
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report.
- Labor and materials costs being approximately consistent with the assumptions in the Report.
- Permitting and arrangements with stakeholders being consistent with current expectations as outlined in the Report.
- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders.
- Certain tax rates, including the allocation of certain tax attributes, being applicable to the Project.
- The availability of financing for the planned development activities.
- The timelines for exploration and development activities on the Project.
- Assumptions made in Mineral Resource estimate and the financial analysis based on that estimate, including, but not limited to, geological interpretation, grades, commodity price assumptions, extraction and mining recovery rates, hydrological and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business, and economic conditions.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented.

19.2 MODEL PARAMETERS

The economic model is a series of annual cashflows through the life of the Project modeled in a spreadsheet. The annual cashflow has three primary components; income (discussed in this section) and operating and capital costs, in Section 18.

The discounted cash flow analysis was performed on a stand-alone project basis with quarterly cash flows for years -3 through year 3 and annual cash flows after year 4. The economic evaluation used a real discount rate of 5% and was performed at commencement of construction using Q1 2025, US dollars.

All costs prior to the start of construction are considered as "sunk costs" and not considered in the economic analysis.

This economic analysis is a direct result of the capital cost estimate and is therefore considered to have the same level of accuracy minus 20% to plus 25%.

Table 19.1: Economic Model Parameters

Description	Values
Construction Period (years)	2.5
Economic Life	10.13
Discount Rate	5%
Contingency Capital Costs	15%
Production Inputs	
Gold Recovery (%)	67%
Copper Recovery (%)	80%
Silver Recovery (%)	68%
Mineral Pricing	
Gold Price ($/oz)	$2,100
Copper Price ($/lb)	$4.10
Silver Price ($/oz)	$27.00
Cost Criteria	
Leverage	100% Equity
Royalties	
State of Wyoming Office of State Lands	2.1%
Taxes	
Federal Tax	21.0%
Ad Valorem Production Tax	6.7%
Ad Valorem Property Tax	6.7%
Wyoming Severance Tax	2.0%

19.3 CAPITAL COSTS

The total capital cost is estimated at $316.4 million, including $272.8 million during preproduction, $16.6 million for sustaining capital, and $27 million in initial working capital over the life of the mine. Working capital is recovered over the life of the mine. Note that the preproduction capital does not match Section 18. Some of the initial capital is withheld as retention payments in Year 1 Quarter 1 and captured as sustaining capital. Table 19.2 summarizes the capital cost over the life of the mine.

Table 19.2: Life of Mine Capital Cost Summary

Description	Cost US$M
Site Infrastructure	$4.7
Mine & Mine Facilities	$19.5
Earthwork	$5.7
Concrete	$11.5
Steel	$8.2
Buildings	$28.0
Mechanical	$68.8
Piping	$9.8
Electrical	$23.1
Instrumentation	$4.6
Tailings Storage Facility	$5.5
Process Facilities Construction Indirects	$6.8
Process Facilities Construction Equipment	$3.2
3rd Party QA/QC	$0.5
Pre-Operations Support	$0.9
Process Facilities Spare Parts	$1.4
Initial Fills	$0.7
Freight	$7.2
Contingency	$35.9
Retention Payments	-$3.6
Total Preproduction Capital	**$272.8**
Sustaining Capital - Mining	$2.2
Capital Cost Retention Payments	$3.6
Sustaining Capital - TMF Infrastructure	$8.0
Sustaining Capital - Ore Stockpile	$2.9
Total Sustaining Capital	**$16.6**
Working Capital (initial)	**$27.0**
Total LOM Capital	**$316.4**

19.4 OPERATING COSTS

The LOM operating cost is estimated at $1,026.1 million, or $922.3 per equivalent ounces of gold processed, as summarized in Table 19.3.

Table 19.3: Summary of Operating Costs

Description	Total LOM	Annual Average	Per Eq Oz Au
	US$M	US$M	$ / oz Au Eq
Production Mining	$277.8	$27.4	$249.72
Labor	$95.79	$9.5	$86.10
Fixed Power	$43.80	$4.3	$39.37
Variable Power	$129.55	$12.8	$116.45
Maintenance	$26.59	$2.6	$23.90
Reagents	$48.84	$4.8	$43.90
Grind Media	$107.22	$10.6	$96.38
Water	$6.97	$0.7	$6.26
Consumables	$40.49	$4.0	$36.39
Laboratory	$18.59	$1.84	$16.71
Tailings Haulage	$124.22	$12.26	$111.66
G&A	$106.22	$10.5	$95.48
Total	**$1,026.1**	**$101.3**	**$922.3**

19.5 TAXES, ROYALTIES, DEPRECIATION AND DEPLETION

The CK Gold Project is subject to a production royalty of 2.1% on the gross sales value of the product sold, less deductions for costs incurred for processing, refining, transportation, and related costs. This royalty is paid to the Office of State Lands and Investments, State of Wyoming. Note that the typical value of this royalty is 5% in Wyoming; however, US Gold has received an exception from the Office of State Lands. The concentrate value, less applicable deductions, is multiplied by 2.1% to yield the royalty payment. The Project's net income value already considers the royalty payment.

In addition to royalites, Wyoming imposes both an Ad Valorem Production and Ad Valorem Property Tax. Production taxes are assessed at 6.7% and calculated using the proportionate profits methodology. This methodology is a ratio defined as (direct mining costs) / (total direct costs) less administration costs. The gross sales value of product sold, less deductions for costs incurred for processing, refining, transportation, and royalties is multiplied by the ratio described above and 6.7% to yield the Ad Valorem Production tax. The ad valorem property tax also applies to the real and tangible assets. In this situation the real property is owned by the State. The tangible assets including plant and equipment will be owned by U.S. Gold Corp and would be subject to the tax. The fair market value of the assets less depreciation is multiplied by the assessment ratio, (in this case 11.5% for industrial property). This becomes the taxable value which is then multiplied by the mills levied which has been estimated at 6.7%, or 67 mills.

Wyoming also imposes a 2% severance tax calculated using the proportionate profits methodology described above. The gross sales value of the product sold, less deductions for costs incurred for processing, refining, transportation, and royalties, is multiplied by the (direct mining costs)/(total direct costs) less administration costs and 2% to yield the Severance tax.

A Federal tax rate of 21% is assessed on taxable income. Federally taxable income is gross revenue less operating costs, sustaining capital, depreciation, depletion, property taxes, state severance taxes, and tax losses carried forward.

Deprecation of project capital infrastructure costs for the purpose of federal tax calculation is based on a units of production depletion model. Equipment depreciation is over a period of 7 years. Depletion for federal tax purposes is calculated by using percentage depletion method. For this property the depletion percentage is 15% of the gross revenue less royalties, not to exceed 50% of the taxable income.

A summary of the royalties and taxes is provided in Table 19.4.

Table 19.4:Summary of Royalties & Taxes

Description	Annual Average US$M	Per Ton Produced $/oz Au
Royalties		
State of Wyoming Office of State Lands	$4.3	$39.13
Taxes		
Federal Tax	$5.8	$52.60
Ad Valorem Production Tax	$4.2	$38.27
Ad Valorem Property Tax	$2.2	$20.25
Wyoming Severance Tax	$1.3	$11.42
Total	$17.8	$161.68

19.6 CASHFLOW FORECASTS AND ANNUAL PRODUCTION FORECASTS

The results of the economic analysis are provided in Table 19.5 and Table 19.6.

Table 19.5: Economic Model Results

Key Project Indicators	Value US$M
Pre Tax Economics	
IRR	36.0%
Cash Flow (Undiscounted)	$693.2
NPV 5% Discount Rate	$459.2
Payback (years)	1.7
1st 3 Years Net Profit (Avg)	$153.5
After Tax Results	
IRR	29.5%
Cash Flow (Undiscounted)	$556.9
NPV 5% Discount Rate	$355.9
Payback (years)	2.1

Table 19.6: Project Details	
Key Project Indicator	**Value**
Gold Ounces Sold (000's)	663
Copper Sold (Million Lbs.)	196
AuEq Ounces Sold (000's)	1,069
1st 3 years Avg AuEq Production (000's)	138
Initial Capital ($ Million)	$272.8
Sustaining Capital ($ Million)	$16.6
Avg. Cash Cost of Production ($/oz AuEq)	$922.0
All in Sustaining Cost ($/oz AuEq)	$937.0

Income from concentrate sales is based on the metal grades stored within the resource model and associated with material scheduled for the concentrator during the time period. Concentrator recovery factors are applied to the in-situ, contained metal to yield a total metal contained in the concentrate. Smelter terms were provided by Trafigura and Glencore, with Glencore offering more attractive terms which were applied in the economic model. The total income from metal sales, smelter terms, transportation costs, and royalty payments are subtracted to yield net project income. Table 19.7 shows a summary of metal production and revenue projections for the Project. Table 19.8 shows the cash flow summary for the Project.

Table 19.7: Metal Projections															
$ Values in Millions	Total	Year -3	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
Metal in Concentrate															
Gold (000's oz)	679.5				110.8	102.8	73.2	72.6	72.6	61.1	63.4	60.1	28.4	31.4	3.1
Copper (Millions lbs)	208.3				18.8	25.8	23.6	22.2	21.7	21.6	24.3	23.1	13.9	11.4	1.8
Silver (000's oz)	2039.5				287.3	271.5	198.8	221.1	178.3	173.5	165.6	166.1	173.9	181.2	22.4
Metal Sold															
Gold (000's oz)	662.6				108.0	100.3	71.4	70.8	70.8	59.6	61.8	58.6	27.7	30.6	3.0
Copper (Millions lbs)	195.4				17.8	24.2	22.1	20.9	20.3	20.3	22.7	21.5	13.0	10.7	1.8
Silver (000's oz)	1835.6				258.5	244.3	178.9	199.0	160.5	156.1	149.1	149.5	156.5	163.1	20.1
Refiner Receipts															
Gold	$1,391.4				$226.8	$210.6	$149.9	$148.7	$148.7	$125.1	$129.9	$123.1	$58.2	$64.2	$6.3
Copper	$801.3				$73.0	$99.3	$90.5	$85.9	$83.4	$83.3	$93.1	$88.3	$53.3	$43.9	$7.2
Silver	$49.6				$7.0	$6.6	$4.8	$5.4	$4.3	$4.2	$4.0	$4.0	$4.2	$4.4	$0.5
Deductions															
Transportation Costs	$108.2				$9.1	$13.6	$12.5	$11.2	$11.4	$11.4	$12.9	$12.3	$7.4	$5.6	$0.8
Treatment/Refining	$60.9				$5.5	$7.7	$6.9	$6.4	$6.4	$6.3	$7.1	$6.7	$4.1	$3.2	$0.5
Royalty	$43.5				$6.1	$6.2	$4.7	$4.7	$4.6	$4.1	$4.3	$4.1	$2.2	$2.2	$0.3
Total Revenue	$2,029.6				$286.0	$289.0	$221.1	$217.7	$214.0	$190.8	$202.6	$192.3	$102.1	$101.6	$12.4

Table 19.8: Cash Flow Projections

$ Values in Millions	Total	Year -3	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 Revenue	$2,029.6	$0.0	$0.0	$0.0	$286.0	$289.0	$221.1	$217.7	$214.0	$190.8	$202.6	$192.3	$102.1	$101.6	$12.4	$0.0
Operating Costs	$1,026.1	$0.0	$0.0	$0.0	$108.6	$114.1	$112.6	$115.4	$113.6	$105.8	$102.9	$90.7	$77.0	$75.6	$9.7	$0.0
Net Profit Before Tax	$1,003.5	$0.0	$0.0	$0.0	$177.3	$174.9	$108.4	$102.3	$100.5	$85.0	$99.7	$101.6	$25.1	$26.0	$2.7	$0.0
Capital Costs	$272.8	$16.8	$104.4	$151.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Sustaining Capital	$16.6	$0.0	$0.0	$0.0	$5.4	$2.9	$3.8	$0.0	$0.0	$4.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Working Capital	$0.0	$0.0	$0.0	$27.0	$1.3	-$0.3	$0.9	-$0.5	-$1.9	-$0.7	-$3.1	-$3.4	-$0.4	-$16.5	-$2.4	$0.0
Closure Bond & Closure Costs	$20.8	$0.0	$0.0	$0.0	$2.3	$1.8	$1.1	$1.0	$0.2	$0.8	$1.2	$0.5	-$3.2	-$0.9	$13.0	$3.2
Before Tax Cash Flow	$693.2	-$16.8	-$104.4	-$178.6	$168.4	$170.5	$102.7	$101.8	$102.2	$80.3	$101.6	$104.6	$28.7	$43.4	-$7.8	-$3.2
Cumulative Before Tax Cash Flow	$693.2	-$16.8	-$121.2	-$299.8	-$131.4	$39.0	$141.7	$243.5	$345.7	$426.0	$527.6	$632.2	$660.9	$704.2	$696.4	$693.2
After-Tax Cash Flow	$556.9	-$16.8	-$104.4	-$178.6	$147.0	$139.1	$84.4	$87.9	$89.0	$70.2	$89.6	$90.9	$27.5	$42.3	-$7.9	-$3.2
Cumulative After-Tax Cash Flow	$556.9	-$16.8	-$121.2	-$299.8	-$152.8	-$13.8	$70.6	$158.5	$247.5	$317.7	$407.3	$498.2	$525.7	$568.0	$560.1	$556.9

19.7 SENSITIVITY ANALYSIS

Sensitivity Analysis was performed on the parameters, capital cost, operating cost, and metal price. The following figures, Figure 19.1 through Figure 19.4, show the sensitivity of the project NPV and IRR, respectively, to key changes in project parameters on both a pre-tax and post-tax basis.

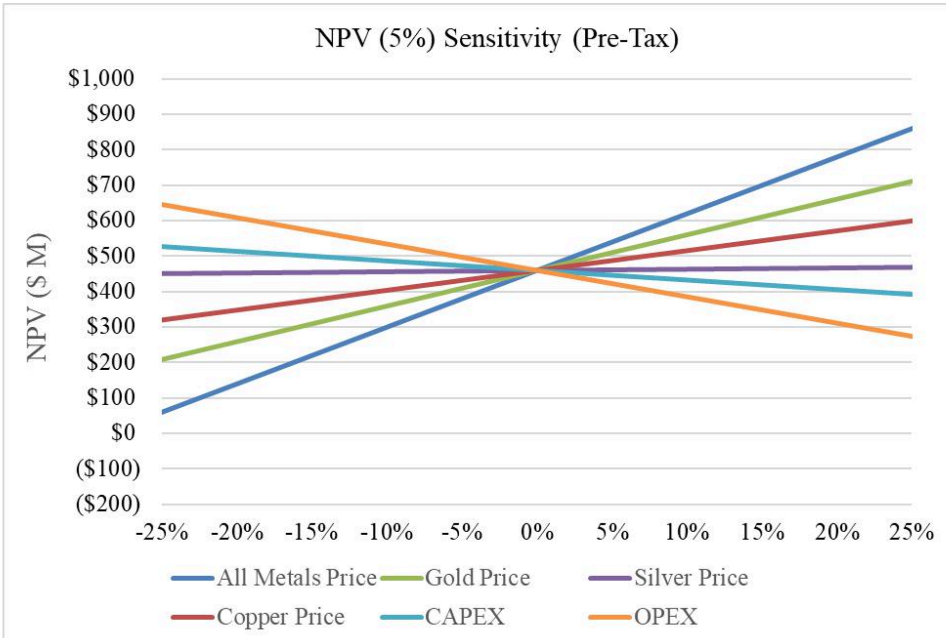

Figure 19.1: Pre-Tax NPV Sensitivity

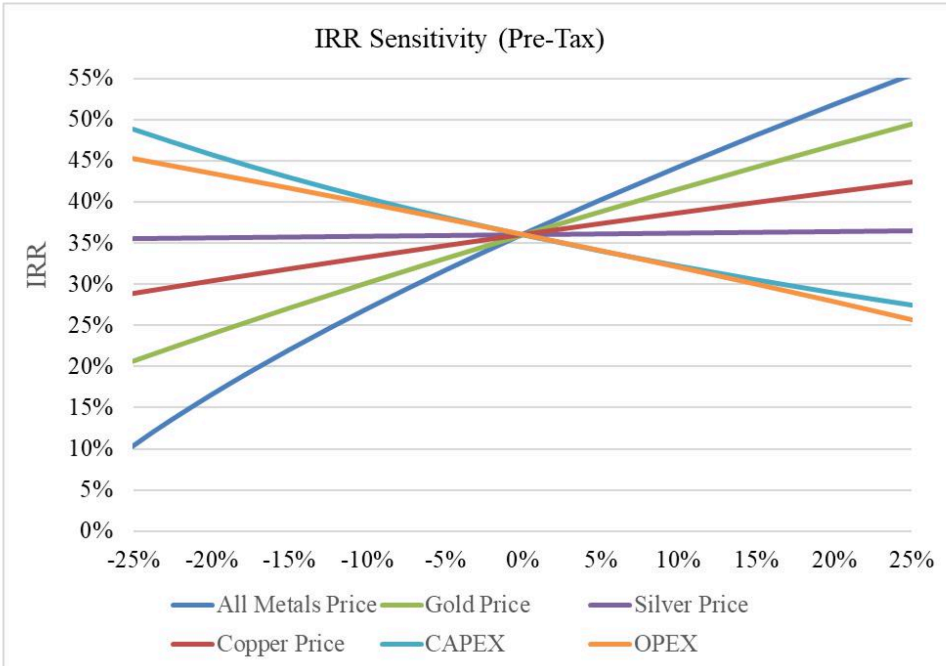

Figure 19.2: Pre-Tax IRR Sensitivity

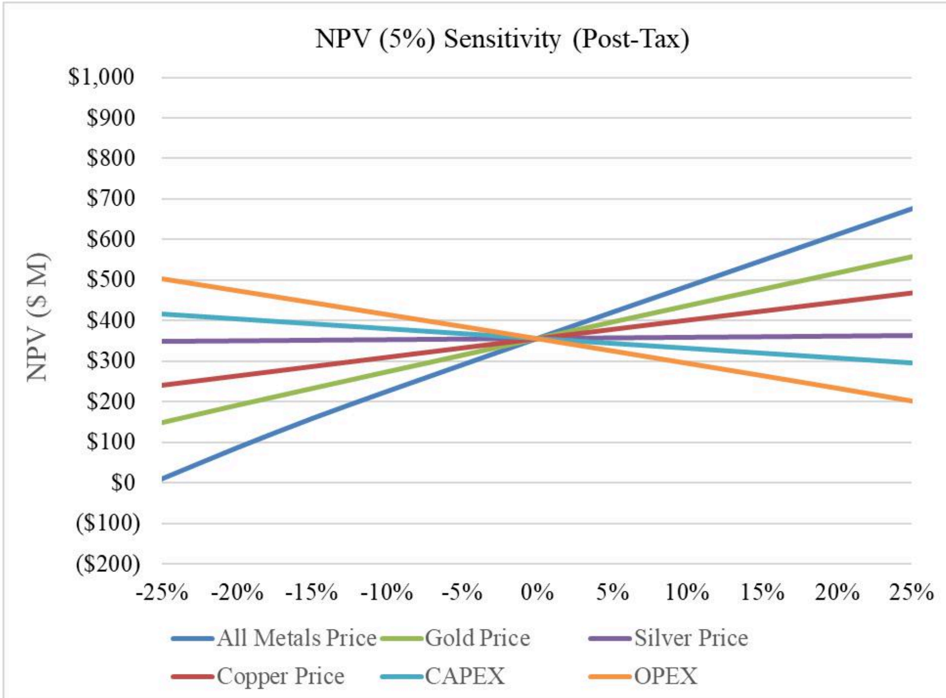

Figure 19.3: Post Tax NPV Sensitivity

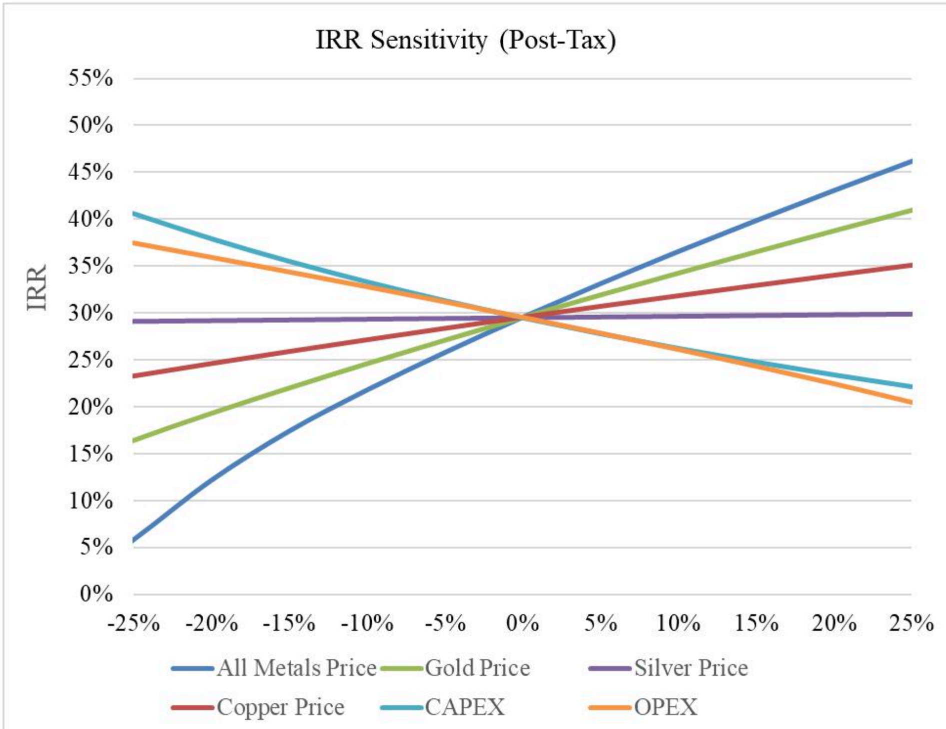

Figure 19.4: Post Tax NPV Sensitivity

A sensitivity analysis on metals pricing indicates additional potential for this Project at higher metals pricing, Table 19.9. Additionally, the sensitivity indicates the robustness of the Project with positive economic outcomes at reduced metals pricing.

Table 19.9: Metal Price Sensitivity							
Metal Pricing		Pre-Tax			Post-Tax		
Gold Price	Copper Price	NPV	IRR	Payback	NPV	IRR	Payback
Au/oz	Cu/lb	$M	%	Years	$M	%	Years
$1,300	3.80	$35	8.1%	5.55	($13)	3.8%	6.98
$1,700	3.90	$240	23.0%	2.71	$177	18.4%	3.44
$2,100	4.10	$459	36.0%	1.73	$356	29.5%	2.12
$2,500	4.30	$678	47.6%	1.37	$532	39.4%	1.63
$3,000	4.50	$945	60.4%	1.10	$745	50.3%	1.31

20.0 ADJACENT PROPERTIES

The text of this Section remains substantially unchanged from the Gustavson Associates report, “S-K 1300 Technical Report Summary CK Gold Project”, dated Dec. 1, 2021.

There is no information from adjacent properties that is material to the CK Gold Project. There are no adjacent properties requiring any disclosure. The area is a historical mining district, however the QPs are not aware of any mineral exploration occurring on adjacent properties. The proximity and similarities of these historic copper-gold deposits does not, on its own, indicate the CK Gold Project should be similarly mineralized.

Approximately two miles to the south of the property is an aggregate quarry that has no material impact to the Project.

21.0 OTHER RELEVANT DATA AND INFORMATION

21.1 AGGREGATE PRODUCTION

In addition to metal concentrate sales from operations, there is also potential to sell granite/granodiorite waste rock to local construction companies as feed material for aggregate production. The material considered for aggregate feedstock has been sampled from existing exploration holes and is representative of the rock selected for aggregate production and rail ballast. Analysis shows that the material is suitable for aggregate production. Production parameters are shown in Section 21.2.

21.2 AGGREGATE MARKET STUDY

The U.S. Gold CK Gold Property is located in southeast Wyoming approximately 18 miles east of Cheyenne in the southern Laramie Mountains. The area is attractive for the quarrying of granite for use as construction aggregate. In the state of Wyoming there are currently three permitted operating granite quarries. These three quarries are located within four miles of the project site location. On average, in recent years, the two operational pits in the area produced a total of approximately 2.9 million tons of granite annually. Having conducted extensive testing on the rock quality under the supervision of Mountain Plains Consulting and associates, aggregate specialists, a study was commissioned by Burgex and completed in August 2024.

The aggregates industry business cycle reacts to levels of activity within commercial and residential construction markets, in public infrastructure projects, as well as other types of construction. Local demand for aggregate is driven by use in infrastructure projects in the Cheyenne metropolitan area. Due to projected low population growth in Wyoming and shortfalls in tax revenue available from the energy sector, demand for aggregate for road and construction projects in southeast Wyoming may be very limited and be met with significant established competition by the currently permitted quarries. The greatest demand for aggregate from the site would be to the south in Colorado along the Northern Front Range where there are multiple metropolitan districts that are in an active phase of population and economic growth. Significant increases in population and employment have been forecasted to varying degrees within the multi-county 100 mile stretch of Interstate-25 (I-25) that links Cheyenne, Wyoming to these high growth areas in Colorado. The proximity of the Project site to northern Colorado's Front Range urban corridor provides a geographically favorable conduit for materials that could be produced in a more advantageous tax and regulatory climate for operators.

Interviews with key operators active in the construction industry, specifically concrete and asphalt producers, were conducted to better understand the potential demand from the Project. Demand for hard rock, such as the subject granite, that meets specifications for transportation projects is high in urban areas such as the Front Range. In these areas competing land uses can make it difficult for operators to obtain a land position sufficient for long-term aggregate production. Two construction companies expressed immediate interest in obtaining aggregate materials from the Project. Based on their estimates, it is reasonable to project that the Project could supply 250,000 st of granite aggregate feedstock the first year of production mining and could ramp up to 1,000,000 st of feedstock annually after the third mining production season.

An estimate of the price that could be obtained for the stockpiled granite feedstock material at the project site was developed based on information in the public domain and information provided via interviews with market participants. Major cost components were considered in processing the stockpiled material into a usable end product. This was necessary to determine at what price competing products from other sites can be sold in the market since the subject granite must remain cost-competitive with other local supply sources near Cheyenne. The main cost centers consist of the following: 1) crushing, screening, stockpiling; 2) loading and scaling; 3) hauling to take off point. Prices for delivered products in Cheyenne were used as the benchmark for the price analysis. Based on the data analyzed in the Burgex Study (August 2024), the price for the average granite materials produced at the mine gate would be anticipated to be $25.23 per ton. Costs to process the stockpiled material at the site are estimated to be $11.88 per ton, see Burgex Study Case C, the Contractor Case Table 21.1. Comparing the average price to the average cost indicates the material could sell with a margin of $13.12 per ton from the Project site. Prices correlate well with information obtained locally related to sales prices from nearby quarry operations.

Table 21.1: Aggregate Cost Buildup

Table 7-2: Lease Purchase Results Summary

Assumption	Value	Note
Equipment Cost	$13,126,600	Principle
EPCM	$1,882,800	Year 1
Useful Life (Years)	7	
Lease Term (Years)	3	
Percentage Toward Principle	80%	
Tax	7%	
Annual Lease Amount	$1,312,660	
Total Lease Amount	$3,937,980	
Annual Lease Amount plus Tax	$1,404,546	Years 2 - 4 applied as an operating cost
Total Lease Amount plus Tax	$4,213,639	
Annual Lease Applied to Principle	$1,050,128	
Total Lease Applied to Principle	$3,150,384	
Remaining Balance at End of Term	$9,976,216	Year 5 applied as capital cost
Total Cost of Lease-Purchase	$14,189,855	
NPV	**$130,960,000**	
IRR	**32%**	

22.0 INTERPRETATION AND CONCLUSIONS

22.1 RESULTS

The results of the CK Gold Pre-Feasibility Study indicate that the property contains a Mineral Resource, and a significant portion of that Mineral Resource converts to a Mineral Reserve. The Project has a positive economic outcome given the data, parameters, and estimates outlined in the TRS. Furthermore, there are significant opportunities to pursue beyond the project's focus on the recovery of copper, gold and silver, The potential of the rock set aside to uncover metal containing ore also holds the potential to be an additional source of revenue. While there is a positive economic outcome at the stated metal prices, improvement in metal prices offer opportunity and decreases would have to be substantial for the project to make a loss. Additionally, the potential of the revenue from non-metal bearing rock further protects the viability of the project should metal prices fall precipitously.

22.1.1 Metallurgical Program

Previous test work at SGS in 2008-2010 and at prior facilities concluded that flotation would be the most appropriate flowsheet to recover copper, gold, and silver into a high value concentrate. The work at KCA (Reno, USA) and Base Metals Laboratory (Kamloops, Canada) has confirmed this. The most recent test work program concluded at BML in September 2022 and the overall body of testwork is now judged to be of suitable depth and quality to act as a valid reference for the Pre-Feasibility Study process design.

Three composites, each 200-300 kg, were prepared for test work, namely a High-Grade Oxide composite (from Hole 4), an Oxide Composite from holes 1-3 plus 5-7, and a Sulfide composite from holes 1-7. Between the oxide and sulfide zones is a narrow band of "mixed" material. As this only represented a small component of the drill core it was included in the sulfide composite. However, as results subsequently showed, the impact was significant. The mineralogy indicated 10-15% of the copper minerals in this "sulfide" composite were not sulfide. A second sulfide composite was prepared from core more remote from the mixed zone and tested at BML in July 2021. This resulted in significantly better copper, gold, and silver recoveries, as shown in Section 10.

Sub-samples of core from each composite were provided to Hazen Research in Denver for comminution test work. This showed the material to be of medium hardness but relatively competent. This supports the selection of a SAG-Ball-Pebble crusher grinding circuit. A primary grind P_{80} of 90 µm appears to be close to optimal.

Open circuit flotation of the High-Grade Composite was successful at KCA, providing high (for an "oxide") recoveries of copper (55%), gold (69%) and silver (40%) to a 25% Cu flotation concentrate. Locked Cycle Tests at BML confirmed these results.

Flotation of the Oxide Composite proved to be more challenging. The mineralogy of this composite showed an abundance of copper minerals, such as chrysocolla, that are non-floating. None the less, flotation was moderately successful in that open circuit rougher and cleaner tests produced a low-grade but high value copper concentrate, 10-15% Cu, that contained over 150 gpt gold and 100 gpt silver. This material constitutes about 6-8% of the deposit. The mine plan could see this material treated on a campaign basis or combined with sulfide. Locked Cycle Tests at BML produced concentrates with 7.9%Cu, over 250 gpt Au, over 200 gpt Ag and a gold recovery of 60%.

The sulfide zone constitutes the majority of the deposit. LCTs on two sulfide composites gave high recoveries of copper (82-88%) and gold (67-74%) to a 21-25% Cu, 76 gpt Au, 82 gpt Ag concentrate. Silver recovery was 59%.

The mineralogy showed that some non-sulfide copper minerals were present in the first sulfide composite, and this had a negative impact on the flotation recoveries. As a result, seven variability samples were selected for test work in the PFS phase. In addition, seven variability oxide samples were tested. These tests were summarized in Table 10.29. With less non-sulfide copper, the copper recovery was over 80%. Gold and silver recoveries showed significant variation. The test data to date does not show any correlation between metallurgical performance and the regrind P_{80} in the range 20-40 µm. Finer regrind needs to be investigated as well as flotation test work on two new low grade composites before the issue of an FS.

Rougher tests investigated the primary grind P_{80} over a range of 50-150u. This and earlier work had concluded the optimum to be between 85u and 100u and most of the testwork has been carried out in that range.

22.2 SIGNIFICANT RISKS

Economic Risk

There is no guarantee that metal prices will continue to support adequate revenues to cover the cost of mining and processing. Additionally, the consumer items, manpower, energy, water, capital equipment could potentially increase to the point that profitable operations would be jeopardized. At appropriate stages the company will investigate securing off-take agreements and contracts for cost items that will protect the viability of the Project in the long-term. Economic risk can be reduced through pursuing additional revenues from potential non-metal rock sales.

Resource Risk

While the resource has been extensively drilled and tested and the nature of the mineralization consistent and well understood, there is a risk that the contained metal in the resource may have been misestimated, that the metallurgical performance is not fully representative of the whole rock mass, and the reported values may not be extracted.

Metallurgy & Processing Risk

In general terms, the Project can be classified as low risk, as it uses a conventional, well-proven process flowsheet with no unusual and/or developmental equipment. The market for a clean copper/gold flotation concentrate is global and relatively stable (subject to macroeconomic/global trends).

Although a metallurgical benefit is likely, the Project has purposely avoided using cyanide to recover additional gold as this approach helps to maintain a responsible social and environmental footprint. Power consumption is low relative to other processing routes (e.g., Hydrometallurgical) and water consumption has been minimized through the application of relatively complex and expensive dewatering technology prior to tailings storage.

However, some metallurgical/process risks remain:

- Head Grades of composites used for SGS, KCA, and some BML testwork programs have tended to be higher grade than the PFS reserve mineralization grades. To address this, a short program of work was appended to the program, to provide additional low-grade data points. These are incorporated into the metallurgical models for the FS.

- Additional variability test work using spatially diverse low-grade composites would help to reinforce the metallurgical models.

- The comminution database is relatively small compared to more complex geometallurgical projects. However, ore hardness is not excessive and results to date have been in a relatively narrow range, indicating straightforward grinding performance. A simple jaw crusher + SAG mill circuit offers a low-capital cost option for comminution, but more SAG mill sizing data would be preferable. Additional SMC and Drop weight testing is recommended.

Future flotation test work should include various combinations of sulfide-oxide ore types. Separate campaigns for oxide and mixed zone material, particularly the high-grade oxide may be required.

The tailing filtration plant is a large, capital-intensive area of the flowsheet. The sizing of such machinery is critical to the project success and as such, more detailed analysis is recommended, perhaps in conjunction with a preferred vendor. Several vendors supply suitable pressure filtration equipment although sizes do vary. Confirmatory testwork at a vendor’s facility would be a helpful de-risking exercise.

The selection and sizing of the jaw crusher, mills, flotation equipment and tailings filters will be the subject of detailed optimization discussions with vendors during the Feasibility and Detailed Engineering phases.

Operations Risk

There are many potential operational risks ranging from the inability to hire, train and retain workers, and professionals necessary to conduct operations, to poor management and exceptional weather events or climate change that could negatively impact operations. While similar operations are conducted in the vicinity and there is no reason to believe these risk factors cannot be eliminated, they still exist. However, there are many working open pit mining and processing operations currently exist in more remote and extreme environmental conditions.

The Red Canyon water source is currently in exploration. The first pilot well drilled delivered promising water volume at a reasonable drill depth. All agreements are in place for water usage and Right of Way. The second well drilled at production size did not hit the water source at the same depth as the pilot well due to hole deviation. Another well has been drilled from a new collar and this well encountered a significant transmissive sand layer in excess of 250-ft and work continues so water quality and pump tests can be completed. The Red Canyon water source location is more favorable than other sources previously considered, but there is a risk of access to the volume of water required until the well field testing is completed.

Environmental Risk

Environmental risks include:

- Community relations: Neighboring residents and landowners could have concerns about real or perceived negative environmental impacts such as traffic, noise, dust emissions, and visual effects. Other stakeholders could have similar concerns due to the proximity of the Project to the Curt Gowdy state park. The Project’s stakeholder engagement has focused on impact assessment (as part of the Industrial Siting permitting process), local project benefits, and impact mitigation measures.

- Permitting: Timelines for receiving water permits from the Wyoming Department of Environmental Quality could be longer than expected due to potential public opposition and legal challenges from activist organizations.

- Many of the Project's surface water management ponds are sized for 10-year or 100-year storm events. The current permitting does not allow for discharge of water from these ponds without testing and proof that water chemistry is at a quality that can be discharged. Additional consideration of surface water management needs to be addressed in following phases. Those options could be proactive treatment options, strategic increase in the size of certain ponds to allow for additional storage, additional pumping capacity to account for extreme precipitation events, or reviewing process water balancing to ensure storage space is available when needed.

- Regulatory: Changes in regulations, including potentially the definition of Waters of the US and possible changes in regulation of dust emissions, could affect project design and permitting. These risks would be mitigated by starting construction prior to the expiration in February 2026 of the current US Army Corps of Engineers' Approved Jurisdictional Determination of Waters of the US, and by proceeding with the air quality permitting under the current regulations.

22.3 SIGNIFICANT OPPORTUNITIES

Resource Expansion

U.S. Gold has focused on the value proposition surrounding the known resource as a prime motivator to create value for its shareholders. The company realizes that additional mineralization continues at depth, evidenced by exploration holes bottoming out in economic grades, as well as opportunities to expand the existing resource to the southeast. Outreach to the University of Wyoming and reliance on experts is aimed at a better understanding of the origins and genesis of the resource. Pursuing such information and prudent expenditures toward exploration as company valuation increases or revenue streams allow will allow resource expansion to be pursued.

Metallurgy & Processing

Flotation test work continued through 2024. Some further improvement and optimization may still be possible for the oxide and mixed material types.

Cyanide leaching of the high-grade flotation tailings gave high extraction of gold and silver. This, together with possible leaching of scavenger concentrates and cleaner tailings may represent a future opportunity. However, the use of cyanide is a potentially controversial topic within the local community and may trigger additional regulatory reviews and increased facility costs. Cyanide alternatives for metal recovery could also be investigated, however the company has not chosen to pursue non-mainstream technologies up to this point.

Filtration of tailing slurry is an expensive and operationally complex area of the plant. A detailed study of this area, together with examination of alternate dewatering strategies would be very beneficial to the Project.

The process plant layout was improved as part of this PFS, with a more compact plant (and building) layout as a result. However, further costs can likely be saved through additional value engineering.

Several new flotation technologies, such as Jameson cells are currently being tested by U.S. Gold and will be evaluated in the Feasibility Study.

Aggregate Production

Work to date has established that the non-mineralized rock is almost certainly in large part an excellent source of aggregate. Additionally, a market study (Section 21.1) suggests that the local demand could accommodate an additional source of supply and this has been confirmed anecdotally by a series of inquiries and from potential consumers to U.S. Gold. This is further supported by the August 2024 Burgex study that summarized work by aggregate specialists and conducted a market study. There is a sound rationale that further value can be obtained from what would otherwise be waste material at the Project, which has already undergone the cost of mining as part of the gold and copper mining operations. Beyond the potential to increase the revenue from the rock mined and sold as aggregate or aggregate feedstock, there are benefits to reducing the waste stored at site, such as the reduced footprint of areas to be reclaimed.

Reduction in Closure Costs

To help increase the local area's long-term water storage capacity, discussions have begun with the Cheyenne Board of Public Utilities about converting the post-mining open pit into a water storage reservoir. The hydrogeological and geochemical study results to date indicate that the pit wall permeability will be low enough overall to contain water with no significant leakage, and the pit wall rock will be geochemically stable enough to preserve water quality to applicable standards. Assuming the ongoing environmental studies confirm these findings, U.S. Gold intends to put forth the concept of converting the pit to a water storage reservoir as the preferred closure concept. At the end of mining, water could be transferred from external sources to the new reservoir to meet the local area's water storage needs. If further studies identify significant obstacles to this pit closure concept, alternative pit closure concepts can be evaluated, or the current closure plan can be implemented. A post-closure monitoring plan will be implemented to verify that closure objectives are met, including the physical and chemical stability of the closed facilities.

23.0 RECOMMENDATIONS

23.1 PROJECT ADVANCEMENT

The project economics reviewed in Section 19 show this project is robust enough to advance to feasibility stage. The recommendations listed below would further advance the understanding of the property and provide the necessary rigor.

23.2 PROJECT DEVELOPMENT

To further assess the viability of the Project and to feed into a more accurate and comprehensive assessment, plans for an EPCM strategy, construction, and operations to support project development should be created. The development of a detailed owner's team for development, contracting strategy and transitional plan to operations should be identified.

23.2.1 Deposit Understanding

As indicated in Section 6 additional drilling should be performed to solidify understanding of the Copper King fault and mineral deposit model. However, the density of drilling and the distribution of metal values suggest a high level of confidence in the stated reserves.

23.2.2 Future Metallurgical Test Work

The geometallurgical models prepared for the Pre-Feasibility Study highlight recovery relationships with head grade and oxidation level. Additional variability testing, together with larger scale work on lower grade samples would be useful.

There is some test work evidence that suggests a finer regrind might be beneficial and this may be worthy of investigation in the Feasibility Study.

Specialty test work with the vendors of tailings and concentrate filtration, including the oxide and sulfide components.

Specific test work on the mixed ore zone.

23.2.3 Ore Processing

There are indications that treatment of tailings to enhance gold recovery could be considered but such additional treatment could come with significant environmental concern, and it is recommended that while the opportunity be assessed, the current flowsheet be maintained.

23.2.4 Design And Engineering

Additional discrete studies should be pursued prior to feasibility to improve and validate capital costs, pursue any potential improvements in process costs or increased recovery. Samuel Engineering would recommend:

- Performing coarse ore floatation test work which, if positive, could reduce operating costs and capital costs.
- Complete a Blasting and Run of mine material hardness, abrasion, and sizing study to confirm primary crushing equipment sizing and type, and perhaps improve the downstream comminution circuit
- Complete the test work on the Jameson cell processing option as a possible recovery improvement option

During the feasibility study additional engineering and should be performed to:

- Update engineering design completion and materials quantities to support S-K 1300 Feasibility Study SME Reporting Guidelines
- Further optimize site layouts and associated civil costs
- Revisit building construction concepts (alternatives like lightweight steel/fabric materials) to further reduce costs
- Final optimization of general arrangements with final (ready to purchase) equipment selections
- Improve the bulk material estimate by proving quantity take offs for major piping and electrical runs,
- Process area benches should be reviewed to balance site earthworks and potentially reduce civil construction costs
- Update major equipment specifications and pricing making them 'Ready for Purchase' at the end of feasibility
- Complete more design definition effort on the fresh water and surface water system to find ways of reducing capital
- Finalize concentrate off-take agreements (MOU) and consider alternative concentrate transport options to smelter

23.3 ENVIRONMENTAL, PERMITTING, AND SOCIAL

Following is a summary of the Environmental, Social, and Permitting recommendations:

- Continue activities needed to obtain the required state and local permits.
- Continue project information disclosure and consultation with local stakeholders, especially focusing on project impact assessment, local project benefits, and impact mitigation measures.
- Conclude the power supply agreement.
- Identify and secure a potential alternative backup water supply source.
- Additional hydrogeologic assessment will need to be performed to determine potential impacts of the Red Canyon well source.
- Continue engagement with the City of Cheyenne regarding the potential post-mining conversion of the pit to a water storage reservoir serving the city.
- Develop and implement a Project Environmental Management System (EMS) consisting of site-specific plans and procedures governing the environmental management of project activities causing potential environmental impacts during construction, operations, closure and post-closure.

24.0 REFERENCES

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25.0 RELIANCE ON INFORMATION PROVIDED BY REGISTRANT

Table 25.1 provides a detailed list of information U.S. Gold (Registrant) provides for matters discussed in this Technical Report Summary (TRS).

Table 25.1: Information provided by U.S. Gold

Category	TRS Section	Reliance
Legal Matters	Section 3 Property Description and Location	Information and documentation regarding mineral titles, surface land agreements, current permitting status, royalties, and other agreements provided by U.S. Gold
General Information	Section 4 Accessibility, Climate, Local Resources, Infrastructure and Physiography	Physical information about the Project was provided by U.S. Gold. Information consisted of consultant reports, and correspondence with U.S. Gold.
General Information	Section 5 History	Historical data provided by U.S. Gold, primarily previous Technical Reports
Technical Information	Section 6 – Geological Setting, Mineralization, and Deposit	Various public and consultant reports. Dworian MSc thesis.
Technical Information	Section 7 – Exploration	Historical project reports and exploration data
Section 8	Sample Preparation, Analysis, and Security	Consultant reports, Hard Rock Mining Annual reports.
Technical Information	Section 13.2 Geotechnical	"Recommended Prefeasibility - Level Geotechnical Slope Designs for the Copper King Open Pit" Authored by Piteau Associates and provided by U.S. Gold.
Technical Information	Section 13.2 Hydrology	Neirbo Hydrology Report provided by U.S. Gold. Dahlgren Water Supply and Yield Analysis Report for the CK Gold Deposit.
Technical Information	Section 15	PFS level site plan and facility design and report by Trihydro and TGI. Mine operating permit application, Industrial Siting, WYPDES and air permits.
Economic Information	Section 16	Marketing memo prepared by Andy Holloway and Mike Mason. Confidential concentrate sale term sheet.
Environmental Matters	Section 17	Pre-permitting work done provided by U.S. Gold. Mine operating permit application, Industrial Siting, WYPDES and air permits.
Commitments to local groups and individuals	Section 17	Pre-permitting work done provided by U.S. Gold. Mine operating permit application, Industrial Siting, WYPDES and air permits.