Whistler Gold-Copper Project

S-K 1300 Technical Report Summary and Initial Assessment with Economic Analysis

Alaska, United States of America

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Prepared for:

U.S. GoldMining Inc.
1830-1030 West Georgia Street
Vancouver, British Columbia, V6E 2Y3, Canada

Prepared by:

Ausenco Engineering Canada ULC
1050 West Pender Street, Suite 1200
Vancouver, British Columbia, V6E 3S7, Canada

Report Contributors:

Moose Mountain Technical Services
#210 1510-2nd St. North
Cranbrook, British Columbia, V1C 3L2, Canada

Date and Signature Page

This technical report summary (the TRS), entitled "*Whistler Gold-Copper Project, S-K 1300 Technical Report Summary and Initial Assessment with Economic Analysis, Alaska, United States of America*" is current as of March 2, 2026, and has been prepared by:

Qualified Person or Consulting Firm	Responsible for the following sections	Signature	Date
Ausenco Engineering Canada ULC	1.1, 1.2, 1.9, 1.12, 1.13, 1.14, 1.15, 1.16, 1.17,1.18, 2, 9.2, 9.3, 10, 14, 15, 16, 17, 18.1, 18.2.1, 18.2.2, 18.2.4, 18.2.5, 18.2.6, 18.2.7, 18.2.8.2, 18.2.9, 18.2.10, 18.3.1, 18.3.2, 18.3.4, 18.3.5, 18.3.6, 18.3.7, 19, 22.1, 22.3, 22.6, 22.7, 22.8, 22.9, 22.10, 22.11, 22.12, 22.13.1.2, 22.13.1.5, 22.13.1.6, 22.13.1.7, 22.13.2.3, 22.13.2.4, 23.1, 23.4, 23.6, 23.7, 23.8, 24, 25.2, and 25.3	"signed"	March 19, 2026
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1 EXECUTIVE SUMMARY

1.1 Introduction

This Technical Report Summary (TRS) was prepared for U.S. GoldMining Inc. (U.S. GoldMining), an indirect subsidiary of GoldMining Inc. (GoldMining) (collectively the "Company"). U.S. GoldMining holds the rights to the Whistler Gold-Copper Project, located 170 kilometers (km) northwest of Anchorage, Alaska. The company is focused on the development and advancement of the Whistler Gold-Copper Project (the Project). There are no active or former major mining operations on the Property.

U.S. GoldMining is listed on the Nasdaq Stock Market as USGO (Nasdaq: USGO), while GoldMining is listed on the Toronto Stock Exchange as GOLD (TSX: GOLD). As a result, U.S. GoldMining is a registrant with the United States Securities and Exchange Commission (SEC) and must comply with subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations of Regulation S-K (S-K 1300). Similarly, GoldMining is a reporting issuer in Canada and must comply with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.2 Terms of Reference

The purpose of this TRS is to report the technical and economic results and updated mineral resource estimate (MRE) for the Project. This TRS was prepared to support the press release titled "*U.S. GoldMining Announces Positive Preliminary Economic Assessment for Whistler Gold-Copper Project, Alaska*" dated March 2, 2026.

1.3 Property Description

The Project is a gold-copper exploration project located in the Yentna Mining District of Alaska, approximately 170 km northwest of Anchorage. The Project comprises 377 State of Alaska mining claims covering an aggregate area of approximately 21,732 hectares (ha). The center of the property is located at 152.566° longitude west and 61.983° latitude north. The Project is located in the drainage of the Skwentna River. The Whiskey Bravo gravel airstrip established adjacent to the Skwentna River is compliant for wheel-based aircraft up to DC-3s. A 24-person camp is equipped with diesel generators, a satellite communication link, tent structures on wooden floors and several wood-frame buildings. Although chiefly used for summer field programs, the camp is winterized.

1.3.1 Mineral Tenure

Rights to the Project were acquired by GoldMining, through its subsidiary U.S. GoldMining, formerly named BRI Alaska Corp., in August 2015 pursuant to an Asset Purchase Agreement (the "Asset Purchase") with Kiska Metals Corporation (Kiska) in exchange for the issuance of 3,500,000 common shares in the capital of GoldMining as disclosed by news releases of GoldMining on July 21 and August 6, 2015. The Project is subject to three underlying agreements, which were assigned to U.S. GoldMining under the transaction.

1.3.2 Royalties and Encumbrances

The first underlying agreement is a Royalty Purchase Agreement between Kiska Metals Corporation, Geoinformatics Alaska Exploration Inc. and MF2 LLC. (MF2), dated December 16, 2014. This agreement grants MF2 a 2.75% net smelter return (NSR) royalty over all 377 claims and extending outside the current claims over an Area of Interest defined by the maximum historical extent of claims held on the Project. The MF2 royalty was subsequently assigned to Osisko Mining (USA) Inc. (OM). U.S. GoldMining can buy back 0.75% of the 2.75% NSR royalty for a payment of US$5,000,000 to OM. Pursuant to a subsequent assignment agreement dated January 11, 2021, the buy-back right was conveyed to Gold Royalty Corp.

The second underlying agreement is an earlier agreement between Cominco American Incorporated and Mr. Kent Turner (whose rights and obligations thereunder were assumed by U.S. GoldMining) dated October 1, 1999. This agreement concerns a 2.0% net profit interest (NPI) to Teck Resources, which was since purchased by Sandstorm Gold, in connection with an Area of Interest specified by standard township subdivision. Sandstorm Gold was acquired by Royal Gold on October 20, 2025.

The third underlying agreement is a royalty agreement dated January 11, 2021, between U.S. GoldMining and Gold Royalty Corp., pursuant to which Gold Royalty Corp. holds a 1% NSR royalty covering the Project.

1.3.3 Surface Rights

Under Alaska Statute AS 38.05.255, the surface uses of land or water included within a state mining location that the owners, lessees, or operators of the location may undertake by virtue of such location are (a) limited to those necessary for the prospecting for, extraction of, or basic processing of minerals and (b) shall be subject to reasonable concurrent uses (Stoel Rives, 2025).

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.4.1 Accessibility and Climate

The Project is in the Alaska Range approximately 170 km northwest of Anchorage and 76 km west of the township of Skwentna. Access to the project area is by fixed wing aircraft to a gravel airstrip located adjacent to the Whistler exploration camp. The project area is between regions of maritime and continental climate and is characterized by severe winters and hot, dry summers. Annual precipitation ranges from 500 to 900 millimeters (mm). Winter snow accumulation usually begins in October and by mid-to-late May the snow has melted sufficiently to allow for fieldwork.

1.4.2 Local Resources and Infrastructure

The nearest public infrastructure for the Project is the town of Petersville, located approximately 106 km east of Whistler; Petersville is connected to Anchorage by an all-weather road and highway. The Project is supported by a 24-person, all-season camp located on the banks of the Skwentna River approximately 2.7 km from the Whistler Deposit and 22 km from the Island Mountain prospect. The camp is connected to the Whistler Deposit by a 6-km access trail.

1.4.3 Physiography

The project is in the drainage of the Skwentna River that forms a large network of interconnected low elevation U-shaped valleys cutting through the rugged terrain of the southern Alaska Range. Elevation varies from about 400 meters above sea level (masl) in the valley floors to over 5,000 masl in the highest peaks.

1.5 History

Mineral exploration in the Whistler area was initiated by Cominco Alaska Inc. in 1986 and continued through 1989. During this period, the Whistler and the Island Mountain gold-copper porphyry occurrences were discovered and partially tested by drilling. In 1990, Cominco's interest waned and all cores from the Whistler region were donated to the State of Alaska. The property was allowed to lapse. In 1999, Kent Turner staked 25 State of Alaska mining claims at Whistler and leased the property to Kennecott. From 2004 through 2006 Kennecott conducted extensive exploration of Whistler region, including geological mapping, soil, rock and stream sediments sampling, ground induced polarization (IP), the evaluation of the Whistler gold-copper occurrence with 15 core boreholes and reconnaissance core drilling at other targets in the Whistler region totaling 12,449 m. Over that period Kennecott invested over US$6.3 million in exploration.

From 2007 through 2008, Geoinformatics drilled thirteen holes for 6,027 m on the Whistler Deposit and five holes for 1,597 m on other exploration targets in the Whistler area. Drilling by Geoinformatics on the Whistler Deposit was done to infill the deposit to sections spaced at 75 m and to test for the north and south extensions of the deposit. Exploration drilling by Geoinformatics in the Whistler area targeted geophysical anomalies in the Raintree and Rainmaker areas, using the same basic porphyry exploration model as Kennecott.

Kiska was formed in 2009 by the merger of Geoinformatics Exploration Inc. and Rimfire Minerals Corporation to advance exploration on the Project. The rights to the property were acquired by Geoinformatics from Kennecott in 2007, subject to exploration expenditures totaling a minimum of US$5.0 million over two years, two underlying agreements, and certain back-in rights retained by Kennecott to acquire up to 60% of the project. In September 2010, Kennecott's back-in right was extinguished after the completion and review of a geophysical and drilling program (the "Trigger Program") whose technical direction was guided by Kiska and Kennecott. From that time forward, Kiska continued to explore the project and completed a total of 48,498 m of drilling, several large geophysical surveys, and an updated Whistler Deposit resource estimate, for a total expenditure of US$29.4 million. Kiska's primary objective was to explore the entire project area and test porphyry targets other than the Whistler Deposit, including Raintree West and the Island Mountain Deposit.

1.6 Geology and Mineralization

Alaskan geology consists of a collage of various terrains that were accreted to the western margin of North America because of complex plate interactions through most of the Phanerozoic. The southernmost Pacific margin is underlain by the Chugach–Prince William composite terrain, a Mesozoic-Cenozoic accretionary prism developed seaward from the Wrangellia composite terrain. It comprises arc batholiths and associated volcanic rocks of Jurassic, Cretaceous, and early Tertiary age.

The Alaska Range represents a long-lived continental arc characterized by multiple magmatic events ranging in age from about 70 million years (Ma) to 30 Ma and associated with a wide range of base and precious metals hydrothermal sulfide bearing mineralization. The geology of the Project is characterized by a thick succession of Cretaceous to early Tertiary (ca. 97 to 65 Ma) volcano-sedimentary rocks intruded by a diverse suite of plutonic rocks of Jurassic to Mid-Tertiary age.

The two main intrusive suites that are important in the Project area are:

1. The Whistler Igneous Suite comprises alkali-calcic basalt-andesite, diorite, and monzonite intrusive rocks approximately 76 Ma with restricted extrusive equivalent. These intrusions are commonly associated with gold-copper porphyry-style mineralization (Whistler Deposit).
2. The Composite Suite intrusions vary in composition from peridotite to granite and their ages span from 67 Ma to about 64 Ma. Gold-copper veinlets and pegmatitic occurrences are characteristics of the Composite plutons (e.g., the Mt. Estelle prospect, the Muddy Creek prospect).

The Project was acquired for its potential to host magmatic-hydrothermal gold and copper mineralization. Magmatic-hydrothermal deposits represent a wide clan of mineral deposits formed by the circulation of hydrothermal fluids into fractured rocks and associated with the intrusion of magma into the crust. Exploration work completed by Kennecott, Geoinformatics, and Kiska has discovered several gold-copper sulfide occurrences exhibiting characteristics indicative of magmatic-hydrothermal processes and suggesting that the project area is generally highly prospective for porphyry gold-copper deposits.

1.7 Exploration

Kennecott completed airborne helicopter geophysical surveys during 2003 and 2004. Results from these airborne surveys were used to interpret geological contacts, fault structures, and potential mineralization in the Whistler, Island Mountain, and Muddy Creek areas. In particular, the airborne magnetic data showed that the Whistler Deposit displays a strong 900 m by 700 m positive magnetic anomaly attributed to the magnetic Whistler Diorite intrusive complex (host to the Whistler Deposit) in addition to a contribution from secondary magnetite alteration and veining associated with Au-Cu mineralization.

Cominco acquired 8.4 line-km of 2D induced polarization (IP) geophysics with results used to target the deposit area with subsequent drilling. From 2004 to 2006, Kennecott completed 39.4 line-km of 2D IP geophysics in the Whistler area. Subsequent lines targeted magnetic anomalies at the Round Mountain, Canyon Creek, Canyon Ridge, Canyon Mouth, Long Lake Hills, Raintree, and Rainmaker prospects. In 2007-2008, Geoinformatics completed 8.8 line-km of 2D IP from six separate reconnaissance lines in the Whistler area targeting airborne magnetic highs. Anomalous results from this survey in the Raintree area led to the Raintree West discovery. In 2009, Kiska completed 224 line-km of a 3D IP geophysical survey. This was executed on two grids (Round Mountain; Whistler Area). This survey reaffirmed that the Whistler Deposit coincides with a discrete 3D chargeability anomaly.

Further exploration was completed by U.S. GoldMining between 2023 and 2025. This work included mapping, rock and silt sampling, drilling, and target definition.

1.8 Drilling and Sampling

A total of 76,493 m of diamond drilling in 267 holes are documented in the Whistler database for drilling on the Whistler Project by Cominco, Kennecott, Geoinformatics, Kiska, and U.S. GoldMining from 1986 to the end of 2024. The drilling is summarized in Table 7-3. Of these drillholes 25,121 m in 57 holes have been drilled in the Whistler Deposit area, 20,803 m in 89 holes have been drilled in the Raintree area, and 15,841 m in 42 holes comprise the Island Mountain resource area. There are 14,727 m in 79 holes in areas outside the three resource areas.

1.9 Mineral Processing and Metallurgical Testing

In 2025, U.S. GoldMining engaged Base Metallurgical Laboratories Ltd. located in Kamloops, (BC, Canada) to perform metallurgical testing. Testwork was conducted on one master composite sample and eight variability samples comprising material from fresh drill cores collected from four drillholes from the 2023 and 2024 drill campaigns.

Laboratory testing included comminution characterization, flotation to recover copper and gold, and cyanidation, with the objective of achieving optimized metal recovery results. Testwork data were provided to Ausenco in June of 2025 by Base Metallurgical Laboratories Ltd.

Comminution data indicated that the Whistler Deposit mineralized material is categorized as very competent with an average Bond Ball Mill Work Index (BWi) of 21.8 kilowatt-hour per tonne (kWh/t) and an average Drop Weight Index (DWI) of 10.9 kilowatt-hour per cubic meter (kWh/m^3) across the eight variability composites.

The results from the 2025 flotation and leach testing support the following conclusions:

- Rougher flotation tests indicate an optimum primary grind size target of P_{80} = 120 microns (µm).
- Flotation testwork included sufficient variability to represent the majority of the head grades present in the current mine plan.
- Recovery to a 25% Cu flotation concentrate is modeled as a function of feed head grades, with average life-of-mine recoveries across the current mine plan of 77.4% Cu, 62.1% Au, and 44.8% Ag.
- An additional flotation tailings leach circuit is expected to extract 74.9% Au and 20.5% Ag from the residual gold and silver present in the flotation tailings. Applying these extractions, together with an assumed 5% soluble metal loss, results in incremental metal recoveries averaging 27.0% Au and 10.7% Ag over the life of mine.
- Overall metal recoveries, combining flotation concentrate and doré products, are expected to average 77.8% Cu, 88.9% Au, and 55.6% Ag over the life of mine.
- Concentrates generated from the recent test work generally contained very low concentrations of deleterious elements, based on a single concentrate sample produced from the final locked-cycle flotation test on the Master Composite. The reported trace element assays do not indicate penalty levels of deleterious elements.

1.10 Mineral Resource Estimate

The Project total MRE includes the Whistler, Raintree West, and Island Mountain deposits and is summarized in Table 1-1 for the base-case cutoff value. The resource is prepared under direction of Independent Qualified Persons (QPs) and in accordance with the United States Securities and Exchange Commission (SEC) regulation S-K subpart 1300 (S-K 1300) for reporting mineral properties (CFR Title 17 § 229.1300-1305).

The resource utilizes pit shells to constrain resources at the Whistler, Island Mountain, and Raintree West gold-copper deposits, as well as an underground potentially mineable shape to constrain the mineral resource estimate for the deeper portion of the Raintree West deposit. The current estimate uses metal prices of US$2,750/ounce (oz) gold price, US$4.35/pound (lb) copper and US$30/oz silver, updated recoveries, smelter terms and costs, as summarized in the notes to Table 1-1. Metal prices have been chosen based partially on market consensus research based on mean prices from 2025 and forecast up to 2026 for long term prices. The metal prices chosen also considered the spot prices and the three-year trailing average prices. For all three metals, the final prices used for this resource estimate are below both the spot metal price and the three-year trailing average, which is considered an industry standard in choosing prices.

The base-case cutoff value for open pit mining is US$13.40/t for all three deposits, which covers the Processing cost of US$11.25/t and the general and administration (G&A) costs of US$2.15/t; this is the marginal cutoff for which mining costs are not included. Cutoff values for underground mining are based on Processing costs plus US$17.10/t for underground bulk mining, to define the marginal cutoff NSR grade. There has been drilling in 2023 and 2024 which resulted in updated geological modelling, resource estimation parameters and an updated resource estimate.

For the mineral resource cutoff value determination, a 3.0% NSR royalty was assumed. This is derived from the sum of a 2.75% royalty to MF2 plus a 1% royalty to Gold Royalty Corp., with an assumption that U.S. GoldMining can negotiate a buy-back of a 0.75% NSR, for a net 3.0% NSR, as is customary to occur for similar project developments. In preparing the resource estimate herein, a sensitivity analysis has also been conducted by the author. Based on such analysis, utilizing a higher 3.75% NSR royalty rate in determining a cutoff value would not materially impact the estimates contained herein and would be de minimis (approximately 0.7% differential of total metal in the Whistler pit on a gold equivalent basis).

These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The QP is of the opinion that issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work. These factors may include environmental permitting, infrastructure, sociopolitical, marketing, or other relevant factors.

As a point of reference, the in-situ gold, copper and silver mineralization are inventoried and reported by intended mining method.

Table 1-1: Mineral Resource Estimate for the Whistler Project (Effective date: March 2, 2026)

Class	Deposit	Cutoff Value	ROM tonnage	In-situ Grades					In-situ Metal			
		(US$/t)	(kt)	NSR (US$/t)	AuEq (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (koz)	Au (koz)	Cu (klbs)	Ag (koz)
Indicated	Whistler		284,203	38.74	0.562	0.409	0.154	1.7	5,132	3,740	964,275	15,808
	Raintree West-Pit	13.40	10,332	35.63	0.517	0.420	0.076	4.8	156	128	15,356	1,321
	Indicated Open Pit		**294,535**	**38.63**	**0.560**	**0.410**	**0.151**	**1.8**	**5,287**	**3,868**	**979,631**	**17,129**
	Raintree West-UG	$40 shell	4,619	58.81	0.853	0.713	0.118	5.4	127	106	12,036	795
	Total Indicated	**varies**	**299,154**	**38.94**	**0.565**	**0.414**	**0.151**	**1.9**	**5,414**	**3,973**	**991,667**	**17,924**
Inferred	Whistler		4,967	38.37	0.556	0.433	0.115	1.2	89	69	12,549	192
	Island Mountain		187,283	29.04	0.421	0.376	0.043	0.9	2,535	2,263	178,368	5,299
	Raintree West-Pit	13.40	18,780	37.83	0.548	0.471	0.057	4.3	289	252	19,475	1,927
	Inferred Open Pit		**211,030**	**30.04**	**0.436**	**0.386**	**0.046**	**1.2**	**2,913**	**2,584**	**210,392**	**7,418**
	Raintree West-UG	$40 shell	79,717	55.32	0.802	0.692	0.102	2.7	2,055	1,773	179,964	6,843
	Total Inferred	**varies**	**290,747**	**36.97**	**0.536**	**0.470**	**0.062**	**1.6**	**4,969**	**4,357**	**390,355**	**14,261**

Notes to the MRE:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.
2. The mineral resource for the Whistler, Island Mountain, and the upper portions of the Raintree West Deposits have been confined by an open pit with "reasonable prospects of eventual economic extraction" using the following assumptions:
   - metal prices of US$2,750/oz Au, US$4.35/lb Cu and US$30/oz Ag;
   - payable metal of 94.8% payable for Au, 96.5% payable for Cu, and 88.2% payable for Ag;
   - refining costs for Au of US$7.50/oz, for Ag of US$1.00/oz and for Cu of US$0.065/lb;
   - off-site costs of US$165.65/t;
   - royalty of 3% Net Smelter Return (NSR);
   - pit slopes are 50 degrees;
   - mining cost of US$2.75/t for waste and mineralized material; and
   - processing costs of US$11.25/t, general and administrative costs of US$2.15/t.
3. The open pits at Whistler and Island Mountain use the 150% NSR case, with the upper portion of Raintree West using the 100% NSR case. The lower portion of the Raintree West deposit has been constrained by a mineable shape within a "reasonable prospects of eventual economic extraction" shape using a $40.00/t cutoff value.
4. Metallurgical recoveries are: 87.8% for Au, 75.4% for Cu, and 49.1% Ag.
5. The NSR equation is: NSR ($/t)=(100%-3%)*((Au(g/t)*87.8%*$78.57/g) + (Cu%*75.4%*$3.88/lb*2204.62 + Ag(g/t)*49.1%* $0.77)).
6. The gold equivalent equation is: AuEq=Au + Cu*0.9361 + 0.0055Ag.
7. The specific gravity for each deposit and domain ranges from 2.76 to 2.91 for Island Mountain, 2.60 to 2.72 for Whistler with an average value of 2.80 for Raintree West.
8. Numbers may not add due to rounding.

1.11 Mining Methods

The Whistler deposit is amenable to conventional drill, blast, load, and haul open-pit mining methods. Open pit mine designs, a mine production schedule, and mine capital and operating costs have been developed for the Whistler deposit at a scoping level of engineering. The Raintree West and Island Mountain deposit resources are not included in this mine plan.

The open pit is designed for approximately 16 years of operations, inclusive of one year of pre-production mining, and one year of low-grade stockpile rehandling to the mill after the open pit is exhausted. The Run-of-Mine (ROM) production contained within the designed open pit, summarized in Table 1-2 with a 0.19 g/t gold equivalent cutoff grade (NSR: $13.40/t), forms the basis of the Whistler Mine Plan. These contents are a subset of the Indicated Mineral Resource Estimate described in Section 11; Inferred class resources have been treated as waste.

Table 1-2: Whistler Plan ROM Production Results

Pit Content	Parameter
Mill Feed	211 Mt
Average Gold Grade	0.44 g/t
Average Copper Grade	0.16 %
Average Silver Grade	1.8 g/t
Waste Material	465 Mt
Strip Ratio	2.2

Mill feed quantities and grades include estimates of mining dilution and recovery based on 20 x 20 x 10 m selective block sizes and an additional 3% dilution applied to account for waste edge contracts on the outer edges of the mineralization. This dilution is balanced with an estimated 97% mining recovery.

The crusher will be fed with material from the pit and stockpile at an average feed rate of 40,000 tonnes per day (t/d).

The open pit is split into four phases, targeting highest to lowest economic value between the pushbacks. The first phase will commence near the center of the deposit, where the highest grade of mineralized material and lowest strip ratio will be encountered and the remaining phases targeting progressively higher strip ratios and lower grades.

Mill feed will be sent to the crusher directly north of the open pit, or to the low-grade stockpile next the crusher. This low-grade facility will be reclaimed to the crusher/mill during and at the end of the mine life.

Preliminary estimates indicate that 55% of the pit waste rock is potentially acid generating (PAG). Another 12% of pit waste rock is undefined and PAG overburden. These PAG quantities will be stored subaqueously within the co-disposal storage facility (CDSF), 2 km northwest of the open pit. The remaining waste and overburden are non-potentially acid generating (NAG) and will be stored at waste rock storage facilities (WRSFs) located directly east and west of the pit. Suitable pit waste rock will also be hauled to the CDSF for dam construction, as needed.

Owner run mine operations will include drilling, blasting, loading, hauling, and pit, haul road and pile maintenance functions. Mobile equipment maintenance operations will also be managed by the owner. Technical services functions will also be managed by the owner and will include geology, engineering and surveying. Mining operations will be based on 365 operating days per year with two 12-hour shifts/day. An allowance of 10 days of no mine production has been built into the mine schedule to allow for adverse weather conditions. The open pit is planned to be electrified in Year 1 of the project, with drilling and loading functions becoming partially electrified.

The mining fleet will include electric and diesel-powered rotary drills with a 228 mm bit size for production drilling; down-the-hole (DTH) drills with 160 mm bit size for wall control drilling; 34 m^3 bucket sized electric powered hydraulic face shovels, 22 m^3 bucket sized diesel-powered face shovels, and 24 m^3 bucket sized wheel loaders for production loading; 230 t payload rigid-frame haul trucks and 40-t articulated trucks for construction and production hauling; plus ancillary and service equipment to support the mining operations. In-pit dewatering systems will be established for the pit. All surface water and precipitation in the pits will be handled by diesel-powered pumps.

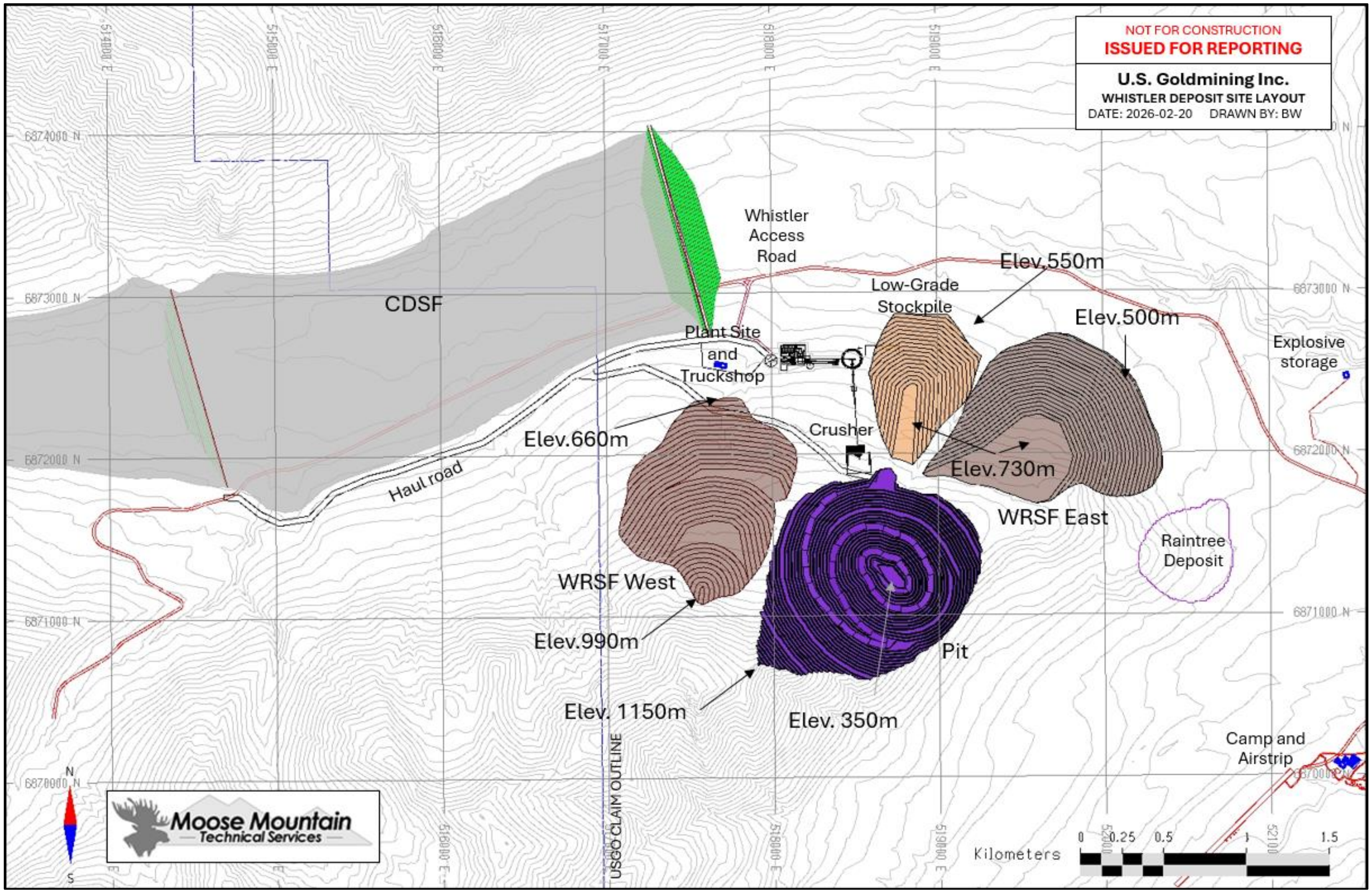
Maintenance on mine equipment will be performed in the field with major repairs to mobile equipment in the shops located near the plant facilities.

Annual mine operating costs per tonne mined range from US$2.05/t to US$3.05/t with a life-of-mine average of US$2.53/t mined.

The initial mine equipment fleet is planned to be purchased via a lease financing arrangement. Expansion and replacement fleet is planned to be traditional capital purchases.

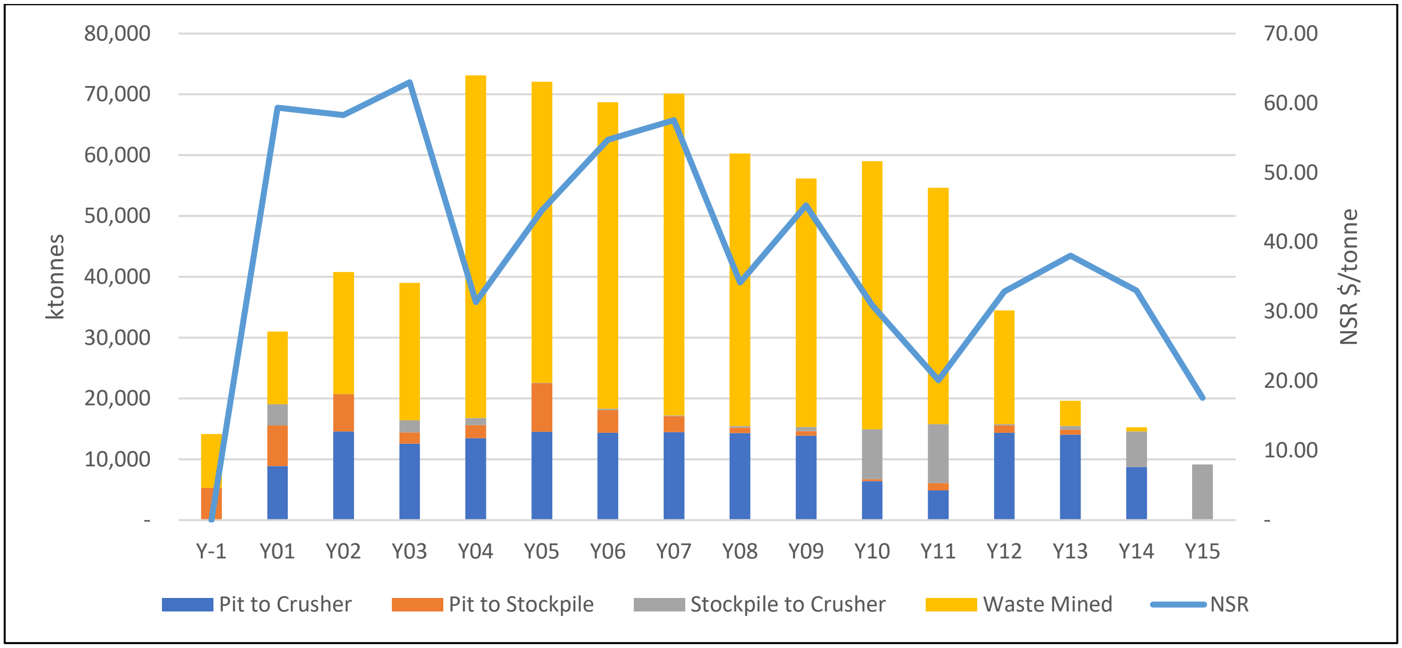
Figure 1-1 shows the mine layout for the pit, stockpiles, and haul roads and Figure 1-2 shows the Whistler Mine Plan Production Schedule.

Figure 1-1: Mine Area General Arrangement



Source: MMTS, 2026

Figure 1-2: Mine Production Schedule, Material Mined and Mill Feed Gold Grades



Source: MMTS, 2026

1.12 Processing and Recovery Methods

The process flowsheet for the Whistler Gold-Copper project was selected based on preliminary metallurgical laboratory testing and preliminary economic modelling. The proposed process plant is designed to treat 40,000 t/d of mineralized material from the Whistler gold-copper porphyry deposit and incorporates conventional, proven technologies for:

- primary crushing of run-of-mine mineralized material and stockpiling
- secondary crushing and screening
- high-pressure grinding roll (HPGR) milling
- ball milling
- copper rougher flotation with regrind
- three-stage cleaning flotation
- copper concentrate thickening and filtration
- gold leaching followed by carbon-in-pulp (CIP) adsorption
- gold desorption, electrowinning, filtration and smelting
- carbon regeneration

- cyanide detoxification
- tailings thickening.

1.13 Infrastructure

1.13.1 On-site and Off-site Infrastructure

The on-site infrastructure required for the Project includes site access roads, WRSFs, stockpiles, infrastructure buildings, diesel storage and distribution, power plant and site electrical distribution, explosives magazine, water management structures, and a CDSFfor storing tailings and PAG waste rock.

The off-site infrastructure required for the project includes the Whiskey Bravo airstrip (existing), power transmission line from Beluga Power Plant, West Susitna Access Road, and ship loading facility upgrades at Port Mackenzie.

The project site is currently accessible by flying into the Whisky Bravo airstrip. The existing exploration camp is located to the east of the airstrip, where 4-wheel drive (4WD) vehicles and side-by-side all-terrain vehicles (ATVs) are available. The site can then be accessed via local exploration roads.

Construction of the WSAR is expected to commence in 2026 or 2027 with completion of the road expected by 2030. Once the WSAR is completed, the Whistler site will be accessible by vehicle from Anchorage or Port Mackenzie. Employees, fuel, reagents, supplies, and concentrate will then be transported via the WSAR.

1.13.2 Co-disposal (Tailings and PAG Waste Rock) Storage Facility

A desktop siting and waste material deposition trade-off study was conducted to evaluate potential sites and disposal methods for tailings and PAG waste rock. Several potential storage sites were identified for slurry and filtered tailings along with PAG waste rock across the site. Ultimately, it was decided to proceed with the co-placement of life-of-mine slurry tailings and PAG waste rock in the CDSF. Slurry tailings (tailings) and PAG waste rock will be permanently stored in the CDSF, located west of the open pit and process plant, while utilizing the natural topography to minimize the need for dam fill material and reduce the overall footprint. The CDSF is designed to hold approximately 214 Mt of tailings and 304 Mt of PAG waste rock. PAG waste rock will be stored subaqueously and/or covered with tailings to eliminate air contact. The primary design objectives for the CDSF are the secure confinement of process tailings, subaqueous and/or tailings cover over the deposition of PAG waste rock to prevent potential acid rock drainage (ARD) and metal leaching (ML), and the protection of regional groundwater and surface water during mine operations and in the long-term post-closure.

1.14 Market Studies and Contracts

No market studies or product valuations were completed as part of this study. Market price assumptions were based on a review of public information, industry consensus, standard practice, and specific information from comparable operations.

Copper concentrates are widely traded and can be marketed directly from producer to smelter or via third-party concentrate trading entities. It is assumed that the concentrate contains negligible deleterious elements that would impact marketability.

The market for gold doré is widely traded and can be marketed domestically or internationally with significant optionality regarding the final customer. It is assumed that no penalties for deleterious elements will be applied to the doré.

Marketing, refining, and transportation costs, along with payability terms, were assumed based on a review of information from comparable recent studies.

1.15 Environmental Studies, Permitting and Plans, Negotiations, or Agreements with Local Individuals or Groups

U.S. GoldMining currently holds multi-year Exploration and Reclamation Permit Number 2778 for Hardrock Exploration – Skwentna River – Yentna Mining District, issued by Alaska Department Natural Resources, Division of Mining, Land and Water in September 2022. The Company has commenced environmental studies and has developed a Stakeholder Engagement Plan and an approved Reclamation Plan.

1.15.1 Environmental Considerations

1.15.1.1 Baseline and Supporting Studies

Several baseline and supporting studies have been completed on the project since 2012, including studies on:

- archaeology
- terrestrial mammals
- aquatic life
- avian survey
- wetlands and soils
- geochemistry
- surface water quality and hydrology.

1.15.2 Permitting Considerations

The Project is located on State of Alaska lands administered by the Alaska Department of Natural Resources (ADNR). The project area is not situated on federal lands, Alaska Native Corporation lands, or tribal trust lands. Mineral tenure is held under state mining claims, and exploration and potential development activities are subject to state regulatory oversight, including authorization through the Application for Permits to Mine in Alaska (APMA) and associated environmental and reclamation approvals. The potential state, federal, and local permit requirements are illustrated in Table 1-3.

Table 1-3: Potential State, Federal, and Local Permit Requirements

Authority	Permit
Federal	
Environmental Protection Agency (EPA)	• Spill Prevention Containment and Contingency (SPCC) Plan
U.S. Army Corps of Engineers (USACE)	• CWA Section 404 Permit (wetlands dredge and fill) • River and Harbors Act (RHA) Section 10 (structures in navigable waters) • RHA Section 9 (dams and dykes in navigable waters-interstate commerce)
Bureau of Alcohol, Tobacco, and Firearms (ATF)	• License to Transport Explosives • Permit and License for Use of Explosives
Federal Aviation Administration (FAA)	• Notice of Landing Area Proposal (existing airstrip) • Notice of Controlled Firing Area for Blasting
U.S. Department of Transportation	• Hazardous Materials Registration
U.S. Fish and Wildlife Service	• Section 7 of the Endangered Species Act, Consultations requiring a Biological Assessment or Biological Opinion
State	
Department of Natural Resources (ADNR)	• Reclamation Plan Approval • Mining License • Temporary Water Use Authorizations
Department of Environmental Conservation (ADEC)	• APDES Water Discharge Permit • Alaska Multi-Sector General Permit (MSGP) for Stormwater • Stormwater Pollution Prevention Plan (part of MSGP) • Sec. 401 Water Quality Certification of the CWA Sec. 404 Permit • Integrated Waste Management Permit • Air Quality Control – Construction Permit • Air Quality Control – Title V Operating Permit • Reclamation Plan Approval • Approval to Construct and Operate a Public Water System
State Office of History and Archaeology (OHA)	• Section 106 National Historic Preservation Act Concurrence
Department of Fish and Game (ADF&G)	• Title 16 Fish Habitat and Passage Permits • Wildlife Hazing Permit
Local	
Alaska Native Corporations	• Land access agreements
Matanuska-Susitna	• Land Use or Development Permits

1.15.3 Social Considerations

The Project is located within the Matanuska-Susitna (Mat-Su) Borough, which contains some Alaska Native Claims Settlement Act (ANCSA) lands owned by the Cook Inlet Region, Inc. (CIRI). No Alaska Native Corporation or tribal lands occur within the project boundary. The broader region is traditionally and contemporarily used by Dena'ina/Tanaina Athabaskan communities, including residents of Tyonek and Skwentna, for hunting, fishing, and travel. These connections are based on customary land use within the surrounding watersheds and access corridors rather than land ownership or jurisdictional authority over the project area (Owl Ridge Natural Resource Consultants, Inc., 2023).

Land use in the vicinity of the project includes active mineral exploration, guided and unguided hunting and fishing, recreational lodges, air-supported access, and historic travel routes associated with early mining activity in South-Central Alaska. These uses occur primarily outside the project footprint but are relevant to permitting and operational planning due to shared access routes, seasonal activity patterns, and regional public interest.

Stakeholder engagement has therefore focused on communities, regional Alaska Native organizations (CIRI and Tyonek Native Corporation), state and federal agencies, recreational and commercial users, and regional trade organizations, with the objective of providing early project awareness, coordinating land use expectations, and supporting future permitting processes. No land tenure conflicts have been identified, and no restrictions on mineral tenure or surface access are currently known that would materially affect exploration activities (Owl Ridge Natural Resource Consultants, Inc., 2023).

1.15.4 Closure and Reclamation Considerations

Closure of the Project will primarily be regulated by the ADNR and Alaska Department of Environmental Conservation (ADEC) under the Alaska Reclamation Act and the Solid Waste Management Regulations. The Act requires that a reclamation and closure plan (RCP) and financial assurance (FA) be provided to the State prior to any mining activity or project development. The RCP details reclamation prescriptions which are designed to minimize or eliminate the risk of pollutants released into the environment. The RCP details conceptual measures and methods to return the site to near pre-mining conditions, protect the environment during reclamation, and support long-term site management.

The project will be closed in two phases: the active reclamation closure phase and the passive post-closure phase. During the active closure phase, closure activities for the mining, process plant, infrastructure, and CDSF will take place. Environmental monitoring will span both the active and passive closure phases, while passive water treatment systems will be considered during the post-closure phase.

1.16 Capital and Operating Cost

The capital and operating costs described in this Initial Assessment (IA) are based on open pit mining operations for the Whistler gold-copper project. The process plant is designed to treat 40,000 t/d of mineralized material, or 14.6 Mt/a, over a mine life of 14.6 years.

1.16.1 Capital Cost Estimate

The capital cost estimate was developed in Q4 2025 to target a level of accuracy of -30% to +50%, which aligns with an Association for the Advancement of Cost Engineering International (AACE International) Class 5 level estimate. The estimate includes mining, processing, on-site infrastructure, off-site infrastructure, project indirects, project delivery, owners' costs, and provisions. The total initial capital costs for the Project are estimated at US$1,278.6 million, including US$56.3 million of capitalized operating costs, and US$213.3 million of contingency. The LOM sustaining costs are estimated at US$381.1 million, while the closure costs are estimated at US$98.7 million. The capital cost summary is presented in Table 1-4.

Table 1-4: Capital Cost Summary

WBS	Description	Capital Cost (US$M)	Sustaining Cost (US$M)	Total Cost (US$M)
1000	Mining	39.7	319.0	358.7
2000	Crushing and Conveyance	120.4	-	120.4
3000	Process plant	354.8	-	354.8
4000	On-site Infrastructure	187.6	14.2	201.8
5000	Off-site Infrastructure	72.6	35.7	108.3
	Total Direct Costs	**775.1**	**368.9**	**1,144.0**
6000	Project Preliminaries	80.5	1.6	82.1
7000	Project Delivery	122.1	-	122.1
8000	Owner's Costs	31.3	-	31.3
	Total Indirect Costs	**233.9**	**-**	**233.9**
	Total Direct + Indirect Costs	**1,009.0**	**368.9**	**1,377.9**
	Contingency	213.3	10.6	223.9
	Subtotal Capital Cost	**1,222.3**	**381.1**	**1,603.4**
	Capitalized Opex	56.3	-	56.3
	Closure Costs	-	-	98.7
	Total Capital Cost	**1,278.6**	**381.1**	**1,755.1**

Notes: Totals may not match due to rounding.

The capital cost estimate is based on budgetary quotations for equipment from recent advanced studies and execution projects, supplemented with Ausenco's in-house database, and informed by Ausenco's experience from similar operations in North America.

The following data were used as the basis of estimate:

- Mining schedules.
- Engineering design by Ausenco, including but not limited to design criteria, equipment lists, and material take-offs (MTOs).
- Budgetary equipment quotes from similar recently completed advanced studies and execution projects.
- Additional data such as Lang factors and indirect costs from similar recently completed studies and projects.

The estimate also adhered to these parameters:

- No allowance was made for exchange rate fluctuations
- No escalation was added to the final estimate
- No growth allowance was included.

1.16.2 Operating Cost Estimate

The total operating costs for the Project are estimated at US$20.82/t or US$4,399.8 million over the 14.6-year mine life. These operating costs do not include pre-production operating costs. The operating cost estimate was developed in Q4 2025 to target a level of accuracy of -30% to +50%, which aligns with an AACE International Class 5 level estimate.

A summary of operating costs is presented in Table 1-5.

Table 1-5: Operating Cost Summary

Cost Area	LOM Total (US$M)	US$/t milled	% of Total
Mining	1,676.7	7.93	43.9
Process	2,325.8	11.00	47.9
G&A	216.1	1.02	4.5
G&A (Other) - Road Toll and Maintenance	181.2	0.86	3.7
Total	**4,399.8**	**20.82**	**100**

Note: Totals may not match due to rounding.

Mine operating costs are built up from first principles assuming an owner managed and operated scenario.

The following was used to determine the project's LOM process operating costs in agreement with the cost definition and estimate methodologies outlined below. This basis considers the development of a process plant designed to treat 40,000 t/d of mineralized material. Process unit operations were benchmarked against similar or comparable processing plants to ensure accuracy of cost estimates.

Assumptions made in developing the process operating cost estimate are listed below:

- Mill is designed to treat 40,000 t/d of mineralized material.
- Process plant operating costs are calculated based on labor, power consumption, and process and maintenance consumables.
- Off-site gold refining, insurance, and transportation costs are excluded, as they are included elsewhere in the financial model.
- Labor rates were sourced from recent execution projects in the region.
  - Workforce will be comprised of local and regional workers.
  - Management and administrative staff will be on a 5/2 rotation (five days in, two days out), whereas process and maintenance staff will be on a 14/14 rotation (14 days in, 14 days out).
  - Management and administrative staff who are not required to be on site will be based out of Anchorage or will work remotely.
- G&A costs were baselined against previous project experience, defined along with specific inputs from U.S. GoldMining.
- No factor for spare parts has been applied to adjust for consumption of fewer spare parts in early years of operation.
- Grinding media consumption rates have been estimated based on the mill feed characteristics.
- Reagent consumption rates have been estimated based on the metallurgical testwork results.
  - Reagents and consumable prices were obtained via email quotes from local vendors.
- Mobile equipment cost includes for fuel, maintenance, and lease price for the equipment.
- The unit rate power cost of US$0.08225/kWh was calculated from published rates by the Chugach Electrical Association (CEA).
- The unit rate fuel cost of US$3.79/gallon based on the regional monthly three-year trailing average published by the Energy Information Administration (EIA).

1.17 Economic Analysis

1.17.1 Economic Summary

The economic analysis is based on 100% indicated resources and 0% inferred resources. The capital and operating cost estimates were developed in Q4 2025 to target a level of accuracy of -30% to +50%, which aligns with an AACE International Class 5 level estimate. The capital cost estimate includes a 21% contingency on the initial capital costs.

The IA has been evaluated using a discounted cashflow (DCF) analysis. Cash inflow consists of annual revenue projections for the Project. Cash outflows such as capital costs, operating costs, taxes, and royalties are subtracted from the inflows to arrive at the annual cashflow projections.

The post-tax NPV at a 5% discount rate ($NPV_{5\%}$), is US$2.04 billion with a post-tax IRR of 33.0%, and an initial payback of 2.1 years. These economic results utilize base-case prices of US$3,200/oz gold, US$4.50/lb copper, and US$37.50/oz silver.

1.17.2 Sensitivity Analysis

A sensitivity analysis was conducted on the base-case pre-tax and post-tax $NPV_{5\%}$ and IRR of the project using the following variables: metal prices, discount rate, total operating costs, initial capital costs, recovery, and head grade. The sensitivity analysis reveals that the project is most sensitive to changes in head grade and commodity prices, and less sensitive to recovery, operating costs, and initial capital costs.

1.18 Conclusions & Recommendations

The study highlights positive economic results based on a conceptual design. Further exploration work is recommended to evaluate the Project's mineral potential. Additional field work, laboratory testwork, and analysis are required prior to advancing to a prefeasibility study (PFS). It is also recommended that the Company initiates environmental baseline studies and engages a permitting consultant along with ramping up engagement with government, local groups, local communities, and regional infrastructure owners. The recommended work is estimated to cost US$68.7 million and is summarized in Table 1-6.

Table 1-6: Recommended Work Programs

PFS Program Component	Total Cost (US$M)
Exploration and Resource Drilling	20.0
Metallurgical Testwork	1.5
Mining Drilling and Testwork	7.6
Process Engineering and Testwork	1.5
Infrastructure Hazard Mitigation	3.8
Infrastructure Geotechnical Program	6.0
Environmental Baseline Programs	15.0
Permitting Consultant	3.0
Government and Community Engagement	3.0
Geochemical Program	2.5
Groundwater and Surface Water Investigations	2.0
Wetlands Mapping and Delineation	0.3
PFS-level Engineering Study	2.5
Total	**68.7**

2 INTRODUCTION

This Technical Report Summary (TRS) was prepared for U.S. GoldMining Inc. (U.S. GoldMining), an indirect subsidiary of GoldMining Inc. U.S. GoldMining holds the rights to the Whistler Gold-Copper Project property, located 170 km northwest of Anchorage, Alaska. The Project location is illustrated in Figure 2-1.

Figure 2-1: Whistler Project Location Map

Source: U.S. GoldMining, 2026

U.S. GoldMining is listed on the Nasdaq Stock Market as USGO (Nasdaq: USGO), while GoldMining is listed on the Toronto Stock Exchange as GOLD (TSX: GOLD). As a result, U.S. GoldMining is a registrant with the United States Securities and Exchange Commission (SEC) and must comply with subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations of Regulation S-K (S-K 1300). Similarly, GoldMining is a reporting issuer in Canada and must comply with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Ausenco Engineering Canada ULC (Ausenco) was retained by U.S.GoldMining to undertake a techno-economic study and to produce an S-K 1300 compliant IA with Economic Analysis on the Project for the Whistler Deposit.

Moose Mountain Technical Services (MMTS) was retained by U.S.GoldMining to support the IA with Economic Analysis by updating the geology, mining, and mineral resource components of the study. The effective date for this TRS is March 2, 2026.

This Report is the second TRS developed for the Project in accordance with United States SEC S-K 1300 regulations. The TRS summarizes an updated Mineral Resource Estimate (MRE) and techno-economic IA. All technical analyses, design information, capital, and operating cost information, permitting and legal assumptions, conclusions and recommendations are consistent between this S-K 1300 TRS and the 2026 NI 43-101 Technical Report that will be published at the same time.

2.1 Terms of Reference

The purpose of this TRS is to report the technical and economic results and updated mineral resource estimate (MRE) for the Project. This TRS was prepared to support the press release titled "*U.S. GoldMining Announces Positive Preliminary Economic Assessment for Whistler Gold-Copper Project, Alaska*" dated March 2, 2026.

All measurement units used in this Report are metric, and currency is expressed in US dollars unless stated otherwise.

The Qualified Persons for the report are listed in Table 2-2. By virtue of their education, experience and professional association membership, they are considered Qualified Person as defined by S-K 1300.

2.2 Qualified Persons

This Report was prepared by Qualified Persons (QPs) from Ausenco and MMTS, both considered third-party firms with mining expertise. Table 2-1 lists the sections of the Report prepared by or contributed to by each firm.

Table 2-1: Third-Party Firms and Section Responsibilities

Third-party Firms	Report Sections
Ausenco	1.1, 1.2, 1.9, 1.12, 1.13, 1.14, 1.15, 1.16, 1.17,1.18, 2, 9.2, 9.3, 10, 14, 15, 16, 17, 18.1, 18.2.1, 18.2.2, 18.2.4, 18.2.5, 18.2.6, 18.2.7, 18.2.8.2, 18.2.9, 18.2.10, 18.3.1, 18.3.2, 18.3.4, 18.3.5, 18.3.6, 18.3.7, 19, 22.1, 22.3, 22.6, 22.7, 22.8, 22.9, 22.10, 22.11, 22.12, 22.13.1.2, 22.13.1.5, 22.13.1.6, 22.13.1.7, 22.13.2.3, 22.13.2.4, 23.1, 23.4, 23.6, 23.7, 23.8, 24, and 25.2, 25.3
MMTS	1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.10, 1.11, 3, 4, 5, 6, 7, 8, 9.1, 11, 12, 13, 18.2.3, 18.2.8.1, 18.3.3, 20, 21, 22.2, 22.4, 22.5, 22.13.1.1, 22.13.1.3, 22.13.1.4, 22.13.2.1, 22.13.2.2, 23.2, 23.3, 23.5, 24, and 25.1

2.3 Site Visits and Scope of Personal Inspection

2.3.1 Sue Bird Site Inspection

Sue Bird, P.Eng., of MMTS, visited the Project site on September 14, 2022, and again on August 6, 2024. During the site visit collar locations at Whistler and Raintree West were validated. The core storage site at both Whiskey Bravo camp and Rainy Pass were visited. The core from each deposit was examined for mineralization with 4 samples for re-assay obtained in 2022 with another 5-sample collected in 2024. Additionally, she observed the condition of the existing camp buildings.

2.3.2 Scott Elfen Site Inspection

Scott Elfen, Ausenco's Geotechnical QP, visited the site on June 26, 2025. The purpose of the visit was to assess the topography, surface geotechnical conditions, and water features of the site and their amenability to support tailings and waste rock structures. Additionally, the QP was to assess site access, the Whiskey Bravo airport, the existing camp, and the overall layout of mining related infrastructure.

2.4 Sources of Information

2.4.1 Information and Data

Sources of information are listed in Section 24, with the sources provided by U.S. GoldMining and its parent company, GoldMining, regarding property ownership and environmental permitting listed in Section 25.

2.4.2 Previous Technical Reports

The previous Technical Report Summary (TRS) was published on October 7, 2024, and had an effective date of September 12, 2024. It disclosed an updated MRE and was prepared by MMTS for U.S. GoldMining.

2.5 Effective Dates

The overall report effective date is March 2, 2026.

2.6 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric, and all currencies are expressed in United States dollars (symbol: US$ or currency: USD) unless otherwise stated. Contained gold metal is expressed as troy ounces (oz), where 1 oz = 31.1035 g. All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-2, and units of measurement are listed in Table 2-3.

Table 2-2: Abbreviations and Acronyms

Abbreviation	Description
AA	atomic absorption spectroscopy
ADNR	Alaska Department of Natural Resources
ALS	ALS Metallurgy Laboratories
APMA	Application for Permits to Mine in Alaska
ARD	acid rock drainage
Au	gold
Az	azimuth
BIF	banded iron formation
BWi	bond ball mill work index
CAD:USD	Canadian-American exchange rate
CDSF	co-disposal storage facility
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CIP	carbon-in-pulp
CoG	cutoff grade
CRM	certified reference material
CWi	Bond crusher work index
DCIP	direct current resistivity and induced polarization
DDH	diamond drillhole
DOT-PF	Department of Transportation and Public Facilities
DML	Dawson Metallurgical Laboratories
DTH	down-the-hole
E-GRG	extended gravity recoverable gold

Abbreviation	Description
EM	electromagnetic
EPA	Environmental Protection Agency
FA	fire assay
FAA	Federal Aviation Administration
FET	federal excise tax
FS	feasibility study
G&A	general and administration
GOLD	TSX ticker symbol for GoldMining Inc.
GPR	gross production royalty
GQCV	greenstone-hosted quartz-carbonate vein deposits
GRAV	gravimetric finish method
HPGR	high-pressure grinding roll
IA	Initial Assessment with economic analysis
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma - optical emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IOCG	iron oxide copper-gold
IP	induced polarization
IRG	intrusion-related gold
IRGS	intrusion-related gold system
ISO	International Organization for Standardization
Kiska	Kiska Metals Corporation
LIDAR	light detection and ranging
LUP	land use permit
MCF	mechanized cut and fill
ML	metal leaching
MRE	mineral resource estimate
MSGP	Multi-Sector General Permit
MTOs	material take-offs
NAD 83	North American Datum of 1983
NAG	non-acid-generating
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	nearest neighbour

Abbreviation	Description
NSR	net smelter return
NTS	national topographic system
OK	ordinary kriging
PAG	potentially acid-generating
PFS	prefeasibility study
PGE	platinum group elements
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
RHA	River and Harbors Act
ROM	run-of-mine
RQD	rock quality designation
SAG	semi-autogenous grinding
SCC	Standards Council of Canada
SD	standard deviation
S_d-BWi	micro hardness or bond ball mill work index on SAG ground material
SEDEX	sedimentary exhalative deposits
SG	specific gravity
SOBIR	Standardization of Back-In Rights
SPCC	Spill Prevention Containment and Contingency
STIP	State Transportation Infrastructure Plan
TSF	tailings storage facility
TSX	Toronto Stock Exchange
UG	underground
USACE	U.S. Army Corps of Engineers
USGO	Nasdaq ticker symbol for U.S. GoldMining Inc.
UTM	Universal Transverse Mercator coordinate system
UV	ultraviolet
VLF-EM	very low frequency electromagnetic
VMS	volcanogenic massive sulfide
WSAR	West Susitna Access Road

Table 2-3: Units of Measurement

Abbreviation	Description
%	percent
% solids	percent solids by weight
CAD	Canadian dollar (currency)
C$	Canadian dollar (as symbol)
$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
µm	micron (micrometer)
cm	centimeter
cm^3	cubic centimeter
d/a	days per year
ft	foot (12 inches)
g	gram
g/cm^3	gram per cubic centimeter
g/L	gram per liter
g/t	gram per metric ton (tonne)
h	hour (60 minutes)
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometer
km^2	square kilometer
kW	kilowatt
kWh/t	kilowatt-hour per tonne
kWh/m^3	kilowatt-hour per cubic meter
L	liter
lb	pound
m, m^2, m^3	meter, square meter, cubic meter
M	million
Ma	million years (annum)
masl	meters above mean sea level
mm	millimeter
Moz	million (troy) ounces
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne

Abbreviation	Description
oz/ton	ounce (troy) per short ton (2,000 lbs)
ppb	parts per billion
ppm	parts per million
t	metric tonne (1,000 kg)
ton	short ton (2,000 lbs)
t/d	tonnes per day
USD	US dollars (currency)
US$	US dollar (as symbol)

3 PROPERTY DESCRIPTION

The Project is in the Alaska Range approximately 170 km northwest of Anchorage as illustrated in Figure 3-1. The center of the property is located at 152.57 degrees longitude west and 61.98 degrees latitude north.

Figure 3-1: Location of the Whistler Project

Source: U.S. GoldMining, 2026

3.1 Project Ownership

The Project comprises 377 State of Alaska mining claims covering an aggregate area of approximately 53,700 acres (217 km^2) in the Yentna Mining District of Alaska. All the claims are owned 100% by U.S. GoldMining and are in good standing. The property boundaries have not been legally surveyed.

An all-season camp facility exists near the confluence of Portage Creek and the Skwentna River, approximately 15 km southeast of the Rainy Pass Hunting Lodge. The camp is serviced with a 1,000 m gravel airstrip for wheel-based aircraft. The camp is equipped with diesel generators, a satellite communication link, tent structures on wooden floors, and several wood-framed buildings.

3.1.1 Property Agreements

GoldMining, through its subsidiary U.S. GoldMining (then known as BRI Alaska Corp.), acquired the rights to the project on August 5, 2015, pursuant to an asset purchase agreement date August 5, 2015, between GoldMining, U.S. GoldMining, Kiska Metals Corporation and Geoinformatics Alaska Exploration, Inc. in exchange for the issuance of 3,500,000 GoldMining shares as set out in Gold Mining's news release of August 6, 2015.

3.2 Mineral Tenure

A full Claims List can be found in Appendix A – Mineral Claims at the end of this report. Annual Labor requirements:

- US$400 for each quarter section MTRS claim
- US$100 each for any other type of claim

Labor must be performed by September 1 of each year, and the statement of annual labor must be recorded by November 30. Excess labor from previous years may be carried forward. The claims are in good standing as of the effective date of this report.

3.3 Surface Rights

Under AS 38.05.255, the surface uses of land or water included within a state mining location that the owners, lessees, or operators of the location may undertake by virtue of such location are (a) are limited to those necessary for the prospecting for, extraction of, or basic processing of minerals and (b) shall be subject to reasonable concurrent uses (Stoel Rives, 2025). The surface rights are not an encumbrance.

3.4 Water Rights

A temporary water rights authorization is included in the Company's multi-year Exploration and Reclamation Permit Number 2778 for Hardrock Exploration – Skwentna River – Yentna Mining District, issued by Alaska Department Natural Resources, Division of Mining, Land and Water. The water rights are not an encumbrance.

3.5 Royalties and Encumbrances

The first underlying agreement is a Royalty Purchase Agreement between Kiska Metals Corporation, Geoinformatics Alaska Exploration Inc. and MF2 LLC. (MF2), dated December 16, 2014. This agreement grants MF2 a 2.75% NSR royalty over all 377 claims and extending outside the current claims over an Area of Interest defined by the maximum historical extent of claims held on the project as indicated on Figure 3-1. The MF2 royalty was subsequently assigned to Osisko Mining (USA) Inc. (OM). U.S. GoldMining can buy back 0.75% of the 2.75% NSR royalty for a payment of US$5,000,000 to OM. Pursuant to a subsequent assignment agreement dated January 11, 2021, the buy-back right was conveyed to Gold Royalty Corp.

The second underlying agreement is an earlier agreement between Cominco American Incorporated and Mr. Kent Turner (whose rights and obligations thereunder were assumed by U.S. GoldMining) dated October 1, 1999. This agreement concerns a 2.0% net profit interest to Teck Resources, since purchased by Sandstorm Gold Ltd., in connection with an Area of Interest specified by standard township subdivision as indicated in Figure 3-2. Sandstorm Gold Ltd. was acquired by Royal Gold Inc. on October 20, 2025.

The third underlying agreement is a royalty agreement dated January 11, 2021, between U.S. GoldMining and Gold Royalty Corp., pursuant to which Gold Royalty Corp. holds a 1% NSR royalty covering the Project.

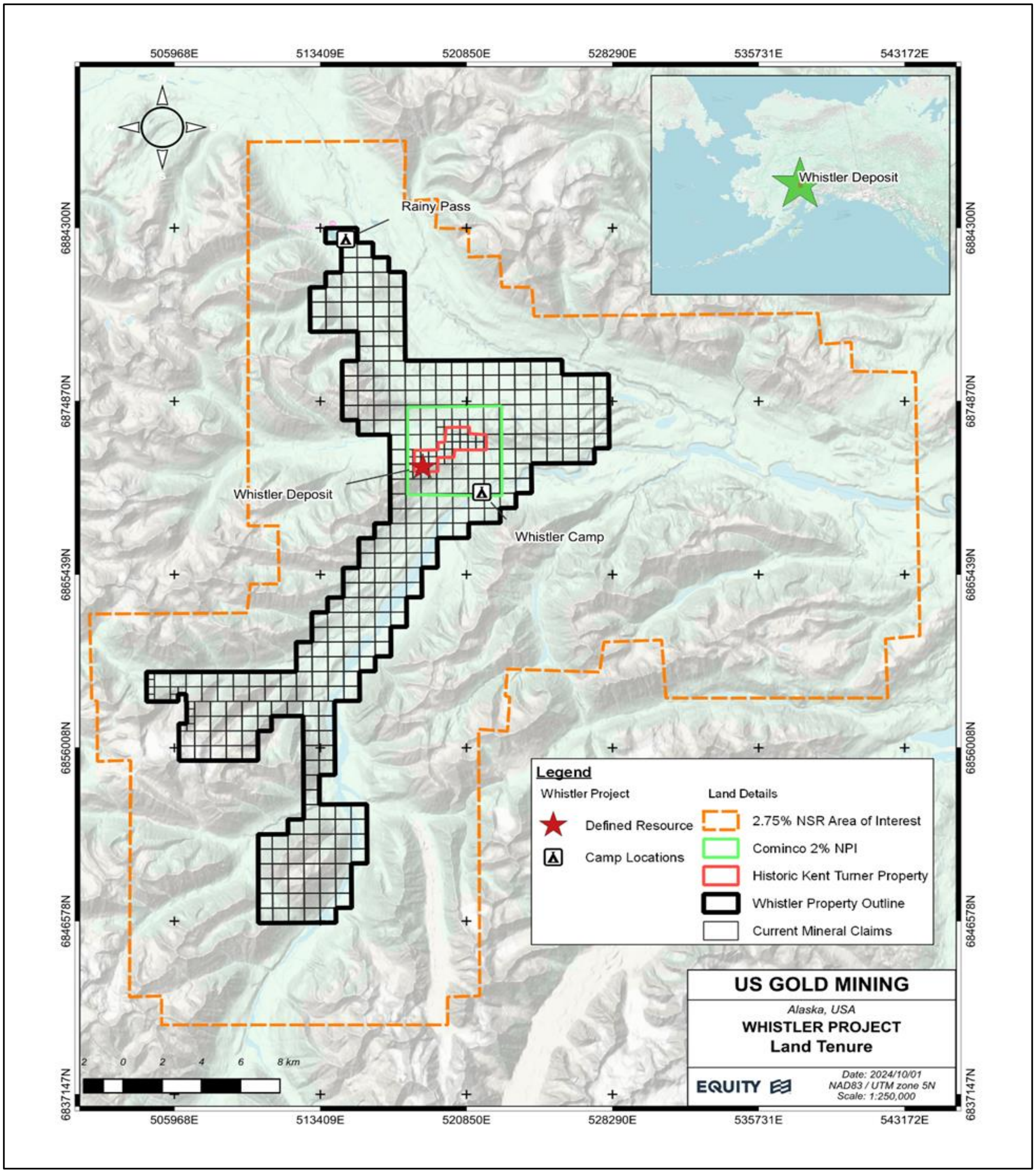
The Company currently holds multi-year Exploration and Reclamation Permit Number 2778 for Hardrock Exploration – Skwentna River – Yentna Mining District, issued by Alaska Department Natural Resources, Division of Mining, Land and Water. U.S. GoldMining has a good understanding of future permitting requirements, however the associated timelines and permit conditions have yet to be determined.

There are no other significant encumbrances, significant factors, or risks relating to the property or work on the property.

3.6 Permitting Requirements and Conditions

Permitting requirements are discussed in Section 17 of this TRS.

Figure 3-2: Tenement Camp



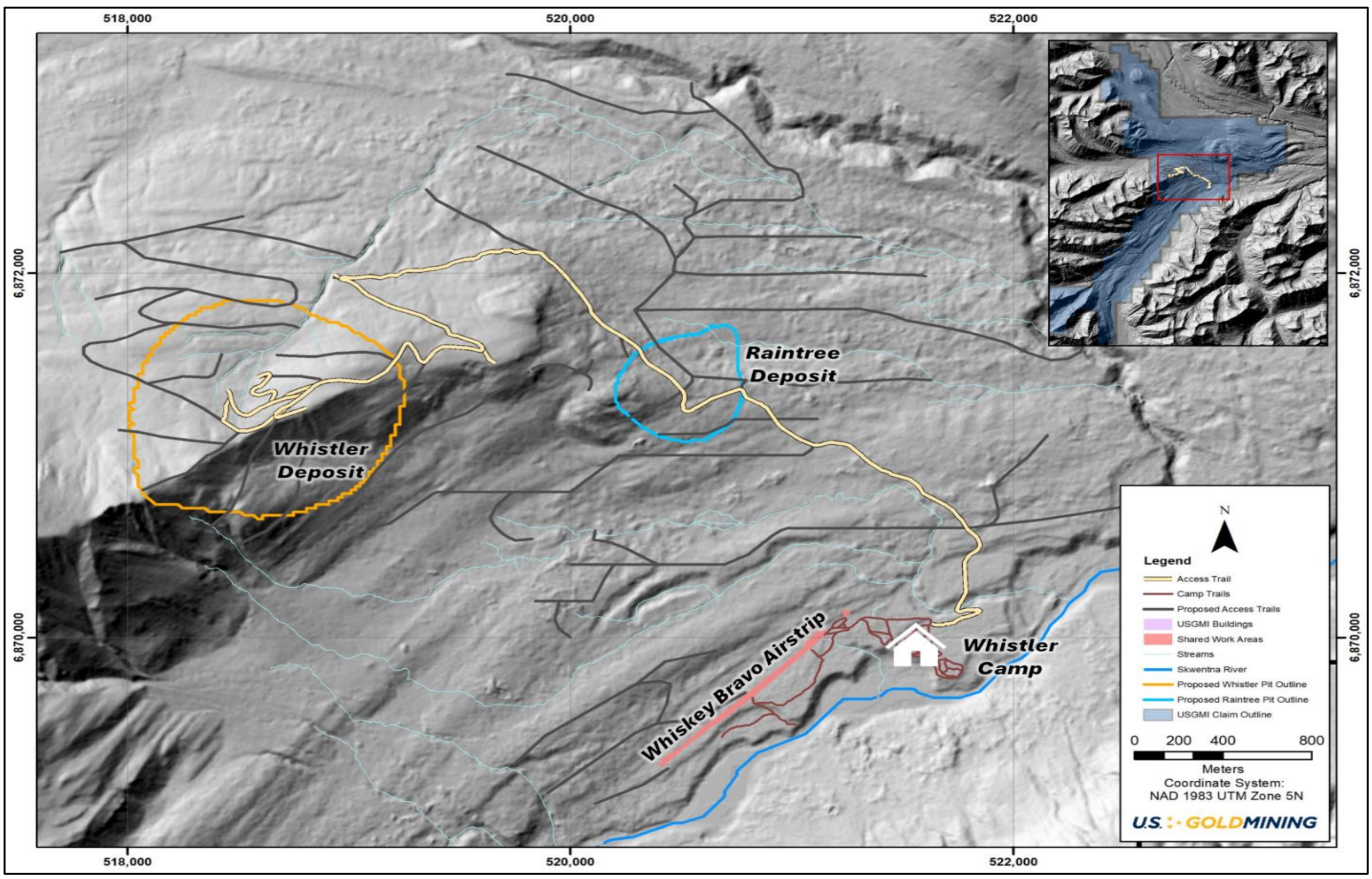
Source: U.S. GoldMining, 2024

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Accessibility

The Project is located within the Alaska Range approximately 170 km northwest of Anchorage and 76 km west of the township of Skwentna as illustrated in Figure 3-1. Access to the project area is by fixed wing aircraft to the Whiskey Bravo gravel airstrip located adjacent to the Whistler exploration camp. In the winter of 2011, Kiska constructed a temporary winter trail to the Project that was then used for the inbound transportation of fuel, earth moving equipment, and bulk items for the camp and exploration programs. A 1,000 m compacted gravel runway, the Whisky Bravo Airstrip (Figure 4-1), provides year-round accessibility to the site. The airstrip is capable of landing up to DC-3 class aircraft and is currently shared with the Estelle Gold Project owned by Nova Minerals Ltd.

Figure 4-1: Layout of Built and Proposed (and Permitted) Infrastructure in the Whistler Area



Source: U.S. GoldMining, 2024

4.2 Climate

The project area is between regions of maritime and continental climate and is characterized by relatively mild winters and warm summers. The maritime climatic influence provides for dry, mild, and temperate summers. Fog and low clouds are common in mid-summer and fall especially around higher-elevation areas. Average summer temperatures range between 5° to 20°C, whereas winter temperatures range from -15° to -5°C. Occasionally, arctic cold fronts will propagate across the Alaska Range from the interior, causing cold dry air to seep into the watershed. These infrequent stationary high-pressure systems can lead to clear days with temperatures dropping to a low of -35° C during the winter. Moderate winds persist during the winter months. Annual precipitation ranges from 500 to 900 mm. Winter snow accumulation usually begins in October and by late May the snow has melted sufficiently to allow for fieldwork.

4.3 Local Resources and Infrastructure

The nearest public infrastructure for the Project is Petersville Bridge, located approximately 106 km east of Whistler. Petersville is connected to Anchorage by an all-weather road and highway. The Project is also located approximately 120 km north of the Beluga gas-powered electricity generation plant and 128 km north of the village of Tyonek on the Cook Inlet coast.

The Project is supported by a 24 person, all-season camp located on the banks of the Skwentna River approximately 2.7 km in a straight line from the Whistler Deposit and connected to it via a 6.4 km long access trail, as illustrated in Figure 4-1. The camp is located 400 m from the northeast end of the Whisky Bravo Airstrip, connected via a gravel access trail.

The Whistler Camp was originally built by Kiska Corp. in 2011, and in 2023 the camp was renovated by U.S. GoldMining to satisfy building codes and all state safety, health and hygiene regulations. The camp is served by a 45-kW single phase generator with a 37-kW single phase backup generator, water well, septic system, showers and flush toilets, and a modern kitchen and dining facility. The camp has eight wood-frame accommodations cabins, kitchen/dining hall, First Aid Tent, a wood-frame water well/generator house and a wood-frame men’s and women’s shower/restroom building. The camp is currently permitted for 24 personnel, but with the addition of extra accommodations, it could be expanded to accommodate up to 50 personnel.

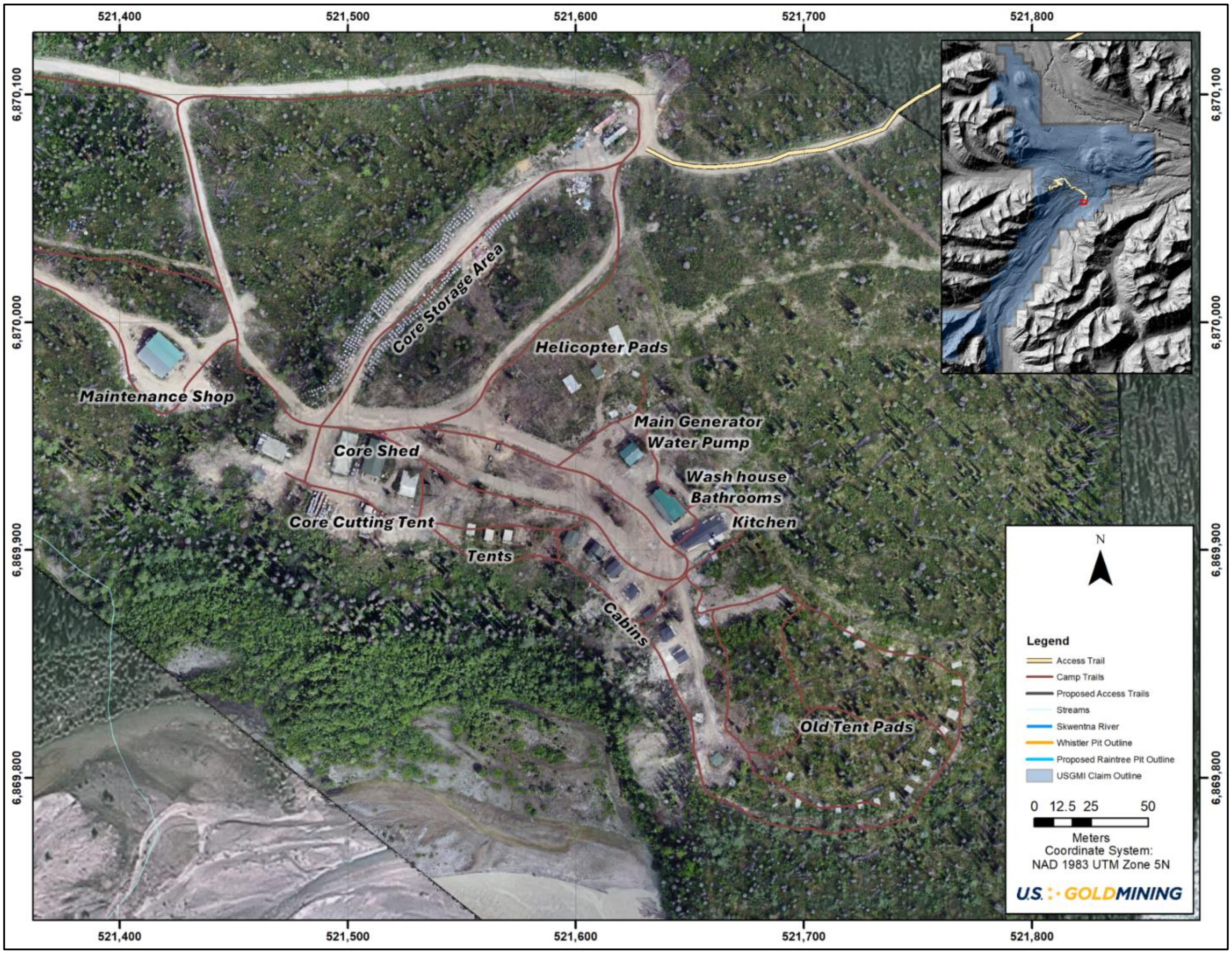
Core processing facilities consist of a well-insulated, well illuminated 7 m x 14 m (98 m^2) wood-frame building, and a core cutting tent that houses a core saw. The core logging facility has a deck that is designed for ease of handling large volumes of core with skid-steer forklifts.

A wood-frame workshop building serves mobile equipment and general camp equipment maintenance. The core cutting facilities and shop are supplied with electricity by separate 20 kW and 2 kW kilowatt generators respectively.

Heavy equipment and ground transport machines at the Project include one Cat D6N bulldozer; one Cat 226B track skid-steer; one Bobcat S175 wheeled skid-steer; one Volvo A-30C haul truck; two Mahindra Roxor 4WD vehicles, and a fleet of smaller ground transportation vehicles including snowmobiles; and side-by-side ATVs.

A core storage area approximately 23 m x 10 m (230 m^2) has been cleared near the core shack. Additional clearings can be made for more storage as the project grows. There are also two wooden-deck helicopter pads with a small building for helicopter support (Figure 4-2).

Figure 4-2: Layout of the U.S. GoldMining Camp and Facilities located adjacent to Whisky Bravo Airstrip



Source: U.S. GoldMining, 2024

The Whisky Bravo Airstrip for the camp is illustrated in Figure 4-4. A 5,000-gallon (18,927 liters) fuel storage facility, comprising ten 500-gallon tanks (1,893 liters), is located at the northeast end of the runway. All tanks are stored in lined containments. All pumping is done through aircraft approved filter systems. A lay-down area is located adjacent to the fuel storage facility for drilling contractor equipment, parts and materials storage (Figure 4-3).

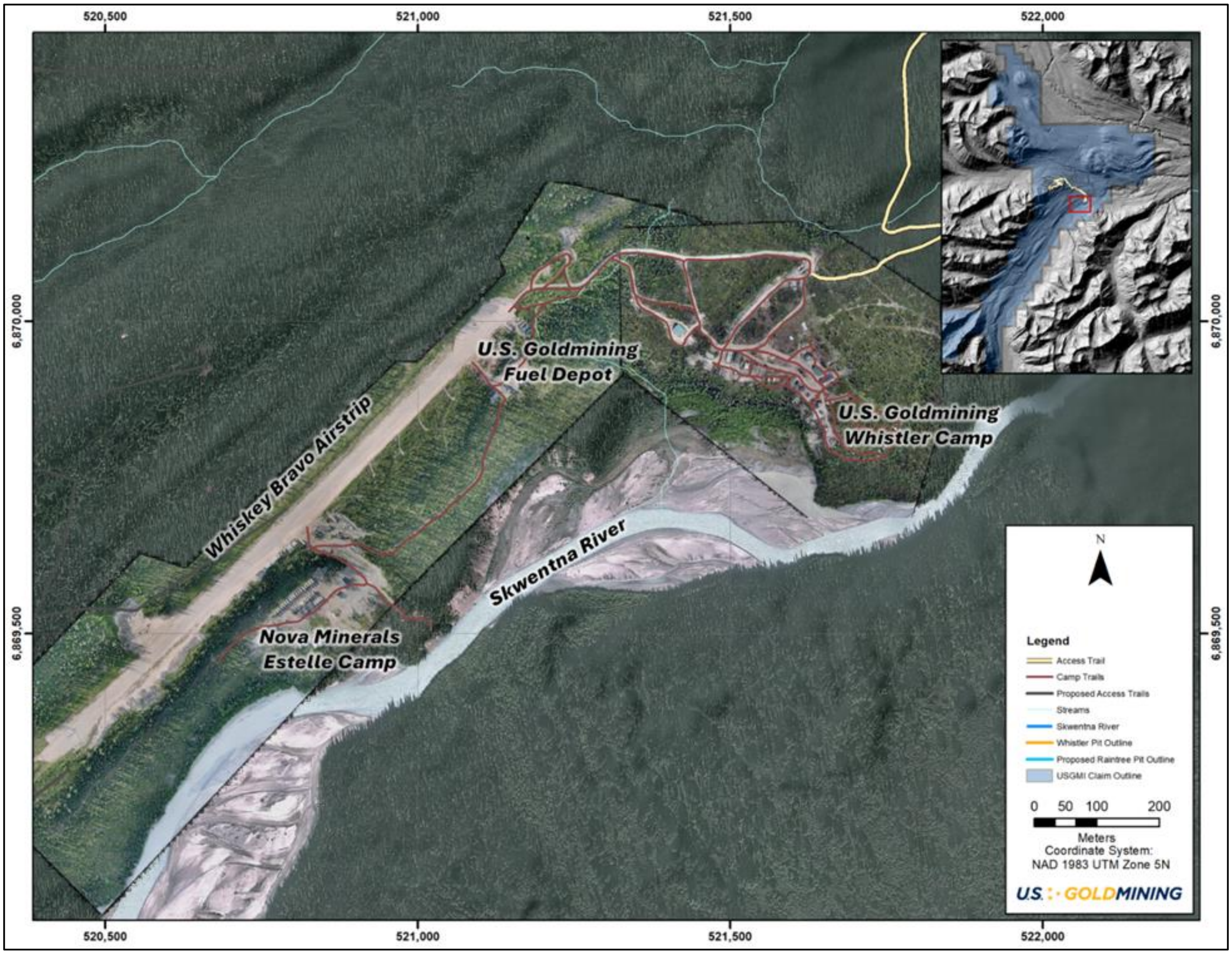
Figure 4-3: Drone Aerial Image Looking Northeast Overlooking Whistler Camp Adjacent the Skwentna River

Source: U.S. GoldMining, 2024

Communications are enabled by Starlink TM satellite system, with routers providing wireless internet communication throughout camp. Cell phone reception is weak at the top of Whistler Ridge, and there is no cell coverage in camp. Operations communications are executed via two-way handheld radios, with a repeater located at the top of Whistler Ridge to ensure adequate radio coverage between camp and the northern side of the ridge.

A winter trail was blazed by previous operators Kiska Corp. and has been used in recent years by Nova Minerals Ltd. which owns the Estelle Project west of U.S. GoldMining's claims, and which also operates from a camp on the southern side of the Whisky Bravo Airstrip (Figure 4-4). The winter road requires annual trail re-establishment, including building of ice bridges over creek and river crossings. The cost of establishing the trail is such that it is only utilized for bulky and heavy equipment which cannot be flown, however for general freight purposes the cost and reliability of air transportation, which has year-round availability, is competitive.

Figure 4-4: Layout of the Whiskey Bravo Airstrip Relative to Whistler Camp



Source: U.S. GoldMining, 2026

Future potential mining operations would require a year-round access road. On October 27, 2021, the Alaska Industrial Development and Export Authority (AIDEA) announced the receipt of US$8.5 million in funds for the advancement of predevelopment work for the WSAR project, which would extend into areas west of Cook Inlet in South-Central Alaska in the vicinity of the Project. During 2022 – 2024 AIDEA undertook road engineering investigations to support road design and test alternatives, environmental baseline surveys and archaeological surveys, and stakeholder consultation. In 2025 AIDEA announced that it had submitted a Department of the Army Individual Permit application for the construction of the WSAR, a 78.5-mile access road across Alaska's Matanuska-Susitna Borough in South-Central Alaska. The Alaska Department of Transportation and Public Facilities (DOT-PF) has also announced that it has included the first 25 miles of the WSAR in its State Transportation Infrastructure Plan (STIP) and has commenced permitting.

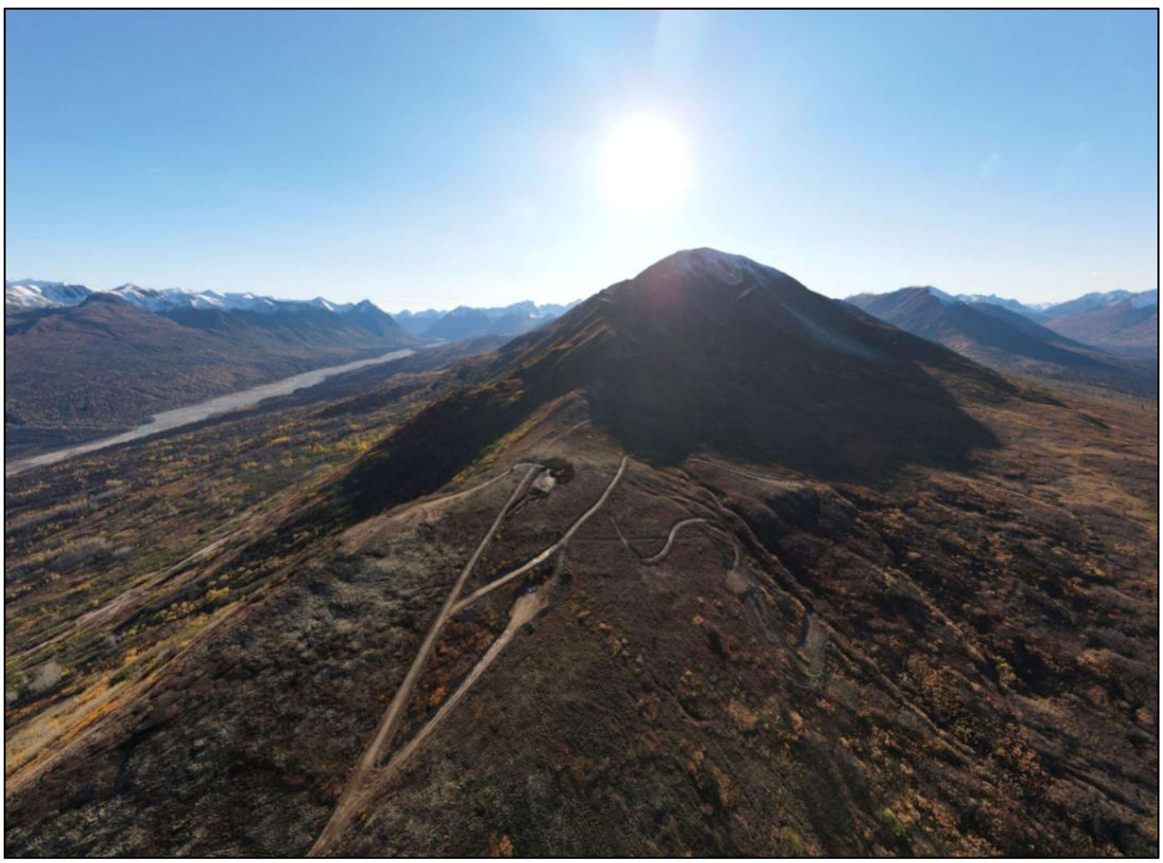
4.4 Physiography

The project is in the drainage of the Skwentna River that forms a large network of interconnected low-elevation U-shaped valleys cutting through the rugged terrain of the southern Alaska Range. Elevation within the property footprint varies from approximately 315 m above sea level in the valley floors to over 1,740 m in the head waters of Muddy Creek. The Alaska Range is a continuation of the Pacific Coast Mountains extending in an arc across the northern Pacific.

The vegetation in the Whistler region is quite variable. The valley floors and lower slopes are usually characterized by dense vegetation giving way above about 750 m elevation to dense tundra shrubs above the timber line (Figure 4-5). At higher elevations, vegetation is absent and, on the peaks, surrounding the Whistler Property, active glaciers with terminal and lateral moraines are present. The timber line is located at elevations varying between 800 to 1,100 m. Bedrock exposures within the project area are scarce except at elevations above 1,000 m and along incised drainage.

The Project mineral claims provide the area that is sufficient for the development of a potential open pit project, including tailings storage, waste disposal, potential processing plant sites and water sources.

Figure 4-5: Drone Aerial Photo Looking Southwest Over the Whistler Deposit with the Skwentna River Valley (left side)



Source: U.S. GoldMining, 2024

5 HISTORY

5.1 Owners and Operators

During the late 1960s, regional mapping and geochemical sampling by the United States Geological Survey (USGS) identified several base and precious metal occurrences over a very large area in the southern Alaska Range including southern portions of the Whistler Project area.

Following the results of that work, limited exploration was conducted in the area during the 1960s and 1980s. Falconbridge (or their operator St. Eugene) was involved in exploring the nearby Stoney Vein in the late 1960s. A local prospector, Arne Murto (deceased), was active in the Long Lake Hills area from at least 1964 and AMAX staked at least four claims over the Lower Discovery showing at Mount Estelle (circa 1982).

Mineral exploration in the Whistler area was initiated by Cominco Alaska in 1986 and continued through 1989. During this period, the Whistler and the Island Mountain gold-copper porphyry occurrences were discovered and partially tested by drilling. In 1990, Cominco's interest waned and all core from the Whistler region were donated to the State of Alaska. The property was allowed to lapse.

In 1999, Kent Turner staked twenty-five State of Alaska mining claims at Whistler, the "Turner Property", and leased the property to Kennecott Exploration Company (Kennecott). From 2004 through 2006 Kennecott conducted extensive exploration of the Whistler region, including geological mapping, soil, rock, and stream sediments sampling, ground induced polarization and they conducted an evaluation of the Whistler gold-copper occurrence with 15 core boreholes (7,948 m) and reconnaissance core drilling at other targets in the Whistler region (4,184 m). Over that period, Kennecott invested over USD$6.3 million in exploration.

In June 2007, Geoinformatics Exploration Inc. (Geoinformatics) announced the conditional acquisition of the Whistler Project as part of a strategic alliance with Kennecott. Between July and October 2007, Geoinformatics drilled seven core boreholes (3,321 m) to infill the deposit to sections spaced at 75 m and to test for the north and south extensions of the deposit.

In August 2009, Geoinformatics acquired Rimfire Minerals Corporation and changed its name to Kiska Metals Corporation (Kiska). In 2009 and 2010, Kiska completed three phases of exploration on the property to fulfill the terms of the Standardization of Back-In Rights (SOBIR) Agreement between Kennecott Exploration Company and Kiska Metals Corporation.

In total, Kiska completed 224 line-km of 3D IP geophysics, 40 line-km of 2D IP geophysics, 327 line-km of cut-line, geological mapping on the 3D IP grid, detailed mapping of significant Au-Cu prospects, collection of 109 rock samples and 61 soil samples, 8,660 m of diamond drilling from 23 drillholes (all greater than 200 m in total length), petrographic analysis of mineralization at Island Mountain, a preliminary review of metallurgy at the Whistler Resource, and metallurgical testing of mineralization from the Discovery Breccia at Island Mountain. This program was executed by Kiska geologists, independent geologists, and multiple contractors, under the supervision of Kiska personnel. All aspects of the exploration program were designed and monitored by a Technical Committee comprised of two Kennecott employees and two Kiska employees. In August of 2010, Kiska delivered a Technical Report (Roberts, 2010) to Kennecott summarizing the results of the completed Trigger Program. In September of 2010, Kennecott informed Kiska that it would not exercise its back-in right on the project and hence retained a 2% Net Smelter Royalty on the property (see Section 3.5 for current status of Royalties and Encumbrances).

From this point forward, Kiska continued to drill and explore the Whistler Project for the duration of the 2010 and 2011 field seasons. The majority of this work included shallow grid drilling (25 m to 50 m top-of-bedrock drilling) in the Whistler Area (also referred to as the Whistler Corridor), conventional step-out drilling from prospects in the Whistler Area, step-out drilling at the Island Mountain Breccia Zone, an airborne EM survey of the Island Mountain area, reconnaissance drilling at Muddy Creek, and minor infill drilling at the Whistler Deposit, followed by the publication of an updated NI 43-101 resource estimate (MMTS, 2011).

A Purchase and Sale agreement between Kent Turner, Kiska Metals Corporation and Geoinformatics Alaska Exploration Inc. dated December 16, 2014, terminated the "Turner Agreement" (an agreement that granted Kennecott and its successors a 30-year lease on twenty-five unpatented State of Alaska Claims; see Figure 3-2) and transferred to Kiska and Geoinformatics, and their successors, an undivided 100% of the legal and beneficial interest in, under, to, and respecting the Turner Property free and clear of all Encumbrances arising by, through or under Turner other than the Cominco American net profit interest.

GoldMining Inc., through its subsidiary U.S. GoldMining (then known as BRI Alaska Corp.), acquired the rights to the Whistler Project in 2015 (see Section 3). In April 2023 U.S. GoldMining listed on the Nasdaq and raised US$20 million to fund the recommencement of exploration activities at the Whistler Project, including drilling which commence in August 2023.

Details of historic exploration completed by each prior operator of the Project is summarized in the following sections. A summary table of the drilling by operator and year with plan maps of the drilling in each resource area can be found in Section 7.5 of this report.

5.2 Surficial Silt, Soil, and Rock Sampling

From 2004 to 2006 Kennecott collected 1,300 rock samples, close to 2,500 soil samples and 103 stream sediments samples in the Whistler, Island Mountain, and Muddy Creek areas. Within this program, a soil grid over the Whistler Deposit returned anomalous Au-Cu results coincident with the magnetic high. Other reconnaissance soil lines in the Whistler area with anomalous Au-Cu results helped to define areas of interest at the Round Mountain, Canyon Creek, Canyon Ridge, Canyon Mouth, and Long Lake Hills prospects. In addition, soil reconnaissance lines at Island Mountain led to the Discovery of the Breccia Zone and broad zones of anomalous Au at Muddy Creek.

In 2009 and 2010, Kiska collected 46 silt samples, 1,417 soil samples and 293 rocks samples, which largely confirmed areas of interest in the Whistler, Island Mountain, and Muddy Creek areas previously defined by Kennecott.

Rock samples consisted of approximately one kilogram of rock collected over a small area surrounding each sampling site using a rock hammer. The sampling location was located using a handheld global positioning system (GPS) unit and marked in the field with a metallic tag. Descriptive information about the geology of the sample was recorded and aggregated into the project database.

Soil samples were collected from the surface soils (generally the B-horizon) by extracting approximately one kilogram of soil into a plastic bag usually with a hand auger. Each sampling site was located using a GPS unit. Descriptive information such as sampling depth and physical attributes were recorded and aggregated into the project database. Typically, field duplicates were collected at a rate of one every twenty samples.

Soil samples were collected along traverses as part of multi-kilometer reconnaissance programs, generally at 100-m spacing. In two areas (Whistler Deposit and Snow Ridge), samples were collected at a more regular 100-m grid spacing. This area is illustrated in Figure 5-1 with the Whistler-Rainmaker terrain shown in Figure 5-2.

Figure 5-1: From the Whistler Deposit area looking north to the Portage Creek valley and Snow Ridge area

Source: MMTS, 2015

Figure 5-2: From the Whistler Deposit area looking south to the Skwentna River valley and Rainmaker area

Source: MMTS, 2015

A summary of all surface exploration work conducted by various operators from 1986 to present is summarized in Table 5-1. Cominco Alaska Inc. is attributed with the discovery of the Whistler Deposit in 1986. The only surface exploration activity documented by Cominco for which U.S. GoldMining has records are 8.4 line-km of 2D IP geophysics over the Whistler Deposit and sixteen diamond drillholes (1,677 m) in the Whistler Deposit.

Table 5-1: Summary of Historical Surface Exploration on the Whistler Project

Operator	Field Seasons	Mapping	Geophysics	Rock	Soil	Silt	Till
Cominco	1986-1989	n/a	8.4 line-km of 2D IP over the Whistler deposit	n/a	n/a	n/a	n/a
Kennecott	2003-2006	Property-wide mapping	39.4 line-km of 2D IP Property-wide AM (400 m line spacing) Snow Ridge AM (79 line-km at 200 m line spacing) Whistler Area AM (1,365 line-km at 50 m line spacing)	1,312	2,446	103	n/a
Geoinformatics	2007-2008	Prospect-scale mapping	8.8 line-km of 2D IP (Whistler area)	20	195	nil	n/a
Kiska	2009-2011	Prospect-scale mapping	40 line-km of 2D IP (Whistler area, Muddy Creek, Island Mountain) 224 line-km of 3D IP (Whistler area) sland Mountain EM (635 line-km at 100 m line spacing)	315	1,425	46	n/a

Note: AM=Airborne Magnetic survey; EM=Airborne Electromagnetic survey; IP=Induced Polarization survey.

5.3 Geological Mapping

The bulk of the detailed geological mapping and interpretation on the property was undertaken by Kennecott and summarized in a report by Young (2006). This work laid the foundation for the geological interpretation of porphyry-style mineralization in the Whistler area (including the Whistler Deposit and the Raintree - Rainmaker deposits), the Breccia Zone at Island Mountain, and Intrusion-Related Au mineralization in the Muddy Creek area. Subsequent prospect-scale mapping was done by Kiska at the Round Mountain, Snow Ridge, Tryone, Lightening, Whistler Ridge, Long Lake Hills, and Island Mountain. Additionally, reconnaissance level regional mapping was conducted along the 3D-IP lines during the 2009 IP survey.

5.4 Airborne Geophysics

An airborne helicopter geophysical survey was commissioned from Fugro Airborne Surveys (Fugro) by Kennecott during 2003. This survey covered the entire property with a high sensitivity cesium magnetometer and a 256-channel spectrometer.

Additional airborne magnetic data were acquired by Kennecott in 2004 over two smaller areas using a helicopter equipped by a Rio Tinto bird operated by Fugro and a Kennecott geophysicist. One area over the Snow Ridge target was investigated at 200-m line spacing (79 line-km). The other grid was flown over the Whistler Deposit and surrounding area using fifty-meter line spacing (1,365 line-km).

Results from these airborne surveys were used by Kennecott to interpret geological contacts, fault structures and potential mineralization in the Whistler, Island Mountain, and Muddy Creek areas. In particular, the airborne magnetic data showed that the Whistler Deposit displays a strong 900 m by 700 m positive magnetic anomaly attributed to the magnetic Whistler Diorite intrusive complex (host to the Whistler Deposit) in addition to a contribution from secondary magnetite alteration and veining associated with Au-Cu mineralization. This observation formed that basis for exploration targeting in the Whistler area, particularly those areas covered by a thin veneer of glacial sediments, such as the Raintree and Rainmaker deposits. These surveys, in addition to 2D Induced Polarization ground geophysical surveys targeted over airborne magnetic anomalies, were instrumental in the discovery of the Rainmaker and Raintree deposits by Kennecott in 2005 and 2006, respectively.

Kiska commissioned a helicopter borne AeroTEM survey over the Island Mountain area by Aeroquest Airborne in June 2011. The principal geophysical sensor was an AeroTEM III time domain electromagnetic system, employed in conjunction with a caesium vapour magnetometer. Navigation was provided by a real-time differential GPS navigation system, plus a radar altimeter and a video recorder mounted in the nose of the helicopter.

The survey was flown on east-west flight lines with a spacing of 100 m. Control lines were flown north-south, perpendicular to the survey lines, with a spacing of 1,000 m. The nominal terrain clearance of the EM bird was 30 m. The magnetometer sensor was mounted in a smaller bird connected to the tow rope 33 m above the EM bird and 20 m below the helicopter. Nominal survey speed was 75 km/h, resulting in a geophysical reading about every 1.5 to 2.5 m along the flight path. The total survey coverage, including tie lines, was 635 km. Mira Geoscience was subsequently engaged to produce a 3D inversion of the data. The survey was designed to target potential zones of disseminated and net-textured pyrrhotite mineralization like the pyrrhotite-associated gold-only zone of mineralization on the flanks of the Breccia Zone. The survey did detect a large 1.5 km long by 1.0 km wide conductivity low anomaly on the southeast side of the Island Mountain area, referred to as the Super Conductor target. This anomaly was subsequently tested by three drillholes that did suggest that the conductivity anomaly may be associated with disseminated pyrrhotite mineralization with elevated gold values, yet further drilling is required to be conclusive and fully test the target.

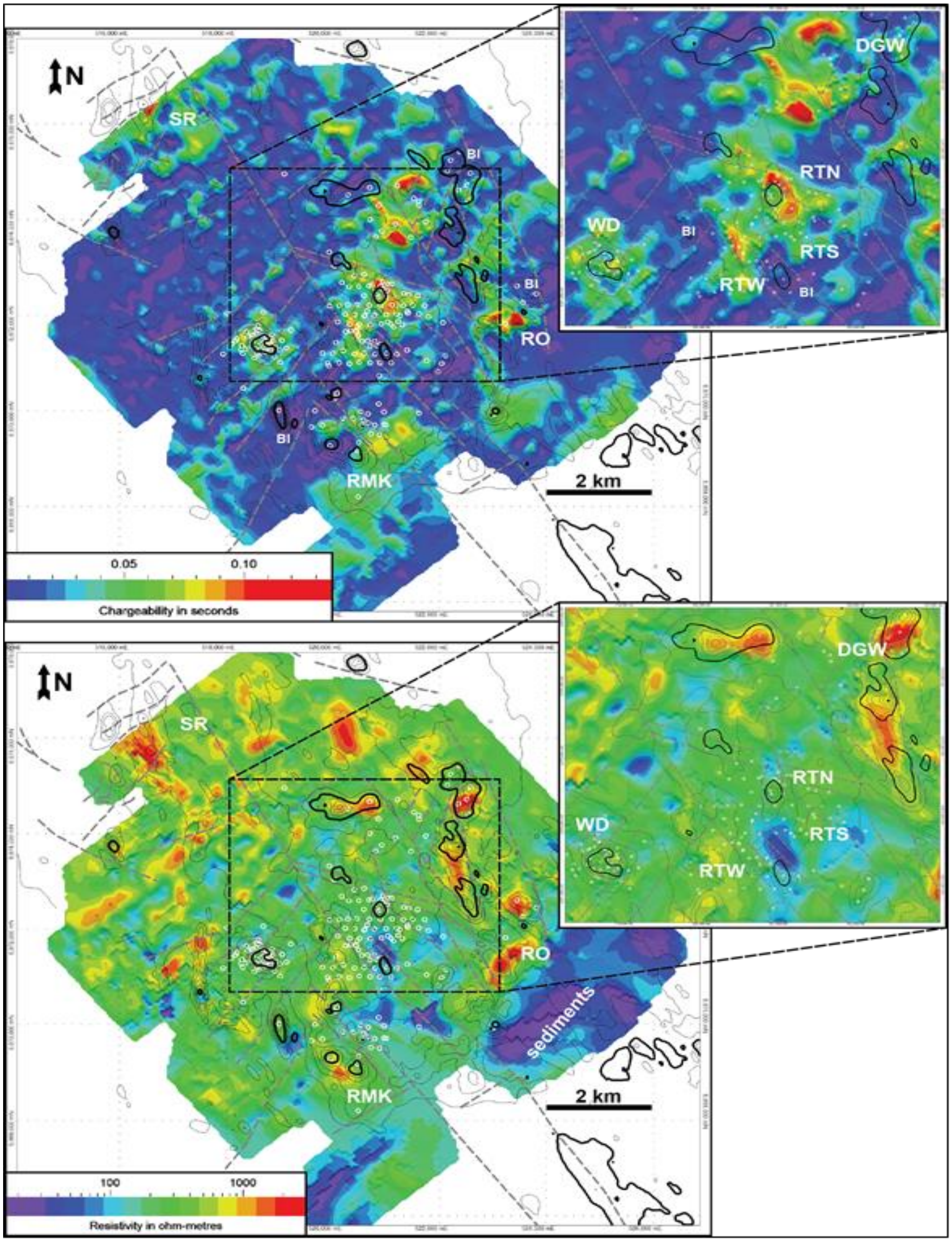
5.5 Ground Geophysics

Cominco acquired 8.4 line-km of 2D Induced Polarization geophysics from six east-west oriented lines centered over the Whistler Deposit discovery outcrops. Anomalous results from these lines were used to target the deposit area with subsequent drilling. From 2004 to 2006, Kennecott completed 39.4 line-km of 2D IP geophysics in the Whistler area. Within this survey, two IP lines were run over the Whistler Deposit magnetic anomaly and showed that mineralization is coincident with a strong chargeability anomaly. Subsequent lines targeted magnetic anomalies at the Round Mountain, Canyon Creek, Canyon Ridge, Canyon Mouth, Long Lake Hills, Raintree and Rainmaker deposits. In 2007-2008, Geoinformatics completed 8.8 line-km of 2D IP from six separate reconnaissance lines in the Whistler area targeting airborne magnetic highs. Anomalous results from this survey in the Raintree area led to the Raintree West discovery.

In 2009, Kiska undertook a significant 2D and 3D IP survey over most of the prospective areas in the Whistler, Island Mountain, and Muddy Creek areas. Kiska commissioned Aurora Geoscience to complete 224 line-km of a 3D IP geophysical survey. This was executed on two grids (Round Mountain; Whistler Area) which were comprised of grid lines ranging from 4 to 9 km long with a line spacing of 400 m. From November to December 2009, the raw data was delivered to Mira Geoscience for detail data quality control and error analysis prior to the construction of a 3D inversion model. This survey reaffirmed that the Whistler Deposit is coincident with a discrete 3D chargeability anomaly and showed that much of the Whistler area contains broad areas of anomalous chargeability (Figure 5-3). In conjunction with the airborne magnetic data, these zones of anomalous chargeability formed the basis for exploration drilling in the Whistler Area in 2010.

In 2009 Kiska commissioned SJ Geophysics to complete 40 line-km of a 2D Induced Polarization geophysical survey. Survey lines were generally semi-straight reconnaissance-type lines over areas of interest at Alger Peak, Island Mountain, and Muddy Creek. The geophysical survey was acquired with a pole – dipole 2DIP technique with 100 m dipoles.

Figure 5-3: Depth slices (100 m) of the chargeability (top) and resistivity (bottom) inversion model of the 3D IP data in the Whistler Area (with contours of the 400 m line spacing AMAG RTP).



Notes: WD=Whistler Deposit; RTW=Raintree West; RTN=Raintree North; RTS=Raintree South; DGW=Dagwood; RMK=Rainmaker.
Source: Roberts, 2011a

5.6 Drilling

5.6.1 Drilling by Cominco Alaska Inc.

Partial records documenting the sixteen shallow core boreholes (1,677 m) drilled by Cominco on the Whistler gold-copper deposit in 1988 and 1989 including descriptions of the core, drilling logs and assay results are described by Couture, 2007.

Kennecott resurveyed the locations of several holes using either a handheld GPS or with a Trimble ProXr receiver providing real-time sub-meter accuracy. Three holes were unable to be located. The core from the Cominco holes was reportedly donated to the State of Alaska in 1990 and may be stored at a core library in Eagle River, Alaska (Couture, 2007).

5.6.2 Drilling by Kennecott

Between 2004 and 2006, Kennecott drilled a total of 31 core holes (9,630 m) on the Whistler Project, with fifteen of those core holes (7,953 m) intersecting the Whistler Deposit. The Kennecott core is partly stored at the site camp with some in a secured warehouse in Wasilla, Alaska. Drilling operations were conducted by NANA-Dynatec and NANA-Major drilling out of Salt Lake City, Utah using up to three drill rigs supported by helicopter. Core size was HQ-diameter in 2004 and subsequently NQ in 2005 and 2006 (Couture, 2007).

Drilling was documented by Kennecott personnel. The collar position of each borehole was laid out with a hand GPS unit, while azimuth and inclination were determined with a compass. Individual collars were subsequently surveyed using a Trimble ProXr receiver providing real-time sub-meter accuracy. Flex It Multi-shot readings at twenty-foot (six meter) intervals were taken to monitor downhole deviation. Magnetic susceptibility and gravity data were also recorded. Drilling, logging, and sampling were directly supervised by a suitably qualified geologist. Core retrieved from drilling was oriented using EzMark or an ACE device. All casing was pulled after drilling. Core recovery, geotechnical point load test, and rock quality determination were collected before the geologist recorded detailed information about lithology, mineralogy, alteration, vein density, and structure. All recorded descriptive data were entered into an acQuire database (Couture, 2007).

Twenty drillholes (4,746 m) were drilled by Kennecott to investigate exploration targets outside the Whistler Deposit. Targets selected for drilling were typically chosen based on a combination of geology, geochemical and geophysical criteria believed to be indicative of magmatic-hydrothermal processes. Selected targets were explored with vertical or angled drillholes to validate the geological model. One or more boreholes were drilled with the intent to identify the potassic core of a magmatic-hydrothermal system known to be associated with better copper and gold sulfide mineralization in this area (Couture, 2007).

5.6.3 Drilling by Geoinformatics

In 2007 and 2008, Geoinformatics drilled twelve holes totaling 5,784 m on the Whistler deposit, two holes of 622 m in the Raintree area and four holes totaling 1,219 m on other exploration targets in the Whistler project area. Geoinformatics used the same drilling contractor and drilling procedures as previously Kennecott except that oriented core was not obtained. Exploration drilling by Geoinformatics in the Whistler area targeted geophysical anomalies in the Raintree and Rainmaker areas, using the same basic porphyry exploration model as Kennecott (Roberts, 2011a).

5.6.4 Drilling by Kiska

During the 2009-2011 Kiska drilling campaigns, diamond drilling was performed by Quest America Drilling and Falcon Drilling Ltd. and supervised by geological staff from Kiska. Drilling was performed by helicopter-portable diamond drill rigs. Drillholes were collared with HQ-diameter tools (6.35 cm) and reduced to NQ diameter tools (4.76 cm) when the rig reached the depth capacity of the HQ equipment. Collar locations were determined with handheld GPS devices by Kiska staff. Downhole surveys for all holes were conducted by the drill contractor at 60-m intervals downhole using a Reflex EZ Shot downhole camera (Roberts, 2011a).

During the 2009-2011 Kiska drilling campaign a total of 188 diamond drillholes were completed for a total of 48,498 m. All drillholes were logged by Kiska geologists at the core logging facility at the Whistler exploration camp. Logged geological information included lithology type, alteration type and intensity, vein types, percent vein volume and vein orientations (to core axis), structures (to core axis), the percent of sulfides and oxides, and magnetic susceptibility at meter intervals. Geotechnical information logged included core recovery and rock quality designation (RQD). All logging data was entered on paper logging forms in 2009 and transcribed digitally info LogChief software in 2010 and 2011 (Roberts, 2011a).

5.6.4.1 Whistler Deposit

A total of 8 holes totaling 5,475 m were drilled on the Whistler Deposit by Kiska. These holes were targeted to infill gaps from the previous drill campaigns and to test the edges and depth of the intrusive complex that hosts the deposit.

5.6.4.2 Raintree West Deposit

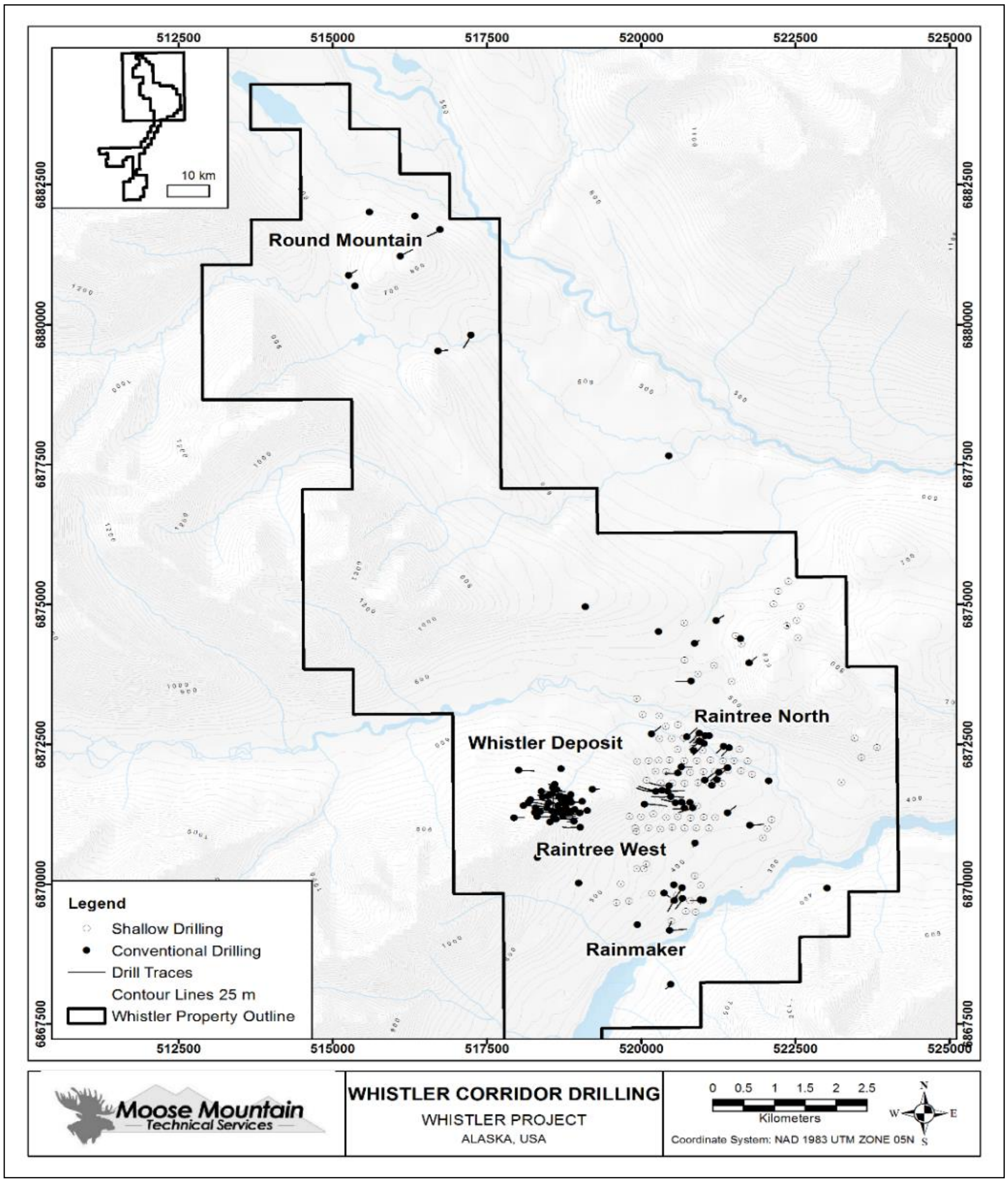
The Raintree West deposit is located 1,800 m to the east of the Whistler deposit in the area formerly called Raintree, just off the nose of Whistler Ridge. The discovery drillhole, RN-08-06, targeted an airborne magnetic high anomaly that is coincident with an IP chargeability high anomaly detected on a 2D IP reconnaissance line that crossed the Whistler area. This hole discovered a significant zone of near-surface (below 5 m to 15 m of till cover) gold-copper porphyry mineralization (160 m grading 0.59 g/t Au, 6.02 g/t Ag, 0.10% Cu). Kiska expanded on this discovery in 2009 with a scissor hole drilled on the same section as RN-08-06 (WH09-02). This was successful at duplicating the gold-copper mineralization zone in RN-08-06, and identified a second, deeper zone of porphyry mineralization on the west side of the Alger Peak fault zone. In 2010, Kiska followed up with an additional four drillholes, and in 2011 further tested the shallow zone and the deep zone with a total of eight holes for a total of 5,997 m. The majority of drillholes in the Raintree area were drilled on east-west sections with section spacing of 100 m.

5.6.4.3 Whistler Area Drilling

A total of 133 exploration holes for 27,464 m of drilling in the Whistler area were completed by Kiska in 2009-2011. A majority of these holes were drilled in the area that includes much of the broad valley floor to the north, east and south of the Whistler Ridge, that includes the parts of the Raintree and Rainmaker prospect areas (Figure 5-4).

Targeting for this drilling program was developed by a technical team comprised of Kiska and Kennecott geologists based on blind geophysical targets heavily weighted by the results of the 2009 3D IP survey (chargeability and resistivity anomalies), airborne magnetic anomalies, anomaly size, and proximity to areas of known mineralization or anomalous surface geochemistry.

Figure 5-4: Whistler Orbit Area Drilling



Source: MMTS, 2026

A majority of these holes intersected andesitic volcanic rocks with moderate to strong sericite-clay-pyrite alteration and occasional sphalerite- and galena-bearing quartz-carbonate veins with banded and colloform epithermal-like textures. The holes were spaced on average greater than 500 m apart and alteration and veining indicate that broad areas in the Whistler Area define the upper, cooler margins of a large porphyry-related hydrothermal system or a cluster of smaller, coalescing porphyry-related hydrothermal systems. Within this broad area, drilling returned Whistler-like, porphyry-style Au-Cu mineralization with significant intercepts at the Raintree West, Raintree North, and the Rainmaker deposits, and anomalous alteration and geochemistry at the Dagwood prospect.

5.6.4.4 Island Mountain Drilling

The 35 out of 42 holes completed by Kiska in the Island Mountain area between 2009 and 2011 targeted the Breccia Zone. The remainder targeted zones of either anomalous surface rock geochemistry and alteration (Cirque Zone) or geophysical anomalies (Super Conductor). Significant results were only returned from the Breccia Zone and are summarized below. The alteration patterns and geochemical pathfinder elements from the other areas may be useful for future drill targeting.

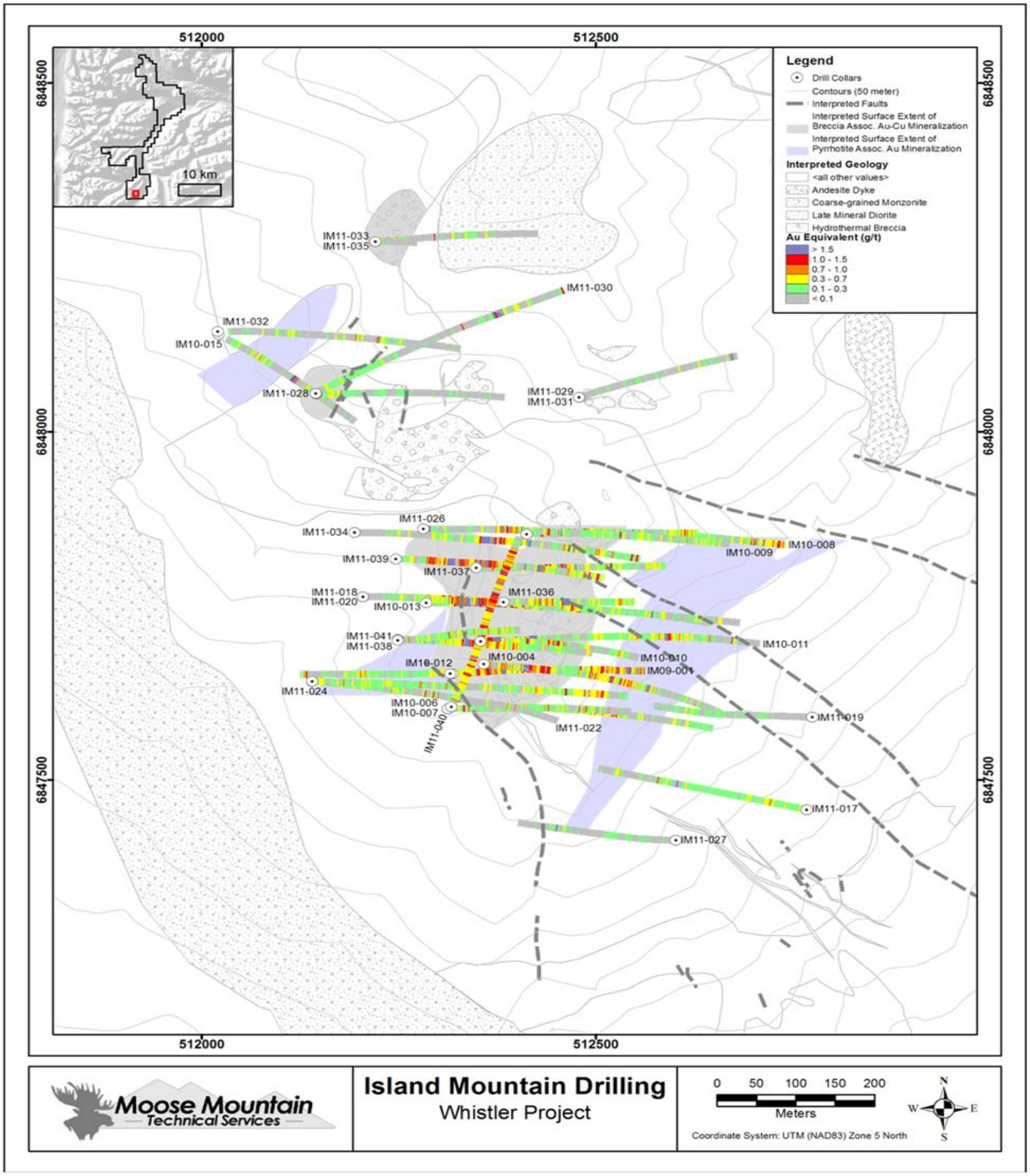
At the Island Mountain Deposit, drilling included in the resource estimate includes 36 drillholes for 14,410 m of drilling. The majority of these holes were completed on seven east-west cross-sections spaced 50 m apart in a 300 m^2 area from 6,847,600 N to 6,847,900 N (Figure 5-5). The lithologies, alteration and mineralization of the breccia-related mineralization indicate that the magmatic-hydrothermal breccia complex defines an irregular pipe-shaped body approximately 300 m by 300 m in plan which from the surface down 500 m. Like the strike of the faults in the area, this breccia complex is subvertical and appears to trend in a northwest-southeast orientation (Roberts, 2011a).

Surface mapping, soil geochemistry and drilling has defined other distinct breccia bodies with zones of alteration, surface anomalism and significant mineralization up to 700 m to the north-northwest of this breccia complex. Significant zones of mineralization are shown in Figure 5-2.

Table 5-2: Summary of Historical Surface Exploration on the Whistler Project

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	Cu (%)
IM10-015	74.3	111.0	36.7	0.27	0.37	0.01
and	166.8	212.9	46.1	1.19	0.53	0.01
Including	168.5	182.2	13.7	3.69	0.56	0.01
and	274.0	276.0	2.0	10.5	2.30	0.04
IM11-030	20.0	63.0	43.0	0.32	1.12	0.03
and	364.1	438.0	73.9	0.72	2.24	0.09
including	364.1	390.0	25.9	1.79	5.05	0.09
IM11-032	104.0	137.0	33.0	0.21	0.62	0.02
and	246.0	300.0	54.0	0.29	0.28	0.01
IM11-033	2.8	58.0	55.2	0.41	1.54	0.03
including	2.8	42.0	39.2	0.56	1.18	0.02
IM11-035	3.0	44.0	41.0	0.44	2.19	0.03

Figure 5-5: Plan Map of Drillholes and Mineralization Style at the Breccia Zone – Island Mountain



Source: MMTS, 2026

6 GEOLOGICAL SETTING, MINERALIZATION, AND DEPOSIT

6.1 Geological Setting

The Project is situated within the Wrangellia Composite Terrane (WCT), one of three composite terranes accreted to the Alaskan portion of the North America Cordilleran margin in the Mesozoic and Cenozoic. This margin records a complex history of terrane accretion, basin formation, basin exhumation, subduction, and multiple pulses of magmatism.

In south-central Alaska, the WCT is comprised of three significant tectono-magmatic assemblages Figure 6-1 (with the black inset box showing the location of the Whistler area and map extent in Figure 6-2). The assemblages include the Paleozoic-Triassic basement rocks upon which the Early to Late Jurassic Talkeetna island arc was built, including volumetrically significant plutonic rocks; the Kahiltna assemblage, consisting of Jura-Cretaceous flysch sediments that formed in basins initiated by the convergence of Wrangellia with the former continental craton; and voluminous Upper Cretaceous and Paleocene-Oligocene igneous rocks, dominantly plutons, that stitch the Wrangellia composite terrane with the inboard autochthonous terranes.

The latter two assemblages dominate the regional geology of the Whistler area.

The Kahiltna assemblage occurs as a broad 100 km by >300 km belt extending across the Alaska Range. This assemblage is comprised of mostly marine sediments with fossils indicating deposition from the Late Jurassic to Early Cretaceous.

The black inset box shows the location of Whistler area and map extent in Figure 6-1.

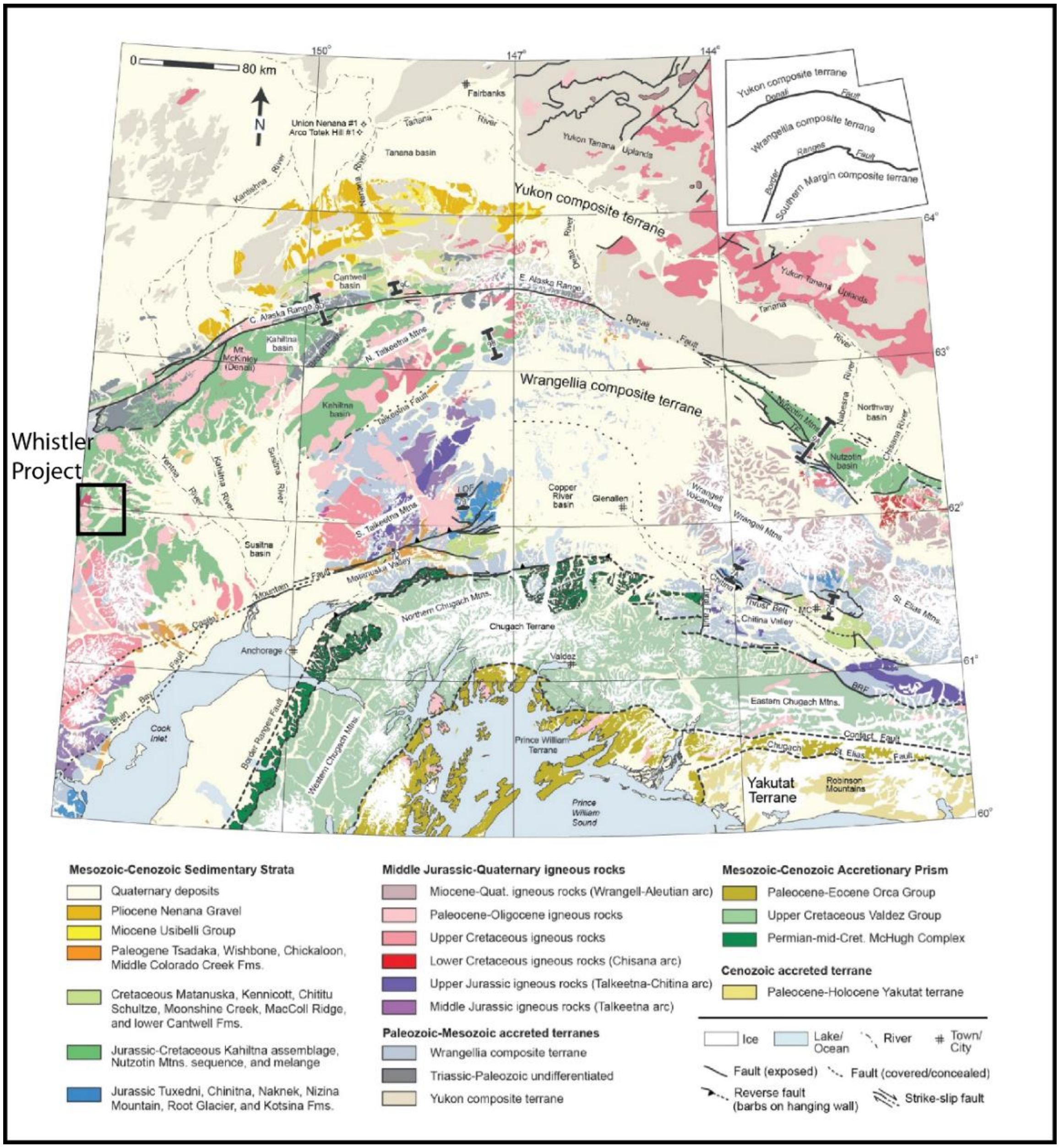
Uplift and shortening of the Kahiltna basin was followed by the construction of a continental-margin arc as defined by an extensive belt of 80 to 60 Ma plutons extending from the Alaska Range south-eastwards into the Coast Range of Canada. In the Alaska Range, these arc rocks are dominated by plutons interpreted to be the deeper roots of subvolcanic and volcanic centers; however extrusive sections are locally preserved.

There are four intrusive suites associated with this epoch of magmatism that are recognized in the Whistler region, including (from oldest to youngest): the Whistler Intrusive Suite or WIS (host to the Whistler Deposit); the Summit Lake Suite; the Composite Suite; and the Crystal Creek Suite, as illustrated in Figure 6-2.

A stratigraphic column in Figure 6-3 illustrates the timing relationship of intrusive suites in the district, and their relationship to host country rocks.

The Whistler Intrusive Suite consists of intermediate to mafic extrusive and intrusive rocks, including diorite porphyries. These diorite porphyries are host to, and genetically associated with, gold-copper porphyry mineralization in the Project area. This is the only suite where comagmatic extrusive rocks and shallow subvolcanic intrusive rocks are recognized in the region. On a district scale the intrusions generally occur as sills and less commonly as dikes and small stocks. Hornblende Ar-Ar dating of Whistler diorite porphyry gives an age of 75.5 ±0.3 Ma (Layer and Drake, 2005) and mapping shows Whistler diorite intruding extrusive andesite. Subsequent U-Pb age dating of zircons from the mineralized diorite porphyry in the Whistler Deposit, and other mineralized porphyries on the Project, indicate igneous ages of 76.36 ±0.3 Ma (Hames, 2014). One of the least altered diorite porphyry intrusions located on the Whistler Ridge has a hornblende Ar-Ar age date of 75.5 ±0.3 Ma (Young, 2005).

Figure 6-1: Regional Geological Map of South-central Alaska



Source: Trop and Ridgeway, 2007

The Summit Lake intrusions are regionally represented by 74 to 61 Ma calc-alkaline granodiorite to diorite, becoming more monzonitic and of alkali-calcic affinity in the Whistler area. East and northeast from Whistler, these intrusions are associated with local gold prospects and have been called the Kichatna plutons and more locally, the "Old Man Diorite."

The Composite Plutons include the Emerald, Mount Estelle, Stoney, and Kohlsaat plutons, and are locally associated with gold mineralization. The Composite Plutons are seen to be somewhat concentrically zoned magmatic series, with an early border phase of alkaline mafic to ultramafic rock, inwards towards less alkaline monzonites to granites. The common age range is 67 to 64 Ma.

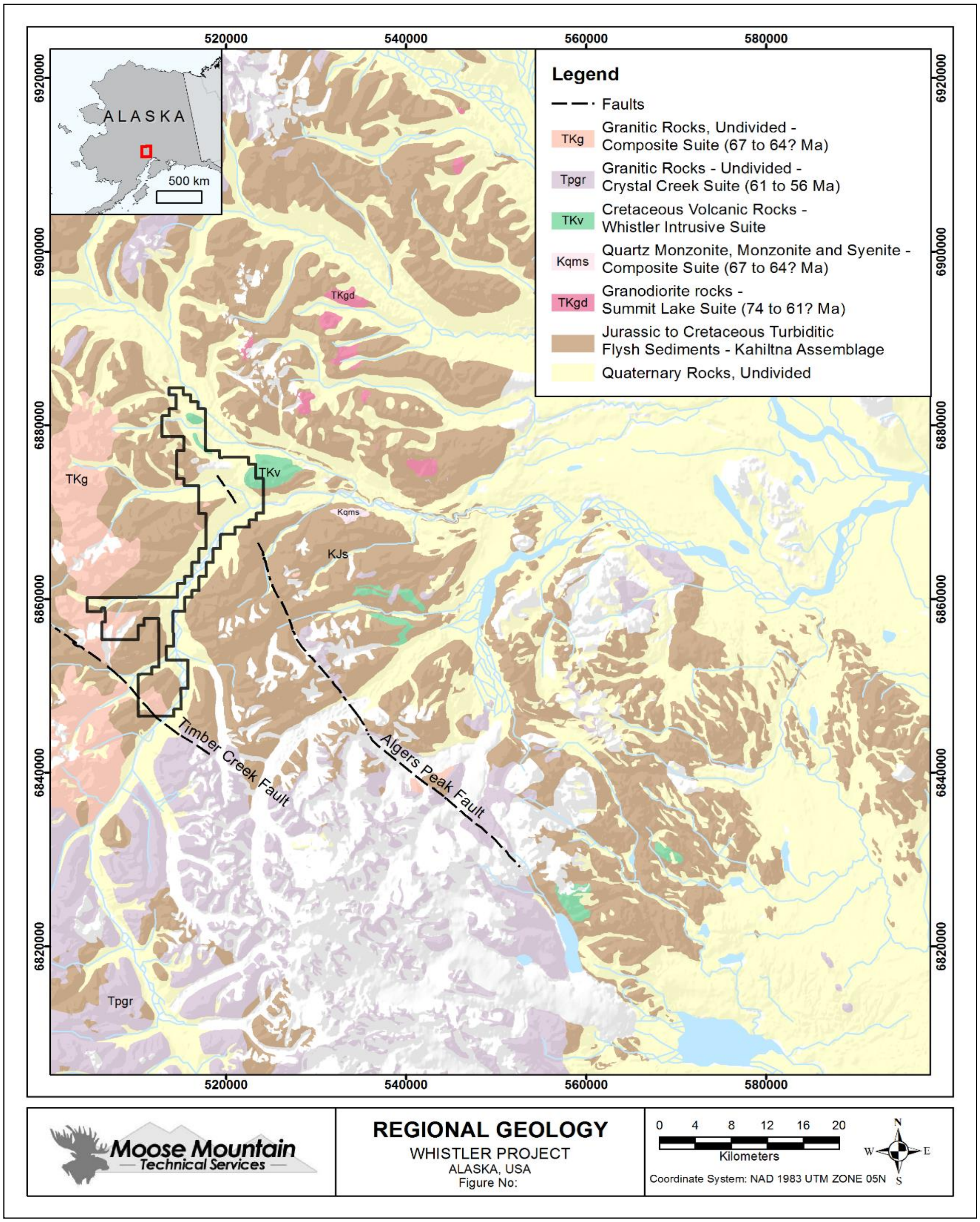
The regional geology of the Whistler project area is shown in Figure 6-2. The Crystal Creek sequence, located south of Whistler, is mainly calc-alkaline granite or rhyolite and ranges in age from 61 to 56 Ma. More mafic rocks, including the 61 Ma Porcupine Butte andesite and Bear Cub (diorite) pluton, may represent higher level/border phases to the Crystal Creek sequence.

Continental arc magmatism in the Latest Cretaceous is responsible for some of the most significant gold and copper-gold deposits in Alaska. These include the Pebble gold-copper porphyry deposit (89 Ma; Schrader et al, 2001), the Donlin Creek gold deposit (70 Ma; Szumigala et al, 2000), the Fort Knox gold deposit (95 to 89 Ma; Mortenson et al, 1995), and the Livengood gold deposit (Late Cretaceous). The property geology of the Whistler area is well-documented and described in detail by Young (2005) and Franklin (2007). A stratigraphic column in Figure 6-3 illustrates the timing relationship of intrusive suites and their relationship to host country rocks at the property scale.

The Whistler Project contains three separate magmatic related gold ± copper ± silver mineral systems identified to date.

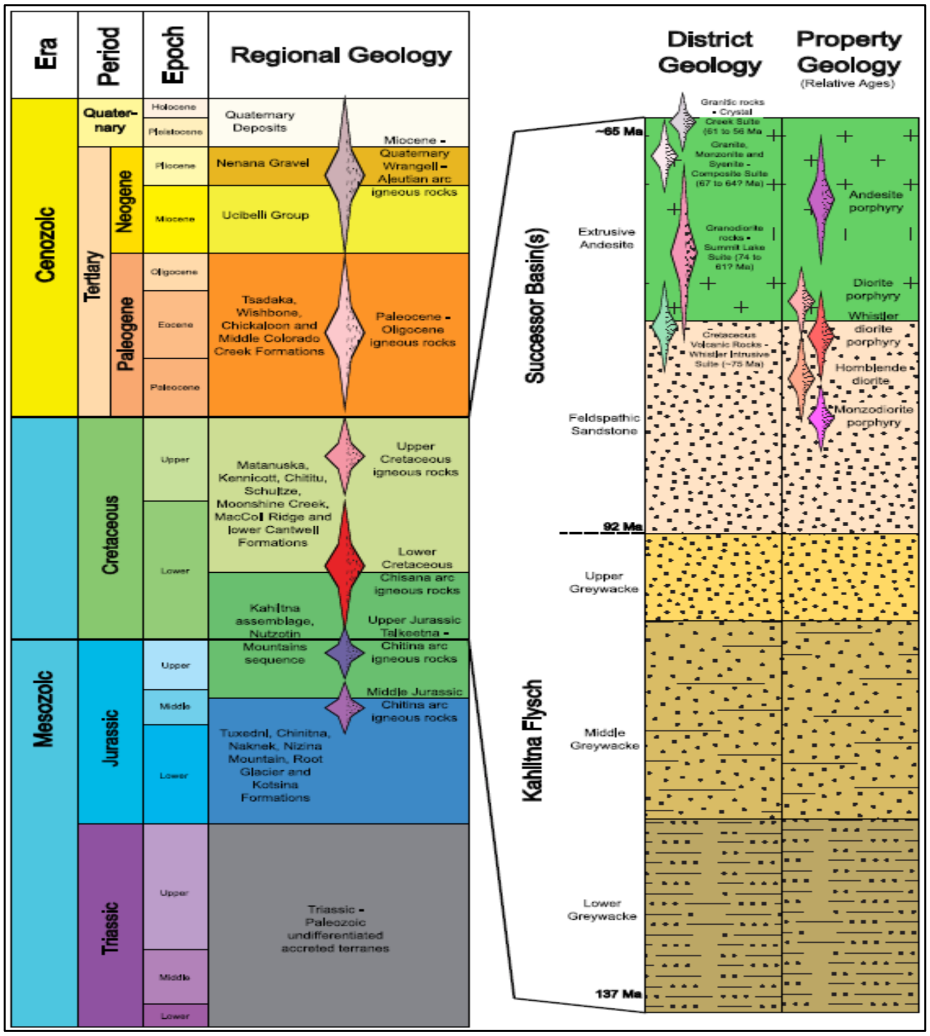
1. **Whistler–Raintree mineral system**, also known as the "Whistler Orbit", which includes the Whistler and Raintree West mineral deposits (see Section 14) which are hosted within the broader Whistler Orbit intrusive center, comprising multiple additional mapped porphyry intrusions spread over an area of approximately 5 x 5 km, interpreted as a classic 'porphyry cluster' with potential for additional gold ± copper ± silver mineralization to be discovered.
2. **Island Mountain mineral system** – encompasses the known Island Mountain deposit plus several additional porphyry or intrusion related gold targets over an area of mapped intrusive rocks with diameter of +3 km.
3. **Muddy Creek mineral system** – a large gold-in-soil geochemical footprint over an area of approximately 6 km x 4 km with an intrusion-related gold geochemical signature.

Figure 6-2: Regional Geology of the Whistler Project Area



Source: Wilson, et al., 2009

Figure 6-3: Stratigraphic Column of the Whistler District and Property Geology

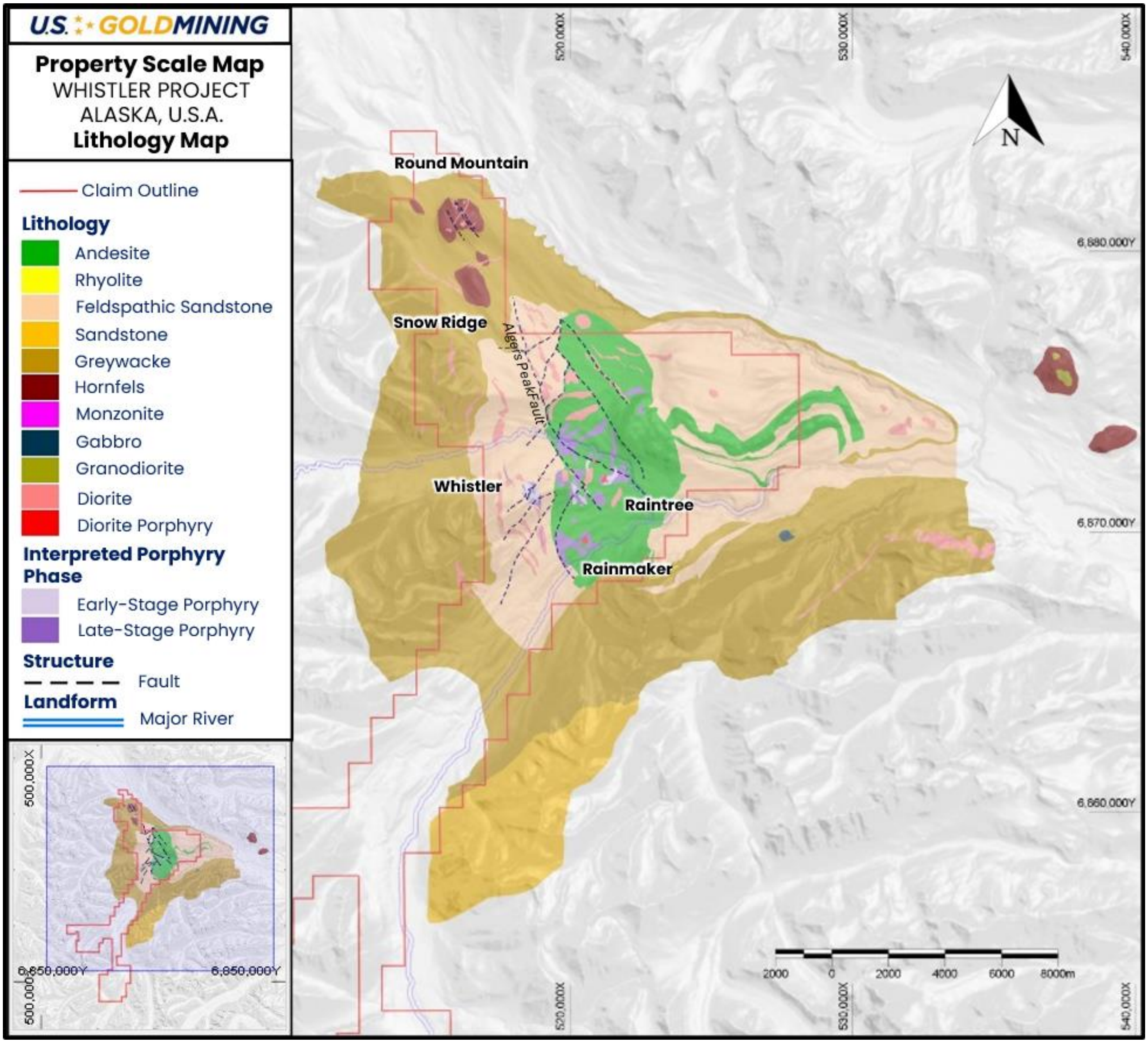


Source: Young, 2005 and Hames, 2014

6.1.1 Whistler – Raintree Mineral System

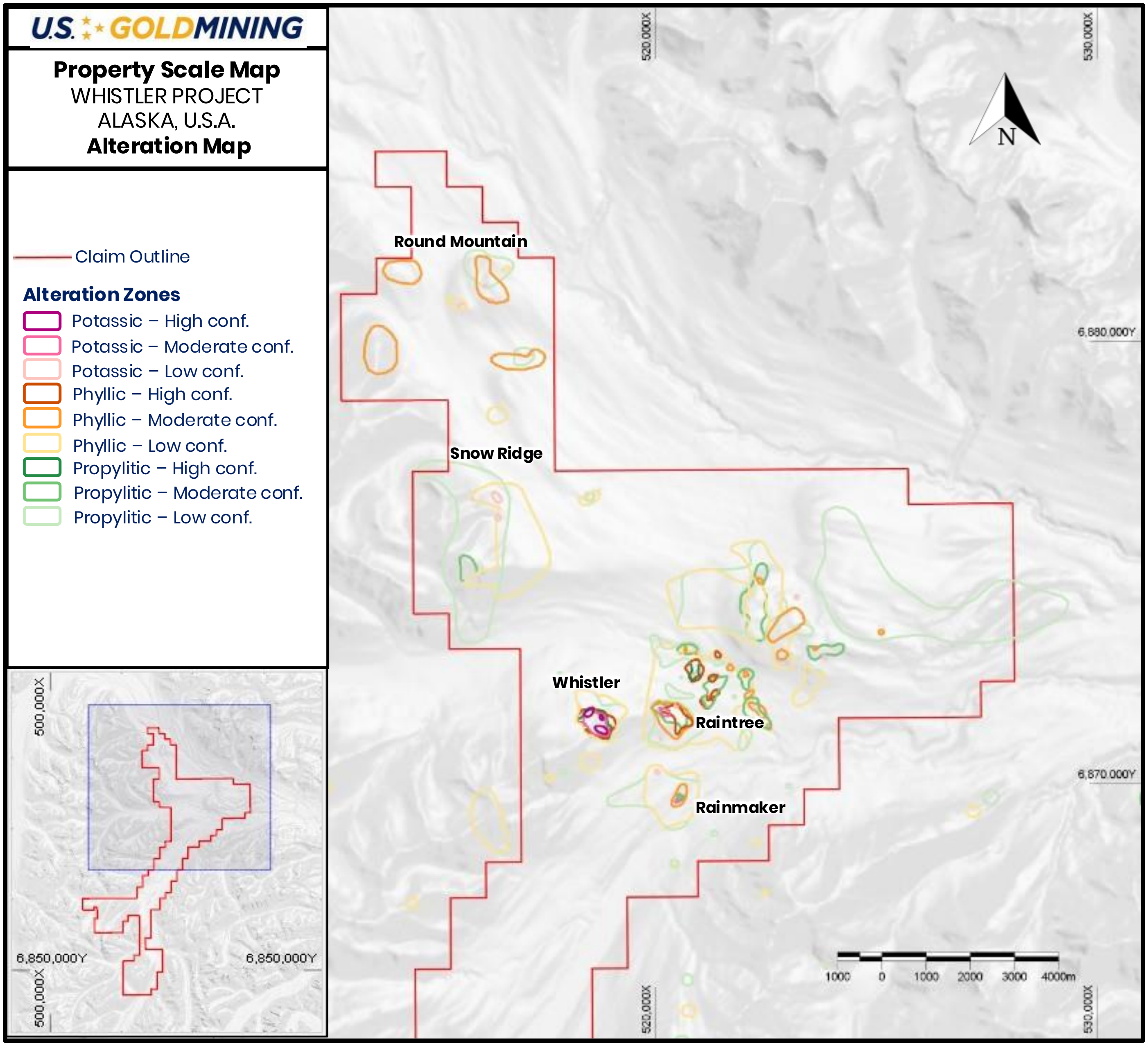
The bulk of the Whistler property is underlain by flysch sediments of the Kahiltna assemblage, while the Whistler Corridor is dominated by a largely fault bounded block of andesitic volcanic rocks, interpreted to represent a local volcanic-dominated basin as illustrated in Figure 6-5 The sedimentary and volcanic rocks are host to a variety of dioritic to monzonitic dykes, sills, and stocks of the WIS. Many of the low-lying areas in this region are covered by 5 to 15 m of glacial till, and hence much of the geological map is based on drilling and interpretation of geophysical data. Relogging of historical drill holes and more recent drilling by U.S. Goldmining has produced an updated bedrock lithology map of the Whistler area with interpreted lithology (Figure 6-4) and hydrothermal alteration assemblages related to porphyry-style mineralization (Figure 6-5).

Figure 6-4: Whistler Orbit Interpreted Bedrock Lithology Map, Whistler Project



Source: Equity, 2025

Figure 6-5: Whistler Orbit Porphyry-Related Mineral Alteration Map



Source: Equity, 2025

The Whistler Deposit is hosted by a multi-phase diorite porphyry intrusion, belonging to the WIS, nested within sediments of the Kahiltna flysch package. Several additional mineral prospects in the Whistler Orbit area (Raintree West deposit, Rainmaker drill target) are hosted by similar diorite porphyry intrusive centers within the volcanic (andesitic) basin. Age dating of mineralized and barren diorite porphyry units on the Whistler ridge indicates that magmatism occurred at approximately 76 to 75 Ma (Layer & Drake, 2005; Young, 2005; Hames, 2011).

The mineralogy and composition of the intrusive rocks and the andesitic volcanic rocks are quite similar, suggesting that they are broadly comagmatic (Young, 2005). Mapping implies monzodiorite porphyry and hornblende diorite suites intruded prior to eruption of extrusive andesites and therefore is older than the Whistler diorite porphyry. Hornblende Ar-Ar dating indicates unmineralized diorite porphyry is likely a later phase of Whistler diorite porphyry (Hames, 2014). Andesitic porphyry is observed to cut all phases of diorite porphyry (Young, 2005) and can be assumed to be the youngest intrusive rock at the Whistler property.

Inversion modeling of the airborne geophysical data suggests that there is a large >5 km diameter wide causative batholith possibly situated 2-3 km below the modern topographic, and that shallower diorite porphyry intrusive centers represent hypabyssal apophyses (stocks and dykes) emanating from this deep causative batholith.

The detailed geology of volcanic stratigraphy, and the distribution and geometry of the WIS intrusive phases, remains uncertain, largely due to glacial till cover, and compounded by the extensive amount of texturally destructive, hydrothermal alteration. Broadly comagmatic volcanic rocks are comprised of coherent andesites and volcanic breccias that define a variety of depositional facies. Based on the occurrence of common argillaceous interflow sediments Young (2005) inferred a subaqueous marine setting for the bulk of the volcanic rocks. In the eastern Long Lake Hills area, volcanic flows are interbedded with feldspathic sandstones, and Young (2005) interpreted this to represent the onset of volcanism in a shallow marine setting.

In addition to these extrusive rocks, a large volume of the volcanic rocks is interpreted to be comprised of porphyritic, subvolcanic units, as either large stocks, sills or dykes. These subvolcanic units can be difficult to differentiate from stratigraphically comformable volcanic rocks, particularly porphyritic flows, and especially in areas of intense texturally destructive phyllic alteration. The stratigraphy of the volcanic rocks is currently unresolved. The current geological map only differentiates ‘least-altered’ from ‘altered’ volcanic rocks based on relogging of historical drill holes and recent top-of-bedrock drilling by U.S. Goldmining (Figure 6-5). All the volcanic and subvolcanic rocks encountered in drilling are magnetic when they are least altered, however magnetism can be destroyed due to sulfidation during phyllic alteration.

In addition to least-altered volcanic rocks, ‘magnetic high’ anomalies also occur in association with northwest-elongated linear to oval-shaped diorite dykes and stocks hosted by flysch sediments and in association with zones of near-surface secondary magnetite alteration and veining, such as the Whistler deposit, and the Rainmaker and Raintree North deposits.

The bulk of the flysch sediments on the Project area have north to northeast striking and steeply dipping bedding orientations due to compressional deformation that resulted in chevron-style folding. Fold limbs are typically moderate to steep or overturned (Young, 2005). A dioritic sill, exposed on the Whistler Ridge, apparently intercalated with Kahiltna sediments, is likewise folded suggesting that an earlier phase of magmatism which pre-dated regional deformation and the main WIS magmatism.

Several northeast-trending faults have been interpreted based on topographic linear features and the truncation and offset of magnetic features. These are the earliest structure features on the property since they are truncated by north-northwest-oriented faults with left-lateral offset, such as the Alger Peak Fault (Figure 6-4).

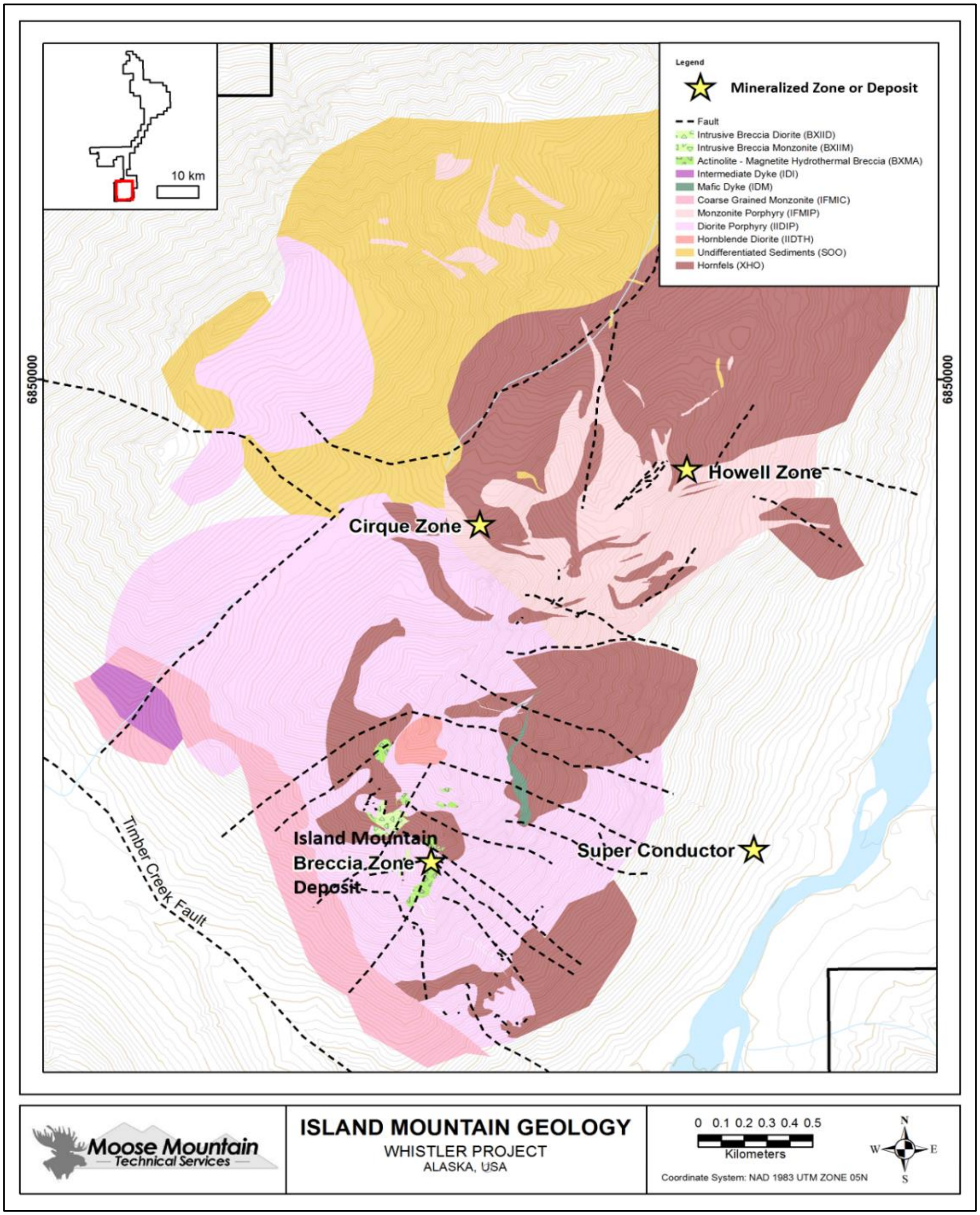
6.1.2 Island Mountain Mineral System

The Island Mountain area is comprised of a suite of nested intrusions, ranging compositionally from hornblende diorite to hornblende-biotite-monzonite, emplaced within flysch sediments of the Kahiltna assemblage as illustrated in Figure 6-6. Texturally, these intrusions range from equigranular to strongly porphyritic, suggesting a relatively high-level of emplacement typical of the porphyry environment.

Unlike the Whistler area, no coeval volcanic rocks are recognized. Based on limited whole-rock geochemistry (Young, 2005) the Monzonite at Island Mountain plots within the silica-saturated alkalic field of Lang et al. (1995) and is the intrusive equivalent of trachyandesite on a total alkali versus silica diagram. This suite of intrusions is mapped as part of the circa 67 to 64 Ma Composite Suite of intrusions, like the Muddy Creek area, however recent age dating suggests some complexity with dates ranging from 77 Ma down to 64 Ma (Gross, 2014). Compared to Muddy Creek, the intrusive rocks at Island Mountain are generally more mafic (diorite and monzonites as opposed to quartz monzonite and granites at Muddy Creek), are magnetite-bearing rather than ilmenite-bearing, are commonly more porphyritic rather than coarse equigranular, lack the strong, pervasive gold-arsenic association, and lack the evenly distributed northwest-oriented sheeted fracture set that typifies mineralized structures at Muddy Creek. For these reasons, it is likely that igneous rocks at Island Mountain represent a unique intrusive suite separate from the Composite Suite.

This unique intrusive center is broadly situated at the intersection between the regionally significant northwest-striking Timber Creek Fault, which can be traced for tens of kilometers, and the Skwentna River valley, postulated as a possible fault zone (Young, 2005). The bulk of the nested intrusions occur on the southeast side of Island Mountain, and this is where sediments in the contact metamorphic aureole of these intrusions are hornfelsed. The hornfels, especially on the southwest corner of Island Mountain, occur as irregular rafts and possibly roof pendants that appear to form a slope-parallel skin of country rock that demarks the roof zone of this intrusive complex. Sediments consist of dark mudstone, shale, thin-to-medium-bedded siltstone and dark grey sandstone and minor dirty calcareous sedimentary beds and a few local thin pebble conglomerate units. These units predominate in the northwest portion of Island Mountain.

Figure 6-6: Property Geology of the Island Mountain Area



Source: MMTS, 2015, modified from Roberts, 2011a.

The earliest recognized intrusive phase is the Island Mountain Diorite Porphyry. This unit has been observed to be cut by all other igneous units and is the host to gold-copper porphyry mineralization associated with intrusive and hydrothermal breccias at the Island Mountain Deposit (previously referred to as the Breccia Zone).

The next most volumetrically significant intrusive phase is a Monzonite Porphyry (IFMIP) that occurs in the northeast corner of Island Mountain, and which is generally the host of gold-copper porphyry-style mineralization at the Cirque and the Howell zones. Unlike the Diorite Porphyry, this unit contains magnetite phenocrysts and is thus well delineated by airborne magnetic survey data.

In the Breccia Zone, Diorite- and Monzonite-cemented intrusive breccias occur as subvertical, 100 to 150 m diameter, subcircular to irregularly shaped pipes that grade into actinolite-magnetite-cemented hydrothermal breccias with pyrrhotite-pyrite-chalcopyrite mineralization, which together define magmatic-hydrothermal conduits that host the bulk of gold-copper porphyry mineralization in this area. Not all the Intrusive Breccia bodies are altered or mineralized, suggesting that either some of these breccias post-date the main phase of mineralization, or that some pre-mineral intrusive breccias were not affected by hydrothermal fluid. Together, these intrusive and hydrothermal breccias have been the focus of most of the exploration drilling at Island Mountain since 2009. A series of these breccias extend discontinuously for 700 m from the Breccia Zone on a north-northwest trend along the southwestern slope of Island Mountain. The Breccia Zone also contains narrow, pencil-like bodies of Coarse Porphyritic Hornblende Diorite that are syn to post gold-copper mineralization.

This corridor of breccias is flanked by strong pervasive albite alteration with local zones of vein and disseminated pyrrhotite that constitutes significant Au-only mineralization within and flanking the Breccia Zone. Similar intrusive and hydrothermal breccias with peripheral sodic alteration and pyrrhotite mineralization occur in areas of gold and copper soil anomalies at the Howell Zone, suggesting the occurrence of multiple magmatic-hydrothermal centers. The Howell Zone remains untested by drilling.

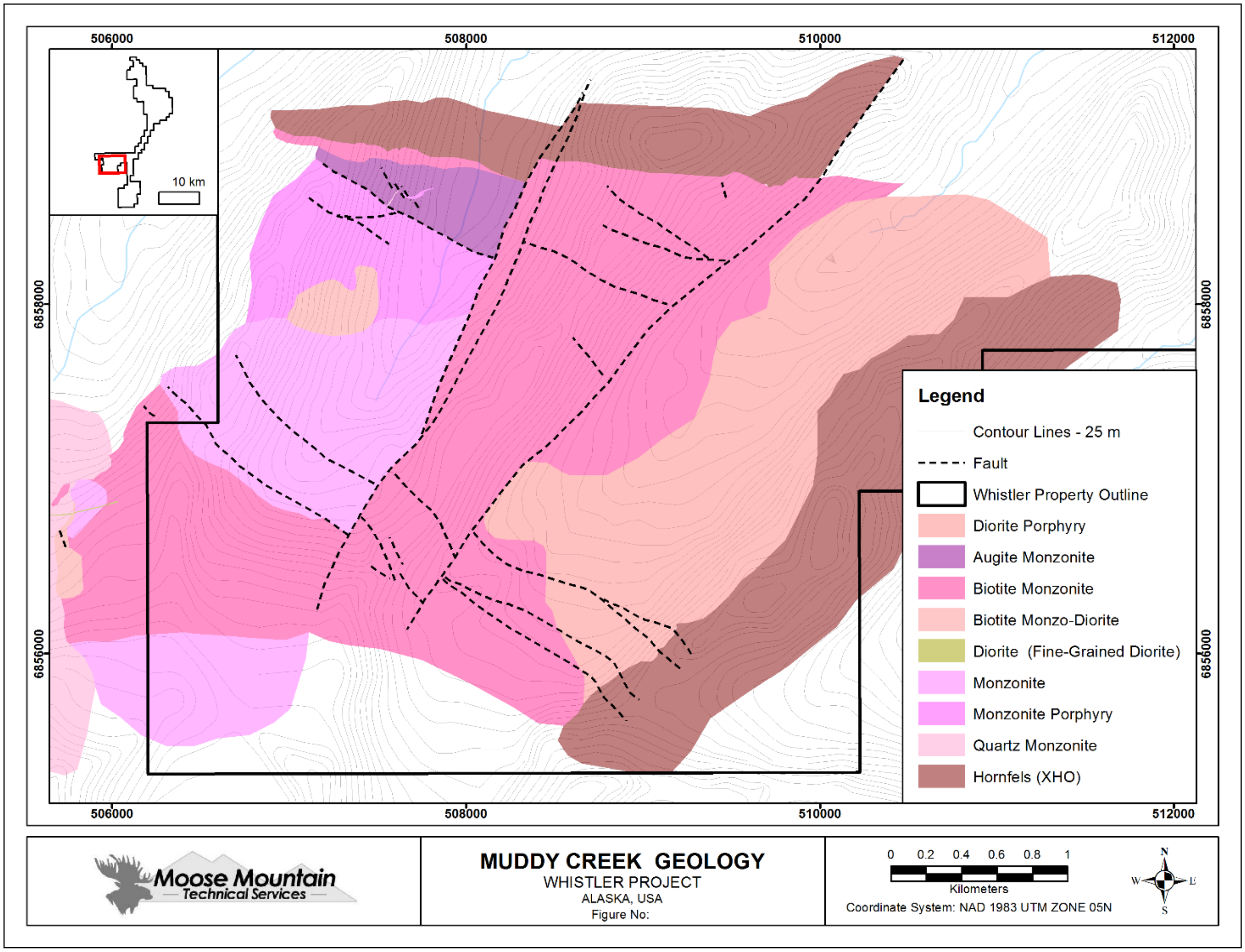
The last volumetrically significant phase of magmatism is represented by a coarse-grained equigranular monzonite that occurs as a northwest-striking dyke or sill exposed near the base of slope on the southwestern side of Island Mountain. This unit lies adjacent and strikes parallel to the regional Timber Creek Fault, suggesting a possible regional control on the emplacement of this unit. Likewise, all the above-mentioned units are cut by narrow, post-mineral, fine-grained mafic to intermediate dykes that generally strike to the northwest and dip steeply.

6.1.3 Muddy Creek Mineral System

Muddy Creek is in rugged terrain along the western edge of the Project and is comprised of several steep, north-east facing U-shaped glacial valleys separated by razor-back ridges with small remnant glaciers at the heads of each valley. This prospect is largely underlain by a monzonitic intrusive complex, part of the Composite Suite (or Estelle Suite) of intrusions that were emplaced within sediments of the Kahiltna Assemblage in the late Cretaceous (Figure 6-7).

An argon-argon analysis of igneous biotite from a granodiorite on the western margin of the intrusive complex returned an age date of 67.4 Ma ± 0.4 Ma (Solie et al., 1991a). A steep, east-west trending contact between the intrusive complex and hornfels sediments is well-exposed in the ridgelines in the northern portion of the prospect and is comprised of a conspicuous and extensive red brown color anomaly. Hornfels also comprises the eastern contact of the intrusive complex.

Figure 6-7: Geological Map of Muddy Creek



Source: MMTS, 2015, modified from Roberts, 2011c.

The bulk of the geological mapping at Muddy Creek was completed by Kennecott and the following descriptions are from Young (2005). The core of the intrusive complex is monzonitic, grading outwards to progressively more mafic and older intrusive phases (Crowe et al, 1991), with pendants of ultramafic rocks at the margins (Millholland, 1998). The pluton intrudes very steeply north-dipping sedimentary rocks of the middle Graywacke Sandstone subunit and Tabular Sandstone unit. Local matrix-supported pebble conglomerate and spherical concretions along Muddy Creek support a correlation with the Tabular Sandstone unit.

Most of the Mount Estelle pluton consists of biotite-monzonite, with an increasing proportion of augite phenocrysts towards the margins. Monzonite is medium- to coarse-grained and idiomorphic granular and occurs at the central and southern portions of the mapped area at Muddy Creek. Mafic minerals, principally biotite books (to 5 mm) and subordinate to absent stubby dark augite generally constitute 15 to 35% of the monzonite. Twinned 3 mm to 1 cm orthoclase phenocrysts are a fundamental component. Groundmass consists of a medium-grained equigranular mixture of feldspar and quartz. Rounded xenoliths are rare, but widespread, and consist of biotitized sediments and more strongly mafic (biotite and augite)-rich intrusive rock of earlier intrusive phases. Intrusion breccia's with rounded clasts are a very local feature as are sinuous to linear aplitic dikes.

6.2 Mineralization

Exploration on the Project by Kennecott, Geoinformatics, Kiska and U.S. GoldMining has identified three mineral deposits containing porphyry-style gold ± copper ± silver ± lead ± zinc mineralization. These include the Whistler, Raintree West, and Island Mountain deposits as shown in Figure 6-8. The Whistler-Raintree and Island Mountain areas host multiple additional porphyry-like prospects defined by drilling, anomalous soil geochemistry, alteration assemblage, veining, surface rock grab sampling, IP chargeability/resistivity anomalies, airborne magnetic anomalies, and/or airborne electromagnetic anomalies. These include Raintree North, Rainmaker, Round Mountain, Puntilla, Snow Ridge, Dagwood, Super Conductor, Howell Zone, and Cirque Zones. The Muddy Creek area represents an additional exploration target with the potential to host low-grade, bulk tonnage intrusion-related gold mineralization.

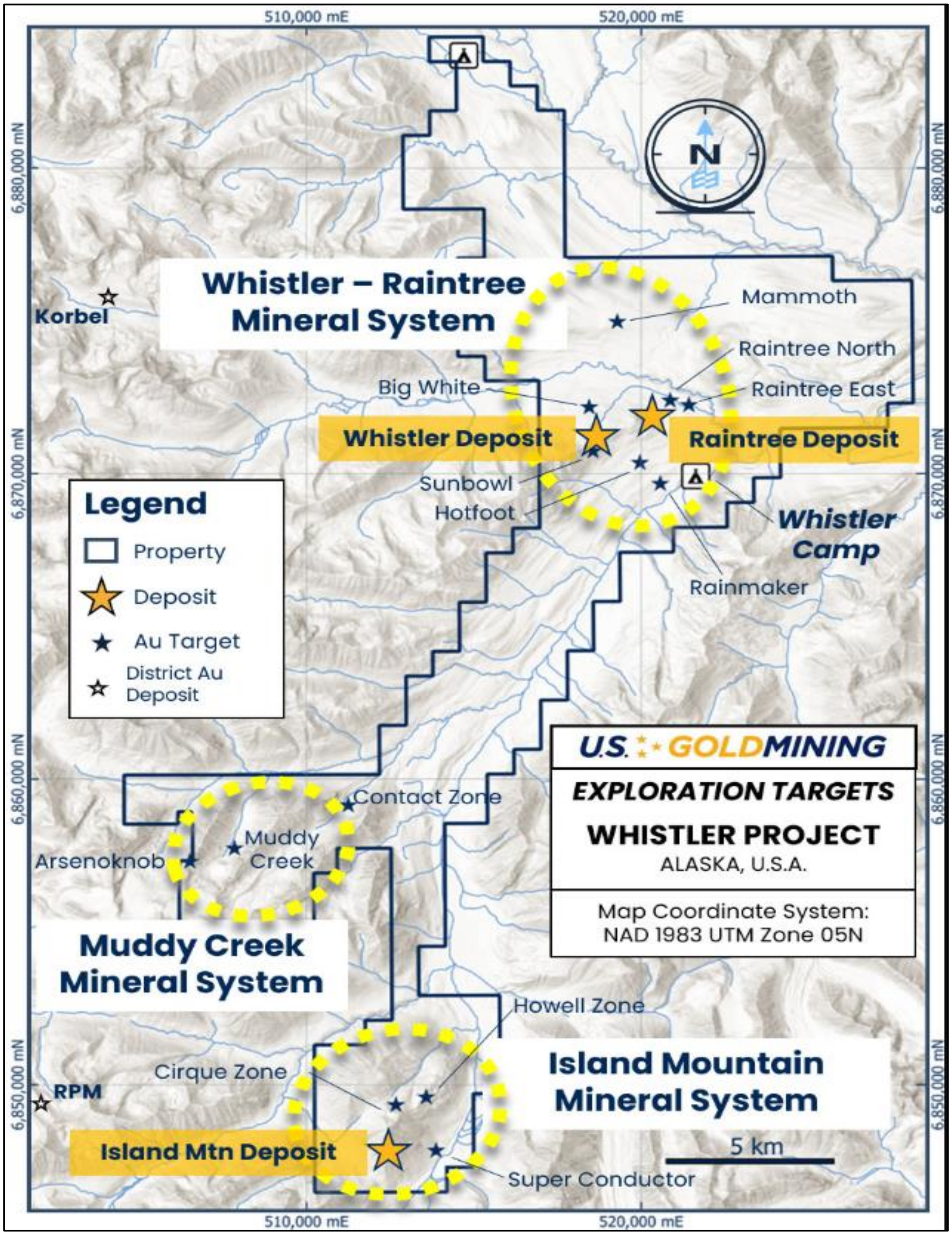
6.2.1 Whistler Property-scale Mineralization Overview

The Whistler Deposit is hosted within the late Cretaceous Whistler Intrusive Suite, a composite of diorite stocks and dykes. Although local variations may exist, the Whistler Deposit and prospects in the Whistler Orbit (Raintree West, Raintree North and Rainmaker) display similar patterns of alteration, vein paragenesis, and mineralization styles. This suggests that these spatially separate porphyry centers may share a common genetic association. These deposits are hosted by pulses of diorite porphyry dykes and stocks nested within broader intrusive complexes. Geophysical inversion models of the airborne magnetic data suggest that these intrusive centers occur as pipe-like features emanating from cupulas along the upper surface of a deep causative batholith.

Mineralization at Island Mountain is hosted by several intrusive phases. The earliest recognized phase is the Island Mountain Diorite Porphyry, which hosts the gold-copper at the Island Mountain Deposit. Mineralization within the Island Mountain Diorite Porphyry is primarily hosted within subvertical Diorite- and Monzonite-cemented intrusive breccia pipes that grade into hydrothermal breccias. The breccia-hosted mineralization also contains narrow, pencil-like bodies of coarse Porphyritic Hornblende Diorite, which are interpreted to be syn- to post-gold-copper mineralization. In addition, gold-copper porphyry-style mineralization found at the Cirque and Howell prospects is hosted in the Island Mountain Monzonite Porphyry.

Mineralization related the Muddy Creek prospect is primarily associated with a late Cretaceous monzonitic intrusive complex part of the Estelle Composite Suite. This complex, also identified as the Mount Estelle pluton, is emplaced into sediments of the Kahiltna Assemblage. The pluton exhibits zonation, featuring a monzonitic core that grades outwards into progressively older and more mafic intrusive phases. Pendants of ultramafic rocks are also noted at the margins. Other minor intrusive features include rare but widespread rounded xenoliths of biotitized sediments and earlier mafic intrusive rocks, as well as very local intrusion breccias and linear aplitic dikes.

Figure 6-8: Whistler Project Mineral Systems and Exploration Targets

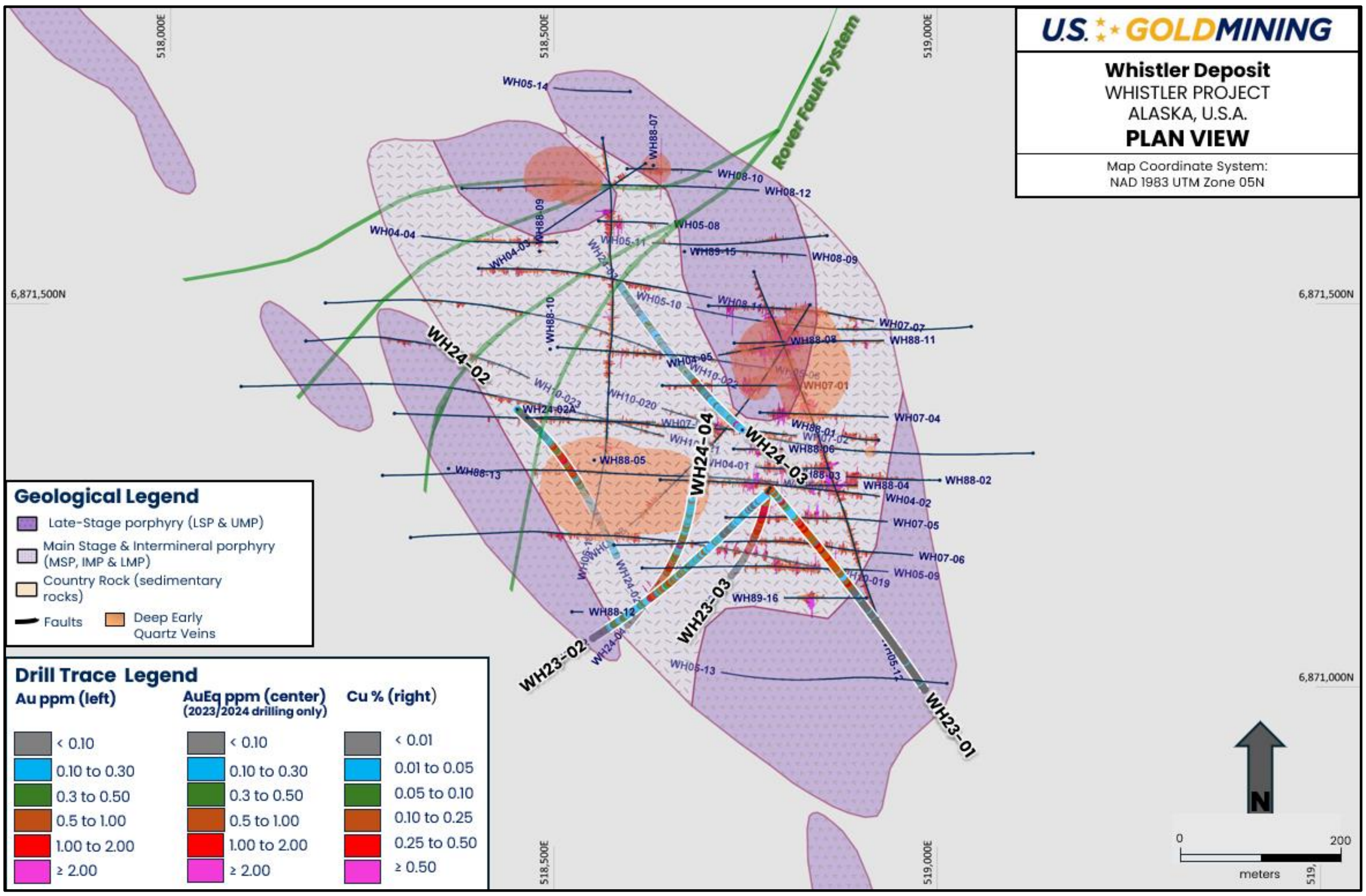


Source: U.S. GoldMining, 2026

6.2.2 Mineralization: Whistler Deposit

Gold and copper mineralization at the Whistler Deposit is hosted by a Late Cretaceous, multi-phase diorite porphyry intrusive complex that intrudes the feldspathic sandstone unit of the Kahiltna assemblage (Figure 6-3 and Figure 6-9). The feldspathic sandstone is comprised of sandstone with minor interbeds of mudstone, siltstone, and conglomerate. Sedimentary bedding in the vicinity of the deposit primarily strikes to the northeast and dips steeply to the northwest.

Figure 6-9: Geological Map of the Whistler Deposit

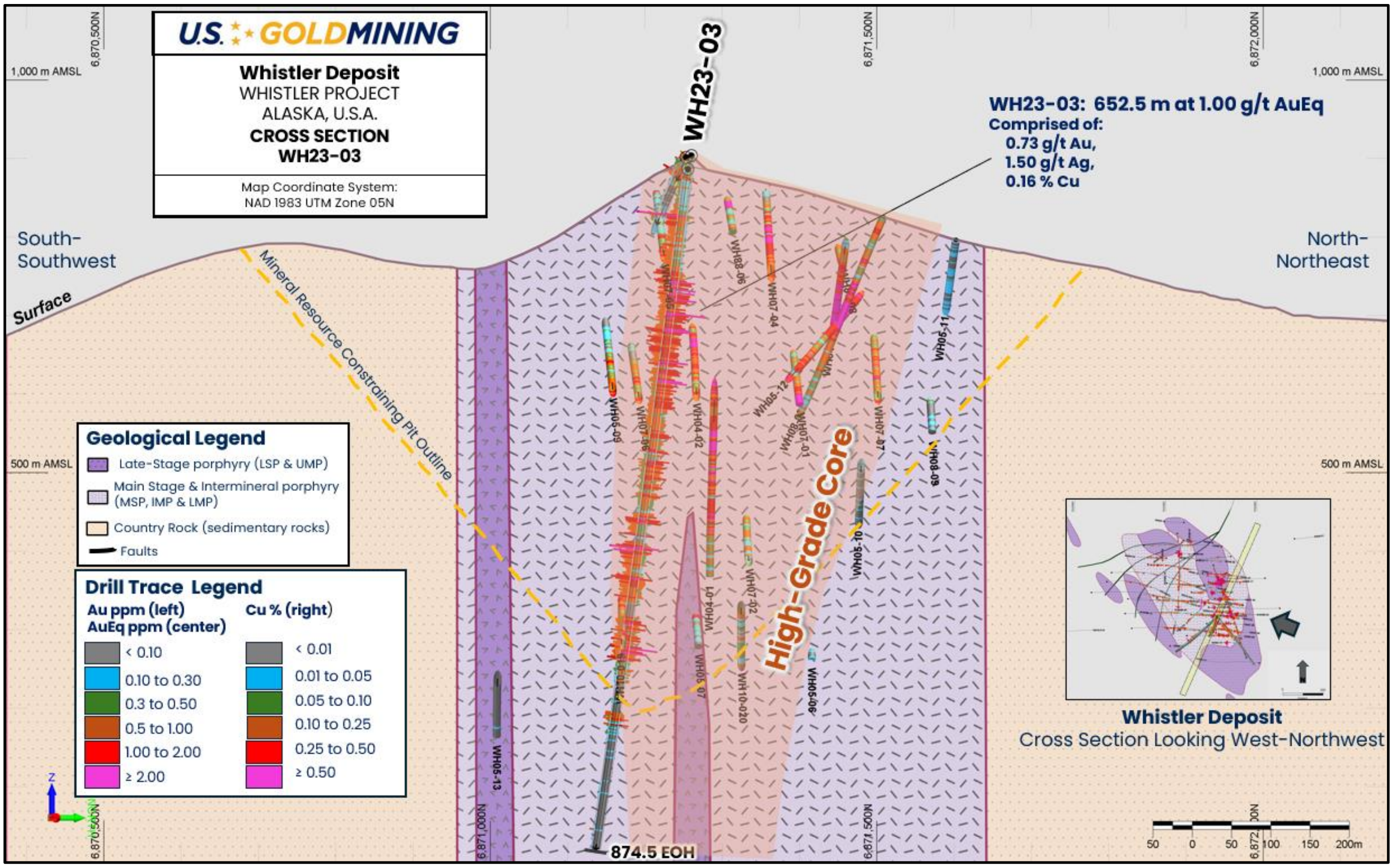


Source: Equity, 2024

6.2.2.1 Porphyry Intrusions

The diorite porphyry intrusive complex is ovoid-shaped and vertically plunging (Figure 6-10). The long axis of the ovoid is 700 m long and oriented in a northwest-southeast direction. The short axis of the ovoid is 500 m wide and oriented in a northeast-southwest direction. Deep drilling indicates that the intrusive complex is open below a depth of 800 m from surface as illustrated in WH23-03 in Figure 6-10.

Figure 6-10: Geological Cross-section – looking WNW of the Whistler Deposit



Source: Equity, 2024

The intrusive complex is composed of at least three diorite porphyry phases that are compositionally and texturally similar: they are comprised of 60%–80%, euhedral to subhedral blocks of plagioclase feldspar phenocrysts (0.2–3.0 mm diameter), 5%–20% hornblende laths (0.2–3.0 mm) that are usually altered to sericite, chlorite, pyrite, or a combination of these, and a fine-grained, granular groundmass of feldspar and minor quartz, that is usually altered to silica, chlorite, sericite, clay or potassium feldspar.

The porphyry events and associated diorite porphyry phases are summarized in Table 6-1. Prior to 2024 previous workers divided the suite broadly into an early Main Stage Porphyry (MSP), a later Inter-mineral Porphyry (IMP) and a late intrusive phase referred to as the Late-Stage Porphyry (LSP). Relogging and geological interpretations of the Whistler deposit by U.S. GoldMining through 2023 and 2024 determined the MSP occurs as narrow dykes within the volumetrically larger IMP as previously defined by Hames (2015). This intrusive timing presented a spatial paradox where the MSP would have intruded the basement rocks first and subsequently be enveloped by a later IMP event. It was subsequently recognized during the 2024 relog program that the previously identified IMP may be an early porphyry phase, now referred to as the Early-Stage Porphyry (ESP). In addition, drilling in 2023 and 2024 identified an additional weakly mineralized Late Mineral Phase (LMP) diorite porphyry that precedes the latter unmineralized LSP and the latest recognized and compositionally distinct Unmineralized Porphyry (UMP) phases. The LMP is poorly characterized, due to its recent recognition, but where it has been observed it forms distinct contacts with prior diorite porphyry phase.

The most significant gold and copper mineralization at the Whistler deposit is hosted by the earlier "productive porphyries" which consists of the ESP and MSP, and to a lesser degree the LMP porphyry phases. The latter intrusive phases which consist of the LSP and UMP make up the "non-productive porphyry" intrusive phase.

Table 6-1: Intrusive Porphyry Events and Phases at the Whistler Deposit

Porphyry Event	Porphyry Phase Name	Early Porphyry Veins (A, AB, B) %	Potassic Alteration	Potassic Intensity	Other Alteration Minerals	Mineralization	Textural Characteristics
Productive Porphyries	ESP	< 5 % (?)	Bio > KFsp	≥ 4 (Strong)	No Mag, Ser ± Clay	Cpy ≥ Py	"rounded" subhedral plagioclase
	MSP	≥ 10%	KFsp > Bio	≥ 3 (Moderate)	Mag ± Bio ± Ab	Cpy > Py	"rounded" subhedral plagioclase, rare qz-eyes
	LMP	None	Bio + KFsp	2 – 3 (Weak – Moderate)	Mag	Cpy ≤ Py	"subhedral" plagioclase, xenoliths of MSP
Non-Productive Porphyries	LSP	None	None	None	Chl ≥ Ser	None	Euhedral plagioclase
	UMP	None	None	None	Gyp, Cal	None	Needle-like hornblende

The MSP phase is most strongly mineralized, characterized by strong potassic alteration and a high abundance of early M-, A-, and B-style porphyry veins (Table 6-1) and corresponding with the highest grading parts of the Whistler deposit. Although the ESP phase may have strong potassic alteration it contains a lower abundance of early porphyry veins, which may correspond to less intense gold-copper mineralization overall. Nonetheless, the ESP (formerly IMP) is the most volumetrically significant porphyry phase, hosting most of the mineralization at the Whistler deposit.

Contacts between the MSP and ESP porphyry phases are indistinct and lack sharp boundaries. Currently, these contacts are determined based on vein types and alteration mineralogy. Due to the compositional and textural similarity of the main stage and early-stage porphyries and hence the difficulty in consistently identifying these stages, and rare occurrence of the LMP phase, the ESP, MSP, and LMP diorite porphyries are currently modeled as a single mineralized porphyry unit. For consistency these phases are therefore referred to as productive porphyry, or collectively the Main Stage Porphyry. Further relogging of drill core, radiometric age dating, and future infill drilling may be able to differentiate these phases more clearly and consistently.

The latter non-productive porphyry suites which includes the LSP and UMP generally are generally below economic cutoff grade or are unmineralized. There are numerous LSP variations characterized by differences in alteration minerals, textures, and accessory minerals. However, these less mineralized porphyries are collectively grouped together in the non-productive porphyry suite. These late intrusions form sharp distinct contacts with the productive porphyry suites, which can be distinguished visually by textural and compositional differences and truncation of metal grade. They occur as narrow, sub-vertical dykes typically from 2 to 10 m wide but may be up to 150 m wide on the south, north and western edges of the MSP. This phase generally has strong pervasive phyllic alteration, and may contain xenoliths or rafts of the MSP, which locally contribute grade.

Table 6-2: Summary of Porphyry Vein Types in the Whistler Porphyry Deposit

Vein Type	Vein Subtype	Main Mineralogy Qtz	Main Mineralogy Mag	Main Mineralogy Py	Other Mineralogy	Alteration Halo	Mineralization	Texture
M	M1	≤ 5 % Qz	≥50 % Mag		± Anhy	Ab-Mag, Ksp-Mag	Cpy ± Bn ± Mo	Banded, sheeted, discontinuous subhedral magnetite
	M2	5-50 % Qz	≥50 % Mag		± Anhy	Ab-Mag, Ksp-Mag	Cpy ± Bn ± Mo	
A	A1	≥50 % Qz	Mag present			Ab-Mag, Ksp-Mag	Cpy	Wormy, diffuse boundary, granular Qz, ductile
	A2	≥50 % Qz	no Mag			Ab-Mag, Ksp-Mag	Cpy	
	A3	≥50 % Qz	no Mag		Mo present	Ab-Mag, Ksp-Mag	Cpy	
A-B	A-B	≥50 % Qz	no Mag			Ab-Mag, Ksp-Mag	Cpy, Py ± Bn ± Mo	e.g. straight edged vein with sugary texture or wormy, sugary quartz vein with a centreline
B	B1	≥50 % Qz	Mag present		± Anhy	± Ab-Mag, ± Ksp-Mag	Cpy, Py ± Bn ± Mo	Sulphide centre line, straight boundaries, 'racing stripe'
	B2	≥50 % Qz	Mag present		Py ± Anhy	Ser, ± Ab-Mag, ± Ksp-Mag	Cpy, Py ± Bn ± Mo	
	B3	≥50 % Qz	no Mag		Py ± Anhy	Ser, ± Ab-Mag, ± Ksp-Mag	Cpy, Py ± Bn ± Mo	
D	D1-2	≤50 % Qz	no Mag	5-50 % Py	± Carb	Ser	Cpy ± Mo ± Sph, ± Gal	Straight, typically sericite halo, Py seams, "E" have colloform Sph, Gal.
	D3 (incl. E)	≤50 % Qz	no Mag	5-50 % Py	Carb	None	Sph, Gal	
	D4-5	≤50 % Qz	no Mag	5-50 % Py	Py	Ser	none	
EDM	EDM	minor Qz	no Mag		Bio, Ksp, Ser	Bio, Ksp, Ser ± And ± Cor	± Cpy ± Bn	Not veins, but are alteration haloes along fractures
SC	SC	No Qz	no Mag		Ser-Chl	Ser	none	Chlorite in SC-veins is elongate in the vein direction and muscovite forms radiating laths.
FeCarb	FeCarb	No Qz	no Mag		Ank	FeCarb, Mag, Hm	none	
G	G	No Qz	no Mag		Gpy	None	none	
C	C	No Qz	no Mag		Cal	None	none	

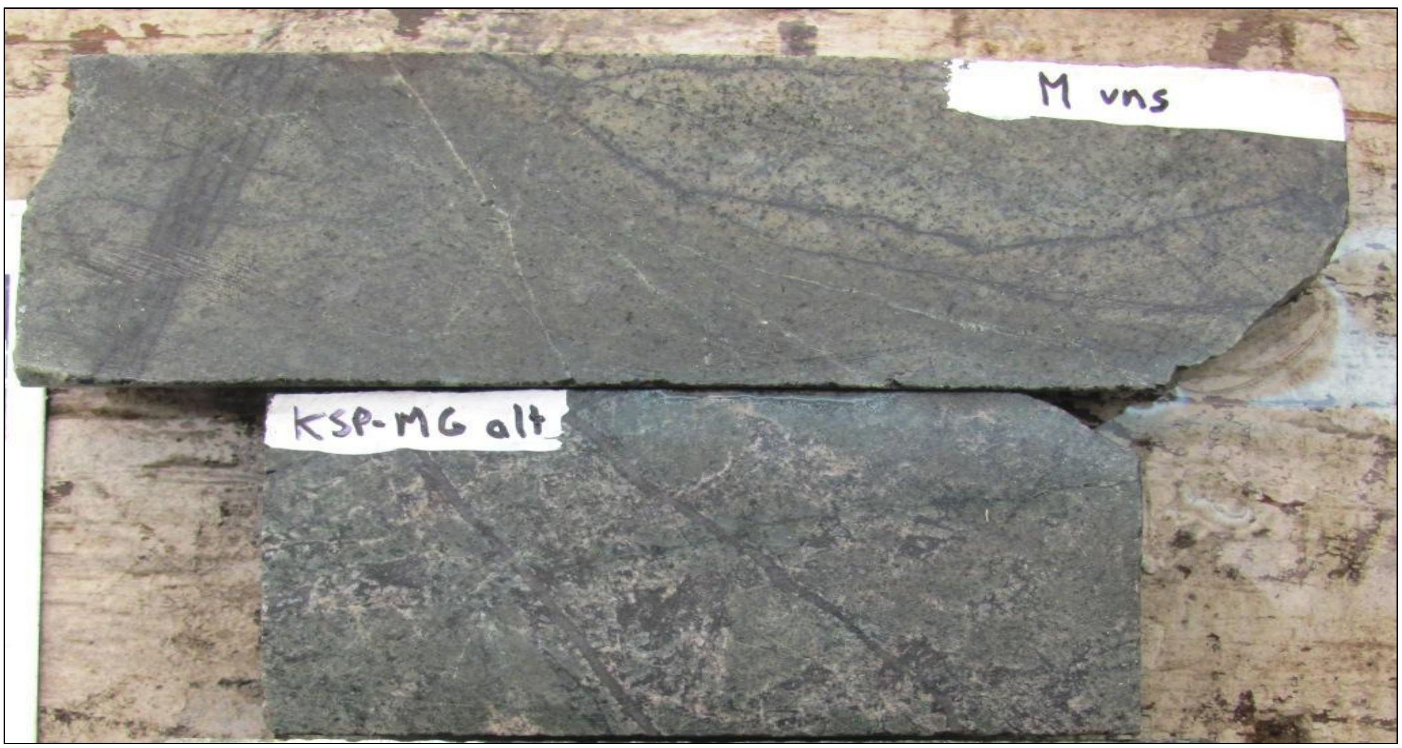
6.2.2.2 Alterations and Veins

The spectrum of A- and B-style quartz veins at the Whistler deposit is summarized in Table 6-2. A-veins are millimeter-wide, sugary quartz ± magnetite with wormy margins. These are generally observed to cut M-veins, however, M-veins have been seen to transition into A-like quartz veins. B-veins are generally comprised of slightly coarser, equigranular quartz with centerline septa of chalcopyrite, and have straight sides. Intense zones of B-style veins form strong stockwork zones are associated with high-grade zones (>1 g/t Au, >0.5% Cu). Potassic alteration and quartz veining may include minor pyrite, yet these zones have relatively low total sulfide content (<1-2%).

The earliest recognized alteration event recognized at the Whistler Deposit and the porphyry prospects in the Whistler area, referred to as "magnetite alteration", occurs as patchy magnetite alteration of mafic minerals (dominantly hornblende and possibly pyroxenes) and narrow, irregular magnetite veinlets (M-veins). Magnetite in this event is occasionally intergrown with trace chalcopyrite. This stage may include the partial replacement of feldspars by secondary K-feldspar, particularly in the selvages to M-veins, and hence may be part of the earliest, weakest stage of potassic alteration (see Figure 6-11). This stage is recognized in both the Main Stage and Early-Stage Diorite Porphyry generally in the core zone of mineralization at the Whistler Deposit. In addition, it has been observed to occur within andesitic volcanic and volcaniclastic rocks within 50 m of similarly altered diorite intrusions in the Whistler Orbit. However, this style of alteration and porphyry veins does not occur within the feldspathic sandstones that host the Whistler Deposit.

The subsequent stage of alteration is potassic alteration, defined by the occurrence of pinkish K-feldspar replacing plagioclase and matrix, which generally occurs as halos to, or pervasively in zones of, A-style and B-style quartz veins. Potassic alteration also includes the replacement of mafic phases by fine-grained secondary "shreddy" biotite, however this is generally difficult to observe due to overprinting chlorite-sericite alteration (see Figure 6-12). Strong potassic alteration (pink rock) is generally accompanied by strong patchy magnetite alteration, and overall, this leads to strong textural destruction such that the rock is mottled pink, black without an obvious porphyritic texture. Potassic alteration is associated with the bulk of gold-copper mineralization, which occurs as chalcopyrite and rare bornite in A- and B-style quartz veins and as fine-grained disseminations in adjacent wall rock (Table 6-2).

Figure 6-11: Magnetite Alteration Observed in the Whistler Deposit



Note: This photo displays the irregular M-veins in dark magnetite alteration of mafics (upper) and pervasive pink-black blotchy k-feldspar and magnetite alteration (lower) with wormy quartz + magnetite + chalcopyrite A-veins (Whistler Deposit). Source: MMTS, 2015

Figure 6-12: Potassic Alteration Observed in the Whistler Deposit

Notes: This image displays a classic B-style quartz vein with a chalcopyrite-filled centerline cutting an irregular, wormy A-style quartz vein (Whistler Deposit, WH 08-08, ~123.0 m). Source: MMTS, 2015

In general, core zones of potassic alteration and Au-Cu mineralization are partially to completely overprinted by chlorite-sericite alteration. This "green rock" alteration is ubiquitous and the most macroscopically obvious alteration in zones of Au-Cu mineralization, even though it is a later event. As shown in Figure 6-13, bright green chlorite replaces secondary biotite and any primary mafic phases remaining, and waxy green sericite replaces feldspars. Pyrite is part of this assemblage, partly replacing mafics and magnetite. Calcite or carbonate may be part of this assemblage, as well as trace epidote. Kennecott referred to this alteration assemblage as “intermediate argillic", which is equivalent to SCC alteration in the porphyry literature (Sillitoe, 2010). Kiska interpreted the chlorite-sericite alteration to be transitional to phyllic alteration, overprinting (telescoping) and immediately peripheral to core zones of mineralization. This pervasive style of alteration is not obviously associated with any veining event, however there is a continuum of glassy quartz veins with pyrite>>chalcopyrite + molybdenite that appears to only occur in zones of chlorite-sericite and phyllic alteration.

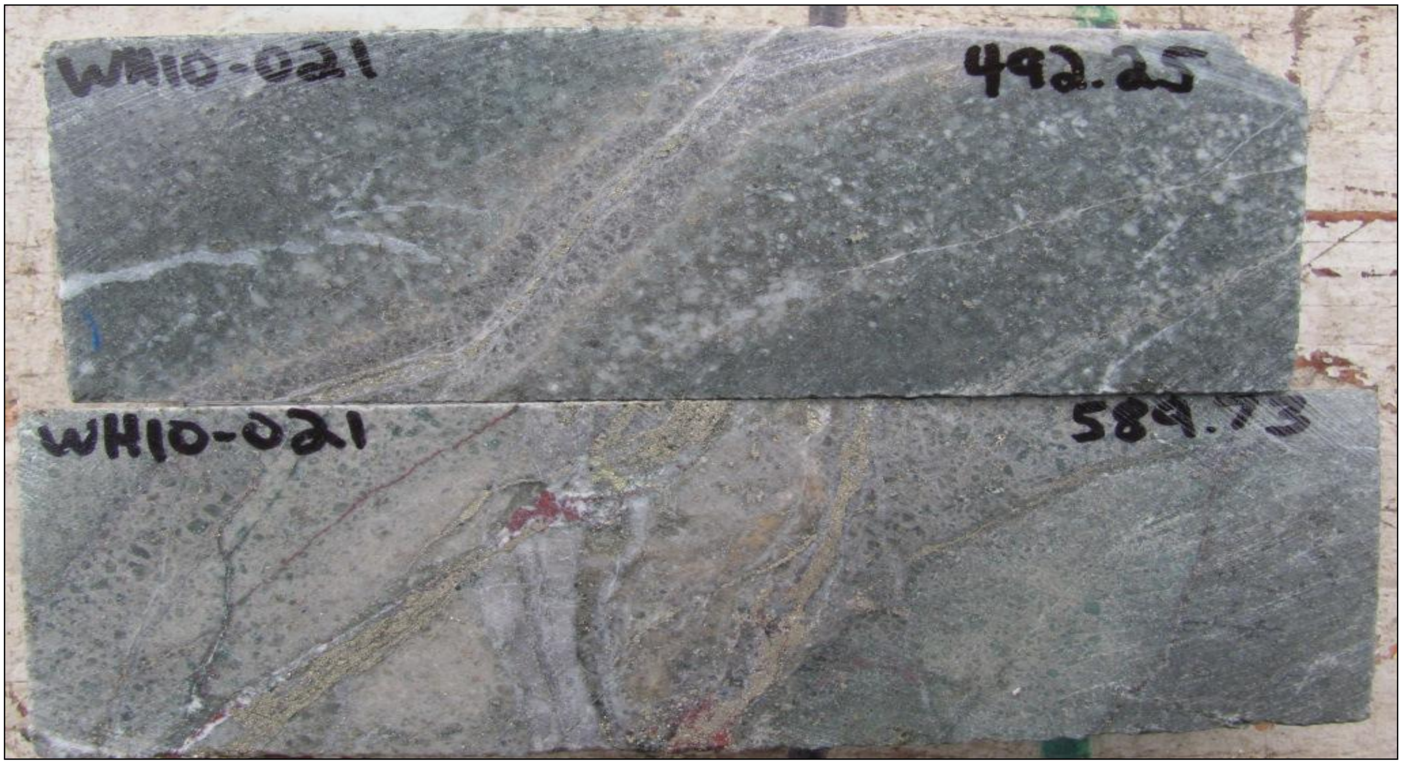
Figure 6-13: Chlorite-Sericite Alteration Observed in the Whistler Deposit

Note: This photo shows chlorite-sericite (+calcite) alteration overprinting potassic–magnetite alteration in a zone of quartz vein stockwork, subsequently cut by later Dpy veinlets with sericitic and iron-carbonate halos (Whistler Deposit). Source: MMTS, 2015

Potassic and chlorite-sericite alteration is variably overprinted by phyllic alteration. The phyllic assemblage consists of sericite + pyrite + quartz. Moderate to strong phyllic alteration is typically bleached grey tan, where mafic minerals are completely to strongly replaced by sericite and pyrite, magnetite is replaced by pyrite, and feldspars are replaced by sericite (and clays). Phyllic alteration commonly occurs in halos to pyritic stringers (Dpy) and quartz + pyrite veins (D-veins). In areas with intense D-style veining, phyllic halos coalesce to give pervasive phyllic alteration, as illustrated in Figure 6-14. Strong to intense phyllic alteration is texturally destructive, which often leads to difficulty in distinguishing intrusive from volcanic rocks. It is also suspected that intense phyllic alteration is grade destructive. At the Whistler Deposit and other prospects phyllic alteration forms an outer and upper, commonly gradational halo to chlorite-sericite alteration, and is also preferentially developed in structural zones, including faults and hydrothermal breccias. Hydrothermal breccias commonly occur along the boundaries of different units (sediment/diorite; volcanic/diorite; diorite/diorite) and are comprised of variably milled wall rock fragments cemented by quartz-sericite-pyrite (pyritic rock flour breccias). These breccias occasionally contain tourmaline.

In the Whistler Area, strong phyllic alteration and high pyrite content (10–15%) is common peripheral to individual porphyry centers extending for hundreds of meters into surrounding volcanic rocks. This has led to significant demagnetization of the volcanic stratigraphy such that the magnetic signature in the area is a function of alteration (dominantly phyllic) rather than primary rock types. In contrast, the phyllic halo at the Whistler Deposit only extends 50 m into the surrounding feldspathic sandstone. In addition to pyrite, porphyry centers in the area are also large sulfur anomalies, in the form of sulfates. Anhydrite appears to span several alteration and vein types: anhydrite occurs within B-type quartz-chalcopyrite veins and within cross-cutting D veins and Dbm veins (see Figure 6-14). Fine-grained anhydrite, of an uncertain alteration affiliation, also replaces feldspars at the microscopic scale. Gypsum locally replaces vein anhydrite and occurs as very narrow and abundant hairline veinlets in zones of strong to intense and pyritic phyllic alteration.

Figure 6-14: Phyllic Alteration Observed in the Whistler Deposit



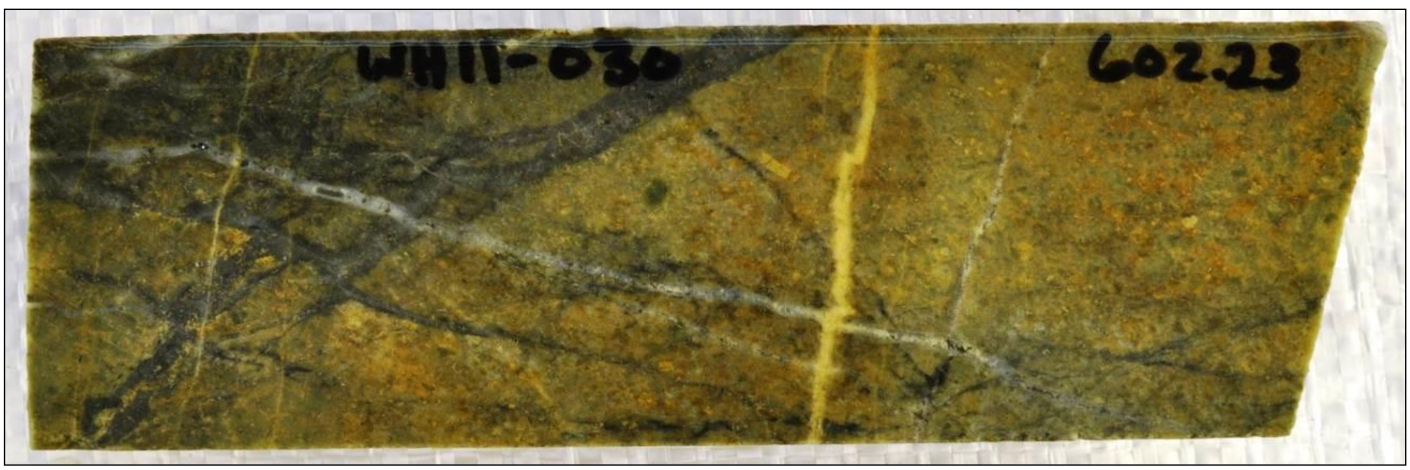
Note: D-style pyrite veins with well-developed phyllic halos (Whistler Deposit), that cut and offset B-style quartz veins (lower sample). Also note the local occurrence of hematite at the intersection of both vein types (magnetite>hematite?). Source: MMTS, 2015

At the Whistler Deposit and other prospects in the Whistler Area, the latest stage of precious and base metal mineralization is associated with quartz-carbonate (dolomite and calcite)-sphalerite-galena ± chalcopyrite veins (Dbm or D-base metal veins as described in Table 6-2). Veins minerals, including sulfides, are medium to very coarse-grained, have local colloform banding, and vein quartz is occasionally chalcedonic as shown in Figure 6-15. These veins have been observed to cut potassic and chlorite-sericite alteration (including Au-Cu mineralization and A- and B-vein stockwork), Dpy and D veins, and sericite-quartz-pyrite cemented hydrothermal breccias as illustrated in Figure 6-16. Based on their cross-cutting relationships, textures, mineralogy and spatial relationship to porphyry centers, these veins are interpreted to have formed concurrently with, or subsequent to, phyllic-stage alteration. That these veins typically cut phyllic-stage hydrothermal breccias and have open-space fill colloform banding, suggests that these veins formed in a much different hydrologic/structural regime (hydrostatic, possible incursion of meteoric waters) relative to Magnetite through to phyllic events. Relative to the Whistler Deposit, these veins are much more abundant in the host rocks to porphyry centers in the volcanic-hosted prospects in the Whistler Area, particularly Raintree West. This observation, in addition to the epithermal-like textures of these veins, supports the notion that other porphyry centers in the Whistler Area may have formed at shallower stratigraphic levels compared to that of the Whistler Deposit.

Figure 6-15: Quartz-Carbonate Vein from Raintree West (WH11-030)

Note: This photo of quartz-carbonate vein from Raintree West (WH11-030) shows well-developed colloform banding and coarse-grained sphalerite and galena. Source: MMTS, 2015

Figure 6-16: Common Vein Paragenesis in all Porphyry Occurrences in the Whistler Area



Note: Common vein paragenesis in all porphyry occurrences at Whistler Deposit include dark grey quartz vein stockwork with chalcopyrite (A- and B-style), cut by quartz-calcite-carbonate-sphalerite-galena veinlet (Dbm veins, top left down to bottom right), cut by narrow Fe-carbonate veinlets with Fe-carbonate alteration halos (Raintree West example). Source: MMTS, 2015

The most significant style of post-mineral alteration is Fe-carbonate alteration as illustrated in Figure 6-14 above. This occurs as pervasive alteration of feldspars in structural zones and as selvages to ankerite veins. Primary igneous magnetite and secondary magnetite are commonly altered to hematite in these zones. Ankerite veins, typically as brittle tension gashes, crosscut all vein styles, including the Dbm veins. The degree and extent of this style of alteration is typically not obvious until the core has weathered for a year or more and is therefore not well-documented in the core logs.

6.2.2.3 Gold and Copper Mineralization

Gold and copper mineralization is characterized by abundant disseminated sulfide and quartz-sulfide vein stockworks including classic porphyry diagnostic ‘A’, B’, D’, and ‘M’ type veins summarized in Table 6-2. Gold and copper mineralization in the MSP is comprised of 1-3% chalcopyrite and trace bornite as grains within magnetite and quartz veins and as disseminations in the host porphyry generally within the halos to these veins. Petrography indicates that gold occurs predominantly as electrum associated with chalcopyrite (Petersen, 2004). This mineralogy and style of mineralization is typical of diorite-hosted gold-copper porphyry deposits (Sillitoe, 2010).

Recent resource modeling has identified a ‘High-Grade Core’ (Figure 6-10) in the MSP defined by coincident approximately ≥0.40 g/t gold and ≥0.20% Cu grade contours and extends approximately 500 m in the north-south dimension, 250 m in the east-west dimension and is 600 m deep (from surface). The high-grade core has the highest gold-copper grades relative to the remainder of the MSP domain, yet the boundaries are geologically gradational.

The high-grade core contains inner zones of strong potassic and magnetite alteration, which is dominantly overprinted by pervasive chlorite-sericite alteration and local phyllic alteration. This domain is also defined by the consistent occurrence and highest concentration of M-veins and mineralized quartz veins (A- and B-veins), which generally range in volume from 1 to 5%. Local higher-grade mineralization occurs in vertical shoots of high-density quartz vein stockwork (locally >20% quartz vein volume) and quartz + magnetite + chalcopyrite cemented hydrothermal breccias. Such zones are interpreted to indicate the hydrothermal fluid ascent paths which brought Au-Cu mineralization up from below. This relationship is illustrated in Figure 6-17 which shows pipe-like high-density B-vein zones coincident with the high-grade zone.

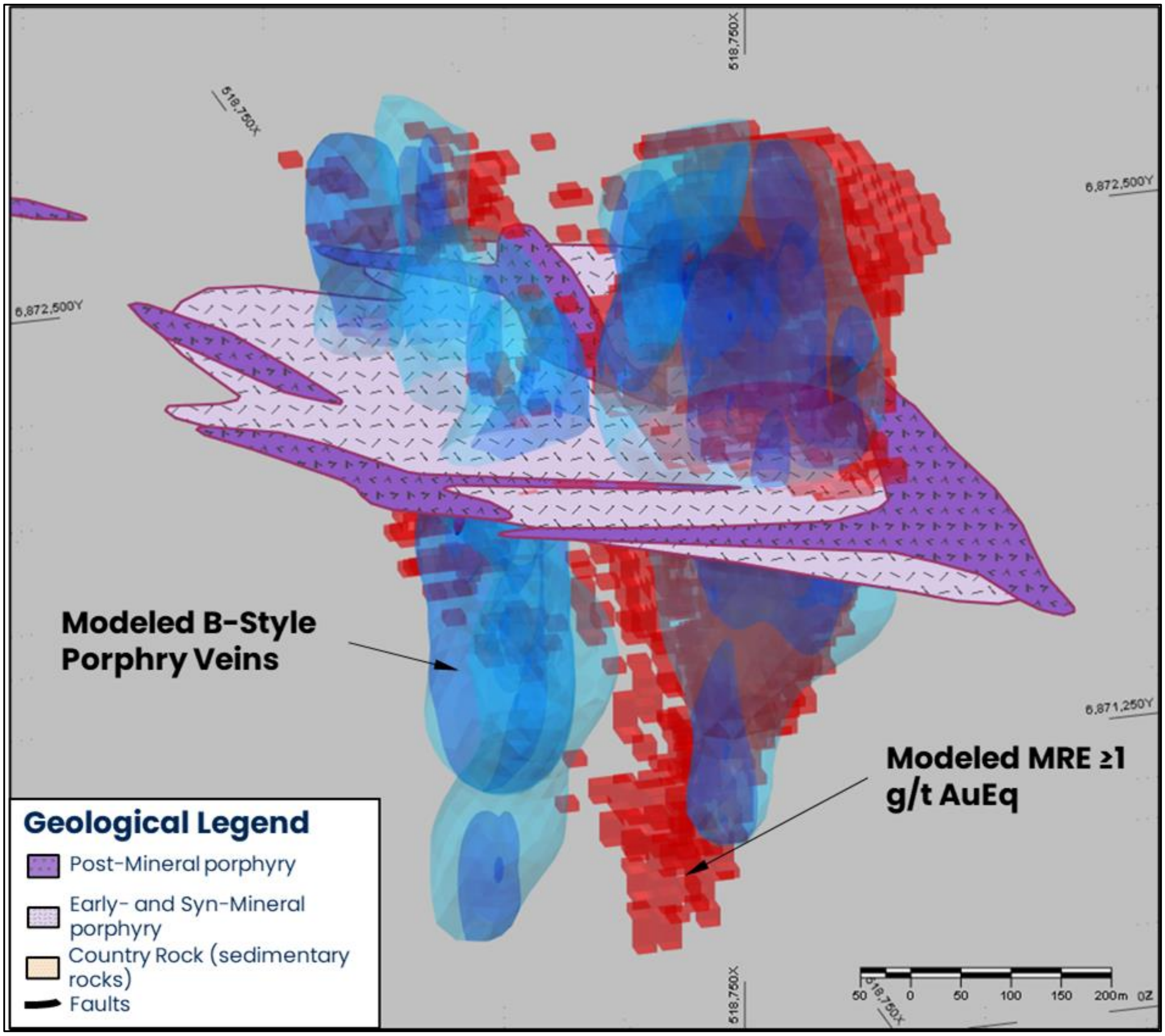
The vein modeling was modeled using classified vein density data collected in 2023 and 2024 drilling and relogging campaigns. The implicit vein modelling shows the B-veins are nested within similarly steeply dipping vein axes of early M-type and A-type porphyry-style veins. Modeling of the A-style porphyry veins shows that the top of the A-style vein zone in the western part of the Whistler deposit is deeper than the eastern part of the deposit which may indicate potential mineralization at depth in the western part of the deposit.

Minor 1- to 10-cm wide quartz-carbonate (ankerite and calcite)-barite-sphalerite-galena ± chalcopyrite veins (Dbm veins) crosscut mineralized and unmineralized portions of the Main Stage Porphyry and are interpreted as intermediate sulfidation epithermal veins that have telescoped on the porphyry system. These sparse veins contain minor Au, Ag, Pb, Zn, and Cu, yet do not contribute significantly to the economic resource.

6.2.2.4 Structure

The structure of the intrusive complex is becoming better constrained with the most recent rounds of drilling (2023 and 2024 totalling 6,240 m core), combined with relogging of historic core, to fill gaps in understanding within the previously widely spaced drilling.

Figure 6-17: 3D View of the Whistler Deposit Looking Northeast



Notes: This figure shows modeled B-style porphyry veins (blue 3D contoured surfaces) and the diorite porphyry phases at the 600-m level plan. Red blocks show interpolated mineralization ≥1 g/t AuEq (from the 2024 Mineral Resource block model), demonstrating the similarity in vein density with gold-copper mineralization and suggesting a deeper target area extends to depth in the northeastern part of the deposit. Source: Equity, 2025

The 'Divide Fault', previously modelled based on drill core intercepts and breaks in the downhole magnetic susceptibility readings, and which formed a hard boundary between eastern and western mineral resource domains (Domain 1 & 2 respectively in Bird, 2022), has been removed from the current structural model of the Whistler Deposit. Successive relogging campaigns and 2023-2024 drilling has failed to find convincing evidence of the existence of this fault. While a diffuse / gradational 'break' in geology can be discerned parallel to the plane of the previously interpreted Divide Fault, this 'transition zone' is likely to represent an interfingering complex intrusive contact between MSP and LSP dykes. Thus, the transition zone is also no longer considered a hard boundary suitable for constraining resource interpolation. Several other previously modelled faults as described in earlier geological interpretations of the Whistler Deposit, have also been removed from the current geological model for the Whistler deposit, as their existence is tenuous and their impact on either controlling or offsetting / juxtaposing mineralization was minor.

A newly interpreted fault system crosscuts the northern part of the deposit (Figure 6-9): the 'Rover Fault' and associated splays. Fault structures in the deposit are commonly associated with narrow zones of strong to intense sericite, clay, pyrite, and carbonate alteration. The Rover Faults are interpreted as post-mineral, with vertical displacement.

6.2.3 Mineralization: Raintree West

The Raintree West deposit occurs 1,500 m to the east of the Whistler Deposit, just off the nose of Whistler Ridge. It occurs below a thin veneer of glacial till (5 to 15 m) and hence is not exposed at surface. Outside of the Whistler Deposit, Raintree West is currently the most advanced deposit in the Whistler Area based on drill meters, with a total of 8,761.62 m since the original discovery hole drilled by Geoinformatics in 2008 (Figure 6-18). The discovery drillhole, RN-08-06, targeted an airborne magnetic high anomaly that is coincident with an IP chargeability high detected on a 2D IP reconnaissance line that crossed the Whistler Area. This hole discovered a significant zone of near-surface (below 5 m of till cover) gold-copper porphyry mineralization (160 m grading 0.59 g/t gold, 6.02 g/t silver, 0.10% copper).

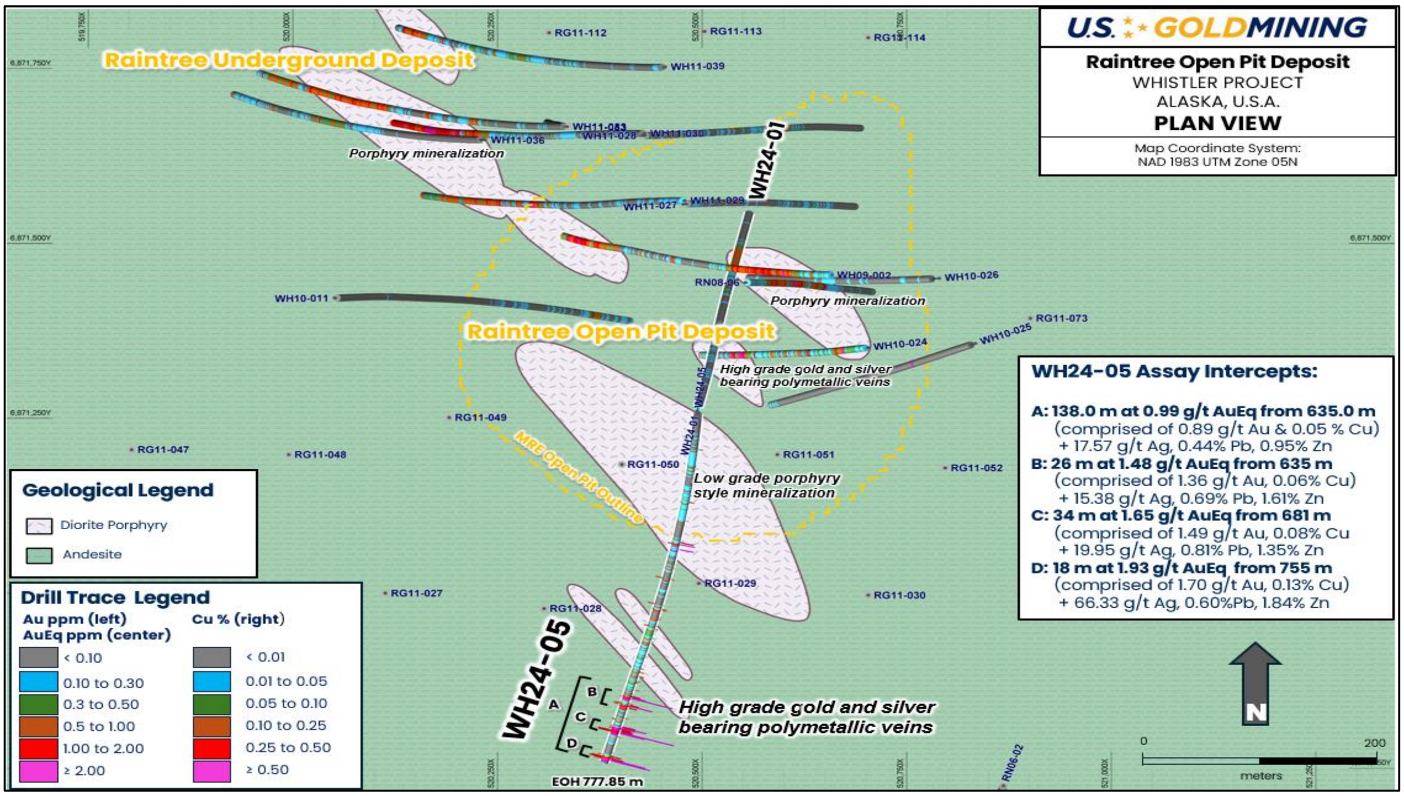
Mineralization at Raintree West occurs as two main types: early, porphyry-style gold-copper mineralization hosted by diorite porphyry stocks and consisting of quartz and magnetite stockwork veining, with vein and disseminated chalcopyrite associated with potassic alteration; and later cross-cutting silver-gold-lead-zinc mineralization in quartz-carbonate veins (Dbm) that contain pyrite, sphalerite, galena, and chalcopyrite, with occasional banded epithermal-like textures. The early gold-copper mineralization is best developed within, and controlled by, early diorite porphyry intrusions (akin to Main Stage Porphyry at the Whistler Deposit), whereas the later silver-gold-lead-zinc veins surround and locally overprint the porphyry mineralization and are most abundant in the host volcanic rocks in zones of strong to intense phyllic alteration vertically above and adjacent to the diorite porphyries. In places, 25 to 50-m-wide diorite porphyry dykes cut both types of mineralization and are barren (akin to Late-Stage Porphyry (LSP) at the Whistler Deposit).

Current drilling at Raintree West has defined two significant zones of gold-copper porphyry mineralization: a near-surface zone on the east side of the Alger Peak fault; and a deep zone on the west side of the fault (Figure 6-18).

The near-surface porphyry gold-copper mineralization is coincident with a northwest-elongate airborne magnetic high anomaly that measures 250 m long and 150 m wide, which pinches to the northwest and southeast. Drilling has only intersected this mineralization on two 100-m spaced east-west sections (6,871,350 mN and 6,871,450 mN). Gold-copper mineralization occurs from the top of bedrock to a maximum depth of approximately 170 m, where it is either truncated by post-mineral diorite porphyry intrusions or faulting, and has a true width of approximately 150 m. Gold-copper mineralization is closed to the north, and potentially open to the south, however grade diminishes, and the airborne magnetic high anomaly pinches out just south of the most southerly hole (WH10-025).

The deep zone of porphyry gold-copper mineralization on the west side of the fault has a maximum apparent width and vertical extent of 300 x 300 m at its widest (6,871,650 N), is open to depth, and occurs at its shallowest at 470 m below surface. This deep zone of mineralization can be traced along a northwest-trending strike extent for at least 325 m where it appears fault bound to the northwest and is open to depth to the southeast. The mineralization is essentially blind to the airborne magnetic data and the 3D IP due to the limited depth penetration of these techniques.

Figure 6-18: Plan Map of Geological Interpretation at the Raintree West Deposit Highlighting 2024 Drilling



Source: Equity, 2025

Porphyry mineralization at Raintree West is essentially like that at the Whistler Deposit with respect to veining and alteration, although Raintree West is mantled by intensely altered volcanic rocks with epithermal-texture quartz-carbonate veins. These veins (Dbm), interpreted to have formed in a shallow environment post-dating the main phase of porphyry gold-copper mineralization, may have developed through hydrothermal/thermal downward collapse onto to earlier formed high-temperature porphyry system, contributing base and precious metals to the mantle of volcanic rocks and porphyry mineralization.

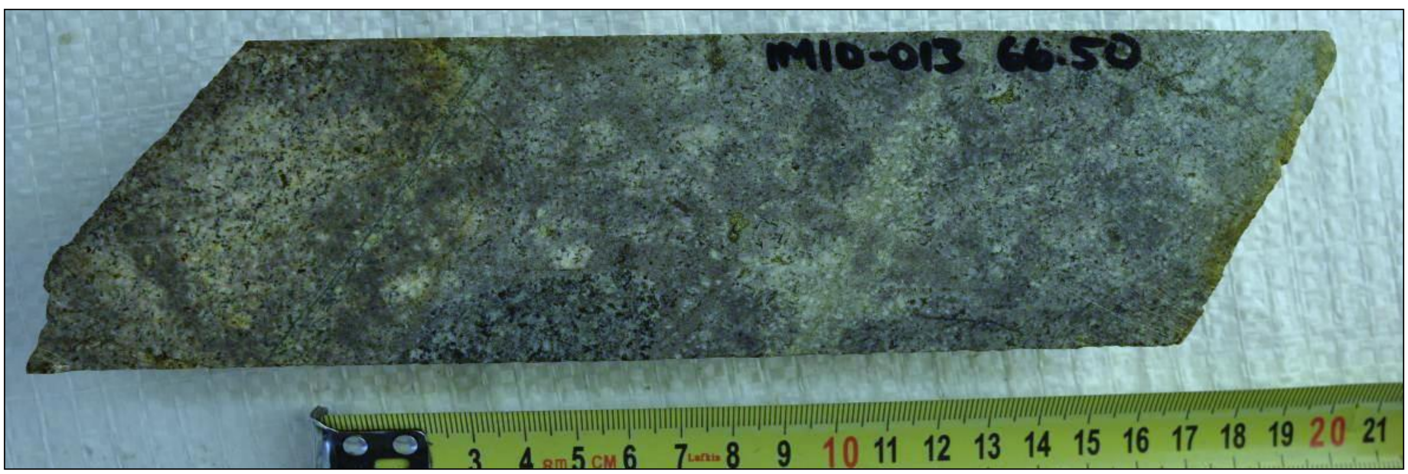
Recent drilling in 2024 intersected a new diorite intrusion in the southwestern part of the Raintree West open pit deposit area. WH24-05 intersected high-grade gold and silver polymetallic mineralization in a series of Dbm-style veins southwest of the newly identified intrusions. Additional work will need to be conducted to follow up this newly identified mineralization.

6.2.4 Mineralization: Island Mountain

The Island Mountain deposit area is host to several mineralized zones interpreted to represent a cluster of individual porphyry centers within this large intrusive complex. These include the Breccia (Island Mountain deposit), Cirque and Howell Zones, and other prospects defined by surface geochemistry and geophysical anomalies that require further field assessment. Exploration activity and most diamond drilling by Kiska have concentrated on mineralization associated within the Breccia Zone on the southwest slope of Island Mountain. Here, at least three styles of significant gold and copper mineralization are currently recognized: gold-copper mineralization hosted by k-feldspar altered monzonitic intrusive breccia; gold-copper mineralization hosted by intrusive and hydrothermal breccias associated with strong sodic-calcic alteration; and gold-only mineralization associated with vein and disseminated pyrrhotite (pyrrhotite-gold).

At the Breccia Zone, the first two styles of mineralization occur within a 300-m diameter, subcircular, subvertical breccia pipe, which appears to have been a conduit for inter-mingled intrusive and hydrothermal breccias hosted by the Diorite Porphyry. Gold-copper mineralization hosted by the k-feldspar altered monzonitic intrusive breccia is volumetrically smaller than the subjacent hydrothermal breccias and is interpreted as being the earliest stage of mineralization, since this breccia body is cut by actinolite veinlets. Mineralization is associated with trace to 2% disseminated chalcopyrite in the k-feldspar-altered intrusive cement of the breccia, as illustrated in Figure 6-19.

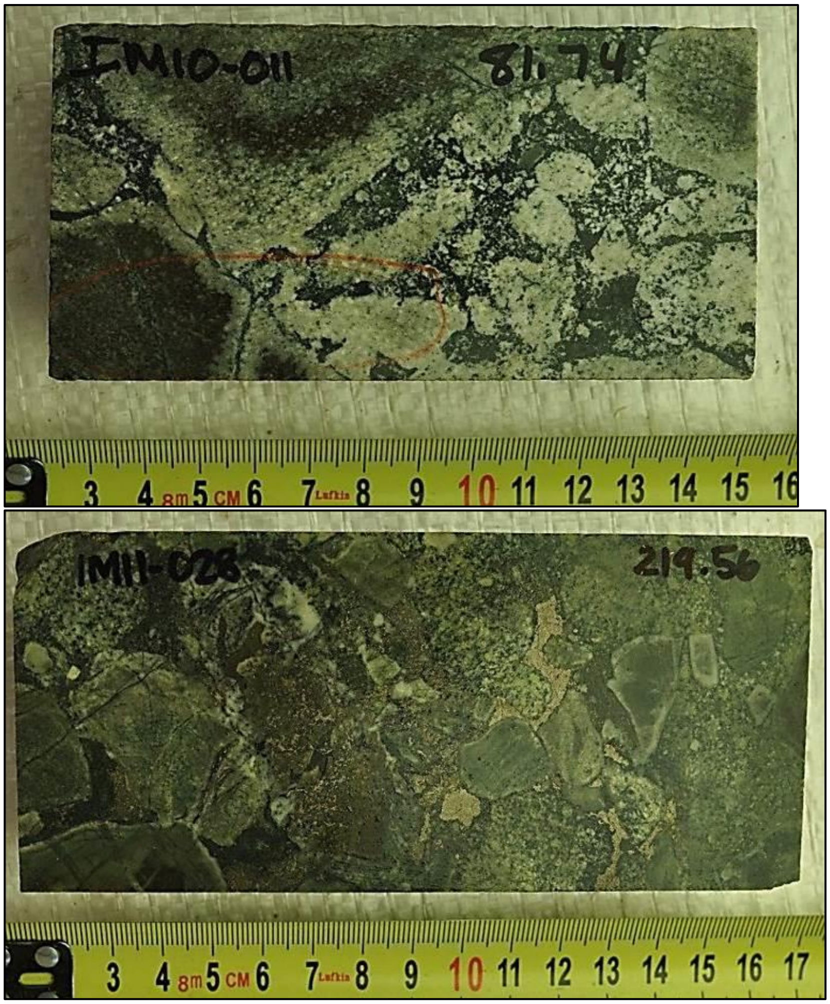
Figure 6-19: Monzonite-matrix Intrusive Breccia with Patchy Albite Alteration, Silicification and Disseminated Chalcopyrite



Source: MMTS, 2015

The bulk of gold-copper mineralization at the Breccia Zone is hosted by intrusive and hydrothermal breccias with strong sodic-calcic alteration with pyrrhotite as the predominate sulfide and trace to 1% chalcopyrite. Chalcopyrite is most abundant in the matrix of the hydrothermal breccias and is commonly intergrown with pyrrhotite and actinolite ± magnetite. Pyrrhotite, ranging from 1 to 5%, occurs as disseminations within the breccia matrix and as large blebs cementing the matrix as illustrated in Figure 6-20. The deportment of gold in the breccia zone is not known. Weaker gold-copper mineralization extends 50 to 75 m beyond the breccia zone and is associated with actinolite stockwork veining.

Figure 6-20: Textures of Actinolite-Magnetite Hydrothermal Breccia (BXMA) in the Breccia Zone

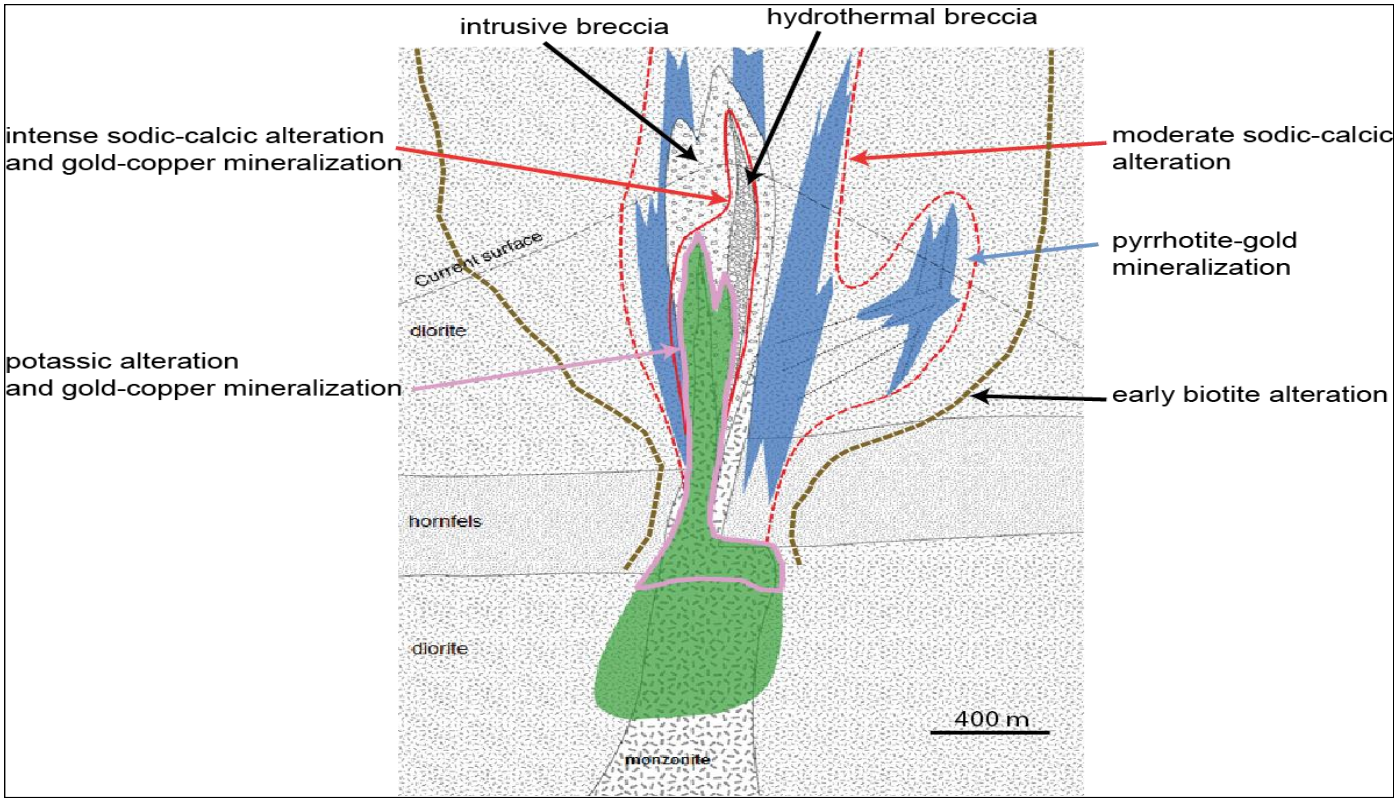


Note: The Actinolite-magnetite hydrothermal breccia (BXMA) textures show strong albitization in monomict breccia (upper photo), and a pyrrhotite matrix in polymict breccia (lower photo). Source: MMTS, 2015.

Gold-only mineralization in the Breccia Zone (referred to as pyrrhotite-gold mineralization) occurs 100 to 200 m peripherally to the intrusive-hydrothermal breccia body and occurs in association with vein and disseminated pyrrhotite within the Diorite Porphyry. Pyrrhotite veins occur in irregular, possibly sheeted sets, and are typically 1 to 10 mm wide and have pyrrhotite-rich (up to 15 to 20%) net-textured vein selvages (e.g., replacing the igneous matrix of the Diorite Porphyry). Petrography and SEM studies indicate that gold occurs as electrum intergrown within marginal to pyrrhotite grains. The orientation and continuity of these veins is currently undefined.

The relationship between the breccia-hosted gold-copper mineralization and the pyrrhotite-associated gold-only mineralization is not fully understood. The current working hypothesis is that the gold-copper and gold-only mineralization are associated with the same hydrothermal fluid, such that copper was precipitated in the hotter parts of the system within the hydrothermal breccia, and copper-depleted, gold-bearing fluids persisted into cooler, structural zones beyond the breccia and were subsequently precipitated as illustrated schematically in Figure 6-21. (Rowins, 2011).

Figure 6-21: Schematic Model of Breccia Zone Alteration and Mineralization



Source: Roberts, 2011b

6.2.5 Mineralization: Muddy Creek

Gold mineralization at Muddy Creek is hosted throughout the core of the plutonic complex and is controlled by northwest-striking and steeply southwest-dipping veinlets of sulfides and quartz, ranging from millimeters to locally centimeters in width, manifested as rusty-weathering sub-parallel fracture sets, commonly spaced one meter or more apart (Figure 6-22). These veinlets may contain any combination of chalcopyrite, arsenopyrite, pyrite, stibnite, pyrrhotite and native gold, with minor amounts of galena, sphalerite and molybdenite. Moderate sericitic alteration is typically restricted to centimeter-wide selvages to these veins, whereas the bulk of the interleaving rock is relatively unaltered and unmineralized. Cone sheets and circular onion skin-type joints that resemble bubbles or miarolites also carry gold mineralization, and elevated gold and copper values are also found in cm-scale pegmatites. Coarse to very coarse-grained feldspar-quartz pegmatite with chalcopyrite and subordinate molybdenite occur along joint planes and intersections, centered in aplitic dikes and at the cores of circular joint sets or cone sheets. Lastly, massive sulfide veins occur locally along Muddy Creek in hornfelsed sedimentary wall rock. Previous workers report gold in all mineralization types to range from ppm to more than 1 oz/t in select samples (Millholland, 1998).

Figure 6-22: Detailed View of Biotite-Monzonite Northwest of Muddy Creek,

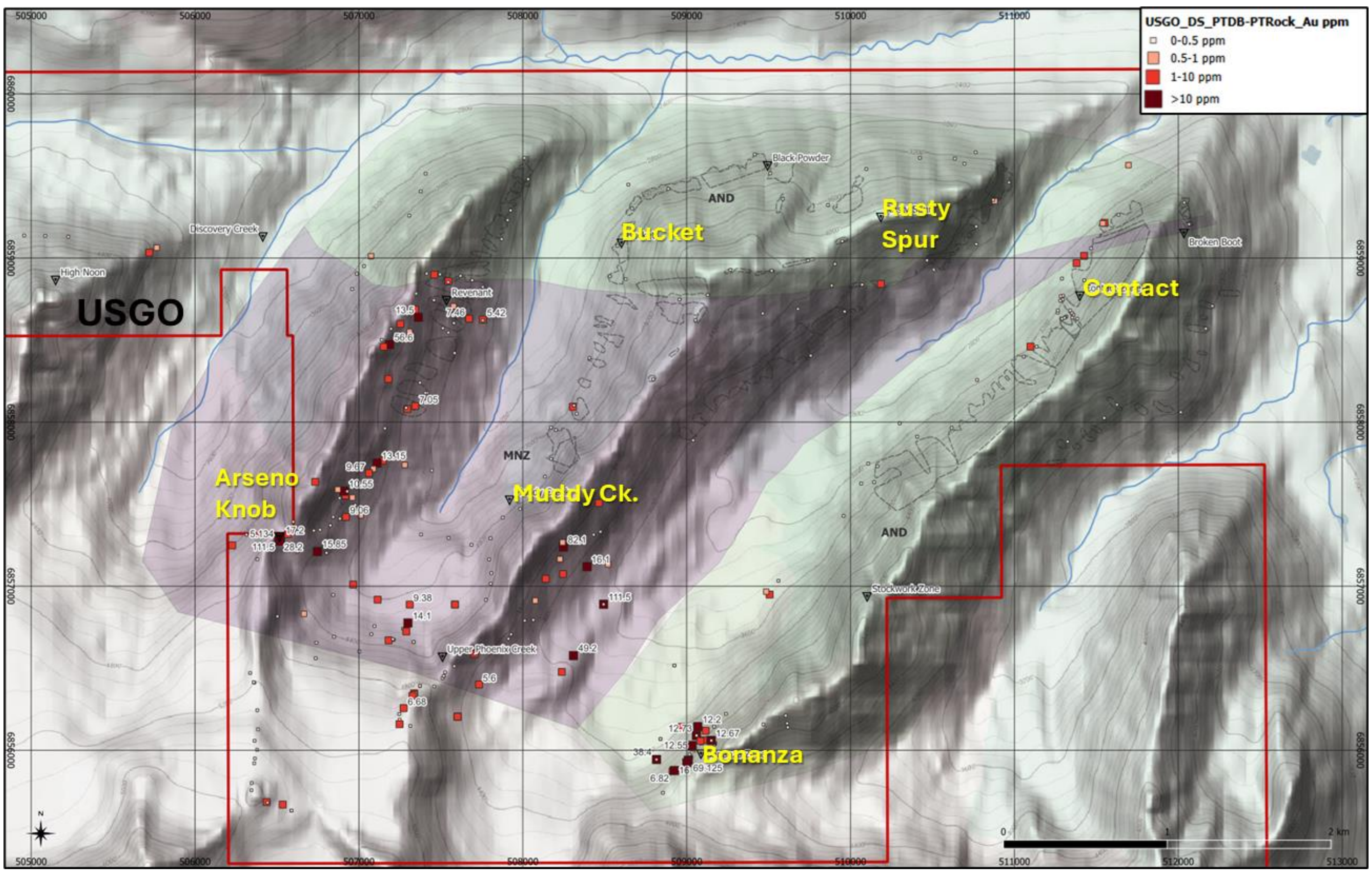
Note: In this photo, biotite-monzonite is cut by subvertical limonite-stained fracture fillings of chalcopyrite-arsenopyrite (~1-3 per meter). Source: MMTS, 2015

Accessory minerals associated with mineralization in veins include vuggy quartz and K-spar, with greatly subordinate ilmenite, tourmaline, apatite, beryl, and possibly corundum. Unlike most other mineral types of the Whistler region, magnetite is completely absent and the only measurable magnetism in hand samples is imparted by ilmenite and pyrrhotite.

Previous exploration has largely been focused on areas where the vein/fracture density is highest. This includes structural zones near the top of Discovery Creek, Phoenix Creek, Prospect Creek, and Muddy Creek that occur along the strike extent of a significant northwest-striking fault zone. Two diamond drillholes drilled by Kiska in 2011 focused on a high-density vein/fracture zone at the top of Prospect Creek. Here drilling returned a highlight result of 0.44 g/t gold over 44.2 m from 297.0 downhole (MC11-002). Only three holes in total, across two target areas, were completed, thus the full extent of mineralization is unknown due to a lack of systematic drilling.

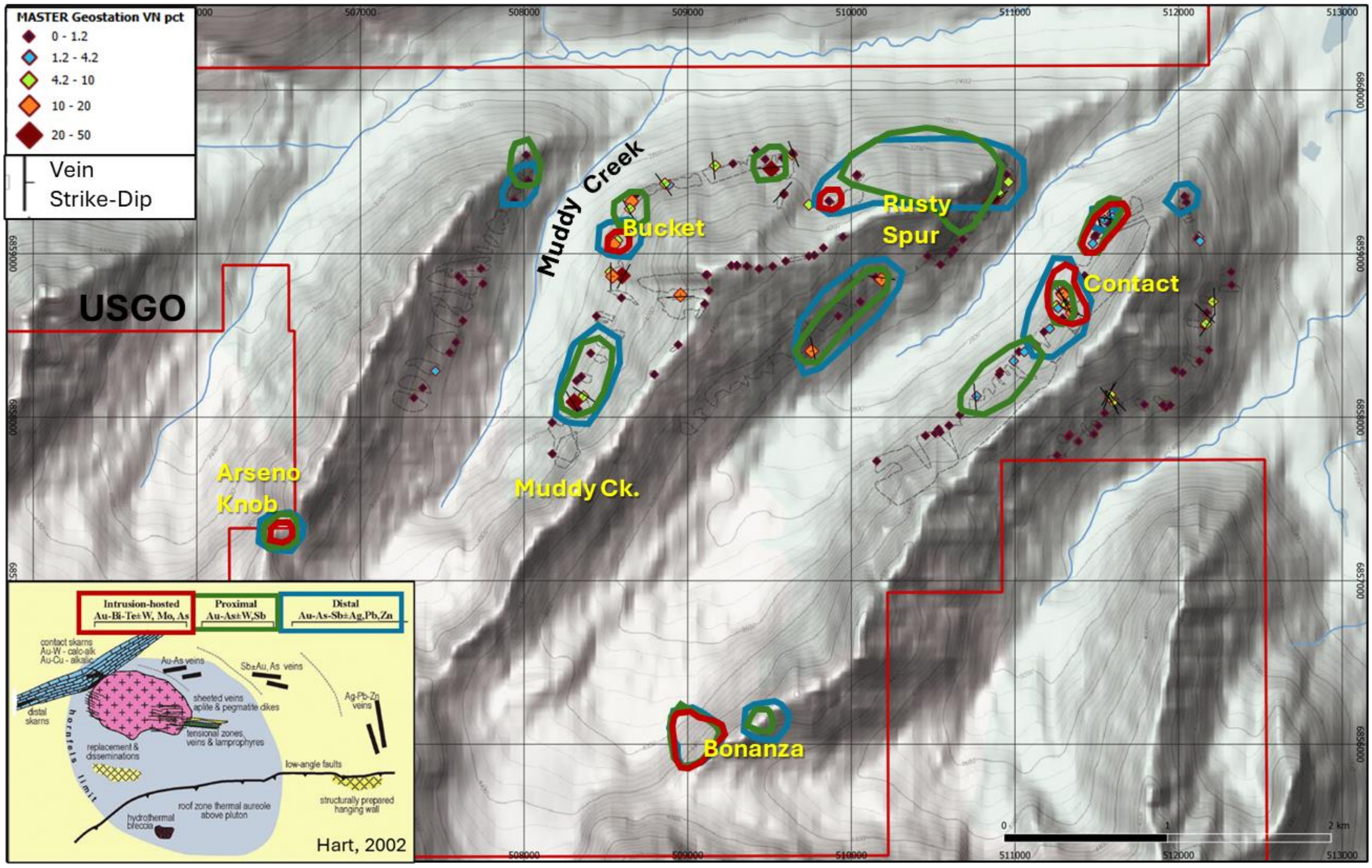
Geological mapping and nominally spaced systematic rock sampling completed by U.S. Goldmining Inc. in 2025 aimed to elucidate the geometry of the intrusion related gold system and identify vectors to higher vein density and potentially more consistent gold mineralization. Mapping identified new gold-bearing zones at the Contact, Rusty Spur and Bucket showings in the eastern portion of the prospect area and defined multiple zones of oxidized veins with minor arsenopyrite and pyrite (Figure 6-23). Veins were steeply dipping with a north-northwest trend, with the highest vein density areas observed near the contacts of the host monzonite with andesite and sedimentary country rock. 2025 sampling found 'proximal' vein assemblages, comprised of anomalous W, Sb, and As in conjunction with Au (Figure 6-24) and increased vein density, observed at the Bucket, Rusty Spur, and Contact Zone targets during the 2025 mapping program, which highlighted the need for follow-up work at the Muddy Creek prospect. Overall, mapping and sampling of veins shows elevated Au, Bi, and Mo values consistent with metal zonation within the host monzonite that is consistent with other known reduced intrusion related gold (RIRG) systems encountered in the area and more broadly the Tintina Gold Belt.

Figure 6-23: Map of Surface Rock Sampling at the Muddy Creek Area Showing Concentration of Gold in Rocks



Source: U.S. GoldMining, 2026

Figure 6-24: Plan Map of Rock Sampling and Vein Mapping at Muddy Creek.



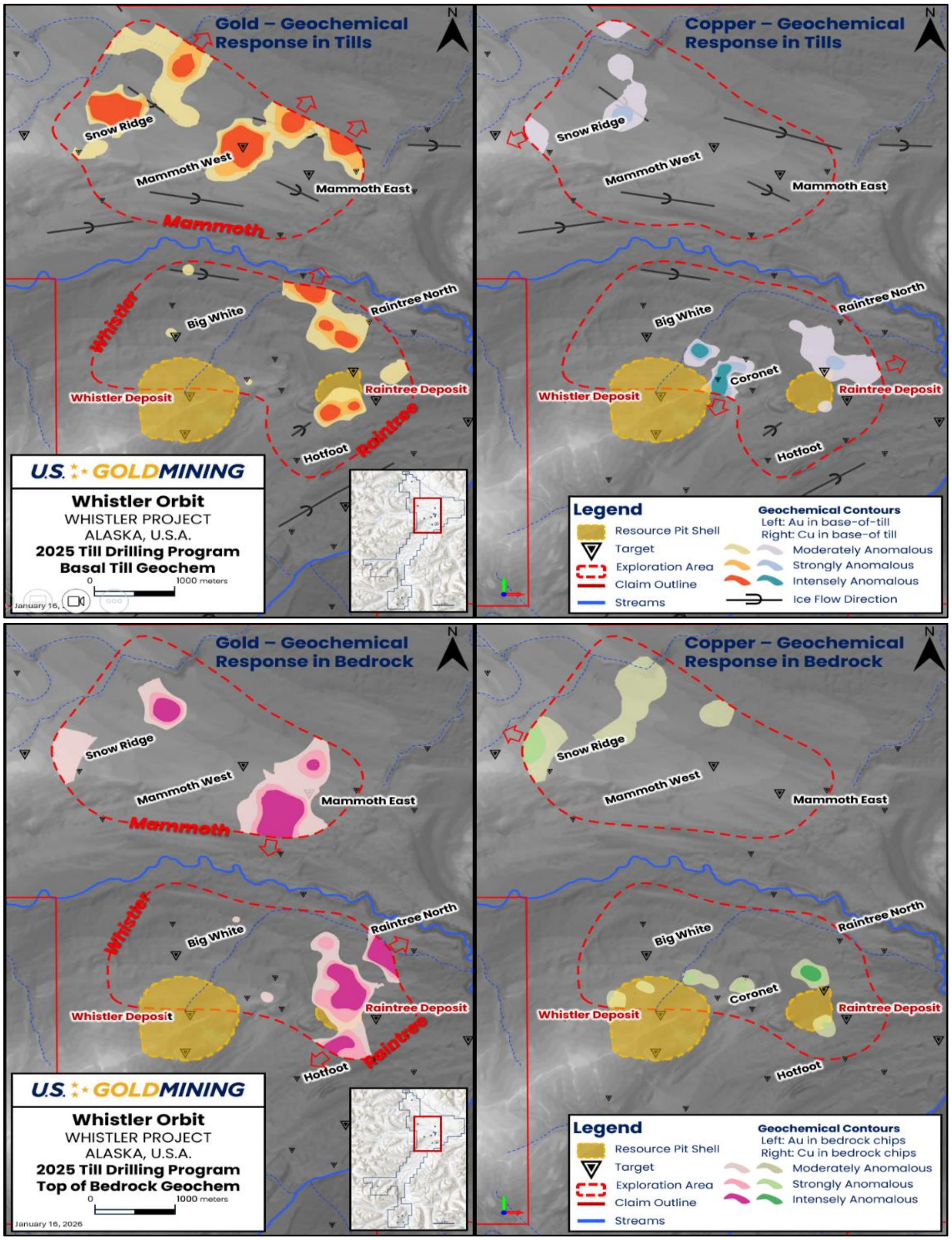
Notes: Diamonds show the total vein density along mapping traverses. Coloured outlines show the zonation of intrusive-hosted metals and hydrothermally associated metals proximal and distal to heat centers in reduced intrusion related gold systems. Source: U.S. GoldMining, 2026

6.2.6 Mineralization: Whistler Orbit

The Whistler Orbit represents a classic porphyry cluster over an area of approximately 7.5 km by 4.5 km comprising multiple porphyry targets surrounding the existing Whistler and Raintree West gold-copper porphyry deposits. The porphyry prospects are mostly overlain by variable thicknesses of glacial till cover.

Bottom of basal till and top-of-bedrock sampling conducted by U.S. Goldmining in 2025 identified four new copper-gold target areas in the Whistler Orbit area (Figure 6-25). The Mammoth & Snow Ridge are two new large gold ± copper anomalies located 2-3 km north of the Whistler deposit comprised of broad intense gold anomalies in tills and bedrock. The Raintree area showed widespread gold ± copper anomalism adjacent to the existing Raintree West deposit and throughout the broader target area. Hotfoot is an emerging new target defined by an intense bedrock gold anomaly associated with a large magnetic anomaly 1 km south of Raintree West deposit.

Figure 6-25: Geochemical Response of Gold and Copper in Tills and Bedrock



Note: Geochemical response in basal till samples (left: gold; right: copper). Bottom: Geochemical response in top-of-bedrock samples (left: gold; right: copper). Source: U.S. GoldMining, 2026

6.3 Deposit Types

Successive campaigns of exploration on the Whistler Project over a 40-year period, by Kennecott, Geoinformatics, Kiska and U.S. GoldMining, has identified three major mineral systems containing numerous exploration targets for porphyry-style gold-copper deposits (including the Whistler, Raintree West, and Island Mountain deposits) and gold-only intrusion related mineralization at the Muddy Creek area. Neighboring explorers have also reported gold-antimony mineralization, and the USGS has identified potential for other critical minerals in the district, including tungsten.

The Whistler, Raintree West, and Island Mountain deposits and their exploration criteria conform to the porphyry deposit model as described in Sillitoe (2010). All the porphyry deposits in the Whistler Area share similar styles of alteration, mineralization, veining and cross-cutting relationships that are generally typical of porphyry systems associated with relatively oxidized magma series (A- and B-type quartz vein stockwork, chalcopyrite-pyrite assemblage, presence of sulfates, core of potassic alteration with well-developed peripheral phyllic alteration zones). The Whistler area also hosts multiple additional porphyry-like prospects defined by drilling, anomalous soil samples, alteration, veining, surface rock samples, IP chargeability/resistivity anomalies and airborne magnetic anomalies. These include Raintree North, Rainmaker, Dagwood, Round Mountain, Puntilla, Canyon Creek, and Snow Ridge prospects. Thus, the Whistler-Raintree area, also known as the 'Whistler Orbit', is considered to be a classic porphyry cluster comprising multiple high-level magmatic apophyses emanating from a common deep causative batholith.

In contrast, Island Mountain has significantly different alteration, veining and sulfide assemblages associated with mineralization, principally the occurrence of pyrrhotite and to a lesser extent arsenopyrite associated with Au-Cu mineralization, Au-Cu association with strong sodic-calcic alteration, lack of significant sulfates, very minor hydrothermal quartz and weak to insignificant phyllic alteration. For these reasons, the porphyry system at Island Mountain may belong to the "reduced" subclass of porphyry copper-gold deposits (see Rowins, 2000).

The Muddy Creek area represents an additional exploration target with the potential to host a bulk tonnage, reduced intrusion-related gold (RIRG) deposit. Previous exploration by Millrock Resources Inc., and more recently by Nova Minerals Ltd., on claims directly adjacent to the Muddy Creek area, which are geologically analogous, have returned encouraging preliminary results. Like Island Mountain, the Muddy Creek mineralization is distinct from the Whistler Porphyry systems and shares more similarity with IRG systems characteristic of the Tintina Gold Belt. The intrusive complex at Muddy Creek is predominantly monzonitic grading to more mafic marginal phases yet is generally more felsic in composition relative to the diorites of the Whistler Area. Mineralization is restricted to sheeted vein zones with narrow millimeter scale veinlets and pegmatitic veinlets of quartz, feldspar, tourmaline, and sulfides that include arsenopyrite, minor chalcopyrite and pyrite-pyrrhotite. Gold mineralization is largely confined to the minute veinlets whereas the intervening intrusive rocks are largely unaltered and unmineralized.

7 EXPLORATION

Between 2023 and 2024, U.S. GoldMining undertook a comprehensive work program at its Whistler Project. The work included compilation and harmonization of all historical data, drill core relogging and geochemical sampling, with site and infrastructure improvements. The company conducted mapping, rock sampling, and prospecting at the Muddy Creek prospect and glacial drift sampling at the Mammoth prospect. Historical core storage at the Rainy Pass airstrip was catalogued and the historical disturbances and waste at the site were remediated. In 2025 U.S. Goldmining conducted base-of-till and top-of-bedrock sampling utilizing a specialized heli-portable auger drill to explore under glacial till cover in the Whistler Orbit area. Additionally, they conducted a systematic mapping and sampling program in the Muddy Creek prosect area to define mineral system vectors.

7.1 Surface Exploration

A summary of all surface exploration work conducted by U.S. GoldMining from 2023-2025 is provided in Table 7-1.

Table 7-1: Summary of U.S. GoldMining Surface Exploration on the Whistler Project

Field Seasons	Relogging and Sampling	Mapping	Geophysics	Rock	Soil	Silt	Till
2023	1,765.2 m at Raintree West & Whistler (Relogging)	Recon.-scale mapping	Magnetics 3D inversion modelling	n/a	na/	n/a	n/a
2024	11,371.1 m at Whistler (Relogging)	Prospect-scale mapping	-	-	-	-	27
2025	169 auger scout drill holes at Whistler Orbit (Sampling)	Prospect-Scale Mapping	-	102	-	6	169

Notes: AM=Airborne Magnetic survey; EM=Airborne Electromagnetic survey; IP=Induced Polarization survey.

7.2 Logging and Database Work

Between 2023 and 2024, U.S. GoldMining undertook a comprehensive work program at its Whistler Project. The work included compilation and harmonization of all historical data, drill core relogging and geochemical sampling, with site and infrastructure improvements. Key activities included diamond drilling at the Whistler, Rainmaker, and Raintree West prospects, along with comprehensive relogging of 13,132 m of core in 164 holes from Whistler and Raintree West. This work culminated in a revised geological model to enhance the understanding of the deposits.

Historic core storage at the Rainy Pass airstrip was catalogued and the site was remediated of historical disturbance and waste.

7.3 Geological Mapping

The company conducted mapping, rock sampling, and prospecting at the Muddy Creek prospect and glacial drift sampling at the Mammoth prospect. Additionally, they conducted a systematic mapping and sampling program in the Muddy Creek prospect area to define mineral system vectors.

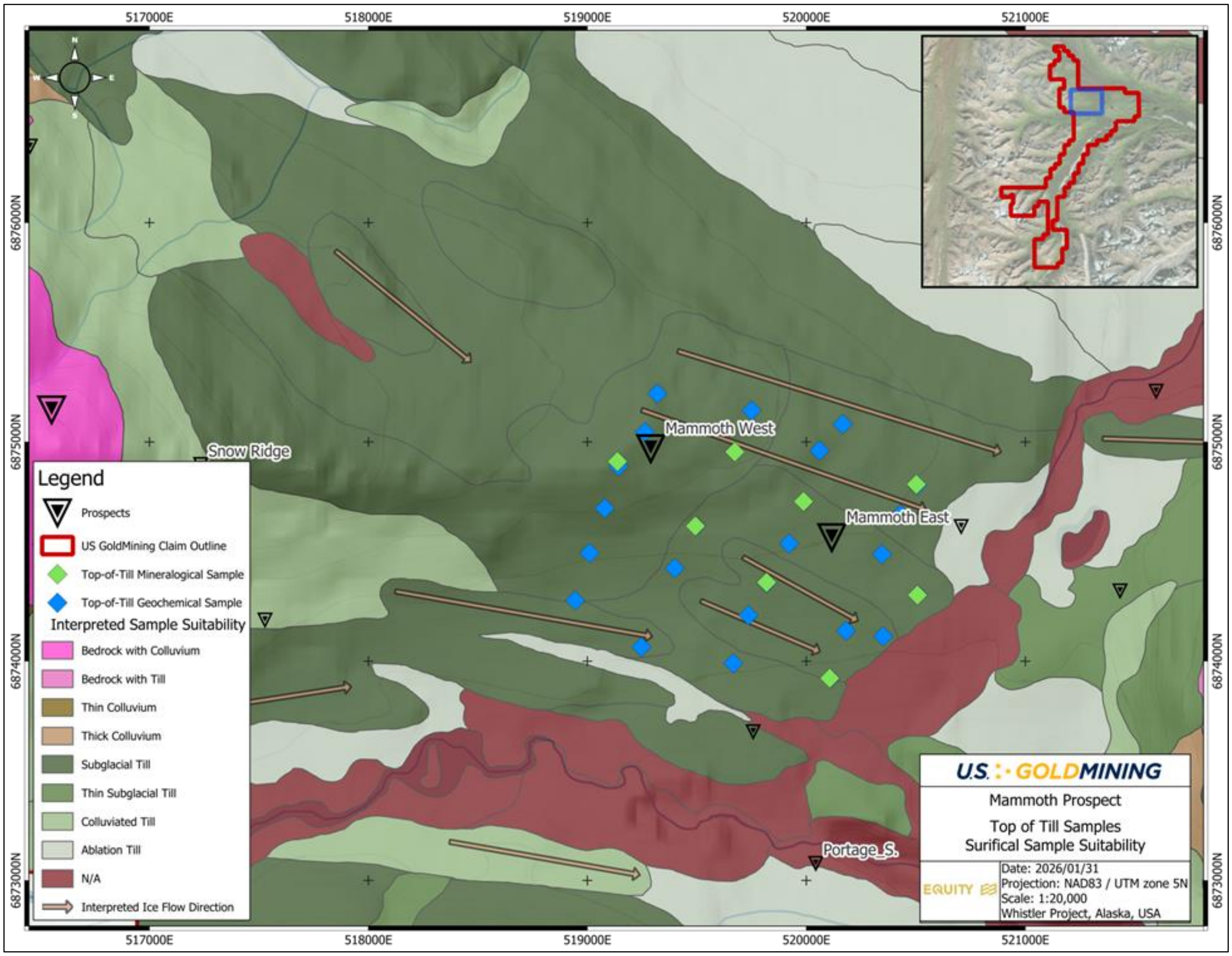
In 2025 U.S. GoldMining completed prospect-scale mapping and rock sampling that identified high-density gold-bearing vein zones with lateral metal zonation consistent with intrusion related gold mineralization at the Muddy Creek prospect. Systematic geochemical sampling comprising 6 silt samples and 102 rock samples at 200 m nominal spacing on traverses across target areas at the Muddy Creek prospect further supported the thesis for high-temperature intrusion-related mineralization via pathfinder trace element geochemistry.

7.4 Glacial Till Sampling

7.4.1 Top-of-Till Sampling

In 2024, Palmer Environmental Consultants Group created a surficial geology and surficial sampling suitability map for the Whistler project area. Twenty-seven Top-of-Till geochemical samples and eight mineralogical samples were collected at the Mammoth target (Figure 7-1). Samples were taken on a designed 200 m × 420 m offset grid perpendicular to local interpreted ice-flow direction over a magnetic high feature. Geochemical samples were assayed for gold and ICP-MS multi-element geochemistry, and indicator minerals and gold grain counts were completed on the mineralogical samples.

Figure 7-1: Sample types collected from Top-of-Till sampling in 2024

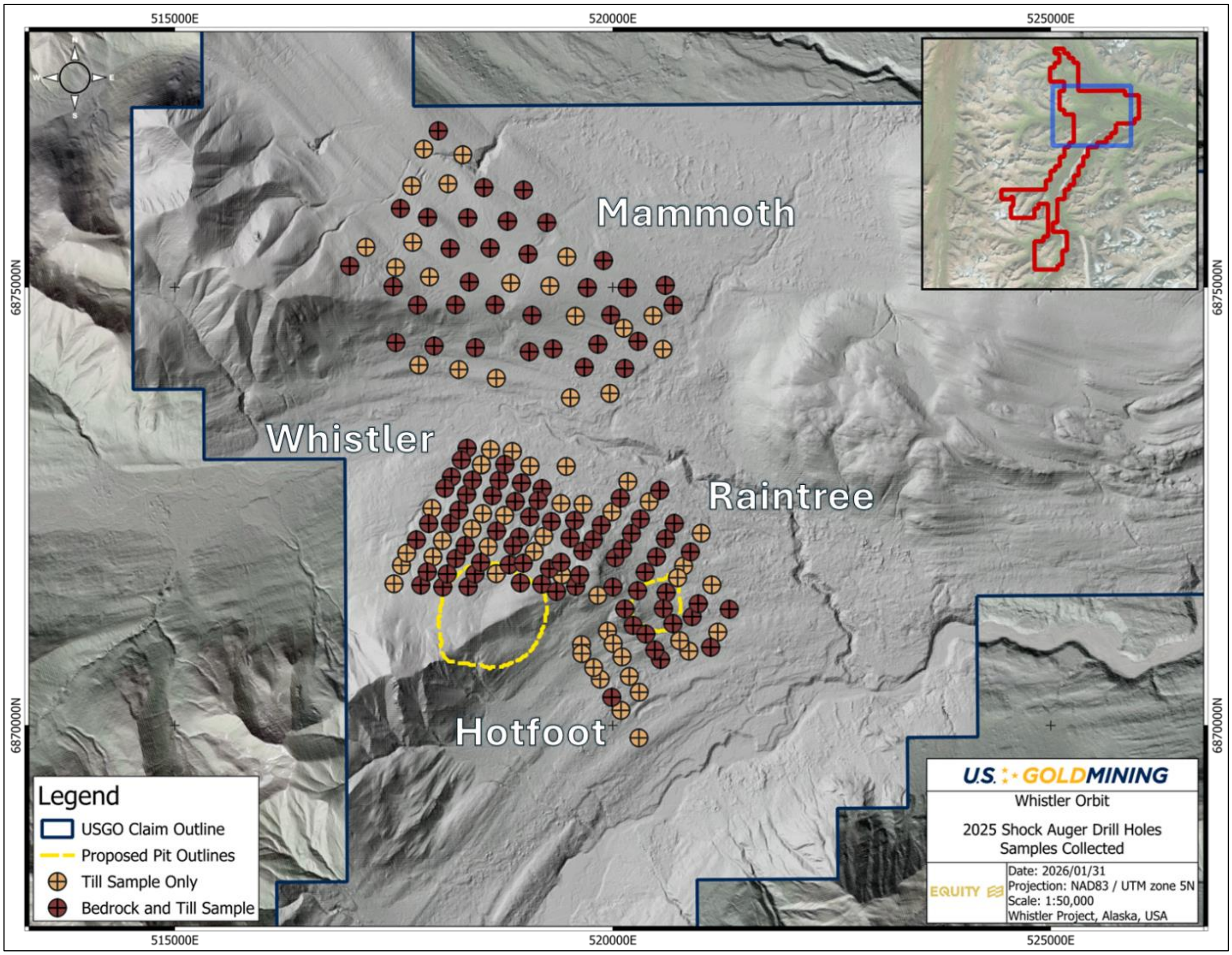


Source: U.S. GoldMining, 2026

7.4.2 Base-of-Till and Top-of-Bedrock Sampling

In 2025 a total of 621 m of shock auger drilling was completed over 172 holes at 169 unique sites. Shock auger drilling in 2025 was conducted by SLR Consulting (Canada) Ltd. (SLR) based in Whitehorse, Yukon, Canada. SLR supplied one helicopter-portable ShockAuger drilling system. Drill collar locations were planned in four grids offset perpendicular to local interpreted ice-flow direction: Whistler, Raintree West, Mammoth and Hotfoot (Figure 7-2). Work completed in 2025 is detailed in Table 7-2.

Figure 7-2: Sample types collected from Shock Auger Drilling in 2025



Source: U.S. GoldMining, 2026

Table 7-2: Summary of Auger Drilling on the Whistler Project

	Total Holes Drilled	Till Samples Collected	Bedrock Samples Collected
Mammoth	85	26	33
Raintree West	63	12	28
Whistler	66	5	41
Hotfoot	62	4	5

7.5 Drilling

A total of 76,493 m of diamond drilling in 267 holes are documented in the Whistler database for drilling on the Whistler Project by Cominco, Kennecott, Geoinformatics, and Kiska from 1986 to the end of 2024. The drilling is summarized in Table 7-3. Of these drillholes 25,121 m in 57 holes have been drilled in the Whistler deposit area, 20,803 m in 89 holes have been drilled in the Raintree West area, and 15,841 m in 42 comprises the Island Mountain resource area. There are 14,727 m in 79 holes in areas outside the three resource areas.

Details on drilling by previous operators can be found in Section 5 of this report.

Table 7-3: Summary of Diamond Drilling on the Whistler Project

Operator	Year	Whistler		Raintree West		Island Mountain		Outside Resource Areas		Total	
		#DH	m	#DH	m	#DH	m	#DH	m	#DH	m
Cominco	1988	13	1,306	-	-	-	-	-	-	13	1,306
	1989	3	370	-	-	-	-	-	-	3	370
	Subtotal	16	1,677	-	-	-	-	-	-	16	1,677
Kennecott	2004	5	1,997	-	-	-	-	1	310	6	2,307
	2005	9	5,251	-	-	-	-	9	1,692	18	6,943
	2006	1	705	4	1,115	2	269	4	1,109	11	3,199
	Subtotal	15	7,953	4	1,115	2	269	14	3,111	35	12,449
Geoinformatics	2007	7	3,321	-	-	-	-	-	-	7	3,321
	2008	5	2,462	2	622	-	-	4	1,219	11	4,303
	Subtotal	12	5,783	2	622	-	-	4	1,219	18	7,624
Kiska	2009			1	479	2	601	2	438	5	1,518
	2010	7	5,247	7	2,890	12	5,434	10	3,103	36	16,674
	2011			73	14,473	26	9,537	48	6,295	147	30,305
	Subtotal	7	5,247	81	17,842	40	15,572	60	9,836	188	48,498
U.S. GoldMining	2023	2	1,074	-	-	-	-	1	561	3	1,634
	2024	5	3,388	2	1,224	-	-	-	-	7	4,611
	Subtotal	7	4,461	2	1,224	-	-	1	561	10	6,246
Grand Total		**57**	**25,121**	**89**	**20,803**	**42**	**15,841**	**79**	**14,727**	**267**	**76,493**

Figure 7-3 through Figure 7-5 are plan views of each deposit illustrating the drillholes by year for Whistler, Raintree West, and Island Mountain respectively. The resource pit outline is shown in black on all figures, with the underground resource confining shape in grey for the Raintree West deposit.

7.5.1 Whistler Deposit Drilling

A total of 7 holes totaling 4,461 m was drilled on the Whistler Deposit by U.S. GoldMining from 2023-2024. These holes were targeted to infill gaps from the previous drill campaigns and to test potential extensions of mineralization on the western and southern margins of the Whistler Deposit, and to depth.

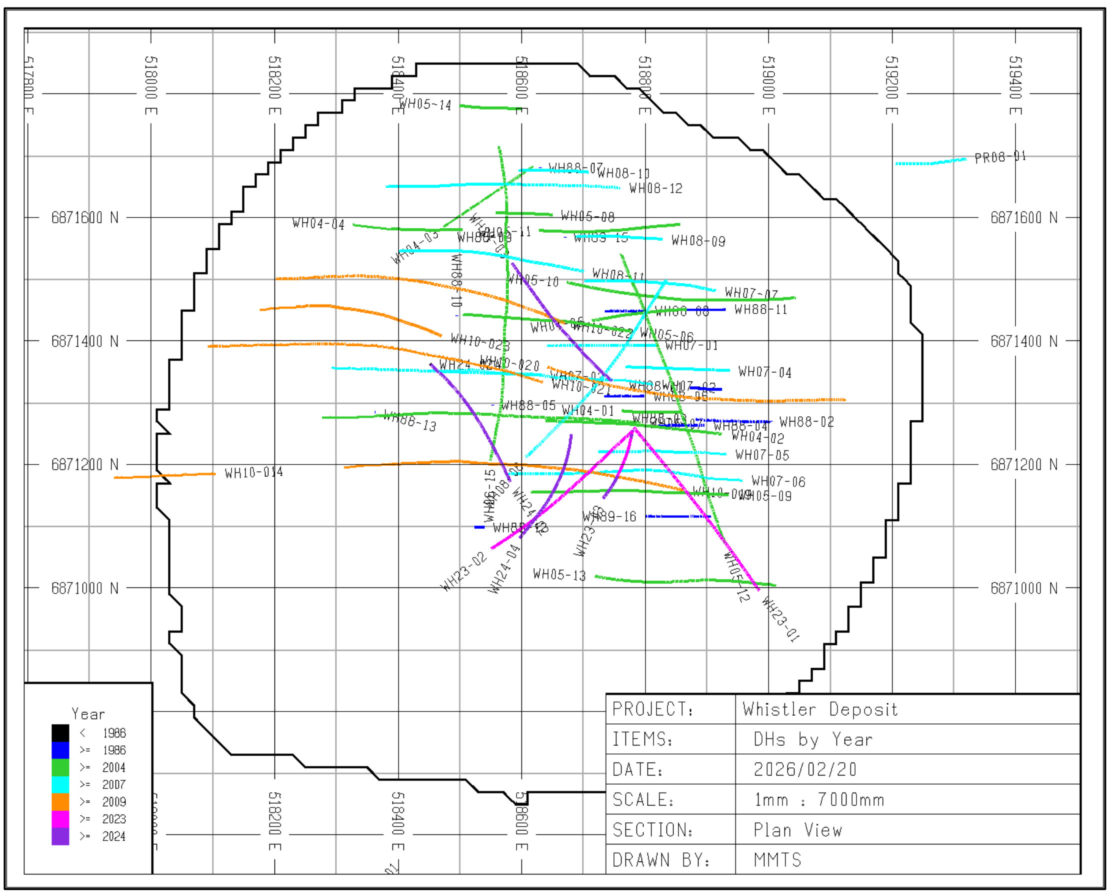
7.5.2 Raintree Area Exploration Drilling

A total of 2 holes totaling 1,224 m was drilled on the Raintree West target by U.S. Goldmining in 2024. These holes tested the modeled geology of the deposit and a southwestern extension of mineralization.

7.5.3 Whistler Area Exploration Drilling

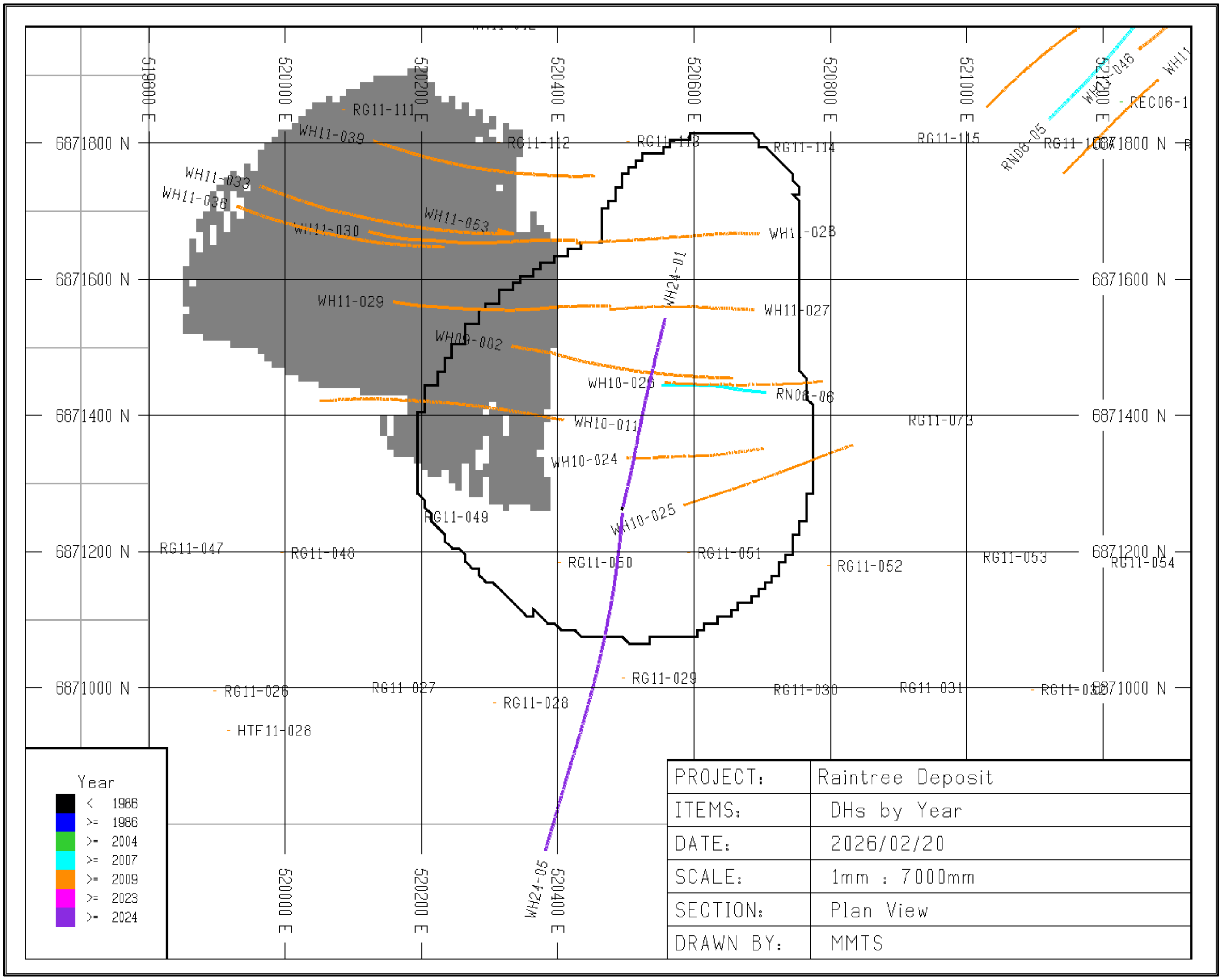
One exploration hole to 560 m depth was drilled at the Rainmaker South target 2023. The hole intersected late-stage porphyry throughout.

Figure 7-3: Plan View of Drillholes by Year/Owner – Whistler



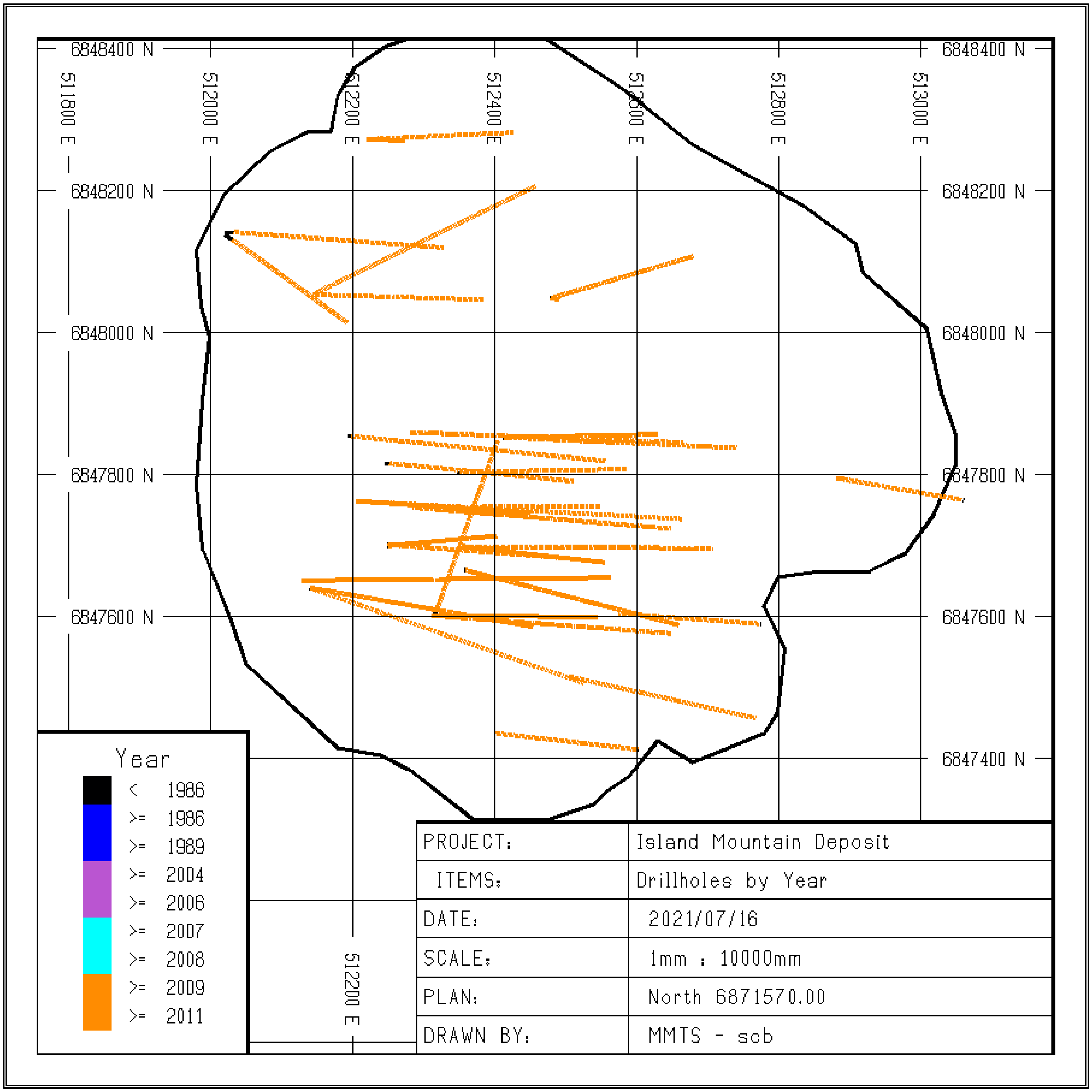
Source: MMTS, 2026

Figure 7-4: Plan View of Drillholes by Year/Owner – Raintree West



Source: MMTS, 2026

Figure 7-5: Plan View of Drillholes by Year/Owner – Island Mountain



Source: MMTS, 2021

8 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section provides an overview of the sample preparation, analyses and security procedures used by the pre-U.S. GoldMining/GoldMining operators of the Project. This section summarizes the verification work and practices employed by each of these operators for which records are available. The independent QP responsible for Section 8 of this report, Sue Bird, P. Eng., believes that these practices are consistent with industry standards and sufficient for their use in mineral resource estimation as detailed herein.

8.1 Sample Preparation and Analyses

8.1.1 Sample Preparation and Analysis-Cominco

There is no available documentation that describes the sampling used by Cominco. The core is not available for data verification. The sample preparation and analytical procedures used by Cominco are not known. Core samples were assayed for gold, silver, and copper and occasionally for a suite of eight other metals (arsenic, cobalt, iron, manganese, molybdenum, nickel, strontium, and zinc) at an unknown laboratory. No certificates of these analyses are available. It is unknown if quality control samples were inserted into the sampling stream, if they were, no records of these samples were available.

8.1.2 Sample Preparation and Analysis – Kennecott and Geoinformatics

Sample preparation protocols for drilling programs on the Project documenting procedures describing all aspects of the field sampling and sample description process, handling of samples, and preparation for dispatch to the assay laboratory, were initially developed by Kennecott and subsequently adopted by Geoinformatics (SRK, 2007).

All soil, rock chips, core, and stream sediments samples were organized into batches of samples of the same type for submission to Alaska Assay Laboratories Inc. in Fairbanks, Alaska (AAL) for preparation using standard preparation procedures. The AAL laboratory is part of the Alfred H. Knight group, an established international independent weighing, sampling, and analysis service company (SRK, 2007).

Kennecott used two primary independent laboratories for assaying samples prepared by AAL. The samples collected during 2004 were assayed at AAL; however, all prepared pulps collected in 2005 and 2006 were submitted to ALS Chemex Laboratory in Vancouver, British Columbia for assaying. The ALS Chemex Vancouver laboratory is accredited to ISO 17025 by the Standards Council of Canada and participates in a number of international proficiency tests, such as those managed by CANMET and Geostats (SRK, 2007).

It is reported (SRK, 2007) that Kennecott used two secondary laboratories for check assaying. ALS Chemex re-assayed 191 pulp samples from the 2004 sampling programs, and Acme Analytical Laboratories Ltd. of Vancouver, British Columbia (Acme) was used as a secondary laboratory in 2005 and 2006. Acme (now Bureau Veritas) is an ISO 17025 accredited laboratory.

Core samples were prepared for assaying using industry standard procedures. Splits of 500 g of coarsely crushed core samples were pulverized to ninety percent passing a -200-mesh screen. Splits of 250 g samples were pulverized to 85% passing a -150-mesh screen. In 2004, 30 g pulp samples were assayed by Alaska Assay Laboratories in Fairbanks for gold by fire assay with atomic absorption finish (AA), and for a suite of nine metals by aqua regia digestion with inductively coupled plasma (ICP). Core and rock samples collected after 2004 were assayed by ALS Chemex for gold by fire assay with AA finish on thirty-gram subsamples and for a suite of thirty-four elements (including copper and silver) by aqua regia digestion and ICP-AES on 0.5 g subsamples. Elements exceeding concentration limits of ICP-AES were re-assayed by single element aqua regia digestion and atomic absorption spectrometry (SRK, 2007).

Kennecott included quality assurance/quality control (QA/QC) samples with all samples submitted for assaying. Each batch of twenty core samples submitted for assaying contained one sample blank, one of three project-specific certified reference materials (CRMs), a field duplicate and a coarse crushed duplicate. These QA/QC samples were inserted blind to the assay laboratory except for the coarsely crushed sample duplicates that were inserted by the preparation laboratory (SRK, 2007).

Geoinformatics used the sample preparation and assaying protocols, and quality control measures developed by Kennecott. All samples collected by Geoinformatics were submitted to Alaska Assay Laboratories for preparation. Pulps were submitted to ALS Chemex by the preparation laboratory for assaying using the same tests described previously (SRK 2008).

Two sample blank materials were collected locally by Kennecott. An andesite rock (OPPBLK-1) collected on outcrop (522,399 m east and 6874,144 m north; NAD27, Zone 5) and porphyritic andesite (WP-BLK-1) intersected in borehole 04-DD-WP-01 (SRK, 2007).

For the Project, Kennecott fabricated three in-house CRMs (WPCO1, WP-MG1 and WP-HG1; from coarse rejects from two boreholes drilled at Whistler (WP04-04-17 and WH04-01-17) that were used through 2010. Coarse rejects from core samples were selected to create three composite samples yielding low, medium and high copper and gold values. Each composite sample was prepared at AAL to yield homogenized pulverized samples for inclusion in the sample stream. Five samples of each standard were then submitted to five commercial laboratories for round-robin assaying. Each standard sample was assayed twice at each laboratory yielding fifty assay results that were analyzed to determine the expected values and standard deviation for QA/QC analysis (Franklin, et al 2006).

8.1.3 Sample Preparation and Analysis – Kiska

Kiska geologists marked out samples for assay after logging the drill core, typically 2 to 3 m in length, honoring lithological and alteration contacts. In general, the drillholes were sampled top to bottom, excepting holes that were partially sampled due to a lack of significant mineralization. After the sample tags were inserted into the core boxes, the core was photographed wet and dry before being cut in half with a diamond saw. One half was submitted for assay, one half was retained (Roberts, 2011a).

In 2009, Kiska used AAL in Fairbanks as the primary assay lab but switched to ALS Chemex for the 2010 and 2011 drilling, both laboratories were independent of Kiska. At AAL samples were dried then crushed to 70% passing 10 mesh, a 250-g split was pulverized to 90% passing 150 mesh. A 30-element suite was conducted by three-acid digestion with ICP-AES and gold was analyzed using 30-g samples by fire assay with AAS finish (Roberts, 2011a).

At ALS Chemex samples were crushed to 70% passing 2 mm, split, and pulverized to 85% passing 75 µm. Gold was analyzed with a 30-g sample by fire assay with AA finish, 33-element analysis and grade were done with four-acid digestion on ICP-AES finish.

Kiska included QA/QC samples at the rate of one CRM, one blank, and one field duplicate (quarter-core) in each batch of 20 samples which were blind to the laboratory. CRMs purchased from Ore Research & Exploration and silica sand was used for blanks. A sample tag was included for a lab duplicate (Roberts, 2011).

8.1.4 Sample Preparation and Analysis – US GoldMining

For the Project drill core sampling program in 2023 and 2024, samples were taken from the NQ/HQ core by sawing the drill core in half, with one half sent to Bureau Veritas Commodities Canada Ltd. (BV) in Fairbanks for sample preparation, then to BV's analytical laboratory in Vancouver, Canada for assaying, and the other half of the core is retained at the site for future reference. Sample lengths downhole were generally 2.0 m, except where samples were taken to honor geological contacts.

BV is a certified commercial laboratory and is independent of U.S. GoldMining. BV holds ISO/IEC 17025:2017 certification as issued by the Standards Council of Canada. The Company has implemented a quality assurance and quality control program for the sampling and analysis of drill core samples, including duplicates, mineralized standards and blank samples for each batch of core samples. The gold analyses were completed by lead collection fire assay fusion with AAS finish (FA430 method) on 30 grams test weight. Copper, silver and other base metals (total suite of 45 elements) assays were assayed by four-acid digestion and ICP-MS analysis (MA200 method) on 0.25 grams test weight.

8.2 Security and Chain of Custody

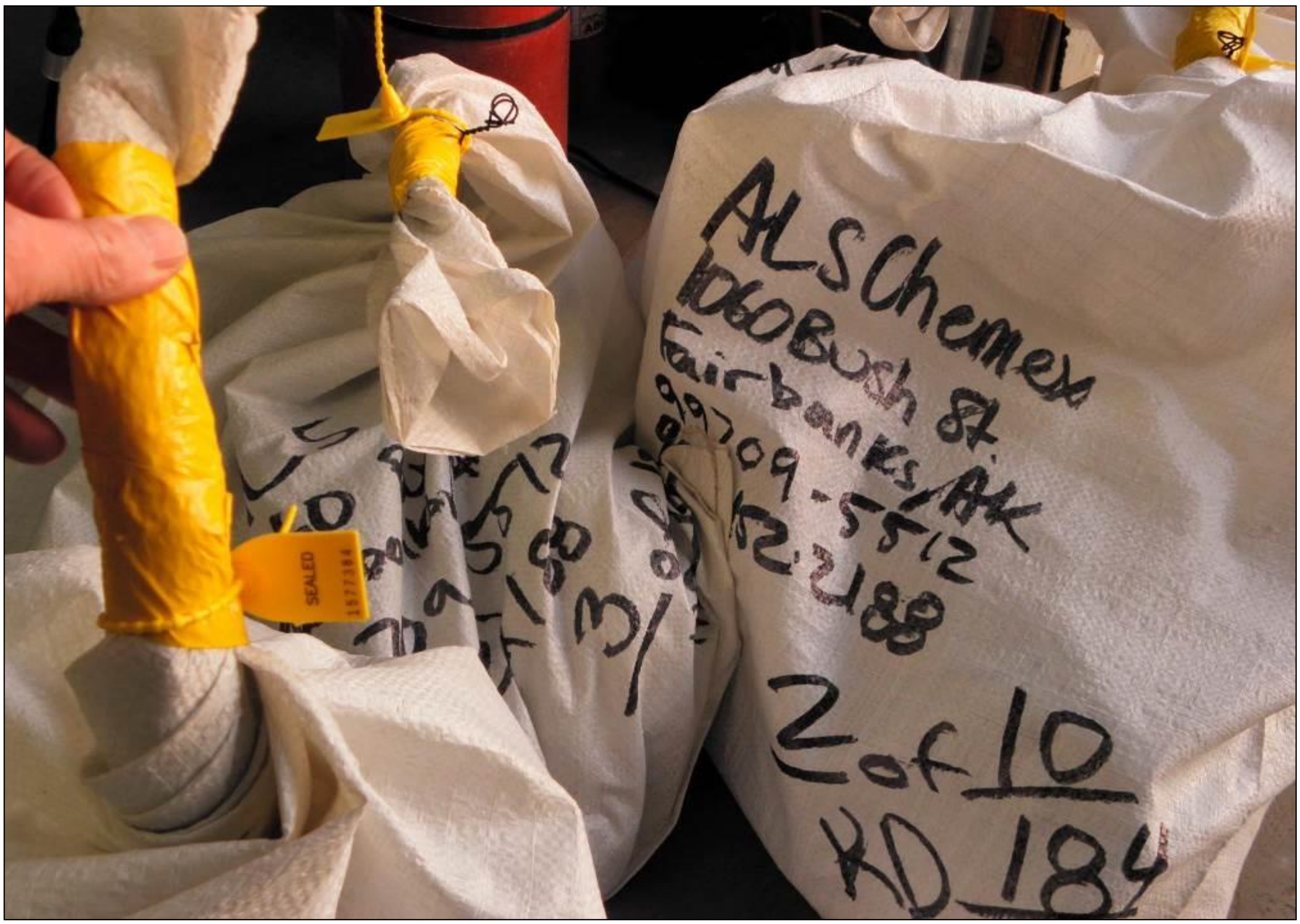
Kennecott devised a documented chain-of-custody procedure to monitor and track all sample shipments departing the base camp until the final delivery of the pulp to the assaying laboratory. Geoinformatics is reported to have adopted all procedures developed by Kennecott. These procedures included the use of security seals on containers used to ship samples, detailed work, and shipping orders. Each transfer point was recorded on the chain-of-custody form up to the final delivery of the pulp to the assay laboratory (SRK, 2007).

Kiska used rice bags closed with security tags to contain the samples for submission as shown in Figure 8-1. The bags were loaded onto Regal Air flights direct to Anchorage and met by an Alaska Minerals representative who delivered them initially to Lynden transport to be shipped to the ALS preparation lab in Fairbanks, AK, or later directly to the ALS preparation lab Anchorage, AK. Prepared pulp samples were shipped to the ALS lab in North Vancouver for assay. Chain-of-custody tracking was documented on the form shown in Figure 8-2 (Roberts, 2011).

Following core processing, U.S. GoldMining's samples were bagged and tagged into separate bags and securely sealed with a zip tie. Samples were bagged in groups of approximately five samples (with a maximum weight of 50 lbs/rice bag) and placed into a rice bag sealed with a security tag. Once finalized, sealed rice bags were put into bulk bags and shipped to Anchorage, Alaska, in batches of up to 100 samples. Sample were backhauled to Anchorage by Desert Air or Regal Air. If samples were shipped by Regal Air, Alaska Minerals Inc. received the samples and transported them to the Desert Air hanger. The samples were securely stored at Desert Air's hanger until being picked up by Carlisle

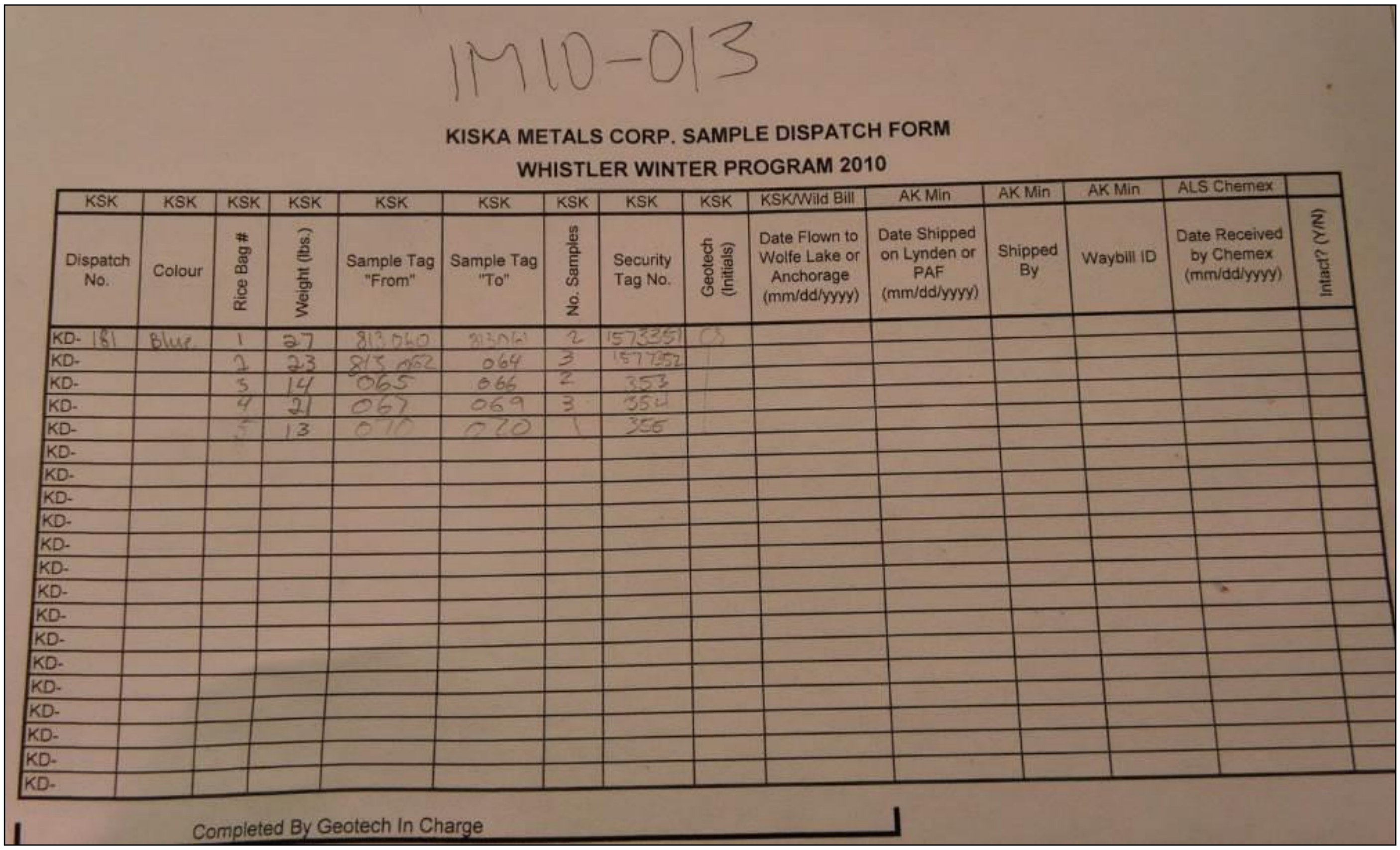
Transportation to be shipped to BV in Fairbanks, Alaska, for sample preparation. Chain-of-custody documents with security tag and sample numbers are checked by the BV management in Fairbanks upon arrival. Upon confirmation of the security tag and sample numbers, the chain-of-custody documents are signed by the BV Fairbanks lab manager and sent back to site management prior to the beginning of sample preparation at the laboratory. An example of the chain-of-custody tracking documents is shown in Figure 8-3. After sample preparation in BV Fairbanks, samples were shipped by BV Fairbanks to BV Vancouver for analysis.

Figure 8-1: Sample Bags with Security Tags



Source: Roberts, 2011a

Figure 8-2: Sample Dispatch Form



IM10-013

KISKA METALS CORP. SAMPLE DISPATCH FORM
WHISTLER WINTER PROGRAM 2010

KSK	KSK	KSK	KSK	KSK	KSK	KSK	KSK	KSK	KSK/Wild Bill	AK Min	AK Min	AK Min	ALS Chemex	
Dispatch No.	Colour	Rice Bag #	Weight (lbs.)	Sample Tag "From"	Sample Tag "To"	No. Samples	Security Tag No.	Geotech (Initials)	Date Flown to Wolfe Lake or Anchorage (mm/dd/yyyy)	Date Shipped on Lynden or PAF (mm/dd/yyyy)	Shipped By	Waybill ID	Date Received by Chemex (mm/dd/yyyy)	Intact? (Y/N)
KD- 181	Blue	1	27	813060	813061	2	1573351	CS						
KD-		2	23	813 062	064	3	1577352							
KD-		3	14	065	066	2	353							
KD-		4	21	067	069	3	354							
KD-		5	13	070	070	1	356							
KD-														
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Completed By Geotech In Charge

Source: Roberts, 2011a

Figure 8-3: U.S. GoldMining Sample Dispatch Form

Security Tag Receiving Form

Project Code:	USGO23-01
Shipped From:	Anchorage, AK
Via:	Carlile Transport
Date Shipped:	2023/09/26
Laboratory:	Bureau Veritas

Date Received:	10/16/23
Lab Job Number:	FBK23002648
Shipment Code:	USGO23-01_04

Bag Number	Seal Number	Sample From	Sample To	Sample From2	Sample To2	Total	Weight (lbs)
1	4107078	3273501	3273504			5	25
2	4107079	3273505	3273508			5	25
3	4107080	3273509	3273509	3273511	3273513	4	20
4	4107081	3273514	3273517			5	25
5	4107082	3273518	3273519	3273521	3273522	4	20
6	4107083	3273523	3273526			5	25
7	4107084	3273527	3273529	3273531	3273531	4	20
8	4107085	3273532	3273535			5	25
9	4107086	3273536	3273539			5	25
10	4107087	3273541	3273544			5	25
11	4107088	3273545	3273548			5	25
12	4107089	3273549	3273549	3273551	3273553	4	20
13	4107090	3273554	3273557			5	25
14	4107091	3273558	3273559	3273561	3273562	4	20
15	4107092	3273563	3273566			5	25
16	4107093	3273567	3273569	3273571	3273571	4	20
17	4107094	3273572	3273575			5	25
18	4107095	3273576	3273579			5	25
19	4107096	3273581	3273584			5	25
20	4107097	3273585	3273588			5	25
21	4107098	3273589	3273589	3273591	3273593	4	20
22	4107099	3273594	3273597			5	25
23	4107100	3273598	3273599	3273601	3273602	4	20
24	4107101	3273603	3273606			5	25
25	4107102	3273607	3273609	3273611	3273611	4	20
26	4107103	3273612	3273615			5	25
27	4107104	3273616	3273619			5	25
28	4107105	3273622	3273624	3273626	3273626	4	20
29	4107106	3273627	3273627	3273631	3273634	5	25
30	4107107	3273636	3273639			5	25
31	4107108	3273640	3273643			5	25
32	4107109	3273644	3273647			5	25
33	4107110	3273648	3273651			5	25
34	4107111	3273652	3273655			5	25
35	4107112	3273656	3273659			5	25
36	4107113	3273660	3273663			5	25
37	4107114	3273664	3273667			5	25
38	4107115	3273668	3273671			5	25
39	4107116	3273672	3273676			6	30
					Total:	186	930

Please examine the shipment carefully for evidence of tampering. Check that:
a) the sides and/or bottom of the bag have not been cut or resealed,
b) the closure at the top of the bag is not distorted or torn, and
c) there is no evidence of the seal being removed from or returned to the bag.

If any of the above features are noted, DO NOT OPEN.
Notify Equity Exploration Consultants Ltd. at 604-688-9806 IMMEDIATELY.

Received in good order by: [Signature] Michael Romeric

Signature Name

Please email the signed form to EvanM@equityexploration.com

Page 1 of 1

Source: U.S. GoldMining, 2024

8.3 QA/QC Summary

For this 2026 iteration of the report, MMTS reviews both the new 2023 and 2024 geochemical data and its QA/QC portion and all historical data back to 2004. Several misclassifications and data mix-ups were corrected in the process, particularly for the data pre-2023, which results in updated tables, graphs, and interpretations compared to the 2024 report. Lab-internal precision control data is partially being included and described.

The total number of assayed intervals and the respective QA/QC samples classified as Certified Reference Materials (CRM or STD), un-certified reference materials (2004 only), blanks, duplicates and check assays in the provided database is given in Table 8-1. It shows the QA/QC rate at 20.5% for Whistler, 24% for Raintree and 27.2% for the Island Mountain portion of the database. The QA/QC rates refer to the percentage of total analyses and include the lab-internal duplicates and replicates but not the lab-internal blanks and standards which were not reviewed for this report.

The QA/QC insertion rates for all three deposits meet or exceed industry standards. Not included in Table 8-1 are >900 Cu-only results from 2004 when a Cu analysis methods comparison program between AAL (primary lab, ICP-2A) and ALS Elko (Cu-AA45) was completed.

Table 8-1 also contains 'unknown' duplicate data from 2004 to 2006 when coarse duplicate and field duplicate designations could not be confirmed from certificates or historical reporting.

QA/QC data for copper and gold are presented here. The analysis of the QA/QC samples split by deposit follows in Sections 0 to 8.3.3.

Table 8-1: QA/QC Sample Summary (All Areas and Years)

Deposit	Year	Samples	Blanks	CRMs	Field Dups*	Umpire	QA/QC all	% QA/QC
Whistler	1986-1989	697	n/a	n/a	n/a	n/a	n/a	n/a
	2004	920	44	44	45	206	339	26.93%
	2005	2,600	132	132	262	67	593	18.57%
	2006	353	22	22	44	0	88	19.95%
	2007	1,347	75	77	138	0	290	17.72%
	2008	1,267	71	71	115	0	257	16.86%
	2010	1,726	106	107	335	0	548	24.10%
	2023	1,011	51	50	160	42	303	23.06%
	2024	1908	125	120	289	103	637	25.03%
	Whistler All	**11,829**	**626**	**623**	**1,388**	**418**	**3,055**	**20.53%**
Raintree	2006	383	22	25	46	0	93	19.54%
	2008	312	18	18	29	0	65	17.24%

Deposit	Year	Samples	Blanks	CRMs	Field Dups*	Umpire	QA/QC all	% QA/QC
	2009	263	15	16	51	0	82	23.77%
	2010	1,171	74	76	234	0	384	24.69%
	2011	5,040	313	317	1006	0	1636	24.51%
	Raintree All	**7,169**	**442**	**452**	**1,366**	**0**	**2,260**	**23.97%**
Island Mountain	2006	85	5	6	11	0	22	20.56%
	2009	293	16	18	54	0	88	23.10%
	2010	2,350	156	146	456	0	758	24.39%
	2011	2,378	203	196	643	0	1042	30.47%
	Island Mountain All	**5,106**	**380**	**366**	**1,164**	**0**	**1,910**	**27.22%**
Total		**24,104**	**1,448**	**1,441**	**3,918**	**418**	**7,225**	**23.06%**

* all available field duplicates, coarse duplicates, and pulp duplicates, including the respective lab-internal reproducibility control data.

As is common with projects that have been developed by various owners or operators over multiple decades, several laboratories with different analytical methods, a wide range of CRMs as well as multiple blank materials have been utilized at Whistler, Raintree West, and Island Mountain. Already described in some detail under Section 8.1, the following laboratories in Table 8-2 were contracted for sample preparation and geochemical analysis of drill core samples over the last 20 years.

Table 8-2: 2004-2024 Assay Laboratories and assay methods for Au and Cu

Year	Primary lab		Cu method	CU DL (ppm)	Au method	Au DL (ppm)
2004	American Assay	AAL	ICP-2A	1	FA30	0.003
2005	ALS Chemex	ALS	ME-ICP41a	5	Au-AA23	0.005
2006	ALS Chemex	ALS	ME-ICP41a	5	Au-AA23	0.005
2007	ALS Chemex	ALS	ME-ICP41	1	Au-AA23	0.005
2007	ALS Chemex	ALS	ME-ICP41a	5	Au-AA23	0.005
2008	Alaska Assays	AKA	ICP-4A	2	FA30 AAS	0.01
2009	Alaska Assays	AKA	ICP-3A	2	FA30 AAS	0.01
2010	ALS Chemex	ALS	ME-ICP61	1	Au-AA23	0.005
2010	ALS Chemex	ALS	ME-MS61	0.2	Au-AA23	0.005
2011	ALS Chemex	ALS	ME-ICP61	1	Au-AA23	0.005
2023	Bureau Veritas	BV	MA200	0.1	FA430	0.005
2024	Bureau Veritas	BV	MA200	0.1	FA430	0.005

The project's QA/QC database currently contains six different designations for blank material (see Table 8-3), of which OPPBLK-1 and WP-BLK-1 have been described under Section 8.1.2. The variability in granularity and hardness of the utilized materials as well as in natural background concentrations in andesitic to basaltic rocks contributed to significant scatter in blank plots, particularly for copper, potentially masking true inter-sample contamination during perp work at the respective labs. Several follow-up-checks on theoretical 'failure' data points and their preceding samples confirmed contamination to be an unlikely cause for these outliers.

Table 8-3: Blank Material Details 2004-2024

Blank name	Years	Source	Comment
Blank	2009	outcrop	Basalt from property outcrop
Blank_SS	2010-2011	n/a	Quartz sand
Blank_Whistler	2007-2008	core	Barren core of WH05-04
OPPBLK-1	2004-2006	outcrop	Andesite from outcrop
VIGORO_blank	2023-2024	purchase	limestone crush
WP-BLK-1	2005-2006	core	Porphyritic andesite from core

To control accuracy of the reported assay results, between 2 and 5 of the following blind reference materials were inserted into the sample streams of each drilling and sampling campaign, starting in 2004. Table 8-4 lists all standards, while Figure 8-6 and Figure 8-7 in the following sections detail their respective performances for each project.

Table 8-4: CRM Detail 2004-2024

CRM name	Years	EV Au ppm	Au SD	EV Cu %	Cu SD	Comment
HMM	2004	0.35*	n/a	1.2*	n/a	* estimate, unknown material
MHH	2004	1.3*	n/a	0.44*	n/a	* estimate, unknown material
UC-2	2004	1.28*	n/a	0.79*	n/a	* estimate, unknown material
UC-5	2004	0.38*	n/a	0.69*	n/a	* estimate, unknown material
OREAS-50c	2010-2011	0.836	0.028	0.742	0.016	Cu 4A, Au FA
OREAS-52c	2010-2011	0.346	0.017	0.344	0.009	Cu 4A, Au FA
OREAS-52Pb	2010	0.307	0.017	0.3338	0.0067	Cu 4A, Au FA
OREAS-53Pb	2010	0.623	0.021	0.546	0.013	Cu 4A, Au FA
OREAS-54Pa	2010-2011	2.9	0.11	1.55	0.02	Cu 4A, Au FA
OREAS 501d	2023-2024	0.232	0.011	0.272	0.009	Cu 4A, Au FA
OREAS 503e	2023-2024	0.709	0.018	0.531	0.016	Cu 4A, Au FA
WP-CO1	2005-2009	0.481	0.026	0.2802	0.0057	project-specific CRM
WP-HG1	2005-2009	4.693	0.19	0.616	0.0133	project-specific CRM
WP-MG1	2005-2009	1.715	0.123	0.2594	0.0052	project-specific CRM

The apparent standards utilized in 2004 by Kennecott (HMM, MHH, UC-2, and UC-5) could not be confirmed to be certified reference material and have therefore been excluded from the normalized gold and copper graphs in the following sections. Standard deviations (SD) as listed in Table 8-4 refer to the 'in-between labs' standard deviation calculations as reported on the respective certificates (COA) by OREAS.

8.3.1 QA/QC Whistler Deposit

8.3.1.1 Whistler Blanks

The summary of the blind gold assays samples of blank material used to assess contamination in the Whistler Deposit sample stream is given in Figure 8-5. The results show an overall 1% failure rate at 10 times detection limit (DL), which is acceptable (5 fails total). The use of locally sourced andesite and porphyritic andesite taken from drill core as blank material by both Kennecott and Geoinformatics in 2004 to 2006 and 'BLANK_WHISTLER' in 2007-2008 may have contributed to the elevated number of 5*DL warning results (3.3% of total), given that this material recorded elevated background values in Cu on occasion as well.

The silica sand utilized by Kiska in 2010 did not produce any warnings or failures but because of its smaller grain size is also unlikely to have gone through the crushing stage during preparation which is a common source of contamination.

The VIGORO limestone crush in 2023-2024 did not record any contamination fails.

Table 8-5: Summary of Gold Assays of Blanks, Whistler Deposit

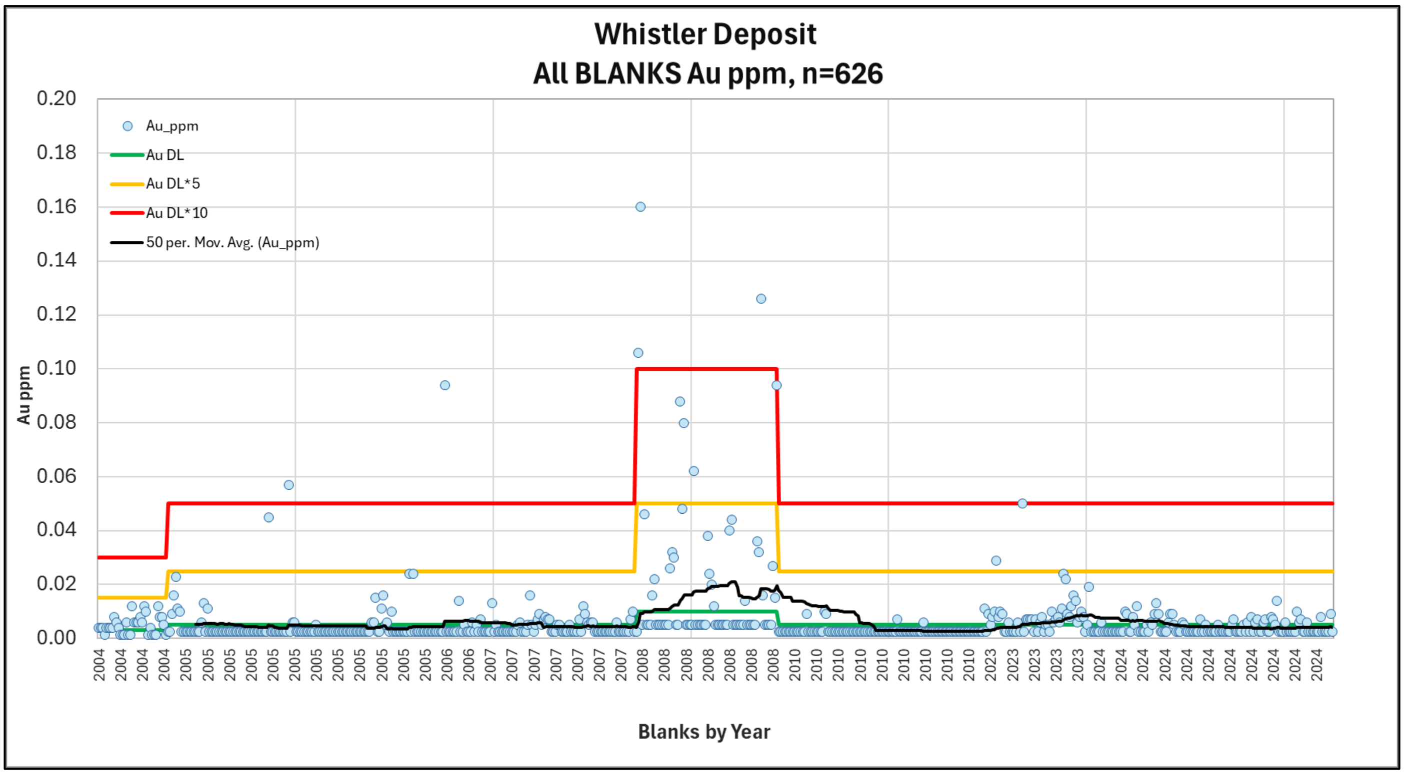
Blank	Year used	Count	>5*DL warning	% >5*DL	>10*DL fail	% >10*DL
OPPBLK-1	2004-2006	144	2	1.4%	2	1.4%
WP-BLK-1	2005-2006	53	1	1.9%	0	0.0%
BLANK_WHISTLER	2007-2008	146	7	4.8%	3	2.1%
BLANK_SS	2010	106	0	0.0%	0	0.0%
VIGORO Blank	2023-2024	176	2	1.1%	1	0.6%
Total	**2004-2024**	**626**	**12**	**1.9%**	**6**	**1.0%**

A sequential plot of gold assays of blanks is presented in Figure 8-4, with the green line representing the DL, the orange line the 5*DL warning threshold, and the red line the 10*DL failure threshold. The blank performance is poorest in 2008 when Alaska Assay was contracted to perform the prep and analysis of the core samples. However, with a fire assay Au detection limit of 0.01 ppm, only three of the elevated assays in 2008 technically classify as fails as per definition of >10*DL.

Of note is that the VIGORO blank Au performance in late 2023 appears to indicate some weak contamination at Bureau Veritas, without reaching the warning or failure threshold. The affected blanks are mostly from drill hole WH23-03 which is strongly mineralized from 0 to approximately 650 m and averages 0.7 g/t Au across that interval. The corresponding blanks average 0.01 g/t Au or 2*DL. MMTS is not concerned about those results from a resource estimation perspective since significant grade smearing is unlikely.

Unsurprisingly, the use of silica sand as blank material in 2010 produced the fewest Au results above DL.

Figure 8-4: Sequential Plot of Gold Assays of Blanks, Whistler Deposit



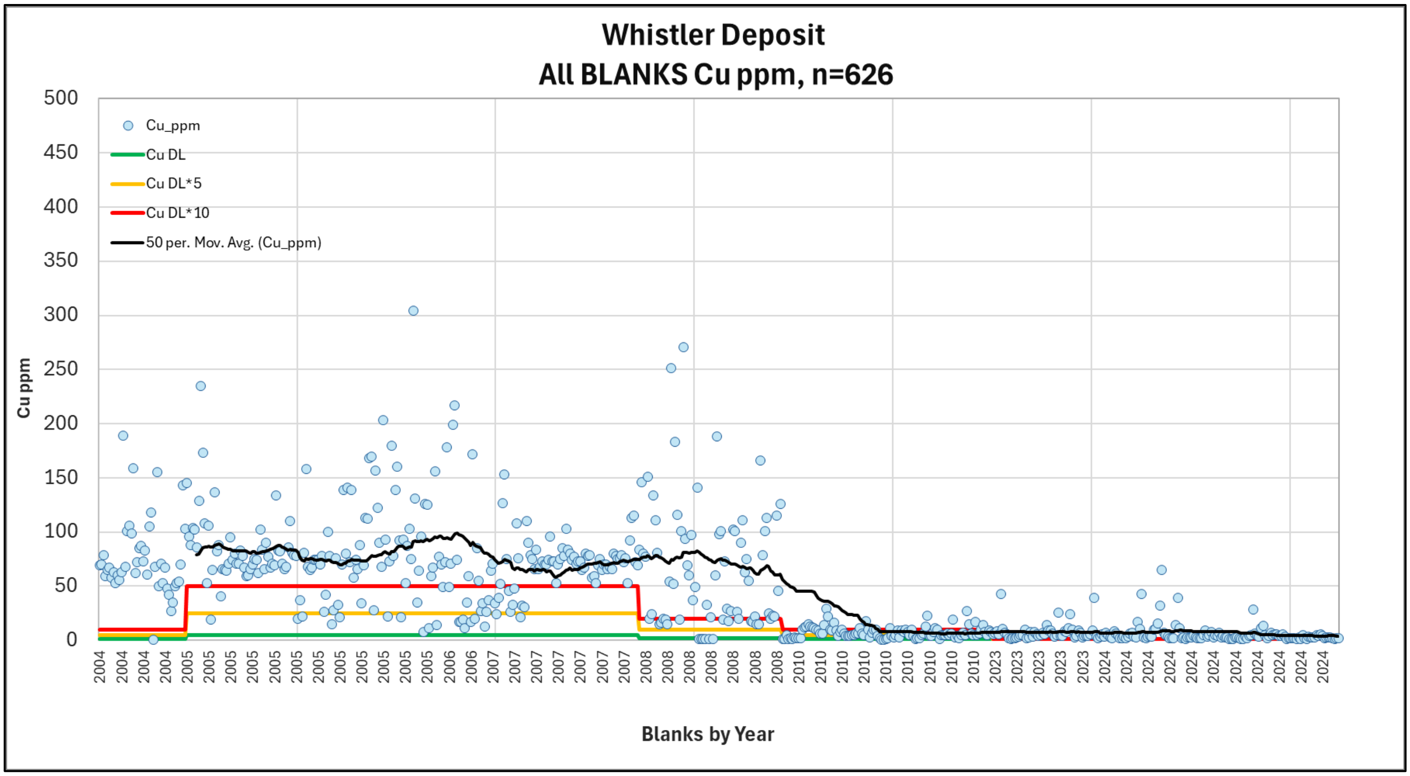
Source: MMTS, 2024

The DL for copper assays at the Whistler Deposit is either 0.1 and 5 ppm depending on the analysis lab and year and applying a criterion of 5*DL or 10*DL results in an extremely high failure rate due to elevated natural copper concentrations in blank material, especially from 2004 to 2008. After spot-checking copper grades or preceding samples to several select high-copper outliers, MMTS is of the opinion that natural variability in copper grades is causing the frequent 'failures' and cross-sample contamination, while certainly possible on occasion, is likely irrelevant.

Starting in 2010, quartz sand was used as blank material which led to background copper levels dropping significantly (yet still well above DL) and for 2023 and 2024, a limestone crush with an equally low copper concentration of <10 ppm was utilized. None indicate any relevant copper contamination. The highest copper result reported was 65 ppm.

The sequential plot of copper assays of blanks in the Whistler Deposit is presented in Figure 8-5.

Figure 8-5: Sequential Plot of Copper Assays of Blanks, Whistler Deposit



Source: MMTS, 2026

8.3.1.2 Whistler CRMs

Gold and copper data for 581 blind CRMs inserted into the Whistler sample stream are available in the database, though one CRM failed to report gold. The results of analysis of these samples are given in Table 8-6 in order of increasing grade of the expected value (EV) and show that the overall failure rate is an acceptable 1.9%, with no gold failures in 2023-2024. A total of 8 of 10 CRMs fall between 0.2% and -2.5% error (%) which indicates a minor negative bias to the laboratory gold assays. Two OREAS standards from 2010 (52Pb and 50c) perform very poorly and their use was discontinued. OREAS-Pb52 with the greatest percentage error (+8.1%) has only been inserted and analyzed twice and its performance is therefore not considered a concern.

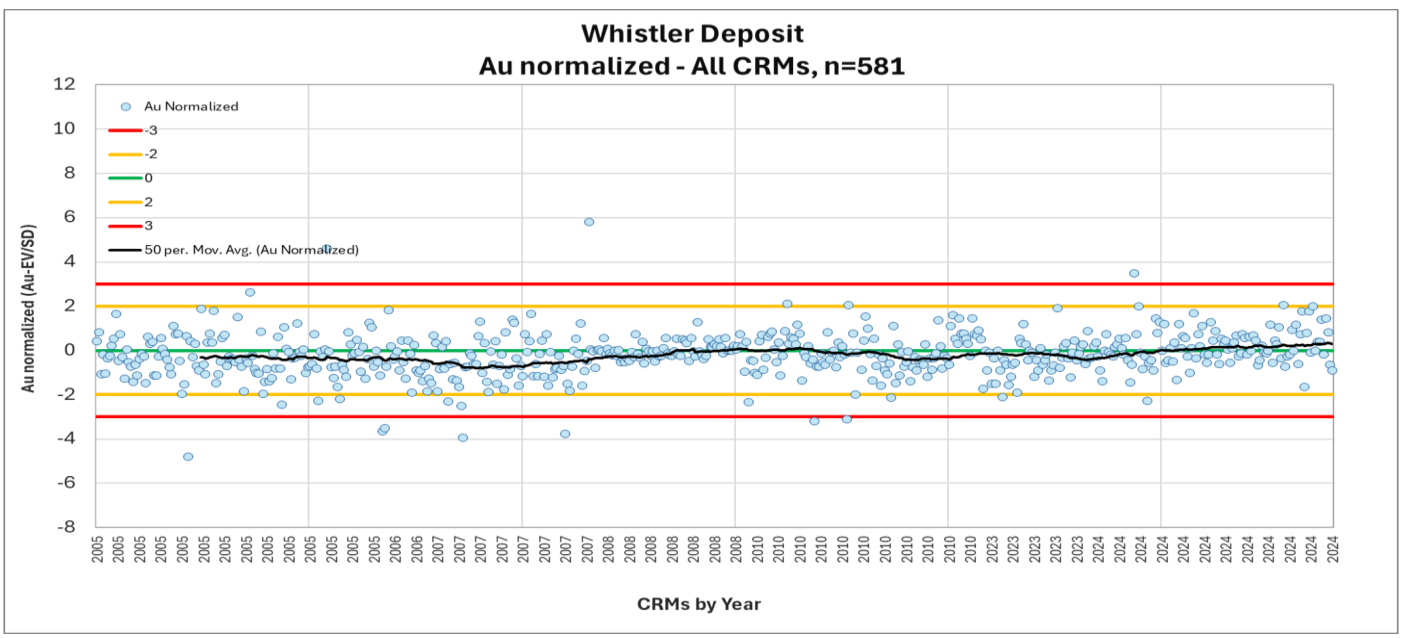
The normalized process control chart showing results for gold is given in Figure 8-5 and shows the acceptable results across all CRMs, highlighted by a 50-sample moving average line that tracks the expected values shown as z-score = zero quite well and demonstrates the aforementioned weak negative bias.

Table 8-6: Whistler Deposit CRM Summary, Gold

CRM	Year used	Count	EV Au (g/t)	AVG Au (g/t)	% Error	Low Fail	High Fail	% Fail
OREAS 501d	2023-2024	90	0.232	0.232	-0.1%	0	0	0.0%
OREAS-52Pb	2010	2	0.307	0.334	8.1%	0	0	0.0%
OREAS-52c	2010	51	0.346	0.343	-1.0%	1	0	2.0%
WP-CO1	2005-2010	136	0.481	0.471	-2.0%	2	2	2.9%
OREAS-53Pb	2010	16	0.623	0.618	-0.9%	0	0	0.0%
OREAS 503e	2023-2024	80	0.709	0.711	-0.2%	0	1	1.3%
OREAS-50c	2010	13	0.836	0.763	-9.5%	1	0	7.7%
WP-MG1	2005-2008	96	1.715	1.674	-2.5%	0	0	0.0%
OREAS-54Pa	2010	25	2.901	2.878	-0.8%	1	0	4.0%
WP-HG1	2005-2010	72	4.693	4.650	-0.9%	3	0	4.2%
Total	**2005-2024**	**581**	**-**	**-**	**-**	**8**	**3**	**1.9%**

Figure 8-6 also demonstrates that Alaska Assays in 2008 was able to measure gold concentration in the blind CRMs most accurately (standards with three different Au grades used that year), while ALS Chemex and BV assays exhibit larger spreads.

Figure 8-6: Whistler Deposit Normalized Process Control Chart, Gold



Source: MMTS, 2026

The summary of copper assays of the CRMs is given in Table 8-7 in order of increasing grade and shows the overall failure rate to be acceptable at 2.6% and the percent error to be negligible at -0.6% when straight averaged. OREAS 52Pb and 50c again perform poorly with high failure rates, while OREAS 54Pa as the highest-grade CRM used at Whistler consistently underperforms. Only 2 of 24 approach or exceed the EV of 1.55% Cu.

Table 8-7: Whistler Deposit CRM Summary, Copper

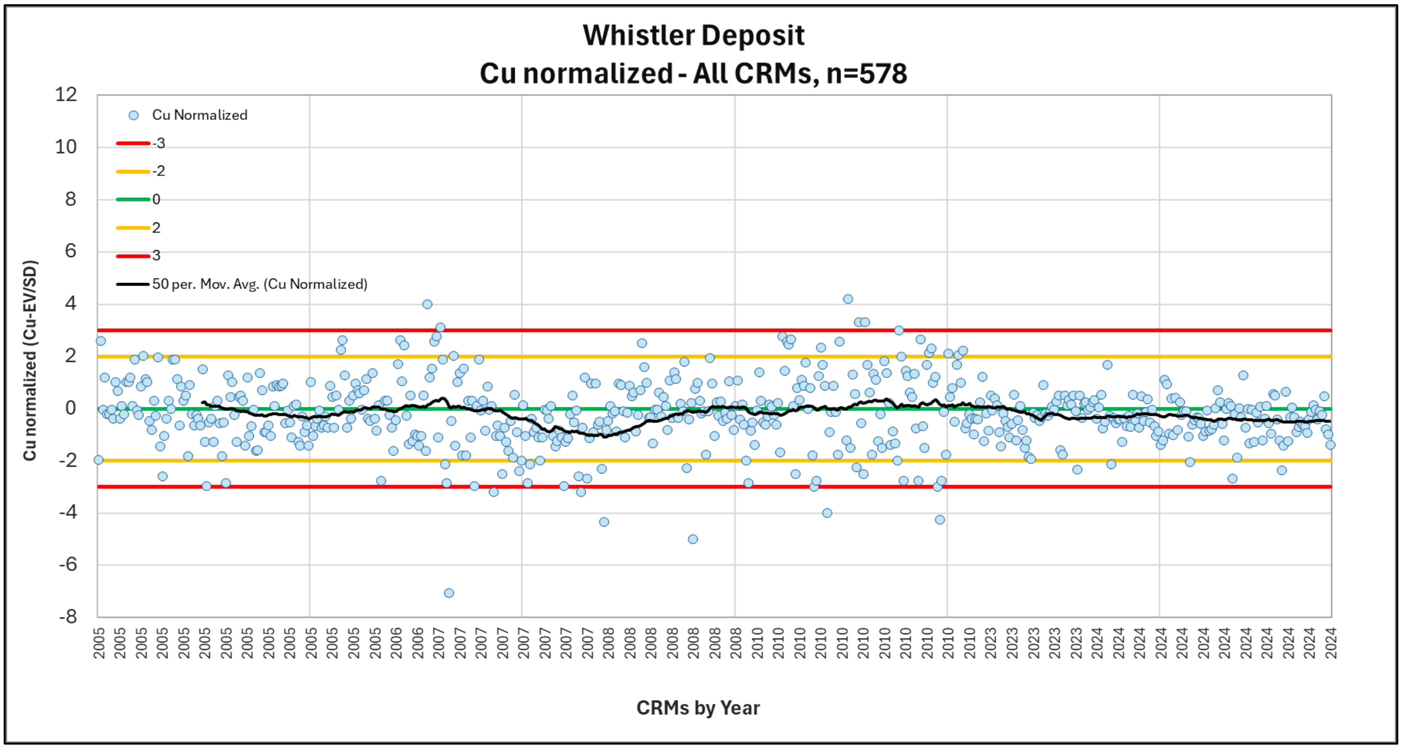
CRM	Year used	Count	EV Cu %	AVG Cu %	% Error	Low Fail	High Fail	% Fail
WP-MG1	2005-2008	96	0.259	0.258	-0.7%	0	0	0.0%
OREAS 501d	2023-2024	90	0.272	0.267	-2.0%	0	0	0.0%
WP-CO1	2005-2010	136	0.280	0.279	-0.4%	5	2	5.1%
OREAS-52Pb	2010	2	0.334	0.345	3.2%	0	1	50.0%
OREAS-52c	2010	50	0.344	0.345	0.3%	1	0	2.0%
OREAS-53Pb	2010	16	0.546	0.539	-1.3%	0	0	0.0%
OREAS 503e	2023-2024	80	0.531	0.528	0.5%	0	0	0.0%
WP-HG1	2005-2010	71	0.616	0.608	1.3%	1	0	1.4%
OREAS-50c	2010	13	0.742	0.766	3.1%	0	2	15.4%
OREAS-54Pa	2010	24	1.557	1.451	-6.8%	3	0	12.5%
Total	**2005-2024**	**578**	**-**	**-**	**-**	**10**	**5**	**2.6%**

The normalized process control chart for copper is given in Figure 8-7 in order of processing and shows the acceptable results with relatively few failures. Of note, however, are the moderate negative biases in normalized copper for 2007 (ALS Chemex) and 2023-2024 (BV). In 2007, three certificates received in November (FA07110167, FA07112984, and FA07127160) delivered consistently low copper results across the three CRMs used at the time and should have been flagged for review and potential re-assaying at ALS.

In 2023 and 2024, the low-range CRM OREAS 501d moderately but consistently underperformed by approximately 2.5% and was also used disproportionately often in comparison to the other CRM used in 2023 (30 to 18 insertions, respectively). MMTS recommends using three standards of different copper and gold concentrations to adequately represent the low, medium, and high expected metal grades at Whistler.

The performance of both gold and copper CRMs in the Whistler Deposit indicates acceptable accuracy (Figure 8-7).

Figure 8-7: Whistler Deposit Normalized Process Control Chart, Copper



Source: MMTS, 2026

8.3.1.3 Whistler Field Duplicates

GoldMining's assay database provides a significant number of field duplicates for the 2010-2024 periods. Unfortunately, for earlier years 2004-2006, the distinction between field duplicates and company-initiated coarse reject duplicates has not been retained, so that a general 'Duplicate' designation has been given to all QA/QC samples that could not be identified as blank or CRM.

The simple statistics of the field duplicates in the Whistler Deposit are given in Table 8-8. It is seen in the averaged absolute relative difference (ARD%) of the gold assays that the % difference of the means is -1.6% indicating there is a small positive bias to the duplicate samples as original samples as compared to the duplicates. There is basically no difference in the average ARD% of the copper assay pairs.

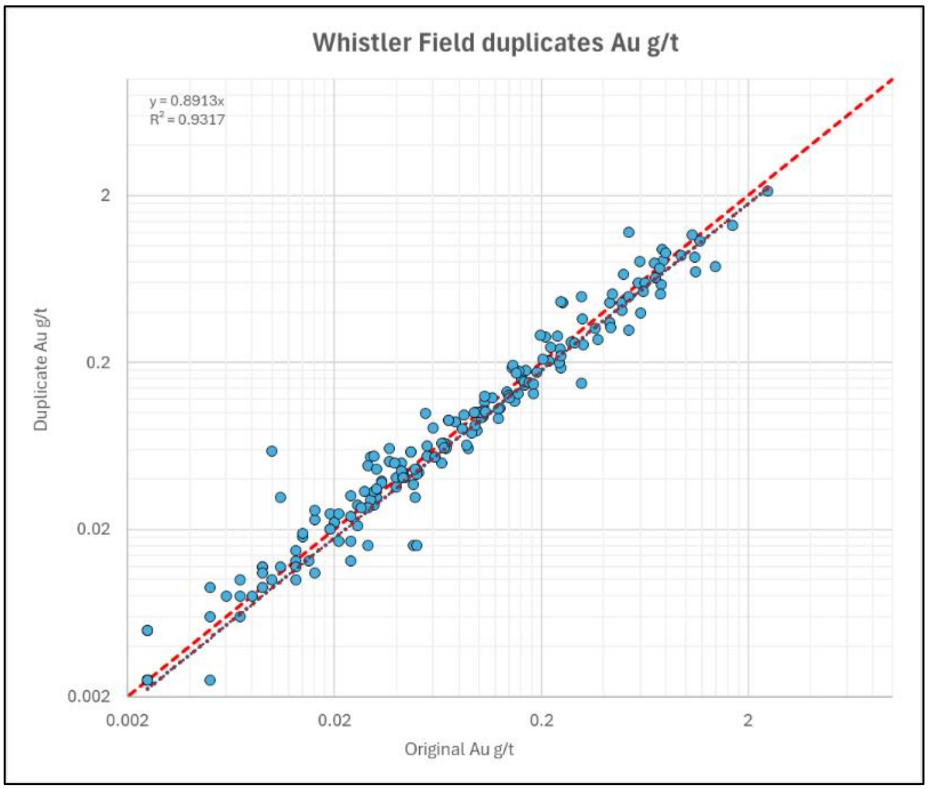
The percentage below 10% Half Absolute Relative Difference (HARD) is 61.1% for gold and 68.4% for copper. The expectation for field duplicates is that 70% or more are below 10%, this is met for either copper or gold, indicating the gold mineralization in Whistler is not highly heterogenous.

Table 8-8: Whistler Field Duplicates Simple Statistics

DUP count	Element	Units	Average			Count >10% HARD	% <10% HARD
			Primary	Duplicate	ARD % Avg.		
193	Gold	g/t	0.196	0.188	-1.59%	75	61.1%
193	Copper	ppm	930	923	-0.17%	61	68.4%

The small statistical original-positive bias of gold assays is also observed in the scatter plot in Figure 8-8 where most of the data pairs plot right around the 1 to 1 line (red dash), with the highest-grade pairs pushing the averages slightly towards the original results. The high correlation coefficient of 0.93 reflects the very good grade distribution of the population and the overall good precision of the higher-grade pairs.

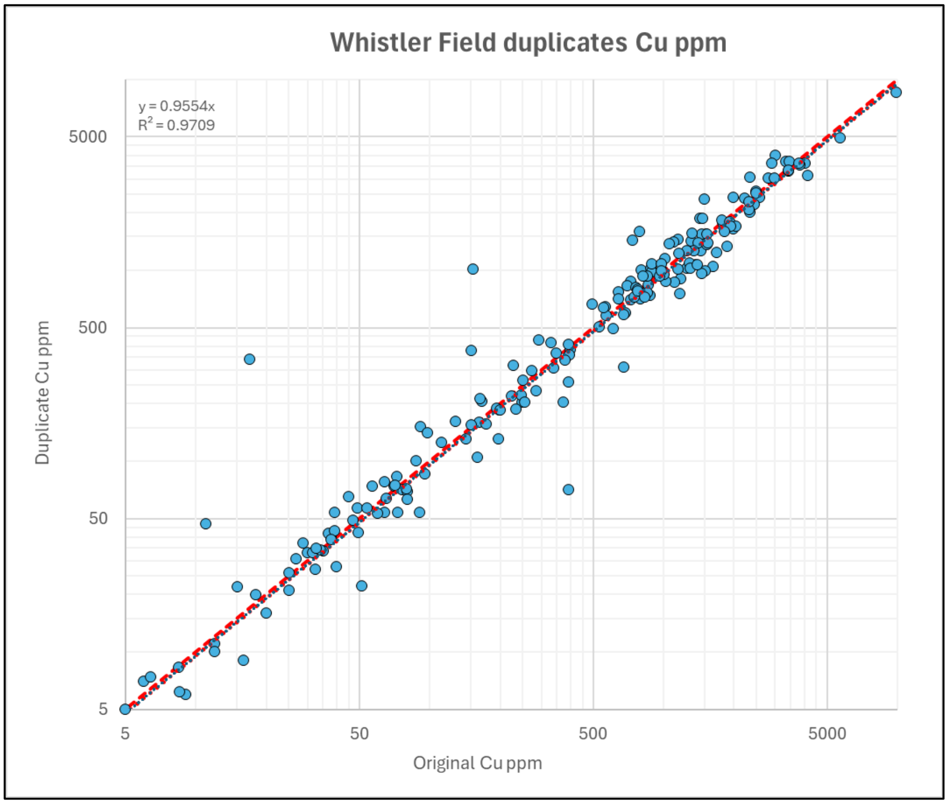
Figure 8-8: Whistler Deposit Field Duplicate Scatter Plot, Gold



Source: MMTS, 2026

The scatter plot of copper field duplicates is given in Figure 8-9 and shows a very good correlation between pairs with slope of best fit line slightly above 1.0 and R2 at 0.97. Again, the grade distribution of the dataset is acceptable, and no significant higher-grade outliers are noted.

Figure 8-9: Whistler Deposit Field Duplicate Scatter Plot, Copper



Source: MMTS, 2026

In addition to the field duplicates, MMTS briefly reviewed the performance of both coarse (367 total pairs) and pulp duplicates (587) for the years 2007-2024 as produced regularly by the respective labs during the size reduction process as part of the lab-internal QA/QC protocols as well as a significant number of client-requested coarse duplicates.

For Au, the R^2 increases from 0.93 (field duplicates) to 0.96 (coarse duplicates) and 0.99 (pulp duplicates), while the rate of the data pairs that plot <10% HARD increases substantially from 61% to 81% to 93%, respectively.

For Cu, the progressions are similar, with 0.97, 0.99, and 0.96 for R^2. The pulp duplicate dataset appears to contain a small number of pairs from 2004 where the ICP lab results reported by AAL might be erroneous, restricted to drillhole WH04-01. The <10% HARD rate improves from 68% to 95% to 98%, respectively.

8.3.1.4 Whistler Check Assays

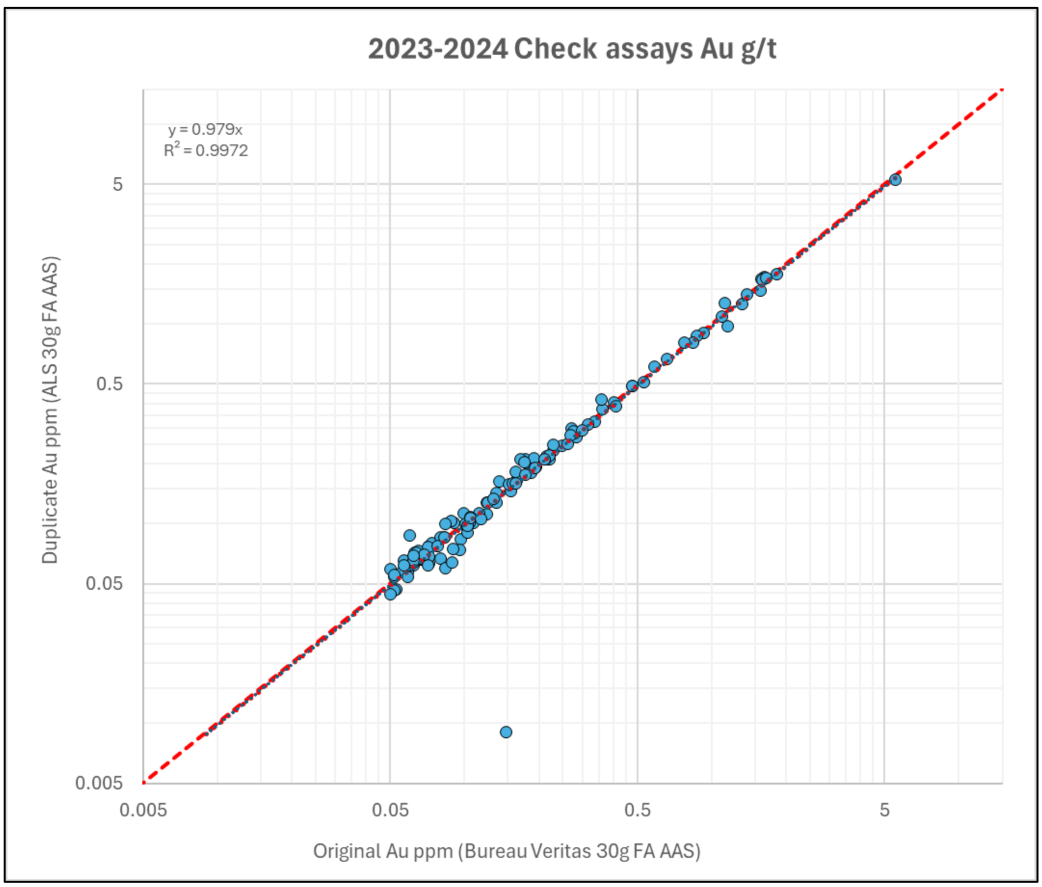
As described and visualized in the 2024 report (MMTS, 2024), approximately 200 hundred pulps from drill hole WH04-05 originally prepared and analyzed by American Assay in 2004 were sent to ALS Chemex in Fairbanks, Alaska, for gold and copper check-assay purposes. Both labs used a 30-g charge for the fire assay Au analysis and the results compare very well.

For copper analysis, aqua regia digestion was requested from both AAL and ALS but the actual instrumentation differed (ICP for AAL and atomic absorption for ALS). As a result, the ALS umpire analyses were reported as consistently higher by about 20%. In response to the strong bias, Kennecott resubmitted all 2004 pulps to ALS Chemex in Elko, Nevada, for additional copper analyses, again requesting Cu-AA45 and Cu-AA46 (overlimit) methods with aqua regia digestion. This most current Cu data is being used for resource estimation.

In 2005, Kennecott contracted ALS Chemex as the primary lab and chose ACME to perform the umpire assaying on a select 93 samples, 81 of which are relevant core samples. They show a significant and consistent bias towards the primary ALS Au results (+16% on average) while the copper data compares well (MMTS, 2024).

For 2023 and 2024, 145 pulps from 8 drillholes, prepared and analyzed by BV as the primary lab, were selected and sent to ALS in North Vancouver for check-assay purposes. The correlation between primary and secondary gold results is very good (Figure 8-10), as is the grade range between 0.05 g/t and 3 g/t. One single low-grade outlier remains unexplained.

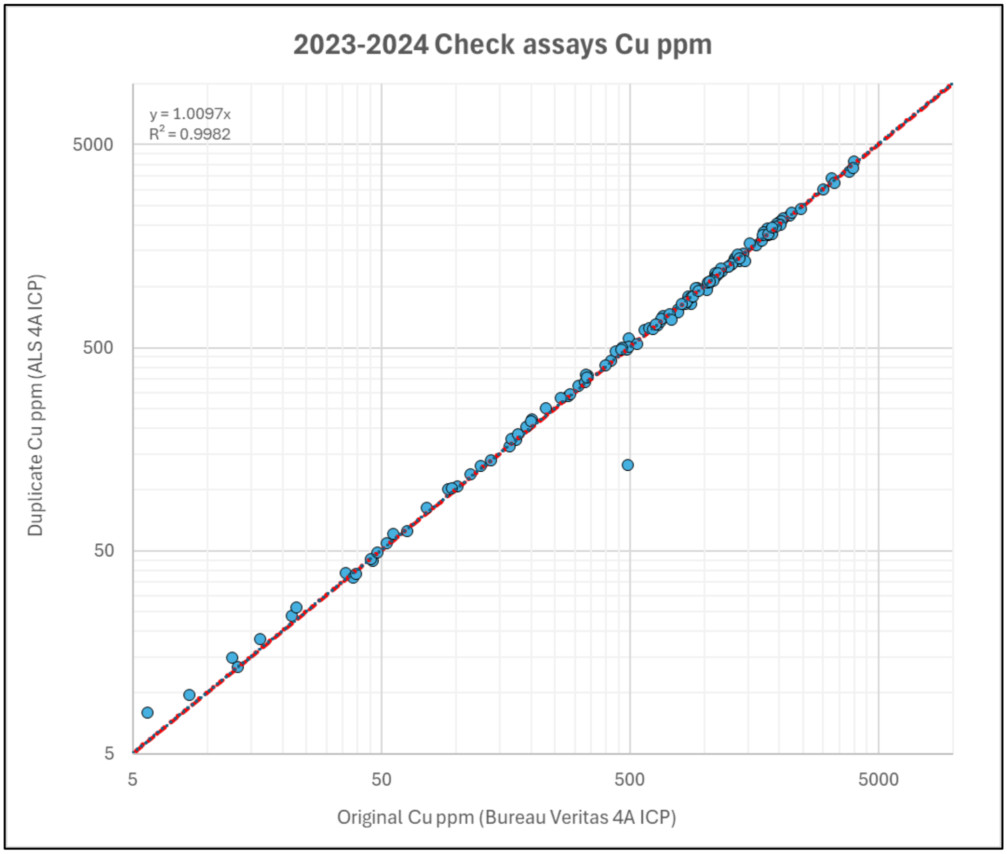
Figure 8-10: Whistler Deposit Check Assays Scatter Plot, Gold



Source: MMTS, 2026

Equally good performance is demonstrated for copper in check assays shown in Figure 8-11. The R^2 is 1 and no bias can be noted across the wide and representative grade range of 5 to 5,000 ppm. One single outlier probably represents a sample or sample number mix-up, as several other metals across the reported ICP spectrum do not match the primary results.

Figure 8-11: Whistler Deposit Check Assays Scatter Plot, Copper



Source: MMTS, 2026

MMTS finds the QA/QC performance of samples taken and analysed at the Whistler deposit acceptable.

8.3.2 QA/QC Raintree West Deposit

The following paragraphs about QA/QC performances and interpretation for the Raintree West deposit largely match the same sections of the 2024 report (MMTS, 2024), except for minor edits after extended data validations and updated graphs.

8.3.2.1 Raintree West Blanks

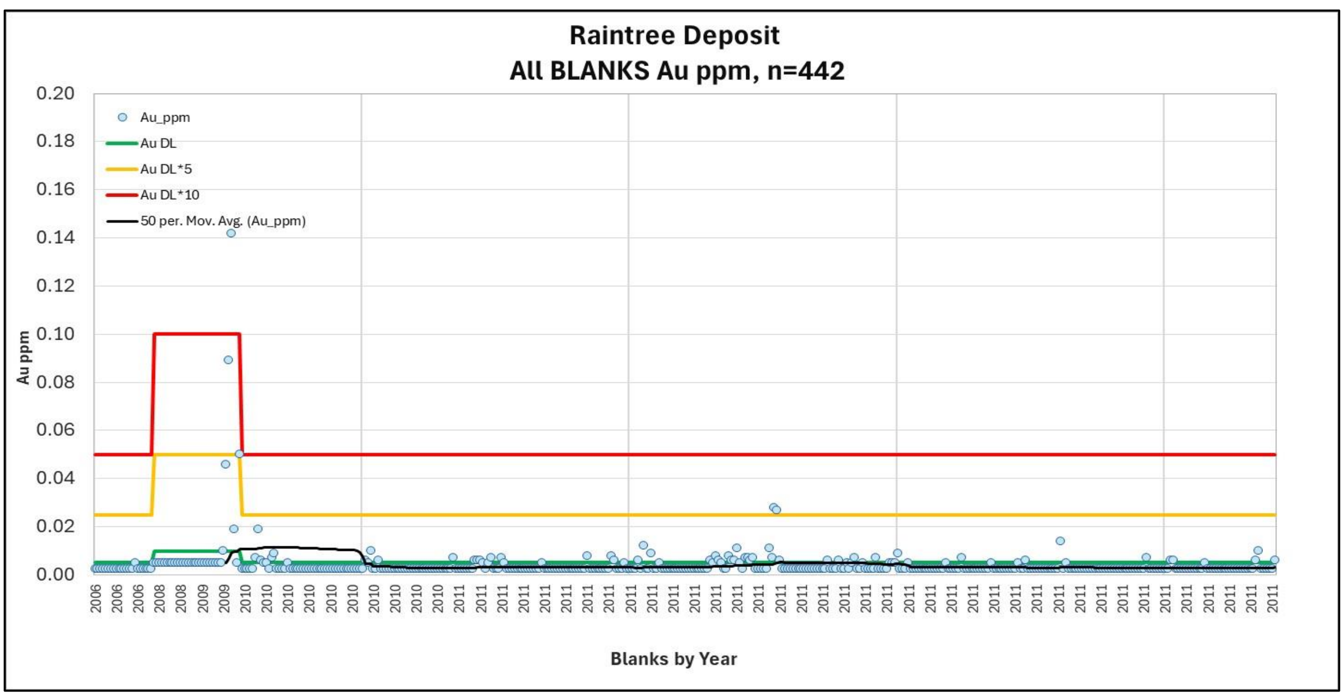
The summary of gold assays of blanks in the Raintree West sample stream is presented in Table 8-9 and shows acceptable results with only 1.1% of samples exceeding the 5*DL warning threshold, and two failures passing the 10*DL level (0.4% of total).

Table 8-9: Summary of Gold Assays of Blanks, Raintree West Deposit

Blank	Year used	Count	> 5*DL warning	% > 5*DL	> 10*DL fail	% > 10*DL
OPPBLK-1	2006	22	0	0.0%	0	0.0%
BLANK	2009	15	2	13.3%	1	6.7%
BLANK_WHISTLER	2008	18	0	0.0%	0	0.0%
BLANK_SS	2010-2011	387	2	0.5%	1	0.0%
Total	**2006-2011**	**442**	**4**	**0.9%**	**1**	**0.2%**

The sequential plot of gold assays of blanks is shown in Figure 8-12 and shows acceptable results indicating cross-sample contamination is not likely to be a problem in the Raintree assay stream. The >0.1 g/t outlier of 2009 in certificate 28678 (Alaska Assays) has been reviewed and preceding sample was found to not have sufficiently high gold concentrations to reasonably explain the results as contamination.

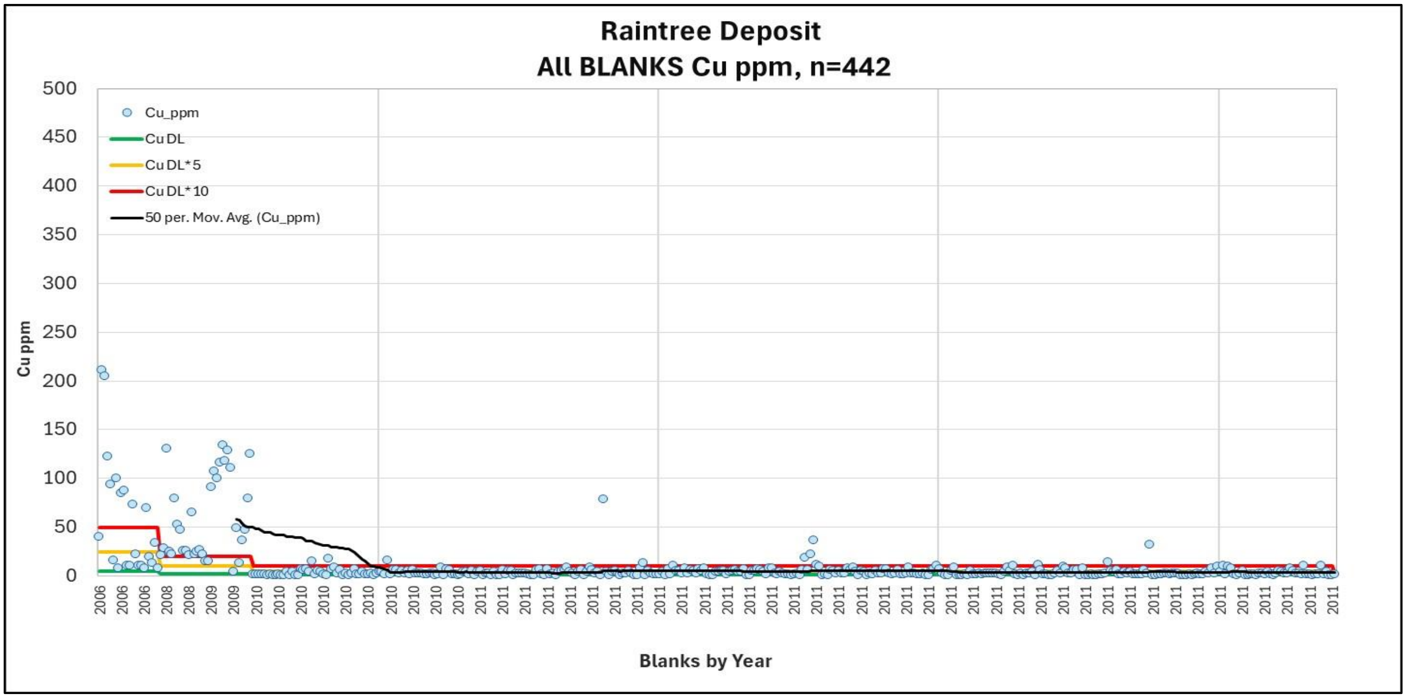
Figure 8-12: Sequential Plot of Gold Assays of Blanks, Raintree West Deposit



Source: MMTS, 2026.

The sequential plot of copper assays blanks is given in Figure 8-13 and, like the Whistler blanks plot Figure 8-5, shows higher assay results in 2006-2009 due to naturally elevated copper in the blank material, as discussed previously. The assays in 2010 and 2011 have only one blank exceeding 50 ppm and are predominantly at 10 ppm and below, demonstrating no evidence of significant contamination in most of the sample stream in Raintree West.

Figure 8-13: Sequential Plot of Copper Assays of Blanks, Raintree West Deposit



Source: MMTS, 2026.

8.3.2.2 Raintree West CRMs

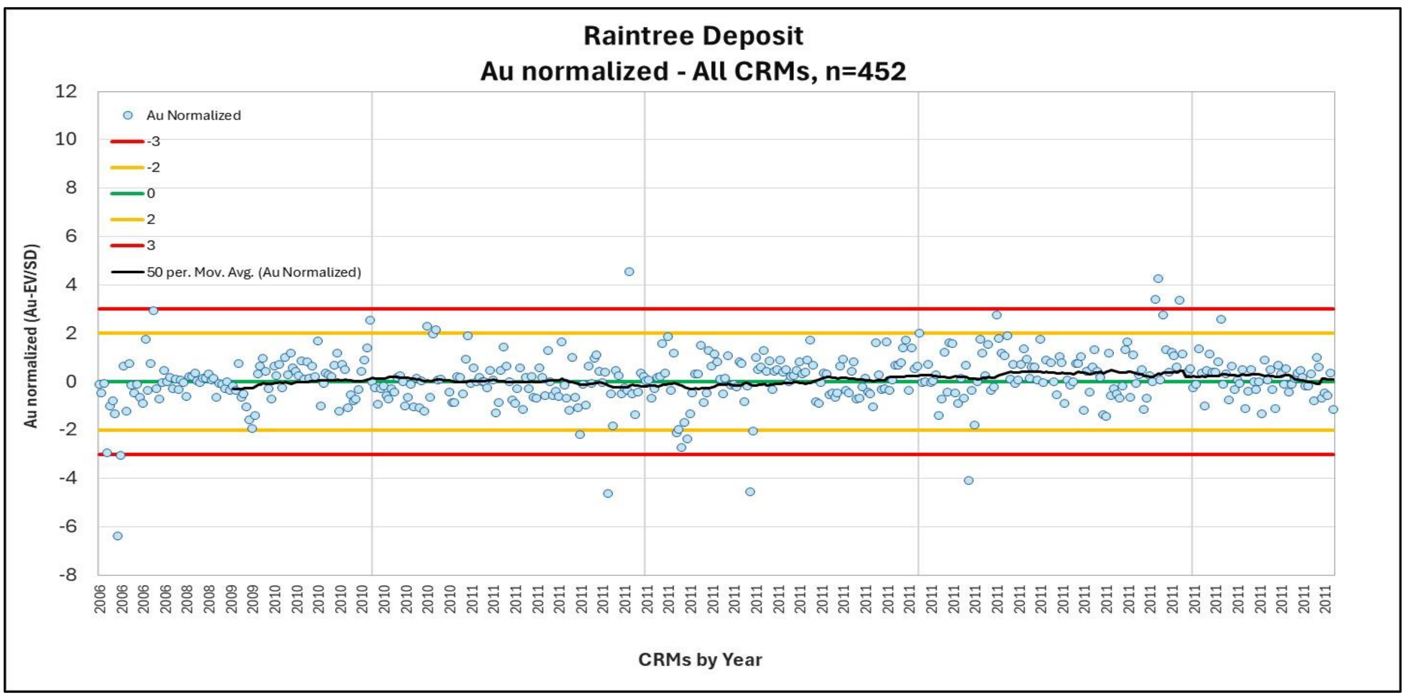
The summary of CRM gold analyses for samples included in drilling in the Raintree West deposit is given in Table 8-10. It is seen that the overall failure rate is 2.9% and there is a marginal overall negative bias of -0.5%. By far the most used CRMs are OREAS 50c and OREAS 52c, both of which performed very well except for a two-week period in August-September of 2011 where ALS Chemex missed on 4 Au CRMs over 4 certificates. The samples in these reports should have been rerun.

Table 8-10: Raintree West Deposit CRM Summary, Gold

CRM	Year used	Count	EV Au (g/t)	AVG Au (g/t)	% Error	Low Fail	High Fail	% Fail
OREAS-52Pb	2010	13	0.307	0.324	5.1%	0	0	0.0%
OREAS-52c	2010-2011	114	0.346	0.343	-0.9%	2	0	1.8%
WP-CO1	2006-2009	21	0.481	0.478	-0.6%	0	0	0.0%
OREAS-53Pb	2010	34	0.623	0.626	0.4%	0	0	0.0%
OREAS-50c	2010-2011	181	0.836	0.835	-0.1%	4	4	4.4%
WP-MG1	2006-2009	22	1.715	1.623	-5.7%	1	0	4.5%
OREAS-54Pa	2010-2011	51	2.9	2.852	-1.7%	1	0	2.0%
WP-HG1	2006-2009	16	4.693	4.659	-0.7%	1	0	6.3%
Total	2006-2011	452				9	4	2.9%

The normalized process control chart of all gold assays of CRMs in Raintree West drilling is presented in Figure 8-14 and shows the very good overall results. Three negative failures are removed from the graphed population as they are likely sample or sample number mix-ups and not representative of the respective labs accuracy performance.

Figure 8-14: Raintree West Deposit Normalized Process Control Chart, Gold



Source: MMTS, 2026.

The results of the 471 copper analyses of CRMs in Raintree West drilling are presented in Table 8-11 and show an overall failure rate of 11.3%, which is significant. The failures are seen to concentrate in three CRMs, OREAS-52c, OREAS-50c and OREAS-54Pa, also the CRMs with the most entries. The overall percent error is slightly negative at -0.7%.

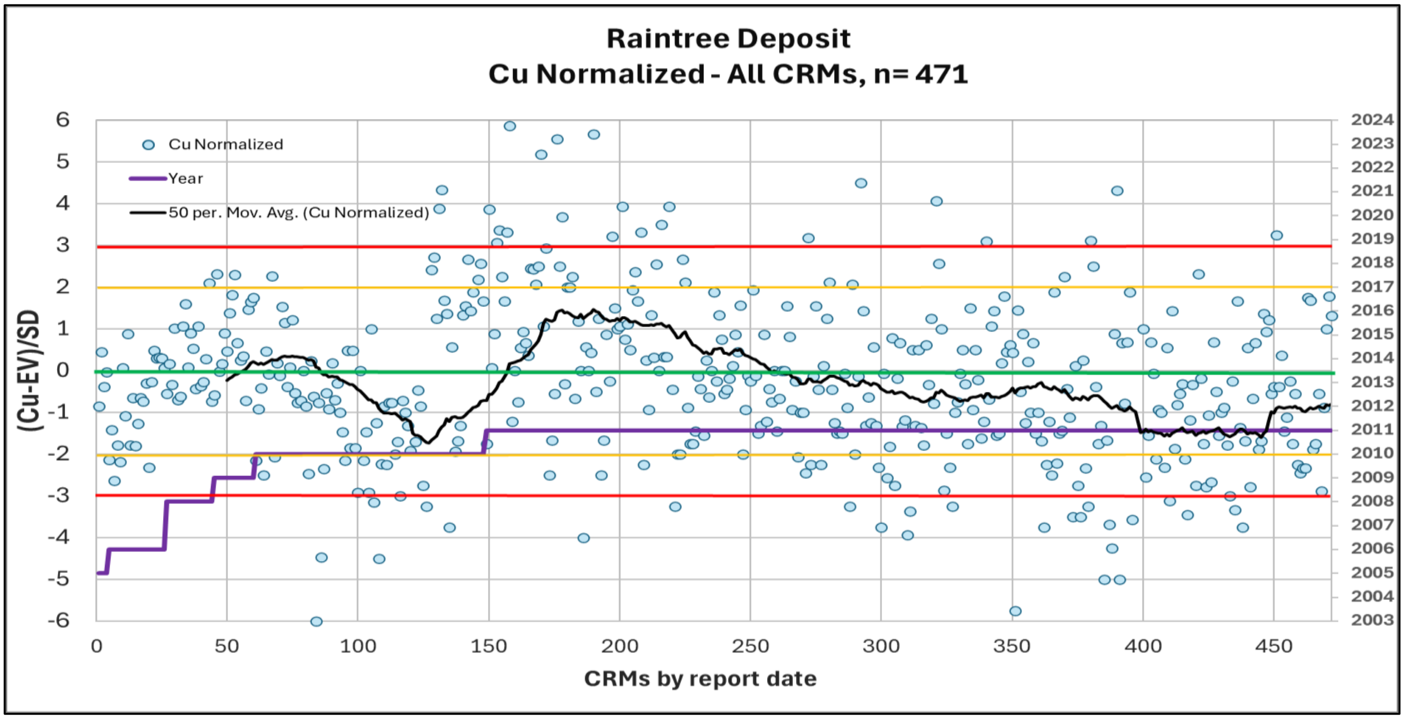
Table 8-11: Raintree West Deposit CRM Summary, Copper

CRM	Year used	Count	EV Cu %	AVG Cu %	% Error	Low Fail	High Fail	% Fail
WP-MG1	2006-2009	22	0.259	0.261	0.7%	0	0	0.0%
WP-CO1	2006-2009	21	0.280	0.277	-1.1%	0	0	0.0%
OREAS-52Pb	2010	13	0.334	0.335	0.5%	0	0	0.0%
OREAS-52c	2010-2011	114	0.344	0.342	-0.5%	4	5	7.9%
OREAS-53Pb	2010	34	0.546	0.531	-2.9%	2	0	5.9%
WP-HG1	2006-2009	16	0.616	0.620	0.6%	0	0	0.0%
OREAS-50c	2010-2011	181	0.742	0.742	0.0%	8	18	14.4%
OREAS-54Pa	2010-2011	51	1.557	1.506	-2.9%	14	0	27.5%
Total	**2006-2011**	**452**	-	-	-	**28**	**23**	**11.3%**

The normalized process control chart is given in Figure 8-15 and shows some significant trends over time. Samples of the 2010 drilling, analyzed by ALS Chemex using ME-ICP61 and OG-62 for CRMs >1% Cu, start by reporting generally higher than expected in April, and over the course of four months, trend noticeably low, without a single CRM approaching or exceeding its expected value for copper in July.

After a three-month hiatus in reporting (August-October 2010), November 2010 and April 2011 ALS certificates again report a significant high bias before trending lower for the rest of the year, with the 50-sample moving average line illustrating a consistent dip below the expected values by the end of May 2011. In this context it is noted that OREAS 54Pa as the CRM with the highest Cu grade at 1.55% EV consistently underperforms, including an approximate 27% failure rate (see Figure 8-17). Two consecutive OREAS 50c inserted into the sample stream of WH11-042 reported especially poorly (below the graphed z-score minimum of -6) and should have triggered a re-assay request at ALS.

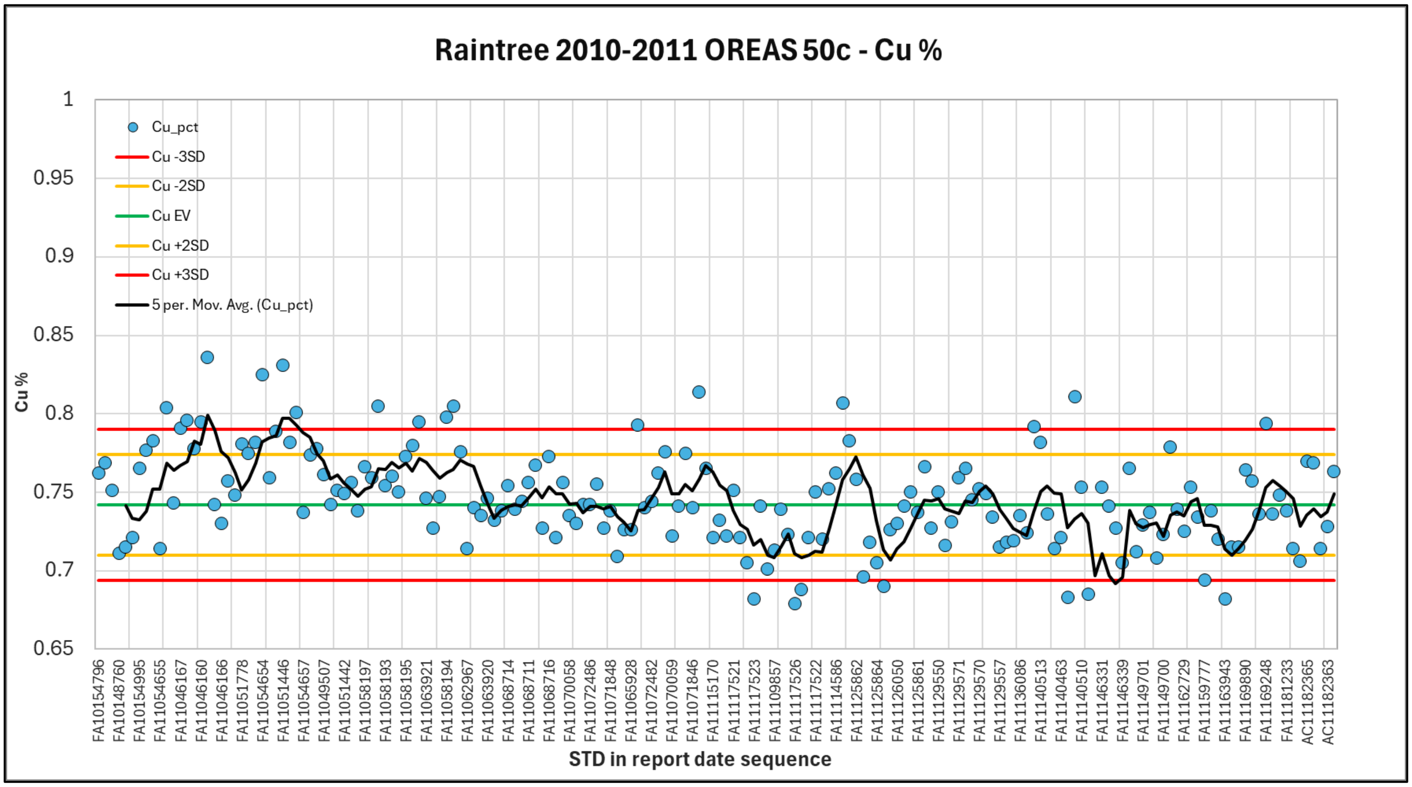
Figure 8-15: Raintree West Deposit Normalized Process Control Chart, Copper



Source: MMTS, 2026.

Results for CRM OREAS-50c, with the most samples at 183 and very high failure rate of >14%, are given in Figure 8-16. The graph shows that despite an initial high bias in the first 50 assays, the overall mean of the population as illustrated by the 5-sample moving average ends up being 0.745, very close to the expected value of 0.742, aided by a comparable distribution of high and low failures and the absence of far outliers.

Figure 8-16: Process Control Chart Raintree West OREAS-50c, Copper

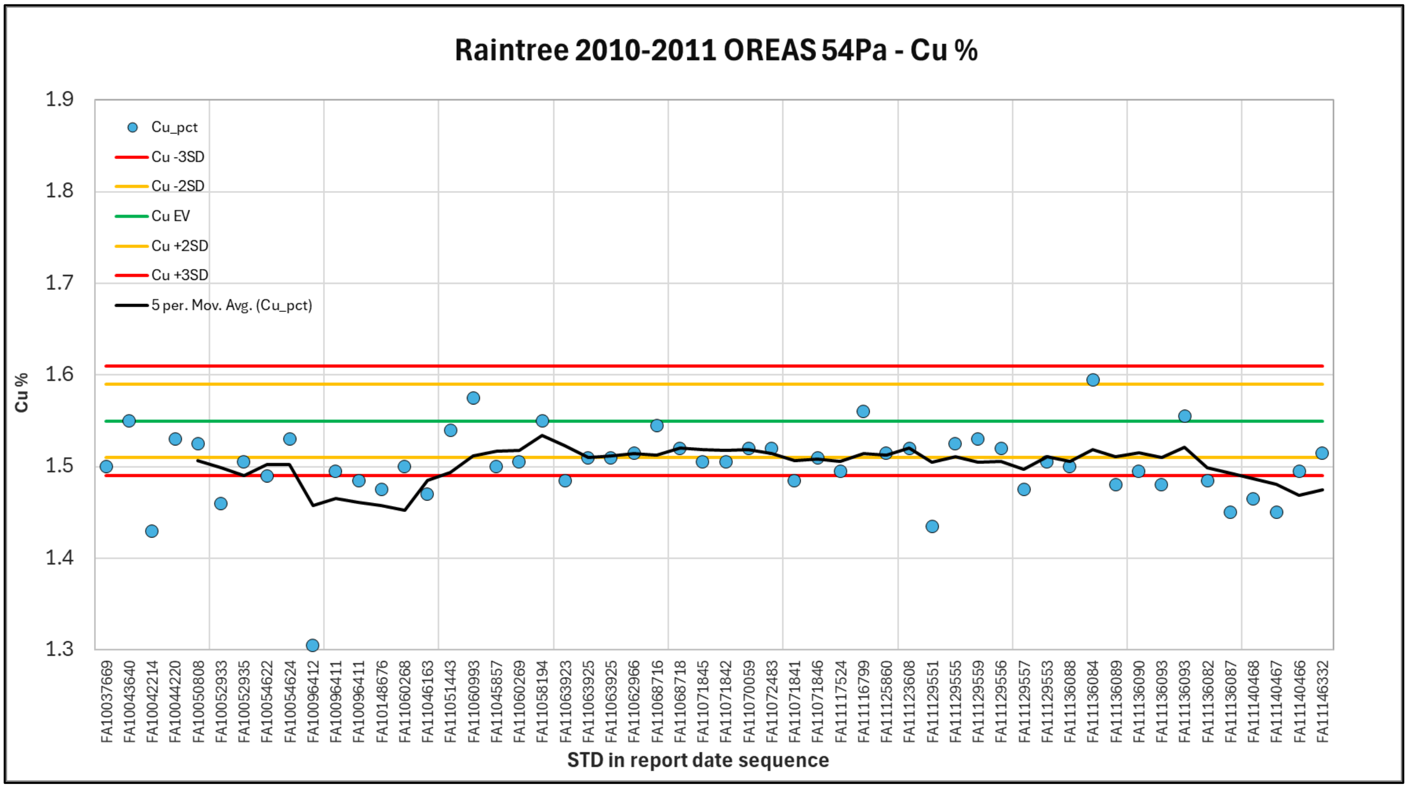


Source: MMTS, 2024

Figure 8-17 for CRM OREAS 54Pb displays a consistent low bias, with only 8 out of a total of 54 results matching or exceeding the 1.55% EV. The poor accuracy of this specific CRM is consistent across the data of all three deposits in this report. Since the EV of OREAS 54Pb is well above the expected grade range at Raintree West, with only 23 core samples in all three deposits recording 1% Cu or higher, and a theoretical low bias in the impacted core samples rendering the resource estimate to be more conservative, MMTS does not consider this a major concern.

In summary, the copper accuracy control in Raintree West shows partially significant bias that previous operators did not address through sample batch reruns at the respective lab. Equally, a significant number of failures should have triggered review and re-assay at the time. Overall, MMTS still views the results as acceptable because 3 of the 5 OREAS CRMs used in 2010-2011 performed relatively well, the failures are not strongly biased and OREAS 54Pa as the poorest control CRM in this dataset is the least applicable CRM in terms of Cu grade.

Figure 8-17: Process Control Chart Raintree West OREAS-54Pa, Copper



Source: MMTS, 2024

8.3.2.3 Raintree West Field Duplicates

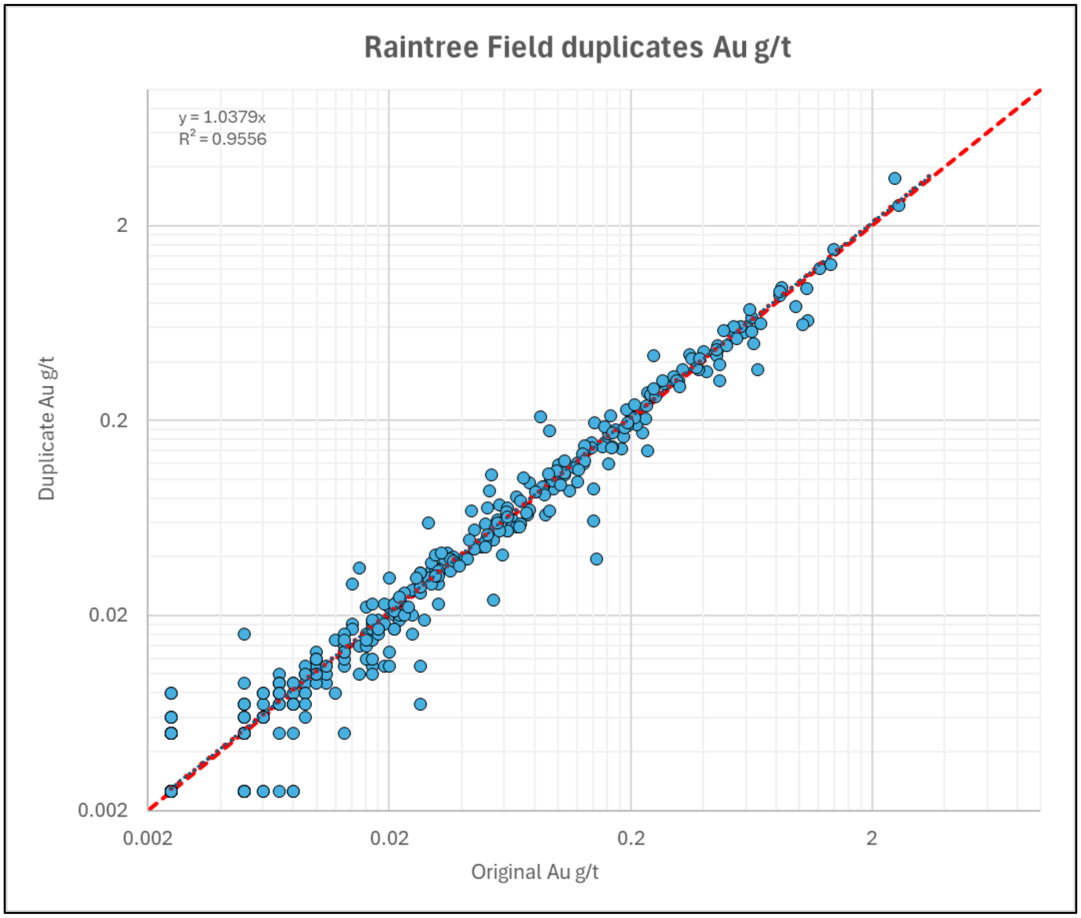
The simple statistics of the field duplicates from drilling in 2006 to 2011 at the Raintree West deposit are given in Table 8-12. Little difference is seen in the means of the gold assays, though the average ARD is slightly negative at -0.7%. The rate of pairs <10% HARD is a respectable 72%. The copper assays show no significant bias at comparatively low Cu grades in the selected duplicate pairs. Both sets of pairs meet the expectation for the HARD statistic, with 83% <10% HARD for Cu aided by the low overall grade.

Table 8-12: Raintree West Field Duplicates - Simple Statistics

DUP count	Element	Units	Average			Count >10% HARD	% <10% HARD
			Primary	Duplicate	ARD % Avg.		
381	Gold	g/t	0.123	0.124	-0.66%	107	71.9%
381	Copper	ppm	245	236	0.26%	64	83.2%

The scatter plot of duplicate pairs of gold assays is given in Figure 8-18, which demonstrates good reproductivity throughout the grade range of DL to +2 g/t Au. Paired with the HARD statistics above, it proves the gold mineralization to not be highly heterogenous or 'nuggety'. The y=1.04 is strongly influenced by the highest-grade pair, which happens to be moderately duplicate-positive.

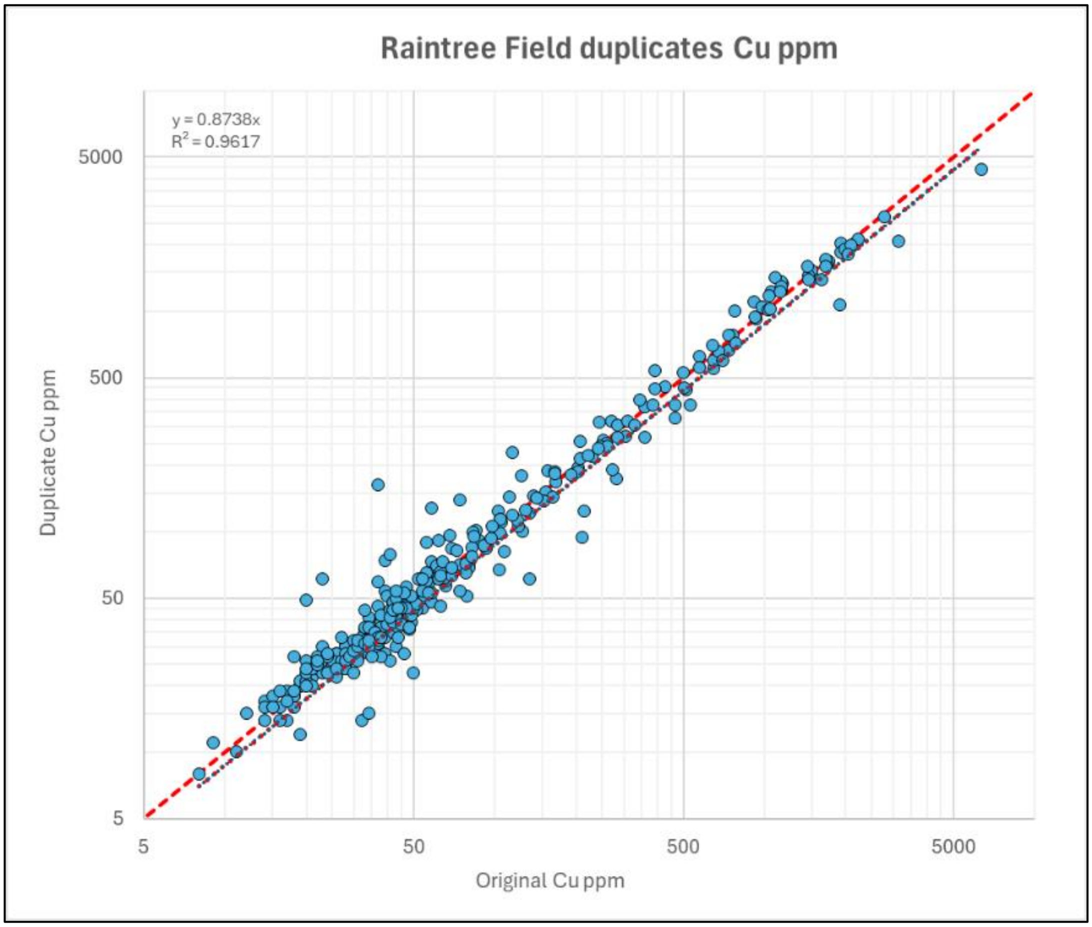
Figure 8-18: Raintree West Deposit Field Duplicate Scatter Plot, Gold



Source: MMTS, 2026

The scatter plot of copper assays of field duplicates is given in Figure 8-19. The graphed assay data confirms the predominantly low-grade nature of the selected duplicates at Raintree West as most of the results plot between 20 ppm and 60 ppm, with a second population forming between 500 ppm and 800 ppm. The correlation is very good at R2 0.96, and the data appears unbiased as the low Y is solely due to the highest-grade pair.

Figure 8-19: Raintree West Deposit Field Duplicate Scatter Plot, Copper



Source: MMTS, 2026

Overall, analysis of field duplicate samples in Raintree West do not show evidence of selection bias at the core sampling level. Only weak to moderate heterogeneity of gold mineralization is interpreted.

Coarse reject and pulp duplicates data was also briefly reviewed. The precision for copper is very good in each dataset (434 coarse duplicates, 195 pulp duplicate results), regardless of laboratory or year.

For gold, however, approximately 10 of the 434 coarse duplicates generate significant scatter at meaningful Au grades while the full dataset is still correlating well with the results of the original samples. These poor results are predominantly being reported by AAL in 2008-2009 and may indicate underlying issues at sample splitting or handling.

A total of 255 Au pulp duplicate assays are available, 48 of which exceed the >10% HARD threshold (18.8%) which is an acceptable result because all but two of them grade <0.08 g/t Au.

8.3.3 QA/QC Island Mountain Deposit

8.3.3.1 Island Mountain Blanks

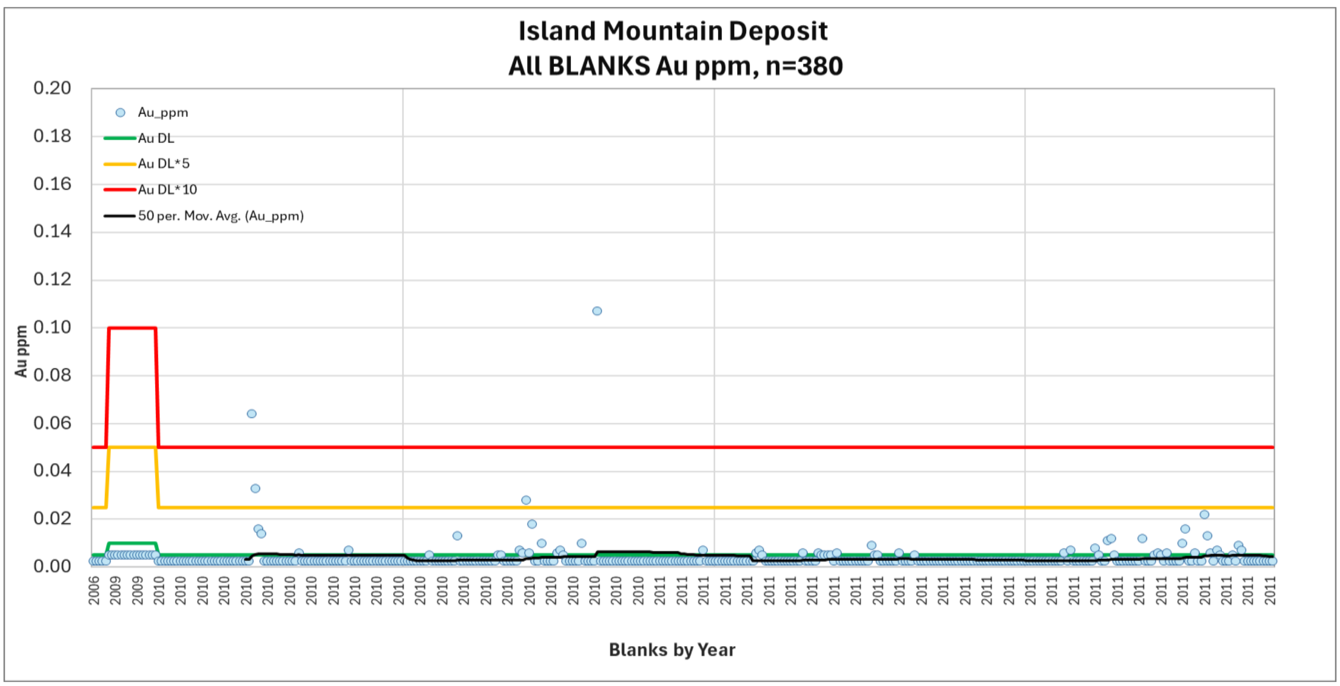
The summary of gold assays of blanks in the Island Mountain sample stream is given in Table 8-13 and shows an overall failure rate of 0.5%. These results are acceptable with little evidence of contamination.

Table 8-13: Summary of Gold Assays of Blanks, Island Mountain Deposit

Blank	Year used	Count	> 5*DL warning	% > 5*DL	> 10*DL fail	% > 10*DL
BLANK	2009	14	0	0.0%	0	0.0%
BLANK_SS	2010-2011	359	4	1.1%	2	0.6%
BLANK_WHISTLER	2009	2	0	0.0%	0	0.0%
OPPBLK-1	2006	5	0	0.0%	0	0.0%
Total	**2006-2011**	**380**	**4**	**1.1%**	**2**	**0.5%**

The sequential plot of gold assays of blank material is given in Figure 8-20.

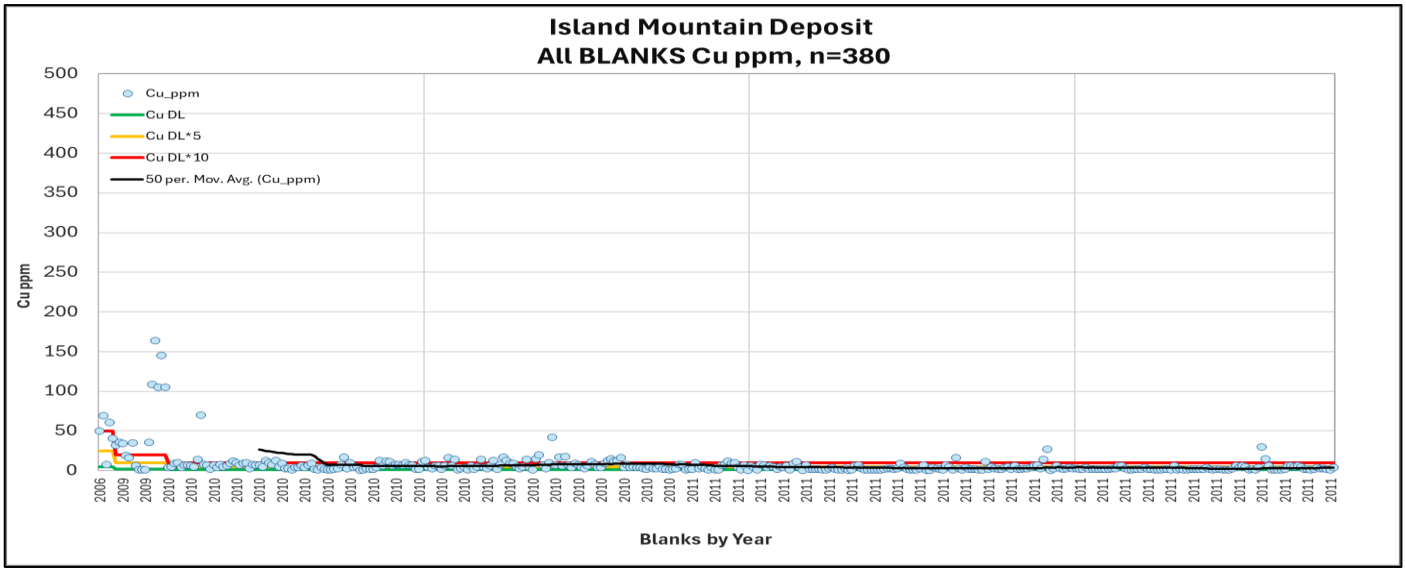
Figure 8-20: Sequential Plot of Gold Assays of Blanks, Island Mountain Deposit



Source: MMTS, 2026

The sequential plot of copper assays of samples of blank material is given in Figure 8-21 and shows 5 results >100 ppm which is inconsequential and probably caused by natural background concentrations. Preceding core samples are not sufficiently mineralized to be the cause of those elevated grades.

Figure 8-21: Sequential Plot of Copper Assays of Blanks, Island Mountain Deposit



Source: MMTS, 2026

8.3.3.2 Island Mountain CRMs

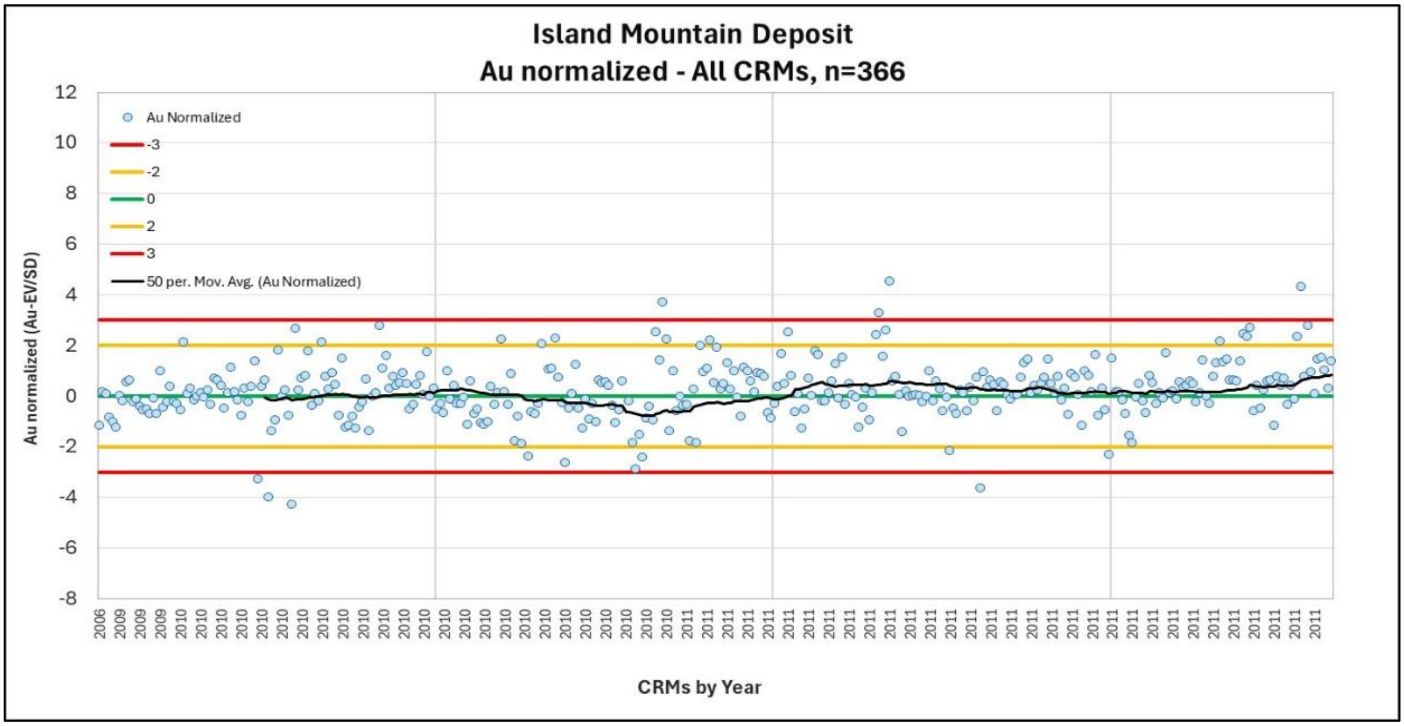
The summary of results of gold assays for CRM samples included in drilling in Island Mountain are presented in Table 8-14. The overall percentage of failures is an acceptable 2.7% and the data is overall unbiased.

Table 8-14: Island Mountain Deposit CRM Summary, Gold

CRM	Year used	Count	EV Cu %	AVG Cu %	% Error	Low Fail	High Fail	% Fail
OREAS-52Pb	2010	21	0.307	0.327	6.2%	0	1	4.8%
OREAS-52c	2010-2011	137	0.346	0.348	0.6%	1	0	0.7%
WP-CO1	2006-2009	10	0.481	0.473	-1.7%	0	0	0.0%
OREAS-53Pb	2010	36	0.623	0.619	-0.6%	3	0	8.3%
OREAS-50c	2010-2011	113	0.836	0.840	0.5%	2	3	4.4%
WP-MG1	2006-2009	7	1.715	1.679	-2.1%	0	0	0.0%
OREAS-54Pa	2010	35	2.901	2.836	-2.3%	0	0	0.0%
WP-HG1	2006-2009	7	4.693	4.666	-0.6%	0	0	0.0%
Total	**2006-2011**	**366**	**-**	**-**	**-**	**6**	**4**	**2.7%**

The normalized process control chart of gold assays in the Island Mountain drilling is presented in Figure 8-22 showing the moving average close to the expected value indicated by the zero z-score and 4 of the 6 low failures very close to the -3SD failure threshold. Two very strong low failures are not shown, one of which is likely caused by an Au data shift in the ALS Chemex certificate (sample 813068 in certificate FA10123328). Both AAL and ALS measured the Au concentrations of the 8 utilized standards quite accurately.

Figure 8-22: Island Mountain Deposit Normalized Process Control Chart, Gold



Source: MMTS, 2026

The summary of results of 366 copper assays of CRMs in Island Mountain is given in Table 8-15 and shows a higher-than-expected overall failure rate of 13.9% with overall percent error of -2.2% indicating a moderate negative bias to the copper assays of the CRMs. Several 2010 Cu results are illogically low and excluded from the normalized graph in Figure 8-23.

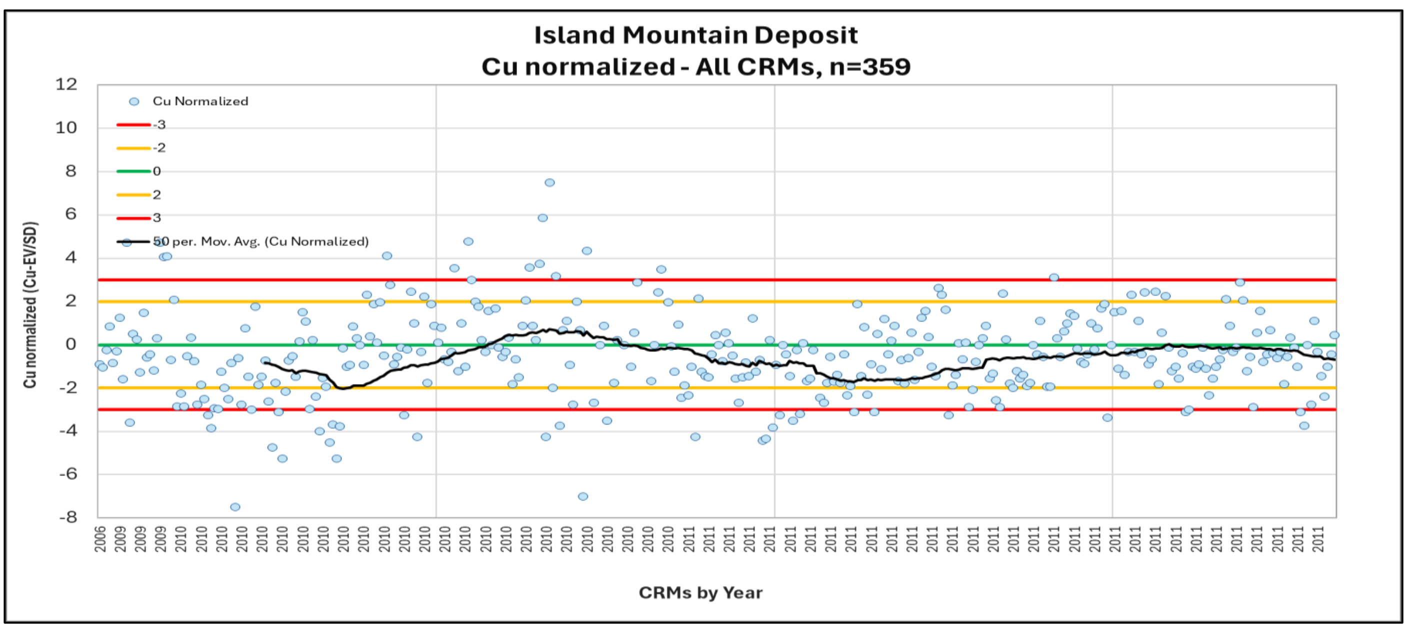
As was noted under 11.3.2.2, CRM OREAS 54Pa again performed very poorly with a 46% failure rate and a general under-performance approaching 7% (ALS Chemex Cu-OG62 method).

Table 8-15: Island Mountain Deposit CRM Summary, Copper

CRM	Year used	Count	EV Cu %	AVG Cu %	% Error	Low Fail	High Fail	% Fail
WP-MG1	2006-2009	7	0.259	0.265	2.0%	0	2	28.6%
WP-CO1	2006-2009	10	0.280	0.278	-1.0%	1	1	20.0%
OREAS-52Pb	2010	21	0.334	0.320	-4.4%	2	3	23.8%
OREAS-52c	2010-2011	137	0.344	0.335	-2.8%	7	4	8.0%
OREAS-53Pb	2010	36	0.546	0.530	-3.1%	3	0	8.3%
WP-HG1	2006-2009	7	0.616	0.628	1.9%	0	1	14.3%
OREAS-50c	2010-2011	113	0.742	0.722	-2.7%	7	4	9.7%
OREAS-54Pa	2010	35	1.557	1.444	-7.3%	16	0	45.7%
Total	**2006-2011**	**366**	**-**	**-**	**-**	**36**	**15**	**13.9%**

The normalized process control chart is presented in Figure 8-23 and shows that that all through 2010 but particularly in June and July 2010, over several certificates, ALS Chemex in Fairbanks, Alaska, was not able to accurately measure copper concentration in any of the blindly inserted CRMs, resulting in a significant low bias early in the campaign before reversing into a period of inconsistent copper results with a weak high bias and several high outliers for the remainder of 2010, particularly October. The 2011 drill campaign results are moderately biased low with several failures between July and September before becoming more consistent and within the ±3 SD line.

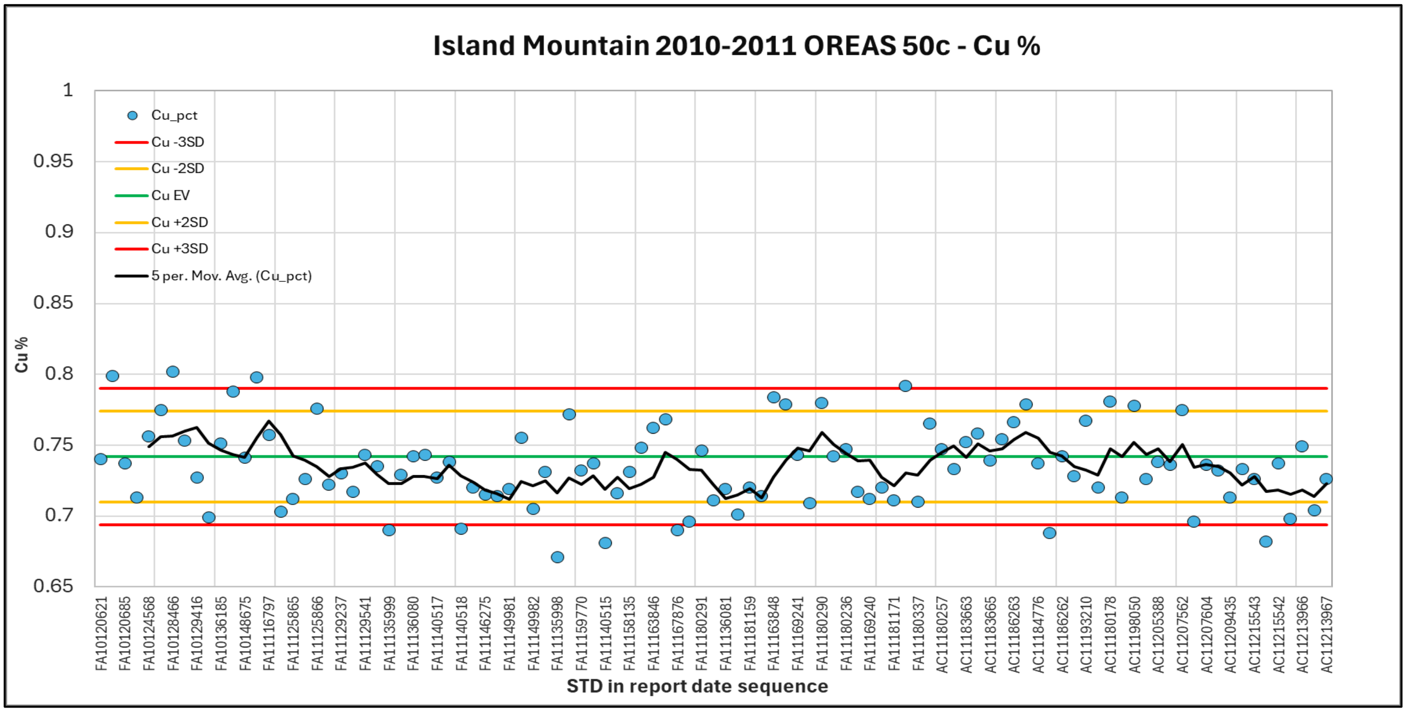
Figure 8-23: Island Mountain Deposit Normalized Process Control Chart



Source: MMTS, 2026

The process control chart for CRM OREAS-50c is given in Figure 8-24 and shows that despite the high failure rate of 10.7% at high volume the results are seen to indicate little bias with the mean close to the expected value of 0.742% Cu. Also, the 11 noted failures are generally very close to the +/-3SD failure threshold.

Figure 8-24: Process Control Chart Island Mountain CRM OREAS-50c, Copper



Source: MMTS, 2024

For drilling in Island Mountain, analysis of the CRMs shows fluctuating results for copper and ALS Chemex should have been requested to investigate their Cu-OG62 Cu grade method performance despite OREAS 54Pa's too-high Cu grade being unsuitable for accuracy control at Island Mountain. The QP sees no indication of bias that would be material to the resource estimate.

8.3.3.3 Island Mountain Duplicates

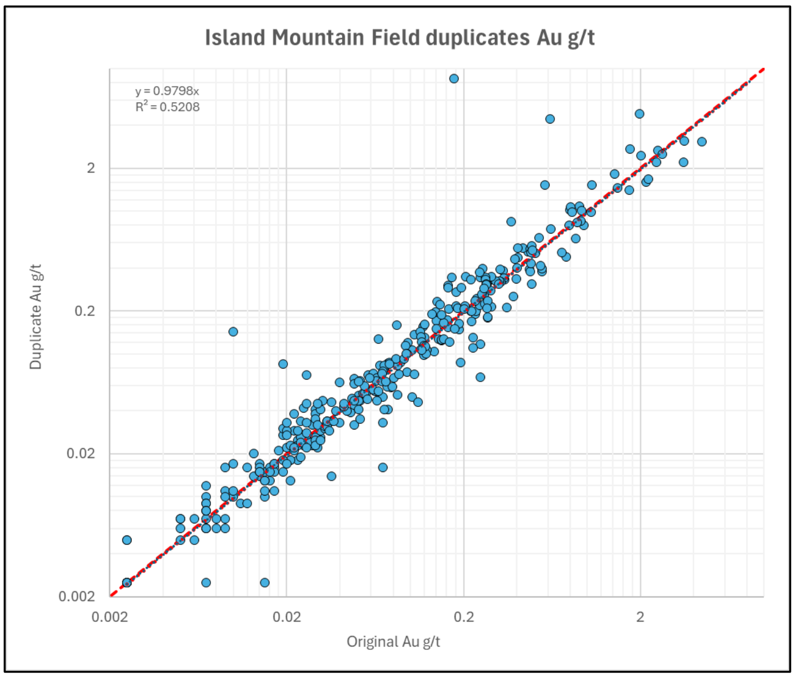
The simple statistics of the gold and copper assays of the field duplicates from drilling in 2010 and 2011 at Island Mountain is given in Table 8-16. The means of the gold assays of the duplicate pairs show a 0.74% ARD% average difference with the originals higher, while the originals of the copper assay pairs are slightly lower than the duplicates, albeit at low grades overall. The ARD% statistic expectation of 70% is more than met for copper and only 59% for gold, indicating higher heterogeneity than Whistler and Raintree West. This is reflected in the significant but generally unbiased scatter around the 1-1 line in Figure 8-25.

Table 8-16: Island Mountain Field Duplicate Simple Statistics

DUP count	Element	Units	Average			Count >10% HARD	% <10% HARD
			Primary	Duplicate	ARD % Avg.		
347	Gold	g/t	0.243	0.243	-2.01%	142	59.1%
347	Copper	ppm	410	402	-1.20%	44	87.3%

The scatter plot of field duplicate assays for gold is given in Figure 8-25 and shows the considerable scatter with low coefficient of correlation at R2 0.52 while the slope at y=0.98 indicates low bias and the overall grade representation is good. However, the data pairs obviously correlate very well, which can be illustrated by the fact that removing the two strongest (duplicate-positive) outliers would improve the R2 to 0.86. The average HARD for Au is 12%.

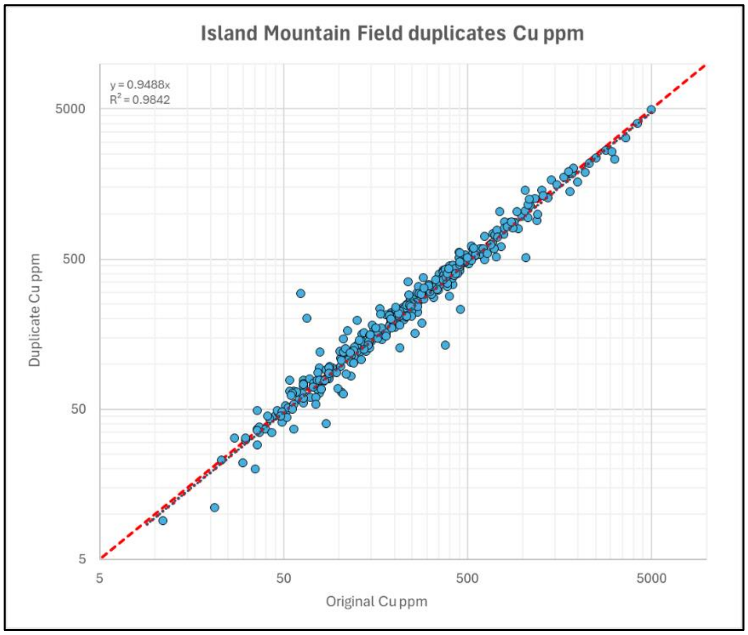
Figure 8-25: Island Mountain Deposit Field Duplicate Scatter Plot, Gold



Source: MMTS, 2026

The scatter plot of copper field duplicate assays is given in Figure 8-26 and shows the excellent correlation of unbiased pairs. The grade distribution in the data appears very consistent between 50 ppm and 500 ppm which is suitable for the deposit. Only 13% of all data fall above the 10% HARD line which is very good and indicates more homogenous, possibly disseminated Cu mineralization at Island Mountain compared to Whistler and Raintree West.

Figure 8-26: Island Mountain Deposit Field Duplicate Scatter Plot, Copper



Source: MMTS, 2026

Overall, analysis of field duplicate samples in Island Mountain do not show evidence of selection bias at the core sampling level. The slightly higher heterogeneity of gold mineralization compared to Whistler and Raintree West is probably a function of the selection of higher-grade field duplicate intervals.

8.4 Sample Preparation, Analyses and Security Conclusions and Recommendations

The QP concludes that the sample preparation, analysis, and security are of sufficient quantity and quality for resource estimation. The author further recommends that:

- For completeness, QA/QC data for silver blanks and duplicates should be collected from the historical database for analysis in future studies that include silver in the resource estimate. None of the CRMs used before 2023 were certified for silver. The lack of silver QA/QC samples is not of material significance currently because silver is a minor contributor to the resource estimate.
- The locally sourced material for blanks used prior to 2009 gives inconclusive results for assessing contamination as it appears to contain trace mineralization. This is particularly pronounced in the Whistler Deposit where most of the sampling was in 2008 and earlier. Future drilling campaigns should use coarse crush limestone as was done in 2023-2024 or alternatively a commercially prepared blank material.
- Three CRMs with varying Au-Cu-Ag concentrations that approach the expected deposit grades be used to better represent low-grade, medium-grade, and high-grade mineralization. The CRMs are to be inserted blindly and at roughly equal ratios.

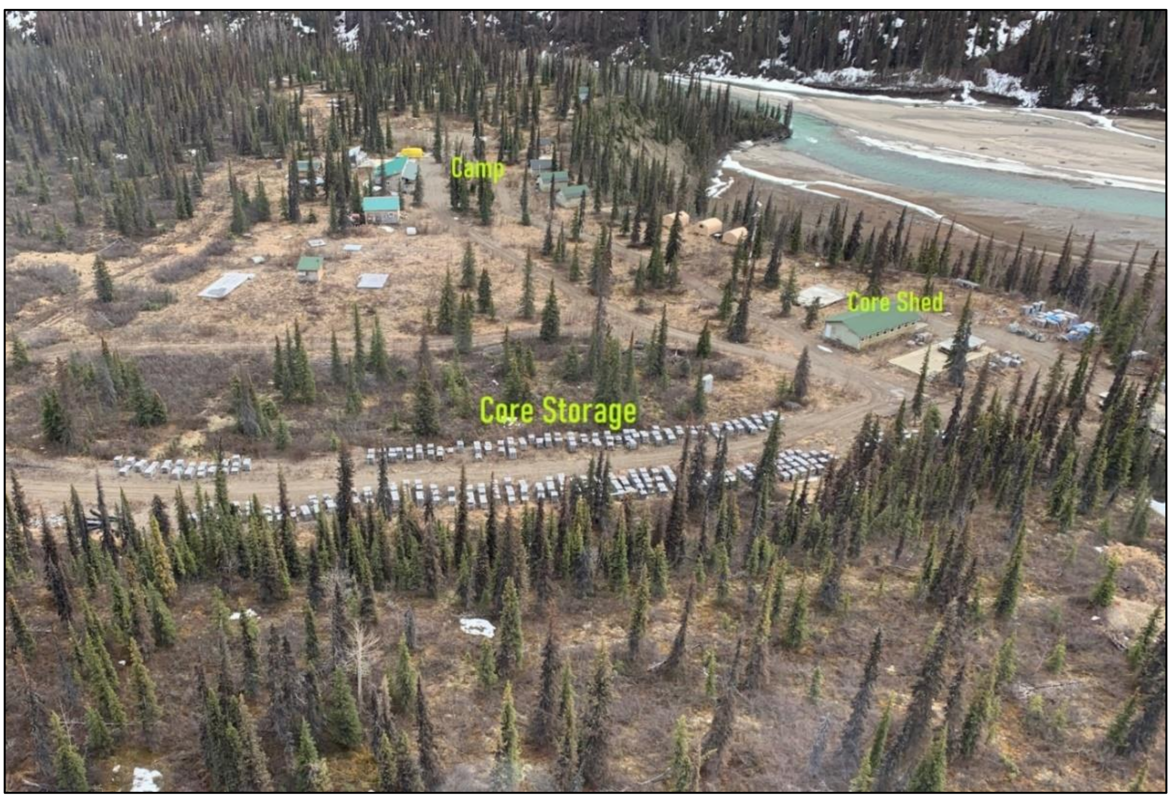
9 DATA VERIFICATION

9.1 Geological and Mineral Resource Data Verification

9.1.1 Site Visit

Site visits have been conducted on September 14, 2022, by Sue Bird, P.Eng. of MMTS and again on August 6, 2024. During the site visits collar locations at Whistler and Raintree West were validated. The core storage at both the Whiskey Bravo camp site was visited. The core from each deposit was examined for mineralization with 4 samples for re-assay obtained in 2022 and an additional 5 samples obtained in 2024. The buildings at the camp have been re-vamped since 2022 and have been used for the 2023 and 2024 drill programs. An aerial view of the camp is given in Figure 9-1. The core storage area is also illustrated in Figure 9-2. It should be noted that much of the Whistler core is also stored at a yard in Sterling, Alaska about 140 miles south of Anchorage. A core logging shed is shown in Figure 9-3.

Figure 9-1: Aerial view of Whistler Camp



Source: MMTS, 2021

Figure 9-2: Drillcore Boxes in Storage Area

Source: MMTS, 2021

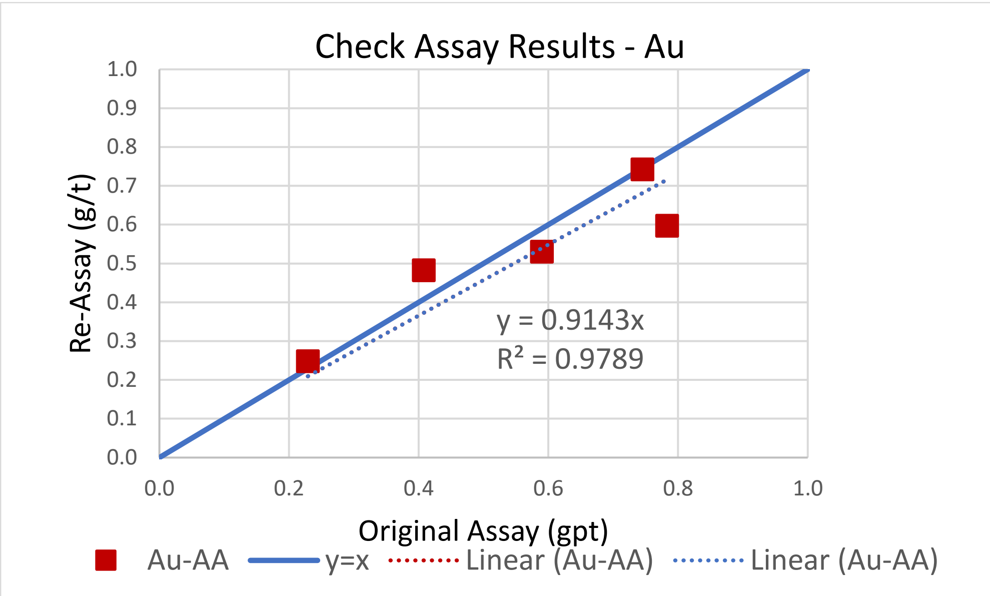
Figure 9-3: Core Logging Shed

Source: MMTS, 2026

9.1.2 Re-Assay Results

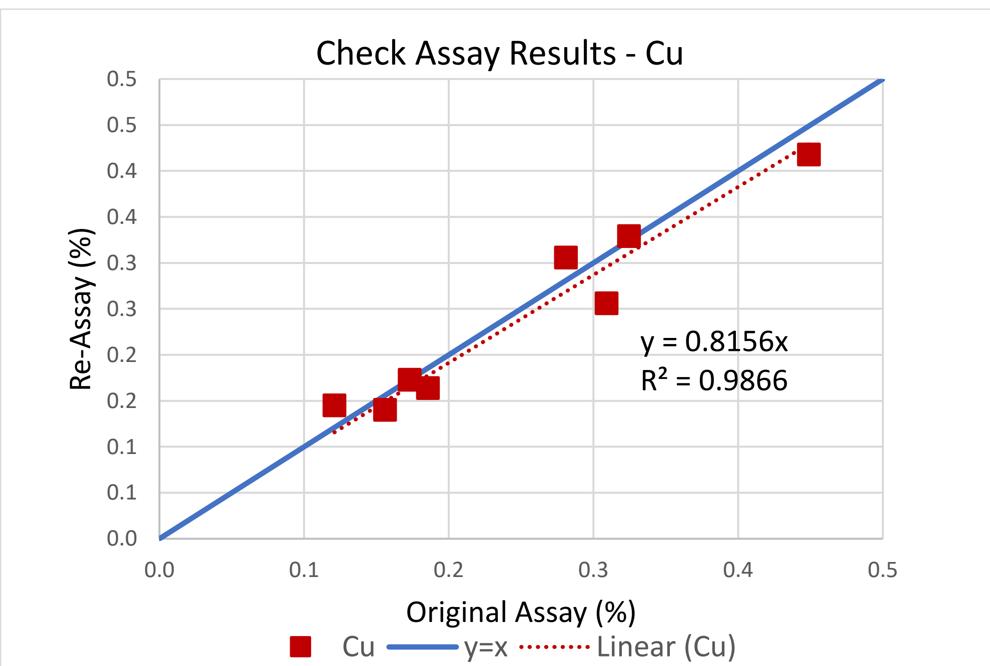
Four intervals of half core were obtained for check assaying, in 2022 and 2024 respectively, for a total of eight samples. The samples were chosen to be of mineralized intervals, with Au grades ranging from 0.223 to 7.160 g/t and Cu grades between 0.121% and 0.449%. Results of this limited check assay program are shown in Figure 9-4 and Figure 9-5 for gold and copper, respectively. Silver had only two samples above detection, both of which had a re-assay value higher than the original. The results indicate slightly lower grades for the higher values of Au. However, it was also noted that the OREAS standards also had lower values than the certified grades, particularly for Au. The results for both Au and Cu are reasonable when considering the outdoor storage area, the general scatter expected for Au and the low results of the CRM material.

Figure 9-4: Check Assay Results from 2022 and 2024 Site Visits – Au



Source: MMTS, 2026

Figure 9-5: Check Assay Results from 2022 and 2024 Site Visits – Cu



Source: MMTS, 2026

9.1.3 Data Audit

The most current version of the assay database was received on May 1, 2025. The database contains 30,128 sampled intervals, excluding QA/QC samples, across all areas of the Project. No errors were noted for overlapping intervals or missing assay data.

9.1.3.1 Certificate Checks and Database Corrections

The assay database as received contained certificate numbers attached to the assay intervals, however for drillholes WH04-01 to WH04-04 two certificates from two laboratories should have been recorded. For validation purposes, MMTS compiled Au and Cu data from 49 randomly selected certificates from years 2004-2023 and proceeded by comparing the resulting 3,071 RAW data against the data in the database (this represents approximately 11% of the data, see Table 9-1).

Only two data issues were detected:

1. 2010 certificate FA10043640: 12 Au data show a small discrepancy, probably related to the proximity to the ALS Chemex Au-AA23 detection limit (samples 755056 to 755073).
2. 2023 certificate FBK23002831: 8 samples were rerun at Bureau Veritas, likely because of a moderately low Cu CRM. The rerun data should replace the original data.

The validation showed that for 2004 the original American Assay data for Au and Cu had been replaced by the re-assay data produced by ALS Chemex data of 2004 and 2005, to which the QP agreed.

Table 9-1: Certificate Check Results

Assayed Intervals	30,128
Intervals Checked	3,071
% Checked	10.2%
Errors	12
% Errors	0.4%
Lab corrections not updated in database	8
Total Findings	20

The amount of data by interval length that is supported by certificate and QA/QC data (blanks CRMs and field duplicates) is given in Table 9-2 and is reported by drilling year, not analysis as previously presented in the QA/QC section. The percentage of assayed length fully supported by certificate and QA/QC in Whistler is 94% because of a limited amount of historical Cominco drill data, in Raintree West and Island Mountain it is 100%.

Table 9-2: Summary of Data Supported by Certificate and QA/QC

Year	Whistler				Raintree West				Island Mountain			
	Assayed Length (m)	Has Certificate (m)	Has QA/QC (m)	% With Certificate and QA/QC	Assayed Length (m)	Has Certificate (m)	Has QA/QC (m)	% With Certificate and QA/QC	Assayed Length (m)	Has Certificate (m)	Has QA/QC (m)	% With Certificate and QA/QC
1986-1989	1,566	-	-	0%	-	-	-	-	-	-	-	-
2004	1,865	1,865	1,865	100%	-	-	-	-	-	-	-	-
2005	5,061	5,061	5,061	100%	208	208	208	100%	-	-	-	-
2006	696	696	696	100%	845	845	845	100%	-	-	-	-
2007	3,243	3,243	3,243	100%	-	-	-	-	-	-	-	-
2008	2,660	2,660	2,660	100%	615	615	615	100%	-	-	-	-
2009	214	214	214	100%	479	479	479	100%	387	387	387	100%
2010	4,500	4,500	4,500	100%	3,164	3,164	3,164	100%	4,956	4,956	4,956	100%
2011	-	-	-	-	13,799	13,799	13,799	100%	8,943	8,943	8,943	100%
2023	2,048	2,048	2,048	100%	-	-	-	-	-	-	-	-
2024	3,504	3,504	3,504	100%	-	-	-	-	-	-	-	-
Total	**21,852**	**20,286**	**20,286**	**92%**	**19,110**	**18,975**	**18,975**	**100%**	**14,286**	**14,286**	**14,286**	**100%**

9.1.3.2 Check Assays

Check assays or umpire assays have been completed for parts of the 2004 and 2005 drilling and assaying campaigns, with the data presented under Section 9.1.3.1. A total of 145 umpire assays for the 2023-2024 dataset have been completed, approximately 4.5% of the core samples taken in these two campaigns.

9.1.3.3 Collar Survey

In 2011, it was reported that collar locations for Island Mountain holes had been re-captured using a Trimble Geoexplorer 6000 GPS instrument (<1 m accuracy) and that the intention was to resurvey most of the holes on the property in 2012 (Roberts, 2011a). DGPS collar resurvey was completed in 2012.

9.1.4 Data Verification Conclusions and Recommendations

The independent QP responsible for Section 9 of this report, Sue Bird, P. Eng., believes the databases are sufficiently validated and verified to support their use in mineral resource estimation for each of the deposit as presented herein. It is further recommended that:

- At least 10% of collar locations in each resource area, to include drilling from all years, be surveyed with GPS equipment with <1 m accuracy. If significant deviations are determined from the recorded values, all collars would need resurvey.
- U.S. GoldMining continue to pursue matching of assay samples to certificates and collection of missing certificates for the Kennecott work completed between 2004 and 2006.
- Future drilling should include third-party check assays and the data should be appropriately maintained.

9.2 Metallurgical Data Verification

The Qualified Person has reviewed the available metallurgical testwork results and supporting technical data used in this study. The review included an evaluation of laboratory procedures, sample preparation methods, analytical results, and the consistency of metallurgical recovery results with the geological domains and mineralization characteristics of the deposit.

Based on this review, the Qualified Person considers the metallurgical data to be reasonable and suitable for the purposes of this Technical Report Summary. While additional metallurgical testwork and engineering studies may be conducted during subsequent stages of project development, the currently available data are considered adequate to support the metallurgical assumptions and technical conclusions presented in this report.

9.3 Geotechnical Data Verification

The QP person has reviewed available topography, mine schedule, mineral processing, and climate data to be reasonable and suitable for the purposes of the level of this report. While geotechnical investigations, high-definition topography, and additional processing test work and optimization of the mine plan may be conducted during subsequent stages of the project development, the current available data are considered adequate to support the geotechnical assumptions and technical conclusions presented in this report.

10 MINERAL PROCESSING AND METALLURGICAL TESTING

10.1 Introduction

The information presented in Section 10 focuses on metallurgical testwork conducted to substantiate the recovery of copper (Cu), gold (Au), and silver (Ag) from mineralized material contained within the Whistler Deposit.

The metallurgical testwork programs conducted on the Whistler Deposit over time were performed by multiple laboratories and have included comminution, flotation, cyanide leaching, and solid-liquid separation tests on mineralized materials and flotation tailings samples across multiple testwork programs. These programs were designed to quantify metallurgical performance and support the evaluation of different processing options. The samples for testing were derived from drill cores that captured spatial variability of the mineralized material available from the Whistler Deposit.

While a summary of historical metallurgical testwork completed on the Whistler Deposit is presented in Section 10.2.1, the flowsheet selection and metallurgical performance projections presented in this report have been developed primarily based on the 2024–2025 metallurgical testwork program performed by Base Metallurgical Laboratories Ltd. in Canada on behalf of the current issuer, U.S. GoldMining. All laboratories involved are independent of the issuer.

10.2 Metallurgical Testwork

Table 10-1 summarizes the current and historical metallurgical testwork programs completed to date for the Whistler Deposit, including the testing laboratory and scope of testwork performed.

Metallurgical testing has focused on the development of a suitable flotation process with tailings leaching to produce a copper concentrate with gold and silver by-product credits, as well as a doré product. Cyanide leaching of flotation tailings was investigated to increase overall gold recovery. The results of the metallurgical testwork have been used to support the recovery assumptions and plant performance projections applied in the Mineral Resource Estimate and economic assessment contained in this report.

Table 10-1: Metallurgical Testwork Summary

Year	Laboratory/Location	Document Reference Number	Testwork Performed
2004 – 2005	Dawson Metallurgical Laboratories, Inc. Salt Lake City (UT, USA)	P-2825 – "Results of Preliminary Metallurgical Test Work Conducted on Three Ore Samples from the Copper and Gold-Bearing Whistler Project." Prepared for: Kennecott Exploration Company (Rio Tinto).	Gravity concentration and Amalgamation Flotation testing (batch and reagents scoping)
2010	ALS Metallurgy Kamloops (BC, Canada)	KM2591 – "Preliminary Flotation and Cyanidation Testing on Composite Samples from the Island Mountain Project" Prepared for: Kiska Metals Corp.	Flotation testing Cyanidation testing
2012 – 2013	ALS Metallurgy Kamloops (BC, Canada)	KM3499 – "Metallurgical Test Program – Whistler Project" Prepared for: Kiska Metals Corp.	Comminution testing (BWi) Flotation testing (batch and locked-cycle) Cyanidation testing
2024 – 2025	Base Metallurgical Laboratories Ltd.	BL1275 – "Metallurgical Testing of Material from the Whistler Project" Prepared for: U.S. Goldmining Inc.	Comminution testing (Axb, BWi, Ai) Flotation testing (batch, variability, and locked-cycle) Cyanidation testing (concentrate and tailings) Thickening testing (solid-liquid separation and flocculant scoping)

10.2.1 Historical Testwork

Under previous ownership, metallurgical testing was carried out in three phases over an eight-year period. The first metallurgical testwork phase was performed in 2004-2005 at Dawson Metallurgical Laboratories, Inc (DML), in Salt Lake City, Utah, under the direction of Kennecott. The second and third phases were performed at ALS Laboratories in Kamloops, British Columbia, under the direction of Kiska Metals. These three phases are described in the following subsections.

10.2.1.1 2004 Whistler Testwork Program (Phase 1)

10.2.1.1.1 Overview

Starting in 2004, Kennecott Exploration Company engaged Dawson Metallurgical Laboratories (DML) located in Salt Lake City, Utah, USA, to perform the preliminary metallurgical testwork under the supervision of Rio Tinto Technical Services. Testwork was conducted on three composite samples differentiated by sample history, particle size, and lead/zinc content. Preliminary laboratory testing included gravity concentration and flotation to recover copper and gold as reported in March 2005 (Nadasdy, 2005).

10.2.1.1.2 Sample Selection and Preparation

From the Whistler drilling program, a total of approximately 180 core interval samples from drillhole WH-04-05-21 (ranging from 2.32 to 328.56 m) were received by DML.

For the Phase 1 testwork program, the first metallurgical test composite (Original Composite) was prepared by combining every other individual assay reject sample, totaling 25 samples, from the 117.6 to 200.2-m interval, as selected by Kennecott. The second composite (New Core Sample) was prepared using the remaining half of the Kennecott's cut core from the same drillhole and represented the material from 140.6 to 155.3 m, excluding selected high-grade lead (Pb)-zinc (Zn) samples. The third composite (Low Lead-Zinc Composite) was prepared from the remaining samples not used in the first composite, with high-grade lead-zinc samples omitted. The comparison of direct head assays, as performed by DML, for each composite is summarized in Table 10-2.

Table 10-2: Head Assay Analysis Comparison

Sample	Cu (%)	Au (ppm)	Ag (ppm)	Fe (%)	S_{Total} (%)	S (%)	Pb (%)	Zn (%)
Original Composite	0.593	2.48	9.0	6.3	2.6	2.4	0.120	0.440
New Core Sample	0.802	3.33	4.0	5.7	2.6	2.5	0.003	0.013
Low Lead-Zinc Composite	0.537	2.52	3.0	5.9	1.8	1.8	0.001	0.011

Source: Dawson Metallurgical Laboratories, Inc., 2005

10.2.1.1.3 Testing

Preliminary metallurgical testwork included gravity concentration and flotation to recover copper and gold. The three mineralized samples designated as: the Original Composite, the New Core Sample, and the Low Lead-Zinc Composite, as described above were tested from September 2004 and March 2005.

Testwork conducted on the Whistler mineralized samples included the following:

1. Original Composite: comparative Bond ball mill work index (BWi) test; gravity centrifugal concentration and amalgamation test; head assay screen test; rougher kinetic-reagent scoping tests; rougher kinetic-pH tests; three-stage cleaning tests at different primary and regrind sizes; and cleaner tests.

2. New Core Sample: gravity concentration and amalgamation test; rougher kinetic series at varied grind sizes; three-stage cleaner test.

3. Low Lead-Zinc Composite: rougher kinetic test; three-stage cleaning tests; a cleaner reagent test with sulfur dioxide (SO_2) added to the first cleaner. A final cleaner test was conducted to generate a third cleaner concentrate for a suite of assays to support smelter evaluation.

10.2.1.1.4 Test Results

The following summarizes the preliminary metallurgical testwork results as reported by DML:

- Gravity Concentration:
  - When ground to a P_{80} of 140 µm, 11 – 15% of gold was recoverable by gravity concentration and amalgamation, with an average grain size passing 400 mesh. Based on these results, gravity separation was excluded from further consideration, and subsequent work focused on flotation recovery.
- Original Composite:
  - The Bond ball mill work index of the Original Composite was estimated at 20 kWh/t (±3 kWh/t) using DML's comparative work index procedure.
  - Reportedly, as a result of sulfide activation, rougher flotation followed by three-stage cleaning produced a low-grade concentrate of 16% Cu, with recoveries of 83% copper and 65% gold, under the final selected flotation conditions of a primary grind size P_{80} of 80 µm and a regrind size P_{80} of 34 µm. Sulfide activation resulted from elevated lead and zinc content, aggressive sample preparation, or sample aging following grinding.
- New Core Sample:
  - Rougher flotation followed by three-stage cleaning produced a concentrate grade of 23% Cu, with recoveries of 84% copper and 60% gold, at grind sizes similar to those used for the tests on the Original Composite.
- Low Lead-Zinc Composite:
  - Rougher flotation followed by three-stage cleaning produced a concentrate grade of 21% Cu, with recoveries of 80% copper and 59% gold, at grind sizes similar to those used for the tests on the Original Composite.

10.2.1.2 2010 Preliminary Metallurgical Testing Program (Phase 2)

10.2.1.2.1 Overview

Starting in 2010, Kiska Metals Corporation engaged G&T Laboratories, later acquired by ALS Metallurgy (ALS), located in Kamloops, British Columbia, Canada, to perform the preliminary metallurgical testing on the Island Mountain deposit. During this period, a review of the Whistler Deposit was also conducted, however, no further metallurgical testwork was performed on the Whistler Deposit. Accordingly, Phase 2 testwork is not applicable to this report and is omitted from this section. The Island Mountain deposit is not included in the mine plan for this study.

10.2.1.3 2012 Whistler Testwork Program (Phase 3)

10.2.1.3.1 Overview

Starting in 2012, Kiska Metals Corporation engaged ALS Metallurgy, located in Kamloops, British Columbia, Canada, to perform the third phase of metallurgical testing. Testwork was conducted on two samples, one from each of the 2008 and 2010 drill campaigns. Each sample was subdivided into high-grade (HG), medium-grade (MG), and low-grade (LG) subsamples based on gold grade. The samples were not verified for aging due to the cancellation of the 2012 drill campaign and the resulting lack of calibration samples.

Phase 3 laboratory testing included comminution characterization, flotation to recover copper and gold, and cyanidation as reported in January 2013 (ALS Metallurgy Kamloops, 2013)

10.2.1.3.2 Sample Selection and Preparation

From the Whistler 2008 drilling program, a total of approximately 609 kg of quarter-core interval samples, divided into two shipments, were received by ALS.

For the Phase 3 testwork program, six sample composites were constructed from the core intervals by grouping samples according to drill campaign and gold grade. The comparison of direct head assays, as performed by ALS, for each composite is summarized in Table 10-3.

Table 10-3: Head Assay Analysis Comparison

Sample	Drill Year	Cu (%)	Au (ppm)	Ag (ppm)	Fe (%)	S_{Total} (%)	S (%)	Pb (%)	Zn (%)
MG Master Composite 1 (MG 08-08 Composite)	2008	0.12	0.53	-	5.80	3.60	-	-	-
MG 10-19 Composite	2010	0.22	0.52	-	2.62	1.93	-	-	-
LG 08-08 Composite	2008	0.08	0.34	-	4.08	2.69	-	-	-
HG 08-08 Composite	2008	0.50	1.78	-	4.94	1.75	-	-	-
LG 10-19 Composite	2010	0.22	0.38	-	3.39	1.69	-	-	-
HG 10-19 Composite	2010	0.17	0.96	-	3.33	1.11	-	-	-

Note: "–" indicates data not available. Source: ALS Metallurgy Kamloops, 2012

10.2.1.3.3 Testing

Phase 3 metallurgical testwork included flowsheet selection, comminution characterization, flotation to recover copper and gold, and cyanidation. The six mineralized samples as described above were tested starting in August 2012. Results were reported in ALS Metallurgy report KM3499 in January 2013 (ALS Metallurgy Kamloops, 2013). No mineralogical work was conducted.

Testwork conducted on the Whistler mineralized samples included the following:

1. Comminution Characterization: A single standard Bond ball mill work index test was carried out on MG 10-19 composite toward the end of the program. A closing size of 106 µm was used. No SAG milling testing (e.g., JK Drop weight or SMC tests) were included in the program, and no Bond rod mill work index tests were completed.
2. Copper-Gold Flotation Testing: Open-circuit batch flotation testing and closed-circuit locked-cycle testing were conducted on the composite samples. In addition, pyrite flotation testing was conducted on the rougher flotation tailings.
3. Cyanidation: Cyanide leaching was performed on the combined three-stage cleaner tailings from two batch flotation tests to investigate additional gold recovery. Leaching was conducted with a 48-hour residence time. No cyanidation testing was performed on the rougher tailings, which represented 14 to 20% of the total gold distribution.

10.2.1.3.4 Test Results

The following summarizes the preliminary testwork results as reported by ALS.

1. Comminution Characterization: The Bond ball mill work index (BWi) was found to be 19.9 kWh/t, placing the Whistler material in the very hard range of ball mill hardness.
2. Copper-Gold Flotation Testing: Open-circuit batch flotation testing consistently achieved copper recoveries of 80-85% to a concentrate grade of 25% Cu. However, gold recovery was lower, ranging from 40-50%, due to lower rougher recoveries and low cleaner recoveries, with significant deportment of gold to cleaner tailings streams. Locked-cycle flotation tests on MG Master Composite 1 and MG Composites 10-19 produced copper recoveries of approximately 92% to a final concentrate grading 25 to 26% Cu. Gold recovery to the same concentrate ranged from 68 to73%. Pyrite flotation of the rougher tailings recovered an additional 6 to 11% of gold and was excluded from further consideration.
3. Cyanidation: Gold extraction through cyanide leaching was approximately 77% for the two tests conducted. As this only increased overall gold recovery by 17%, most of which was recovered through closed-circuit flotation, cyanidation was not pursued further.

10.2.2 Current Testwork

Under current ownership, metallurgical testing has been carried out on the Whistler Deposit. The testwork was performed in 2025 at Base Metallurgical Laboratories Ltd. in Kamloops, British Columbia, Canada (Base Metallurgical Laboratories Ltd., 2026).

10.2.2.1 Overview

Starting in 2025, U.S. GoldMining engaged Base Metallurgical Laboratories Ltd. located in Kamloops, British Columbia, Canada, to perform the current phase of metallurgical testing. Testwork was conducted on one master composite sample and eight variability samples, comprising material from fresh drill cores collected from four drillholes from the 2023 and 2024 drill campaigns.

Laboratory testing included comminution characterization, flotation to recover copper and gold, and cyanidation on selected flotation products, with the objective of optimizing metal recovery. Testwork data was provided to Ausenco in June of 2025 by Base Metallurgical Laboratories Ltd.

10.2.2.2 Sample Selection and Preparation

The drill core samples used for the 2025 metallurgical test program were selected from four drillholes (WH23-01, WH23-02, WH23-03 and WH24-02). The sampling of fresh quarter-core was conducted to produce eight composites, a portion of which was then further homogenized to form a single Master Composite, intended to represent the global average resource grade from the 2024 MRE. The head assays of each composite are summarized in Table 10-4.

Table 10-4: Head Assay Summary

Sample ID	Head Grade Cu (%)	Head Grade Au (g/t)	Head Grade Ag (g/t)	Head Grade S (%)
WC-Var-1	0.16	1.35	1.50	1.19
WC-Var-2	0.11	0.45	0.80	0.92
WC-Var-3	0.18	0.06	0.80	1.52
WC-Var-4	0.29	0.88	2.10	1.15
WC-Var-5	0.28	0.41	1.20	1.94
WC-Var-8*	0.12	-	5.3	4.04
WC-Var-9	0.14	0.22	4.20	1.96
WC-Var-10	0.20	0.78	0.90	1.93
Master Composite	0.17	0.42	0.80	1.26

Note: *WC-Var-8 head assay is only included in the comminution testing and was assayed separately by ICP.
Source: Base Metallurgical Laboratories Ltd., 2025

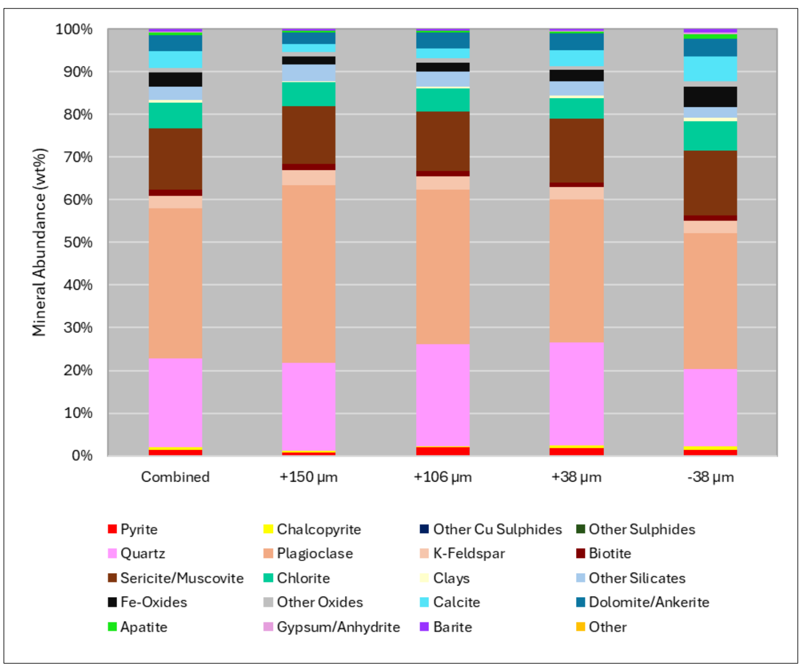
10.2.2.3 Mineralogy

A single mineralogical analysis was performed using QEMSCAN on each composite, ground to a P_{80} of 159 µm. The analysis included mineral liberation assessment, size-by-size mineral distribution, and deportment of copper and sulfur. The analysis of the Master Composite is summarized in Figure 10-1, Figure 10-2, and Figure 10-3.

The mineralogical analysis of the Master Composite indicates the following:

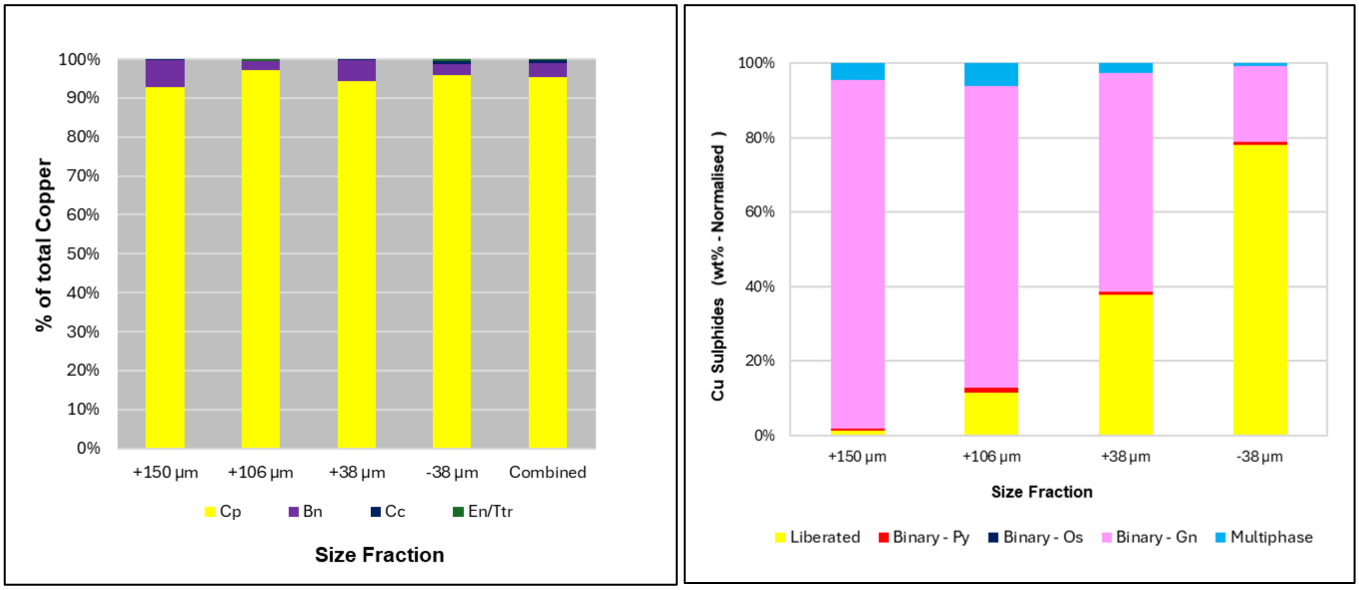
- Pyrite is the dominant sulfide mineral; comprising 1.4% of the total mass. Sulfur deportment indicates that 70% of the sulfur is contained in pyrite, 17% in chalcopyrite, 7% in barite, and 5% in gypsum/anhydrite. At a grind size of 159 µm, Pyrite is 63% liberated, with most non-liberated pyrite associated with the gangue (33%) and only 2% associated with the copper sulfides.
- Chalcopyrite contains 95% of copper. At a grind size of 159 µm, copper sulfides show 55% liberation, with less than 1% associated with pyrite, indicating favorable conditions for selective rejection of pyrite from copper concentrate. The remaining non-liberated copper sulfides (45%) are primarily associated with gangue, indicating the need for further regrinding.
- Copper sulfides account for 0.58% of total mass, with chalcopyrite comprising 0.57%, and the other copper sulfides occur only in trace amounts.
- Size-by-size liberation analysis indicates that a regrind size of less than 38 µm is required to achieve greater than 75% liberation of both copper sulfides and pyrite from gangue.

Figure 10-1: Master Composite Size-by-Size Mineralogy



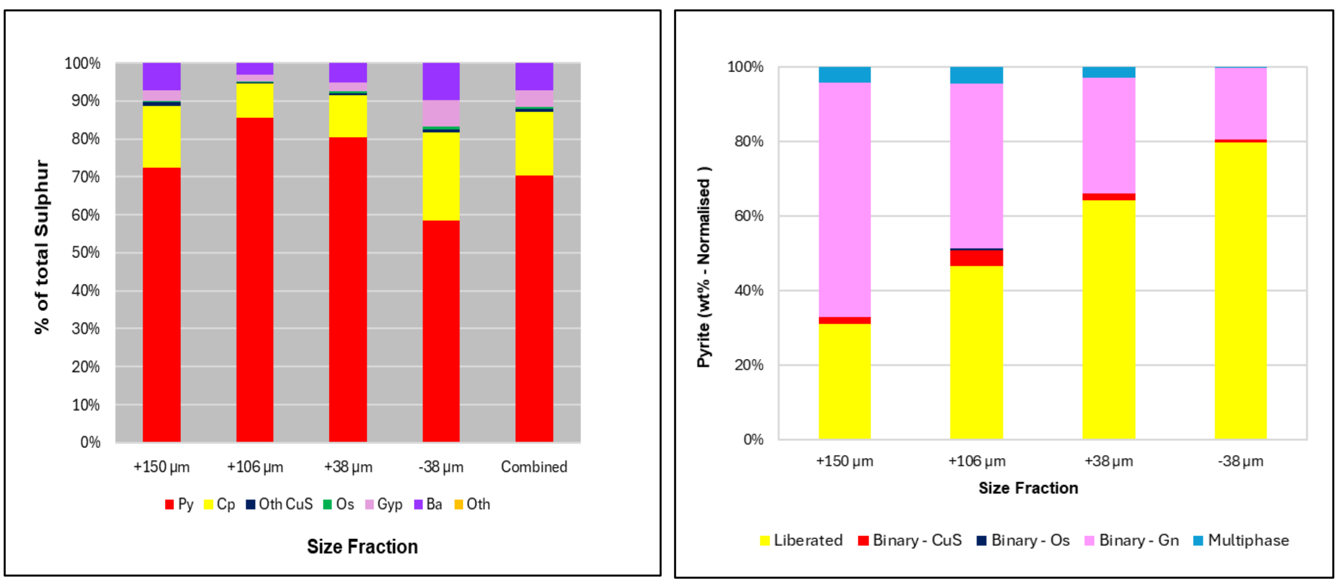
Source: Base Metallurgical Laboratories Ltd., 2025

Figure 10-2: Master Composite Copper Deportment and Copper Sulfides Liberation Analysis



Source: Base Metallurgical Laboratories Ltd., 2025

Figure 10-3: Master Composite Sulfur Deportment and Pyrite Liberation Analysis



Source: Base Metallurgical Laboratories Ltd., 2025

10.2.2.4 Comminution Testing

Comminution testwork, including grinding work indices, JKSimMet parameters (A, b, DWI, t_a, SCSE), and Bond Abrasion Indices, were performed on the Master Composite sample to characterize the variability in competency, hardness, and abrasion of the Whistler Deposit. A summary of the comminution testwork results, including the 75th percentile and average values, is presented in Table 10-5. Based on the comminution characterization results, the Whistler Deposit mineralized material is categorized as very competent. The Bond Ball Mill Work Index (BWi) testing was performed at a 149 µm closing screen size and averaged 21.8 kWh/t across the eight variability samples measured.

Table 10-5: Comminution Test Summary

Sample ID	Relative Density	A x b	t_a	DWI (kWh/m³)	SCSE (kWh/t)	BWi (kWh/t)	Bond Ai (g)
WC-Var-1	2.77	29.2	0.27	9.50	11.7	21.2	0.204
WC-Var-2	2.75	23.7	0.22	11.61	13.0	22.0	0.183
WC-Var-3	2.67	22.6	0.22	11.80	13.0	22.7	0.079
WC-Var-4	2.76	34.2	0.32	8.07	10.8	20.7	0.111
WC-Var-5	2.69	21.4	0.21	12.57	13.4	22.0	0.095
WC-Var-8	2.79	25.0	0.23	11.16	12.7	22.2	0.101
WC-Var-9	2.77	24.8	0.23	11.18	12.7	-	0.161
WC-Var-10	2.73	24.0	0.23	11.36	12.8	-	0.152
75th Percentile	2.77	23.6	0.26	11.76	13.0	22.2	0.167
Average	2.74	25.1	0.24	10.91	12.5	21.8	0.129

Note: The material competence increases as A x b decreases, therefore, the 75th percentile material competence corresponds to the 25th percentile value. Source: Base Metallurgical Laboratories Ltd., 2025

10.2.2.5 Flotation Testing

A total of 28 flotation tests were completed on a combination of Master Composite and variability samples.

Initial flotation testing consisted of four rougher tests followed by six cleaner tests; all performed on the Master Composite. These tests established the design criteria for the final locked-cycle tests and subsequent variability tests. The initial rougher tests were conducted using a range of initial grind sizes (P_{80}) from 92 to 193 µm to select an optimal primary grind size of 120 µm. A secondary grind (regrind) size of 16 µm was also selected to yield the optimal recoveries. The initial cleaner tests were conducted using multiple reagent schemes and dosages to select a combination of hydrated lime, Aero®3894, potassium amyl xanthate (PAX), and methyl isobutyl carbinol (MIBC), dosed to the target levels summarized in Table 10-6.

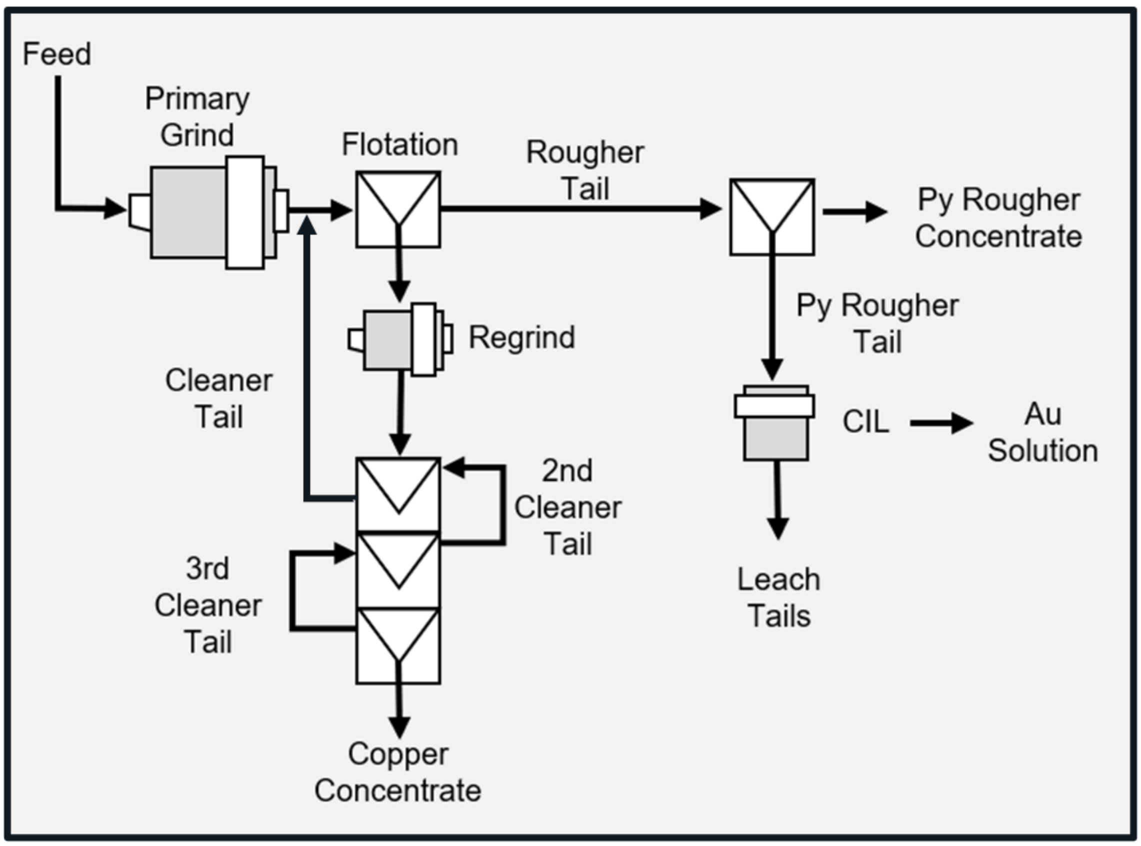
Table 10-6: Flotation Testwork Reagent Dosage Summary (LCT28)

Reagent	Dosage (g/t)
Hydrated lime	370
Aero®3894	47
Potassium amyl xanthate (PAX)	50
Methyl isobutyl carbinol (MIBC)	42

Source: Base Metallurgical Laboratories Ltd., 2025

Four locked-cycle tests were performed on the Master Composite to develop the testwork flowsheet presented in Figure 10-4. The final test circuit consisted of the rougher flotation with the concentrate reporting to concentrate regrind, followed by a three-stage copper cleaner circuit. Tailings from each cleaner stage were recycled to the feed of the preceding stage. The rougher tailings were further treated by a pyrite rougher flotation to investigate the production of a pyrite rich product.

Figure 10-4: Flotation Testwork Process Flowsheet



Source: Base Metallurgical Laboratories Ltd., 2025

Using this flowsheet, the final locked-cycle test (LCT28) produced a copper concentrate of 23.7% Cu at a mass pull of 0.6%, with copper, gold, and silver recoveries of 79.1%, 54.1% and 55.3%, respectively. Following the locked-cycle testing, seven batch, open-circuit, variability tests were performed for both the rougher and cleaner stages to assess performance variability across the deposit and develop a performance projection. The results from the variability testing and the final locked-cycle flotation testwork are summarized in Table 10-7, including recoveries to a saleable copper concentrate at grades below 30% Cu. Variability testwork results indicate limited performance variation with no evidence of outlier behavior. Metal recoveries from the batch flotation tests were as high as 83.7% copper and 74% gold.

Table 10-7: Variability and Locked-Cycle Flotation Testwork Summary

Test ID	Test Type	Sample	Product	Mass Pull (%)	Concentrate Grade (%Cu)	Copper Recovery (%)	Gold Recovery (%)	Silver Recovery (%)
C21	Batch	WC-Var 1	1st Cln Con	0.5	21.6	68.2	49.7	32.9
C22	Batch	WC-Var 2	1st Cln Con	0.4	21.3	64.5	49.2	40.6
C23	Batch	WC-Var 3	1st Cln Con	0.7	25.3	82.2	60.0	57.0
C24	Batch	WC-Var 4	1st Cln Con	0.9	25.4	74.7	61.3	43.1
C25	Batch	WC-Var 5	1st Cln Con	0.8	29.7	83.7	62.3	44.3
C26	Batch	WC-Var 9	2nd Cln Con	0.5	20.5	65.8	54.6	46.9
C27	Batch	WC-Var 10	2nd Cln Con	0.6	29.2	82.5	74.5	49.6
C09	Batch	Master	3rd Cln Con	0.5	24.9	77.5	50.6	41.9
LCT28	Locked-Cycle	Master	3rd Cln Con	0.6	23.7	79.1	54.1	55.3

Source: Base Metallurgical Laboratories Ltd., 2025

10.2.2.6 Direct Cyanidation Testing

To enhance gold recovery, cyanide leaching of both cleaner and rougher tailings was investigated. Cyanidation testwork was conducted on samples derived from the locked-cycle tests (LCT-28 and LCT-11) to assess the feasibility of a supplemental cyanide leach circuit. Cyanidation was conducted on the pyrite rougher tailings (from LCT-28), pyrite concentrate (from LCT-11), and copper first cleaner tailings (from LCT-11). Gold recoveries exceeding 80% were achieved following a 48-hour cyanide leach on the pyrite rougher tailings and copper first cleaner tailings, while the pyrite concentrate reached 42.4% after 24 hours, with no appreciable increase in extraction observed at 48 hours. The results of the cyanidation testwork are summarized in Table 10-8.

Table 10-8: Cyanidation Testwork Summary

Test ID	Leach Feed Source	Flotation Gold Recovery to Leach Feed (%)	Flotation Mass Pull to Leach Feed (%)	Leach Feed Gold Head Grade (g/t)	Gold 24h Extraction (%)	Gold 48h Extraction (%)	Sodium Cyanide Consumption (kg/t)	Lime Consumption (kg/t)
CN11B	LCT11 – Pyrite Rougher Concentrate	13.8	4.2	1.29	42.4	43.4	1.5	2.5
CN11C	LCT11 – Cu 1st Cleaner Tailings	9.4	4.6	0.81	80.4	82.4	4.6	2.2
CN28B	LCT28 – Pyrite Rougher Tailings	35.2	95.4	0.13	75.7	88.7	0.5	1.3

Source: Base Metallurgical Laboratories Ltd., 2025

Cyanidation of the pyrite rougher tailings for 48 hours yielded a net gold recovery increase of 31.2%, complimenting the gold recovery to the copper concentrate, with reagent consumptions of 0.5 kg/t NaCN and 1.3 kg/t lime. Based on the results from LCT-11 cyanidation testing, the leaching of the copper first cleaner tailings resulted in a gold recovery increase of 7.7%. For the final flowsheet, this stream was recycled back to the rougher flotation feed to increase recovery to the copper concentrate, therefore, this recovery increase is not additive to the LCT28 results.

10.2.2.7 Solid-Liquid Separation Testing

A series of dewatering tests were conducted on a slurry sample of Master Composite flotation tailings from LCT28, which had been ground to a flotation feed sizing of approximately 120 µm P_{80}. The final tailings included the cleaner tailings, which had been reground to approximately 16 µm P_{80}. The testwork included flocculant scoping, static settling tests, and dynamic settling tests. Results from selected test conditions are summarized in Table 10-9.

Table 10-9: Static and Dynamic Settling Testwork Summary

Test Type	Test ID	Parameter	Units	Value
Static settling	S2	Grind size, P_{80}	µm	120
		Flocculant type	-	AN913SH
		Flocculant dosage	g/t	40
		pH	-	8.5
		Initial Density	% solids (wt/wt)	13.6
		Final Density	% solids (wt/wt)	54.1
		Free settling	m/h	2.8

Test Type	Test ID	Parameter	Units	Value
Dynamic settling	D-1B	Grind size	µm	120
		Flocculant type	-	AN913SH
		Flocculant dosage	g/t	40
		pH	-	8.5
		Initial Density	% solids (wt/wt)	15
		Final Density	% solids (wt/wt)	58.9
		Rise rate	m/h	4.4
		Loading rate	$t/m^2/h$	0.7
		Turbidity	mg/L	45

Source: Base Metallurgical Laboratories Ltd., 2025

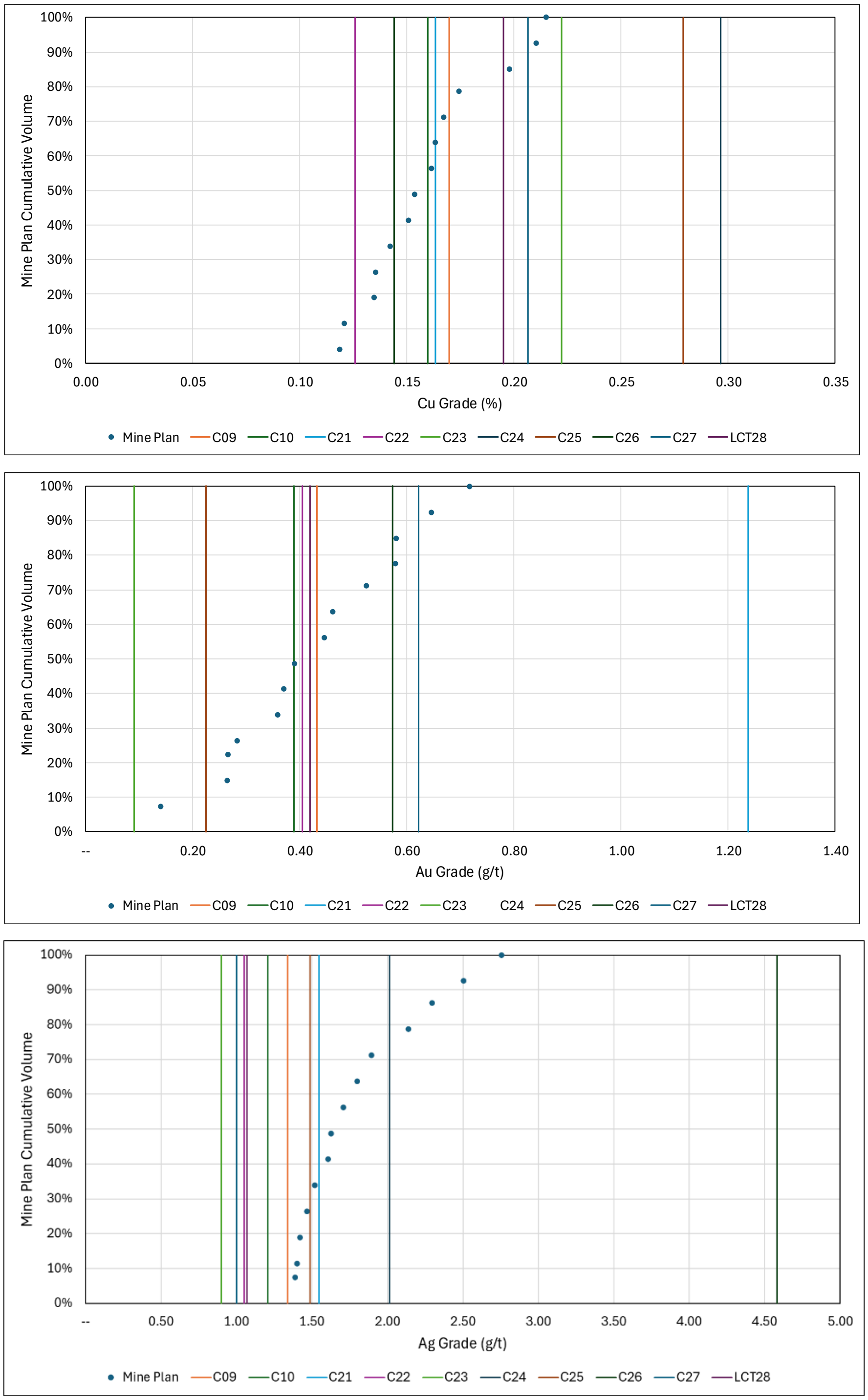
10.3 Metallurgical Variability

The cumulative volume versus grade profiles for copper, gold, and silver in the current mine plan, reported on an annual basis, are represented in Figure 10-5, together with the head grades from each flotation test used in projecting the metallurgical recoveries. These graphs illustrate the range of metal grades present within the mine plan along with the cumulative volume of material that is at or below each grade. Comparing the metal grades present in the mine plan and those represented by the testwork samples leads to the following conclusions:

- The mine plan has a maximum copper grade of 0.21% Cu and a 50^{th} percentile grade of 0.15% Cu. The metallurgical samples represent the upper 89% of copper grades present in the mine plan on a mass weighted basis.
- The mine plan has a maximum gold grade of 0.72 g/t Au and a 50^{th} percentile grade of 0.39 g/t Au. The metallurgical samples represent the lower 85% of gold grades present in the mine plan on a mass weighted basis.
- The mine plan has a maximum silver grade of 2.76 g/t Ag and a 50^{th} percentile grade of 1.63 g/t Ag. The metallurgical samples represent the lower 71% of silver grades present in the mine plan on a mass weighted basis, however, of the ten flotation tests, eight of the tests represent only the lower 34% of silver grades (<1.55 g/t Ag), with a single test conducted at 2.0 g/t Ag.

Overall, the metallurgical samples used for the cleaner variability and locked-cycle flotation tests are representative of the metal grades present across the current mine plan.

Figure 10-5: Comparison of Mine Plan Cumulative Grade Distribution and Flotation Testwork Head Grades.



Source: Ausenco, 2025

10.4 Deleterious Elements

An assay was conducted on the final cleaner concentrate produced from LCT-28 to evaluate the presence of deleterious elements that could affect concentrate quality or marketability. The results of this assay are presented in Table 10-10. The reported trace element concentrations do not adversely affect the viability of economic extraction.

Table 10-10: Concentrate Trace Element Assay Summary LCT-28

Analyte	Unit	Concentrate Grade
Ag	ppm	86.4
Al	%	0.37
As	ppm	134
B	ppm	<10
Ba	ppm	13
Be	ppm	1.2
Bi	ppm	28
Ca	%	0.65
Cd	ppm	16
Co	ppm	43
Cr	ppm	25
Cu	%	>20
Fe	%	>20.00
Ga	ppm	14
Hg	ppm	13
K	%	0.02
La	ppm	12
Mg	%	0.26
Mn	ppm	308
Mo	ppm	2690
Na	%	0.01
Ni	ppm	35
P	%	0.021
Pb	ppm	855
S	%	>20.00
Sb	ppm	254
Sc	ppm	2
Sr	ppm	69
Te	ppm	<20
Th	ppm	<5
Ti	%	0.01

Analyte	Unit	Concentrate Grade
Tl	ppm	<2
U	ppm	23
V	ppm	33
W	ppm	6
Y	ppm	2
Zn	ppm	2273
Zr	ppm	7

Source: Base Metallurgical Laboratories Ltd., 2025

10.5 Recovery Estimates

The proposed flowsheet, selected based on the results of the 2025 testwork campaign, includes primary grinding to a P_{80} of 120 µm, copper flotation with rougher concentrate regrinding to a P_{80} of 16 µm, and recirculation of first cleaner tailings to the rougher feed. The flowsheet excludes a pyrite flotation stage on the copper rougher tailings, with all copper rougher tailings reporting to a gold leach circuit.

The expected metal recoveries to copper concentrate for the Whistler Deposit were estimated by first normalizing the testwork recoveries using the grade-recovery curves to a consistent 25% Cu concentrate grade across all tests. Recoveries from each open-circuit test were then scaled to represent the transition to closed-circuit operation for the final flowsheet. This scaling factor was derived by comparing flotation tests C09 and LCT28. These adjusted recoveries were then fitted to regression curves using nine batch rougher-cleaner flotation tests together with the final locked-cycle test, these test results are summarized in Table 10-11.

Table 10-11: Flotation Testwork Raw Data Used for Recovery Projection

Test Name	Test ID	Feed Grades				Overall Flotation Performance					
		Cu (%)	Au (g/t)	Ag (g/t)	S (%)	Mass Pull (%)	Cu Recovery (%)	Au Recovery (%)	Ag Recovery (%)	S Recovery (%)	Cu Conc. (%)
Cleaner Test	C09	0.17	0.43	1.34	1.07	0.53	77.5	50.6	41.9	14.3	24.9
Cleaner Test	C10	0.16	0.39	1.21	1.09	0.43	68.8	41.5	29.4	11.2	25.8
Variability Cleaner	C21	0.16	1.24	1.55	1.12	0.31	61.3	43.7	32.1	8.9	32.3
Variability Cleaner	C22	0.13	0.40	1.05	1.04	0.25	58.2	42.8	39.7	7.8	29.8
Variability Cleaner	C23	0.22	0.09	0.90	1.10	0.54	80.1	56.4	55.9	16.1	32.9
Variability Cleaner	C24	0.30	0.75	2.01	1.12	0.66	71.1	56.5	41.7	19.9	31.9
Variability Cleaner	C25	0.28	0.22	1.49	1.71	0.69	81.8	60.5	43.5	13.7	33.1
Variability Cleaner	C26	0.14	0.57	4.58	1.89	0.36	60.7	45.9	46.3	6.0	24.2
Variability Cleaner	C27	0.21	0.62	1.00	1.77	0.54	81.6	69.4	48.2	9.8	31.3
Locked-Cycle Test	LCT28	0.19	0.42	1.07	0.99	0.65	81.5	59.7	57.1	18.5	24.4

Source: Base Metallurgical Laboratories Ltd., 2025

The recovery projections for copper, gold, and silver to a 25% Cu concentrate grade are estimated by Equation 1, Equation 2 and Equation 3, respectively.

Equation 1 – Recovery Projection of Copper to a 25% Cu Concentrate

$$Recovery\ Cu\% = 212.5\left(1 - e^{-18.4\ Feed\ Cu\%}\right) - 38\left(1 - e^{-0.4\frac{Feed\ Ag\ g/t}{Feed\ S\%}}\right) - 111.6$$

Equation 2 – Recovery Projection of Gold to a 25% Cu Concentrate

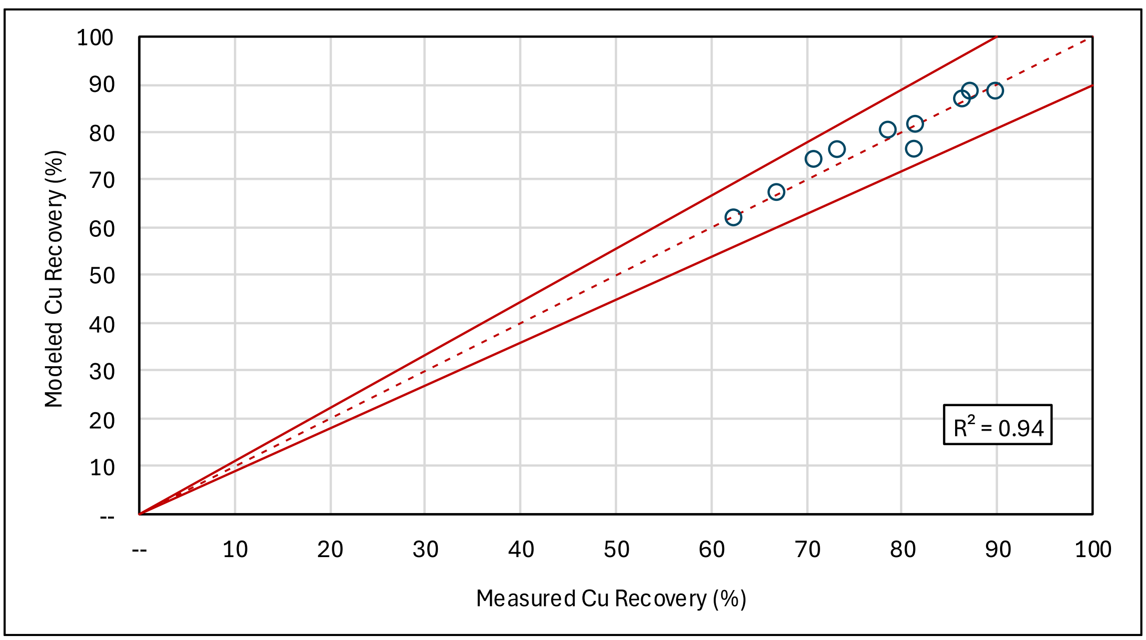
$$Recovery\ Au\% = 4.6\left(\frac{Feed\ Ag\ g/t}{Feed\ S\%}\right)^{-3.3} + 312.3\ Feed\ Cu\%^2 + 45.4$$

Equation 3 – Recovery Projection of Silver to a 25% Cu Concentrate

$$Recovery\ Ag\% = 10.2\left(e^{6.6\frac{Feed\ Cu\%}{Feed\ Ag\ g/t}}\right) + 0.2\left(\frac{Feed\ Ag\ g/t}{Feed\ S\%}\right)^{5.5} + 26.2$$

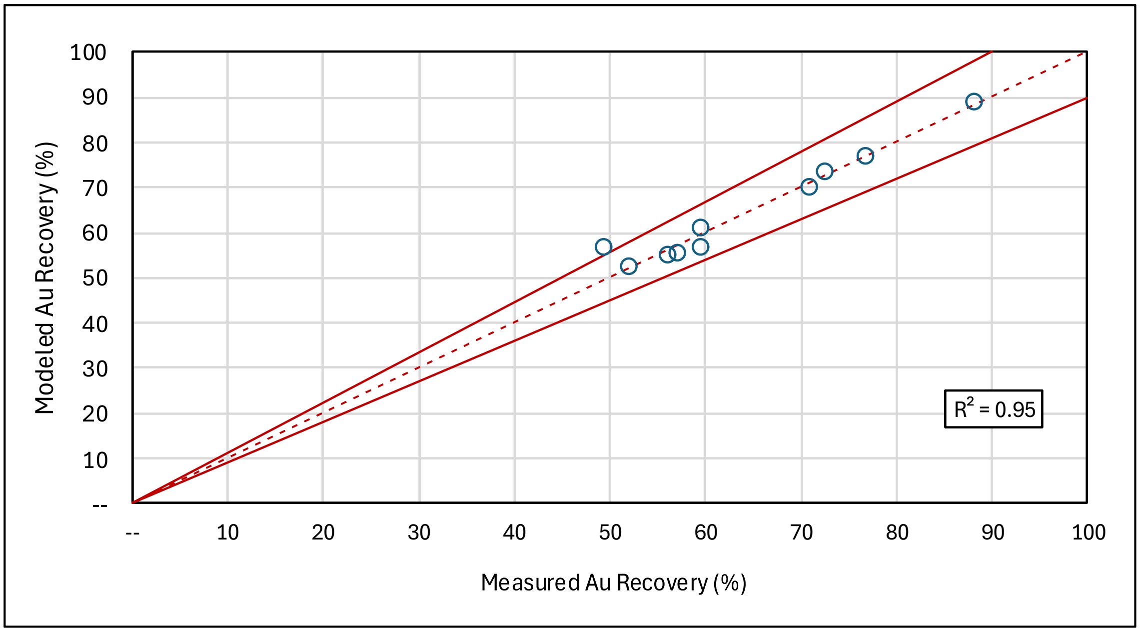
A comparison of predicted and measured testwork metal recoveries for each of copper, gold, and silver to a 25% Cu concentrate are presented in Figure 10-6, Figure 10-7, and Figure 10-8 respectively.

Figure 10-6: Comparison of Modeled and Measured Copper Recovery to a 25% Cu Concentrate



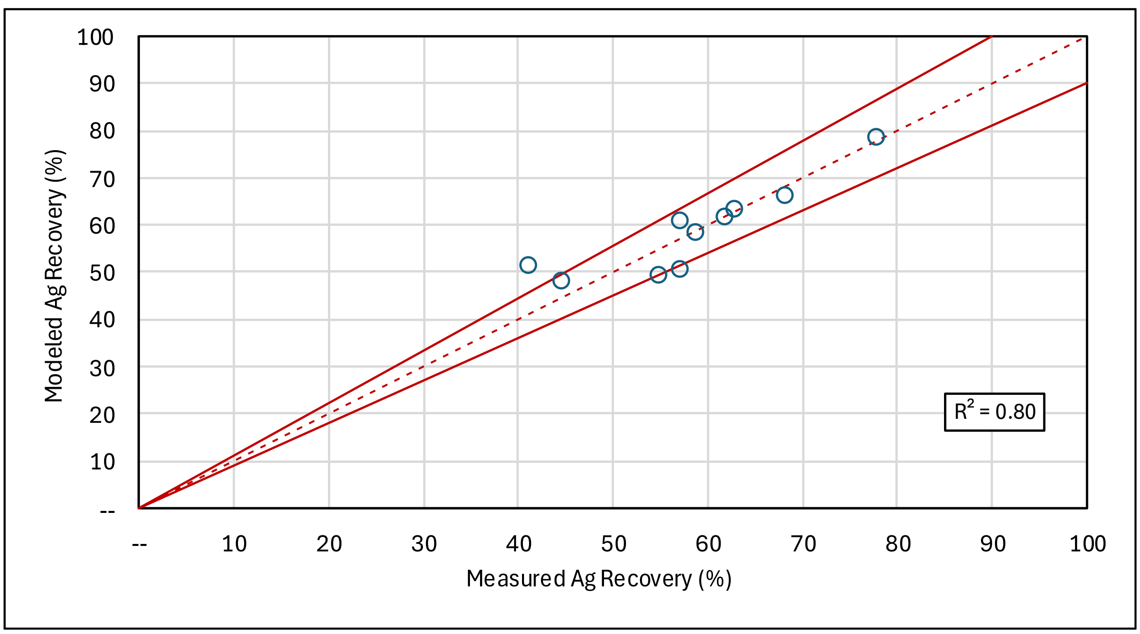
Note: Upper and lower bands are ±10%. Source: Ausenco, 2025

Figure 10-7: Comparison of Modeled and Measured Gold Recovery to a 25% Cu Concentrate



Note: Upper and lower bands are ±10%. Source: Ausenco, 2025

Figure 10-8: Comparison of Modeled and Measured Silver Recovery to a 25% Cu Concentrate



Note: Upper and lower bands are ±10%. Source: Ausenco, 2025

The recovery projections predict that copper recovery to copper concentrate is expected to range from 66% to 84%, with a LOM average recovery of 77.8% when applied to the current mine plan on an annual basis. Similarly, the LOM average recoveries of gold and silver to copper concentrate are 62.1% and 44.8%, respectively.

Gold and silver extraction from cyanide leaching is estimated based on the combined results of the CN11B and CN28B cyanidation testwork, conducted on the pyrite rougher concentrate and pyrite rougher tailings, respectively. Extraction results presented in Table 10-8 were weighted according to each fraction's contribution to the leach feed mass and metal content relative to the flotation feed. This approach yields average leach extractions of 74.9% Au and 20.5% Ag over a 48-hour leach residence time. Accounting for an estimated soluble metal loss of 5% following extraction, the gold leach circuit increases overall gold recovery by 27.0% and silver recovery by 10.7% on a life-of-mine average basis. The combined flotation and leaching circuit results in estimated life-of-mine average overall recoveries of 77.8% Cu, 88.9% Au, and 55.6% Ag.

10.6 Statement on the Adequacy of the Data

The Qualified Person has reviewed the metallurgical testwork results and other technical data available for this study. In the opinion of the Qualified Person, the quantity and quality of the data are sufficient to support the recovery assumptions, and other technical conclusions presented in this section of the Technical Report Summary.

The data have been collected using generally accepted industry practices and laboratory procedures. While additional metallurgical testwork and engineering studies would typically be conducted at more advanced stages of project development, the currently available data are considered adequate to support the level of study presented in this report.

11 MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimate (MRE) for the Project has an effective date of March 2, 2026. The MRE was prepared by Sue Bird, P.Eng., of Moose Mountain Technical Services (MMTS).

11.1 Mineral Resource Estimate

The Project's total MRE includes the Whistler, Raintree West and Island Mountain deposits and is summarized in Table 11-1 for the base-case cutoff grades. The resource is prepared in accordance with the United States Securities and Exchange Commission (SEC) regulation S-K subpart 1300 (S-K 1300). In the opinion of the Qualified Person, the Mineral Resource Estimates reported herein are a reasonable representation of the mineral resources found within the Project at the current level of sampling. The mineral resources were estimated in accordance with §§229.1300 through 229.1305 (subpart 229.1300 of Regulation S-K). This report is an update to the previously publicly disclosed report (Bird, 2022).

The resource utilizes pit shells to constrain resources at the Whistler, Island Mountain, and Raintree West gold-copper deposits, as well as an underground potentially mineable shape to constrain the mineral resource estimate for the deeper portion of the Raintree West deposit. The current estimate uses metal prices of US$2,750/oz for gold, US$4.35/lb for copper and US$30/oz for silver, updated recoveries, smelter terms and costs, as summarized in the notes to Table 11-1. Metal prices have been chosen based partially on market consensus research using mean prices from 2025 and forecast up to 2026 for long term prices. The metal prices chosen also considered the spot prices and the three-year trailing average prices. For all three metals, the final prices used for this resource estimate are below both the spot metal price and the three-year trailing average, which is considered an industry standard in choosing prices.

The base-case cutoff value for open pit mining is US$13.40/t for all three deposits, which covers the Processing cost of US$11.25/t and the G&A costs of US$2.15/t; this is the marginal cutoff for which mining costs are not included. Cutoff grades for underground mining are based on processing + G&A costs plus an additional US$17.10/t for underground bulk mining, to define the marginal cutoff NSR grade. There has been drilling in 2023 and 2024 which resulted in updated geologic modelling, resource estimation parameters and an updated resource estimate.

For the mineral resource cutoff value determination, a 3.0% NSR royalty was assumed. This is derived from the sum of a 2.75% royalty to MF2 plus a 1% royalty to Gold Royalty Corp., with an assumption that U.S. GoldMining can negotiate a buy-back of a 0.75% NSR, for a net 3.0% NSR, as is customary to occur for similar project developments. In preparing the resource estimate herein, a sensitivity analysis has also been conducted by the author. Based on such analysis, utilizing a higher 3.75% NSR royalty rate in determining a cutoff grade would not materially impact the estimates contained herein and would be de minimis (approximately 0.7% differential of total metal in the Whistler pit on a gold equivalent basis).

These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Qualified Person is of the opinion that issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work. These factors may include environmental permitting, infrastructure, sociopolitical, marketing, or other relevant factors.

The sensitivity to the resource by deposits is presented in Table 11-2 through Table 11-4 for the Whistler, Raintree West, and Island Mountain deposits, respectively. As a point of reference, the in-situ gold, copper and silver mineralization are inventoried and reported by intended processing method.

Table 11-1: Mineral Resource Estimate for the Whistler Project (Effective date: March 2, 2026)

Class	Deposit	Cutoff Value	ROM tonnage	In-situ Grades					In-situ Metal			
		(US$/t)	(kt)	NSR (US$/t)	AuEq (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (koz)	Au (koz)	Cu (klbs)	Ag (koz)
Indicated	Whistler		284,203	38.74	0.562	0.409	0.154	1.7	5,132	3,740	964,275	15,808
	Raintree West-Pit	13.40	10,332	35.63	0.517	0.420	0.076	4.8	156	128	15,356	1,321
	Indicated Open Pit		**294,535**	**38.63**	**0.560**	**0.410**	**0.151**	**1.8**	**5,287**	**3,868**	**979,631**	**17,129**
	Raintree West-UG	$40 shell	4,619	58.81	0.853	0.713	0.118	5.4	127	106	12,036	795
	Total Indicated	**varies**	**299,154**	**38.94**	**0.565**	**0.414**	**0.151**	**1.9**	**5,414**	**3,973**	**991,667**	**17,924**
Inferred	Whistler		4,967	38.37	0.556	0.433	0.115	1.2	89	69	12,549	192
	Island Mountain		187,283	29.04	0.421	0.376	0.043	0.9	2,535	2,263	178,368	5,299
	Raintree West-Pit	13.40	18,780	37.83	0.548	0.471	0.057	4.3	289	252	19,475	1,927
	Inferred Open Pit		**211,030**	**30.04**	**0.436**	**0.386**	**0.046**	**1.2**	**2,913**	**2,584**	**210,392**	**7,418**
	Raintree West-UG	$40 shell	79,717	55.32	0.802	0.692	0.102	2.7	2,055	1,773	179,964	6,843
	Total Inferred	**varies**	**290,747**	**36.97**	**0.536**	**0.470**	**0.062**	**1.6**	**4,969**	**4,357**	**390,355**	**14,261**

Notes to Table 11-1 through Table 11-4:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.
2. The mineral resource for the Whistler, Island Mountain, and the upper portions of the Raintree West Deposits have been confined by an open pit with "reasonable prospects of eventual economic extraction" using the following assumptions:
   - metal prices of US$2,750/oz Au, US$4.35/lb Cu and US$30/oz Ag;
   - payable metal of 94.8% payable for Au, 96.5% payable for Cu, and 88.2% payable for Ag;
   - refining costs for Au of US$7.50/oz, for Ag of US$1.00/oz and for Cu of US$0.065/lb;
   - off-site costs of US$165.65/t;
   - royalty of 3% Net Smelter Return (NSR);
   - pit slopes are 50 degrees;
   - mining cost of US$2.75/t for waste and mineralized material; and
   - processing costs of US$11.25/t, general and administrative costs of US$2.15/t.
3. The open pits at Whistler and Island Mountain use the 150% NSR case, with the upper portion of Raintree West using the 100% NSR case.

The lower portion of the Raintree West deposit has been constrained by a mineable shape within a “reasonable prospects of eventual economic extraction” shape using a $40.00/t cutoff value.

4. Metallurgical recoveries are: 87.8% for Au, 75.4% for Cu, and 49.1% Ag.
5. The NSR equation is: NSR ($/t)=(100%-3%)*((Au(g/t)*87.8%*$78.57/g) + (Cu%*75.4%*$3.88/lb*2204.62 + Ag(g/t)*49.1%* $0.77)).
6. The gold equivalent equation is: AuEq=Au + Cu*0.9361 + 0.0055Ag.
7. The specific gravity for each deposit and domain ranges from 2.76 to 2.91 for Island Mountain, 2.60 to 2.72 for Whistler with an average value of 2.80 for Raintree West.
8. Numbers may not add due to rounding.

Table 11-2: Mineral Resource Estimate and Sensitivity – Whistler Deposit

Class	Cutoff Value	ROM tonnage	In-situ Grades					In-situ metal			
	(US$/t)	(kt)	NSR (US$/t)	AuEq (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (Koz)	Au (koz)	Cu (klbs)	Ag (koz)
Indicated	11.25	302,235	37.17	0.539	0.392	0.150	1.7	5,236	3,804	1,000,136	16,519
	13.4	284,203	38.74	0.562	0.409	0.154	1.7	5,132	3,740	964,275	15,808
	15	270,340	40.00	0.580	0.424	0.157	1.7	5,039	3,684	933,927	15,123
	20	225,405	44.48	0.645	0.477	0.165	1.8	4,674	3,459	820,932	13,044
	25	183,491	49.51	0.718	0.539	0.173	1.9	4,235	3,179	699,024	10,914
	30	148,510	54.71	0.793	0.603	0.180	1.9	3,787	2,880	587,698	9,072
	40	98,786	64.84	0.940	0.729	0.192	2.0	2,985	2,314	417,931	6,447
	50	66,618	74.62	1.082	0.849	0.204	2.1	2,317	1,818	299,463	4,562
Inferred	11.25	5,186	37.28	0.540	0.421	0.113	1.2	90	70	12,942	195
	13.4	4,967	38.37	0.556	0.433	0.115	1.2	89	69	12,549	192
	15	4,747	39.48	0.572	0.446	0.116	1.2	87	68	12,171	189
	20	4,078	43.07	0.624	0.491	0.120	1.3	82	64	10,780	169
	25	3,453	46.84	0.679	0.538	0.124	1.4	75	60	9,440	151
	30	2,926	50.30	0.729	0.578	0.127	1.4	69	54	8,180	133
	40	1,936	58.23	0.844	0.682	0.125	1.6	53	42	5,335	97
	50	1,339	64.21	0.931	0.754	0.128	1.5	40	32	3,770	65

Table 11-3: Mineral Resource Estimate and Sensitivity – Raintree West Deposit

Class	Source	Cutoff Value	ROM tonnage	In-situ Grades					In-situ metal			
		(US$/t)	(kt)	NSR (US$/t)	AuEq (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (Koz)	Au (koz)	Cu (klbs)	Ag (koz)
Indicated	Open Pit	11.25	11,232	33.76	0.489	0.397	0.072	4.669	177	143	17729.8	1,686
		13.4	10,332	35.63	0.517	0.420	0.076	4.795	172	139	17,266	1,593
		15	9,786	36.83	0.534	0.434	0.078	4.837	168	137	16,828	1,522
		20	8,282	40.37	0.585	0.479	0.084	4.961	156	128	15,356	1,321
		25	6,775	44.37	0.643	0.530	0.091	5.093	140	116	13,562	1,109
		30	5,231	49.33	0.715	0.595	0.098	5.156	120	100	11,313	867
		40	3,002	60.32	0.875	0.744	0.109	5.282	84	72	7,201	510
		50	1,685	72.70	1.054	0.918	0.113	5.496	57	50	4,190	298
	UG	$40 shell	4,619	58.81	0.853	0.713	0.118	5.355	127	106	12,036	795
	Total	varies	12,901	46.97	0.68	0.56	0.10	5.10	282	233	27,392	2,116
Inferred	Open Pit	11.25	20,902	35.24	0.511	0.437	0.054	4.225	343	294	24,838	2,839
		13.4	18,780	37.83	0.548	0.471	0.057	4.310	331	285	23,641	2,602
		15	17,255	39.91	0.579	0.499	0.059	4.377	321	277	22,558	2,428
		20	13,324	46.60	0.676	0.589	0.066	4.499	289	252	19,475	1,927
		25	10,535	52.99	0.768	0.676	0.071	4.679	260	229	16,444	1,585
		30	7,681	62.50	0.906	0.809	0.076	4.700	224	200	12,903	1,161
		40	4,620	81.13	1.176	1.073	0.083	4.564	175	159	8,454	678
		50	2,965	101.44	1.471	1.357	0.092	4.979	140	129	6,007	475
	Underground	$40 shell	79,717	55.32	0.802	0.692	0.102	2.670	2,055	1,773	179,964	6,843
	Total	varies	93,041	54.07	0.784	0.677	0.097	2.93	2,345	2,025	199,439	8,770

Table 11-4: Mineral Resource Estimate and Sensitivity – Island Mountain Deposit

Class	Cutoff Value	ROM tonnage	In-situ Grades					In-situ metal			
	(US$/t)	(kt)	NSR (US$/t)	AuEq (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (Koz)	Au (koz)	Cu (klbs)	Ag (koz)
Inferred	11.25	214,181	26.94	0.391	0.35	0.04	0.86	2,689	2,388	196,902	5,922
	13.4	187,283	29.04	0.421	0.38	0.04	0.88	2,535	2,263	178,368	5,299
	15	170,368	30.52	0.442	0.40	0.04	0.90	2,423	2,170	166,389	4,930
	20	121,704	35.74	0.518	0.47	0.05	1.00	2,028	1,825	133,082	3,913
	25	84,868	41.56	0.603	0.54	0.06	1.12	1,644	1,482	106,274	3,056
	30	57,000	48.48	0.703	0.63	0.07	1.28	1,288	1,162	83,189	2,346
	40	29,504	61.85	0.897	0.81	0.08	1.55	851	769	53,792	1,470
	50	17,179	74.42	1.079	0.98	0.09	1.74	596	542	35,866	961

11.2 Key Assumptions and Data used in the Estimate

The total Whistler Project area comprises a database of 250 drillholes totalling more than 76,500 m with 267 drillholes within the three deposit block models and other targets in the Project.

A summary of the drillholes within each of the Whistler Project block model areas is provided in Table 11-5 For a summary of all drilling within the entire Whistler project please see Section 7 (Table 7-2) of this report.

Table 11-5: Summary of Whistler Project Drillhole Data within Block Models

Deposit	Year	# Drillholes	Total length (m)	Assayed Length (m)	% Assayed
Whistler	1988	13	1,306	1,220	93%
	1989	3	370	346	93%
	2004	5	1,997	1,865	93%
	2005	9	5,251	5,061	96%
	2006	1	705	696	99%
	2007	7	3,321	3,243	98%
	2008	5	2,462	2,419	98%
	2010	7	5,247	4,500	86%
	2023	2	1,632	1,624	100%
	2024	5	2,827	2,820	100%
	Total	**57**	**25,118**	**23,792**	**95%**

Deposit	Year	# Drillholes	Total length (m)	Assayed Length (m)	% Assayed
Raintree	2006	4	1,115	845	76%
	2008	2	622	615	99%
	2009	1	479	479	100%
	2010	7	2,890	2,807	97%
	2011	73	14,473	13,525	93%
	2024	2	1,216	1,212	100%
	Total	**89**	**20,796**	**19,483**	**94%**
Island Mountain	2006	2	269	267	99%
	2009	2	601	601	100%
	2010	12	5,434	5,396	99%
	2011	26	9,537	9,440	99%
	Total	**42**	**20,796**	**19,483**	**94%**
Total Resource		**188**	**66,710**	**62,759**	**94%**

11.3 Geologic Modelling

Three-dimensional wireframe solids based on geology have been used to constrain the grade interpolations.

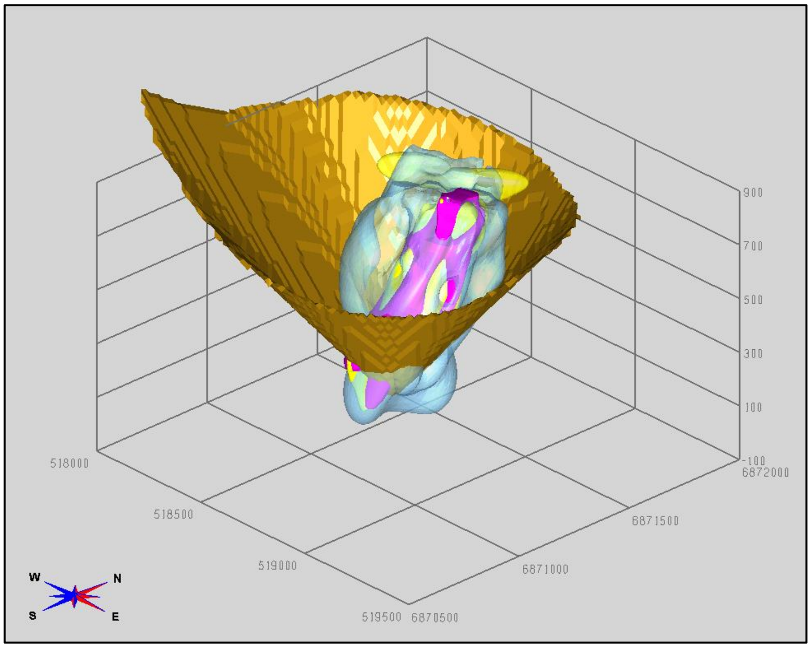
At Whistler, three-dimensional solids of the productive and non-productive diorite intrusions have been created by the site geologists. These were used as a guide to create mineralization shells for both gold-silver and copper mineralization envelopes separately. The shells include a higher-grade “core” with a lower-grade “halo” for both gold-silver shells and for copper shells.

Dykes have not been modelled explicitly because they are too thin both to model and to separate when mining. Therefore, the unmineralized assays within the solids have been included in the interpolations. A three-dimensional view looking northwest of the Whistler higher-grade core domains (gold-silver high-grade domain in red, with copper higher-grade core in yellow) is illustrated in Figure 11-1, also showing the resource pit.

Figure 11-2 illustrates the mineralized domain for Raintree West, looking northeast and plotting the resource pit and underground mineralized shape.

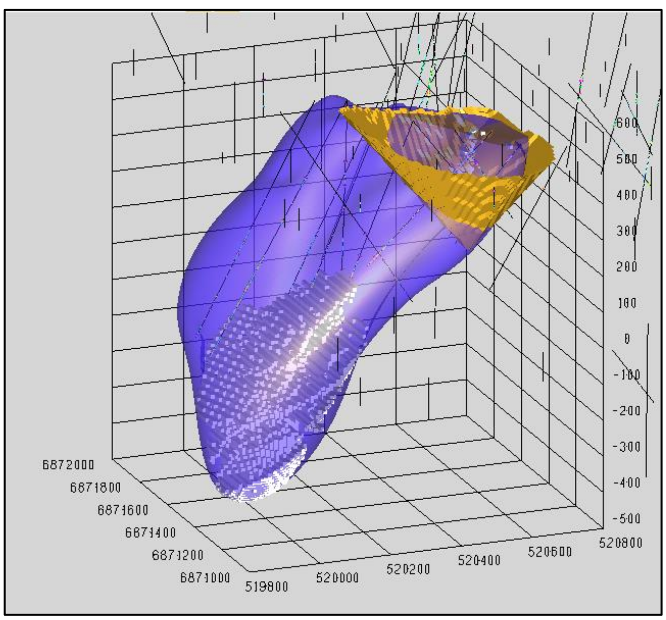
Figure 11-3 illustrates the domains for Island Mountain. There are six subvertical domains (plotted in shades of blue) that are based on lithology as various mineralized dykes. These were combined into one domain for the interpolations. Two domains surrounding the central core at a nominal cutoff of 0.1 g/t and 0.3 g/t AuEq are used to confine the interpolation outside of the dyke boundaries (plotted in yellows). The outline of the resource pit on surface is also plotted for reference.

Figure 11-1: Whistler Deposit – Au-Ag high-grade domain (magenta), Cu high-grade domain (yellow) lower-grade halo (blue)



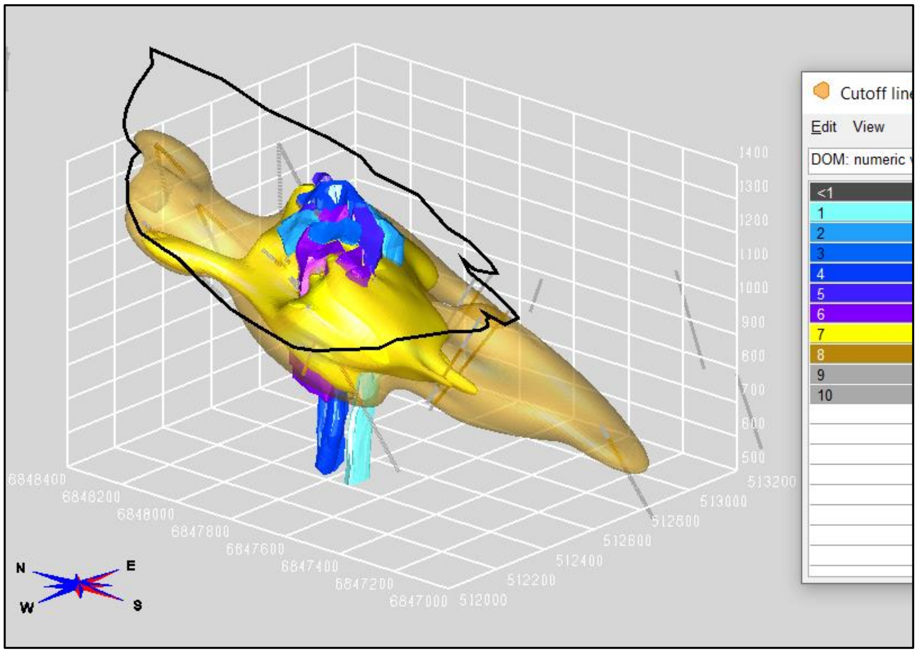
Source: MMTS, 2024

Figure 11-2: Domains Modeled for Raintree West Deposit



Source: MMTS, 2021

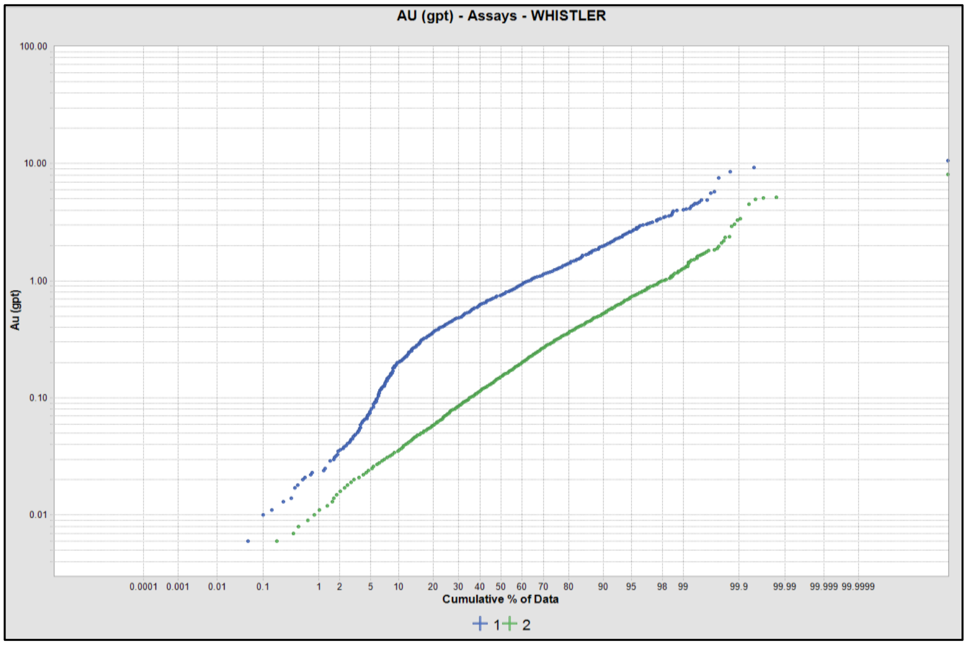
Figure 11-3: Domains Modelled for Island Mountain Capping



Source: MMTS, 2021

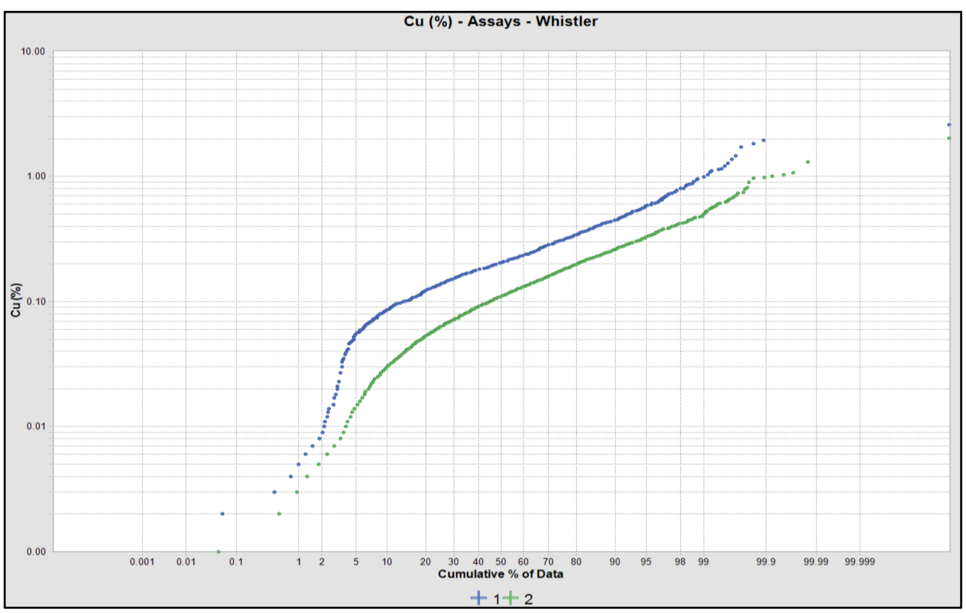
Cumulative probability plots (CPP) are used to define capping values and potential outlier restrictions during interpolations. Figure 11-4 and Figure 11-5 show the CPP plots for gold and copper, respectively for Whistler. Figure 11-6 and Figure 11-7 show the CPP plots for gold and copper respectively for Raintree West and Figure 11-8 and Figure 11-9 are the CPPs for Island Mountain for gold and copper respectively.

Figure 11-4: CPP of Au Assay Data by Domain - Whistler



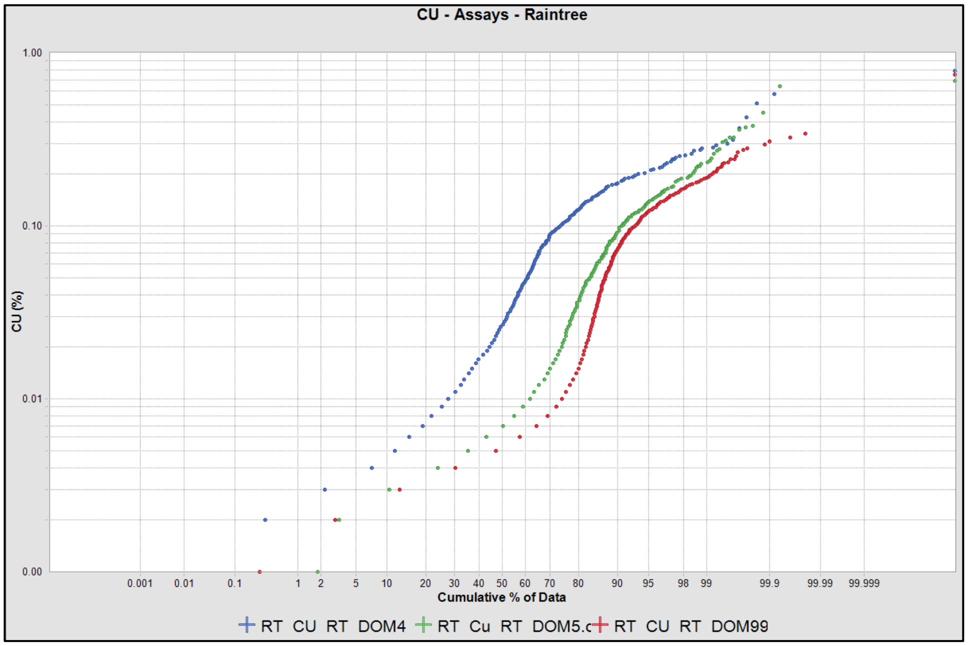
Source: MMTS, 2024

Figure 11-5: CPP of Cu Assay Data by Domain – Whistler



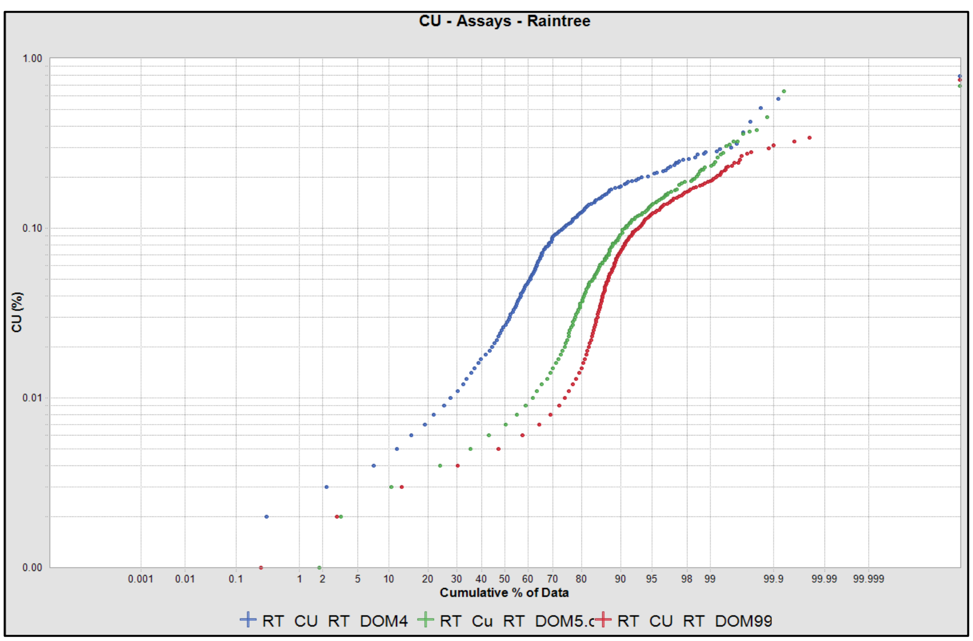
Source: MMTS, 2024

Figure 11-6: CPP of Au Assay Data by Domain – Raintree



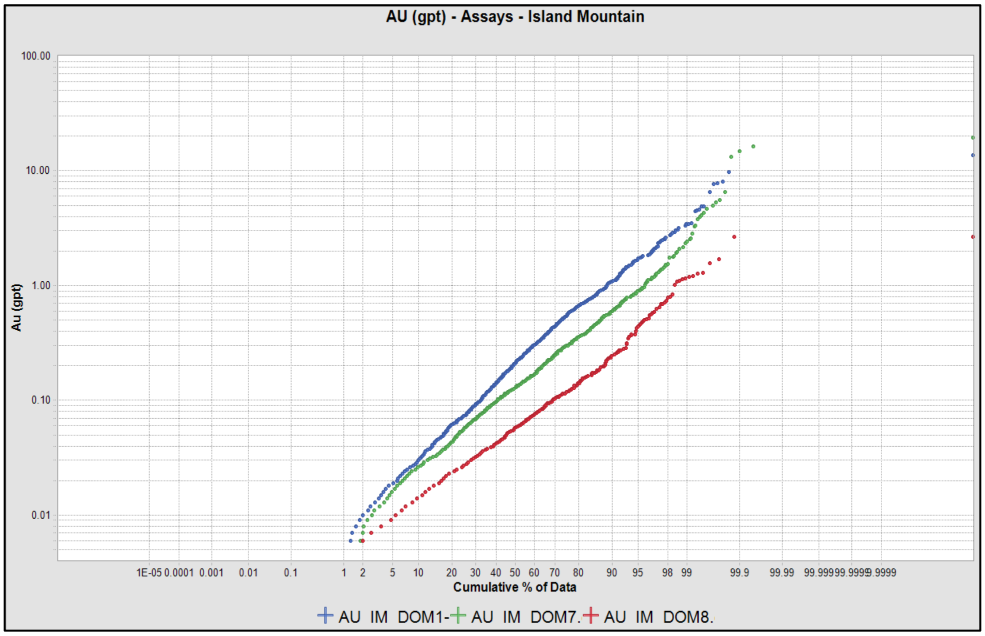
Source: MMTS, 2021

Figure 11-7: CPP of Cu Assay Data by Domain – Raintree West



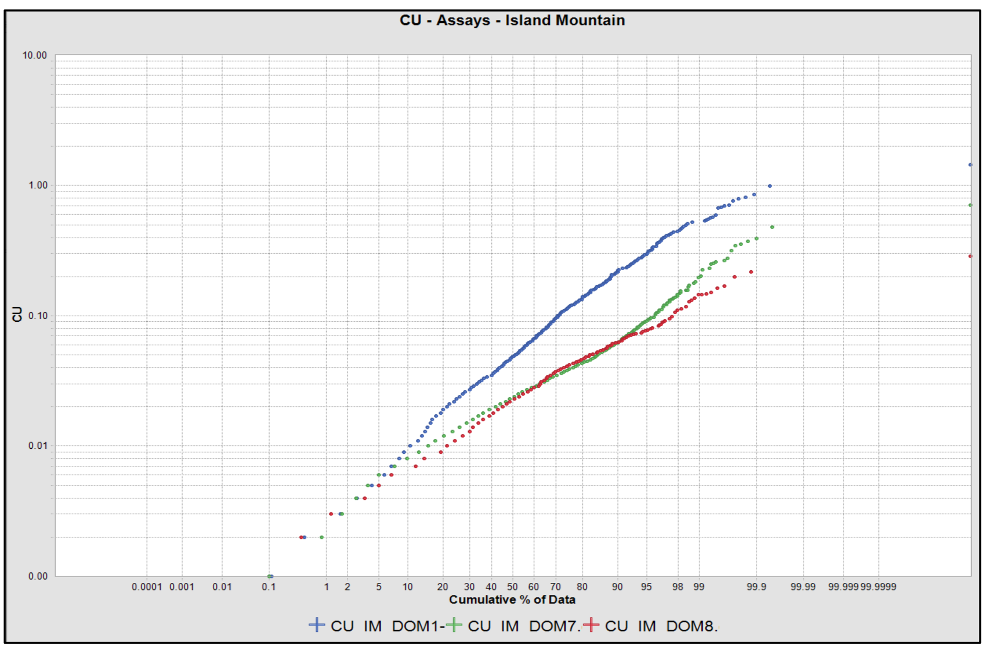
Source: MMTS, 2021

Figure 11-8: CPP of Au Assay Data by Domain – Island Mountain



Source: MMTS, 2021

Figure 11-9: CPP of Cu Assay Data by Domain – Island Mountain



Source: MMTS, 2021

Capping and Outlier values are summarized in Table 11-6. Values above the capping value are equal to the capping value in the assay file prior to compositing. Composite values above the Outlier value are restricted during interpolations to the Outlier value for distance greater than 5 m from the composite interval.

Table 11-6: Summary of Capping and Outlier Restriction Values

Item	Area	Domain	CAP	Outlier
Au (g/t)	Whistler	1	6	2 (pass 1-2)
				1.2 (pass 3-5)
		2	5	4
	Raintree	1	2	10
	Island Mountain	1-6	10	5
		7	10	5
		8	3	5
Cu (%)	Whistler	1	2	0.9
		2	1	0.7
		3	1	0.5 (pass 1)
				0.4 (pass 2-4)
	Raintree	2	2	0.6
	Island Mountain	1-6	1	na
		7	0.6	na
		8	0.3	na
Ag (g/t)	Whistler	1	100	50
		2	100	80
	Raintree	1	100	80
	Island Mountain	1-6	30	12
		7	20	7
		8	20	7

The capped assay and composite statistics of each domain are summarized in the Table 11-7 through Table 11-9 for Au, Cu and Ag, respectively. These tables illustrate that no significant bias has been introduced during the compositing process. They also indicate that the distributions have low CV confirming the choice of linear interpolation methods are appropriate.

Table 11-7: Capped Assay and Composite Statistics by Domain – Au

Source	Parameters	Whistler		Raintree West	Island Mountain		
		1	2	5	6-7	7	8
Assays	Num Samples	2,652	7,626	2,731	1,795	1,999	767
	Num Missing	3	17	1	12	0	1
	Min (g/t)	0.002	0.002	0.003	0.003	0.003	0.003
	Max (g/t)	6.000	8.070	14.150	10.000	10.000	2.660
	Wtd mean (g/t)	0.813	0.234	0.260	0.452	0.253	0.122
	Wtd CV	0.980	1.383	2.067	1.746	2.187	1.899
Composites	Num Samples	813	2,342	1,305	841	917	411
	Num Missing	0	0	1	0	0	0
	Min (g/t)	0.007	0.002	0.003	0.003	0.003	0.004
	Max (g/t)	4.712	4.008	6.068	6.412	4.626	1.167
	Wtd mean (g/t)	0.813	0.234	0.260	0.452	0.253	0.122
	Wtd CV	0.854	1.165	1.562	1.447	1.570	1.409
Difference in Wtd. Means (%)		0.0%	0.0%	0.0%	0.0%	0.0%	0.0%

Table 11-8: Capped Assay and Composite Statistics by Domain – Cu

Source	Parameters	Whistler			Raintree West	Island Mountain		
		1	2	3	5	6-7	7	8
Assays	Num Samples	2,276	286	8,001	2,731	1,795	1,999	767
	Num Missing	0	0	0	1	12	0	1
	Min (g/t)	0.000	0.000	0.000	0.000	0.000	0.000	0.001
	Max (g/t)	2.000	1.000	1.000	0.786	1.000	0.600	0.288
	Wtd mean (g/t)	0.236	0.303	0.117	0.037	0.083	0.032	0.030
	Wtd CV	0.770	0.545	0.834	1.623	1.271	1.160	0.912
Composites	Num Samples	634	86	2,435	1,305	841	917	411
	Num Missing	0	0	0	1	0	0	0
	Min (g/t)	0.003	0.088	0.000	0.000	0.001	0.001	0.003
	Max (g/t)	1.157	0.832	0.832	0.317	0.654	0.397	0.223
	Wtd mean (g/t)	0.236	0.303	0.117	0.037	0.083	0.032	0.030
	Wtd CV	0.640	0.432	0.703	1.489	1.124	0.998	0.826
Difference in Wtd. Means (%)		0.0%	0.0%	0.1%	0.0%	0.0%	0.0%	0.0%

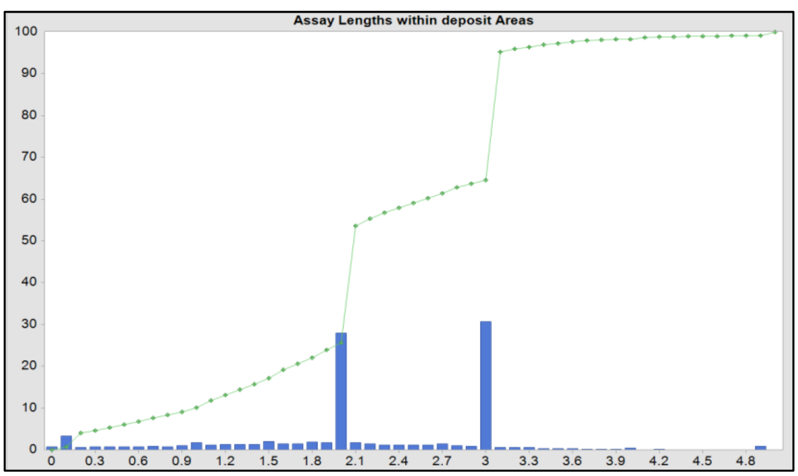
Table 11-9: Capped Assay and Composite Statistics by Domain – Ag

Source	Parameters	Whistler		Raintree West	Island Mountain		
		1	2	5	6-7	7	8
Assays	Num Samples	2,644	7,626	2,731	1,795	1,999	767
	Num Missing	11	17	1	12	0	1
	Min (g/t)	0.050	0.050	0.250	0.250	0.250	0.250
	Max (g/t)	100.000	186.000	200.000	30.000	20.000	14.700
	Wtd mean (g/t)	1.977	1.625	3.305	1.649	0.709	0.627
	Wtd CV	1.851	2.692	2.337	1.339	1.556	1.420
Composites	Num Samples	812	2,342	1,305	841	917	411
	Num Missing	1	0	1	0	0	0
	Min (g/t)	0.050	0.050	0.250	0.250	0.250	0.250
	Max (g/t)	33.468	85.292	83.468	11.180	5.198	3.812
	Wtd mean (g/t)	1.982	1.625	3.305	1.616	0.684	0.602
	Wtd CV	1.203	1.956	1.680	1.028	0.965	0.868
Difference in Wtd. Means (%)		-0.9%	0.2%	0.0%	-2.1%	-3.7%	-4.3%

11.4 Compositing

Compositing of Au, Ag and Cu grades have been done as 5 m fixed length composites. Small intervals less than 2.5 m are merged with the up-hole composite if the composite length is less than 5 m. The length of 5 m is chosen to be half the size of the block height, and longer than most assay lengths, as illustrated in Figure 11-10. Domain boundaries are honored during compositing.

Figure 11-10: Histogram of Assay Lengths



Source: MMTS, 2024

11.5 Variography

Correlograms have been created for each domain within each deposit. A summary of the spherical correlogram parameters is given in Table 11-10 through Table 11-12 for Whistler, Raintree West, and Island Mountain respectively.

Table 11-10: Variogram Parameters – Whistler

Element	Domain	Rotation (GSLIB-MS)		Axis	Total Range (m)	Nugget	Sill1	Sill2	Sill3	Range 1 (m)	Range 2 (m)	Range 3 (m)
CU	1	ROT	330	Major	300	0.1	0.3	0.3	0.3	50	100	300
		DIPN	0	Minor	350					80	160	350
		DIPE	70	Vert	80					15	30	80
	2	ROT	60	Major	180	0.3	0.3	0.3	0.1	50	120	180
		DIPN	-10	Minor	100					60	80	100
		DIPE	0	Vert	100					60	80	100
	3	ROT	310	Major	350	0.2	0.3	0.3	0.2	50	120	350
		DIPN	0	Minor	250					30	80	250
		DIPE	0	Vert	280					15	120	280
AU	1	ROT	330	Major	360	0.1	0.4	0.3	0.2	30	100	360
		DIPN	0	Minor	350					80	160	350
		DIPE	70	Vert	60					15	30	60
	3	ROT	310	Major	220	0.2	0.3	0.3	0.2	40	160	220
		DIPN	0	Minor	180					30	100	180
		DIPE	0	Vert	220					15	100	220
AG	1	ROT	20	Major	80	0.6	0.3	0.1	-	50	80	-
		DIPN	0	Minor	80					50	80	-
		DIPE	0	Vert	80					50	80	-
	3	ROT	0	Major	120	0.5	0.4	0.1	-	50	120	-
		DIPN	0	Minor	120					50	120	-
		DIPE	0	Vert	60					25	60	-

Table 11-11: Variogram Parameters – Raintree West

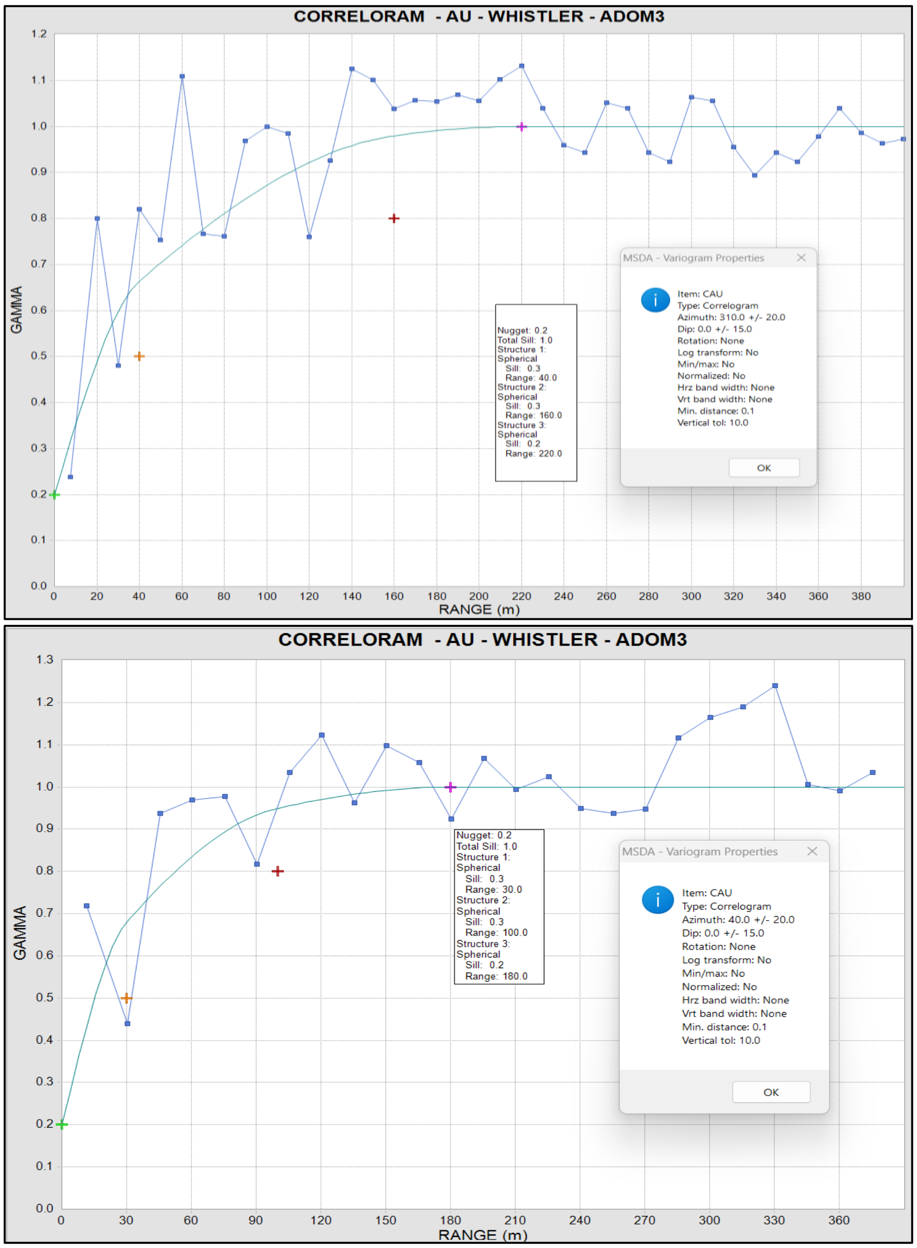
Element	Domain	Rotation (GSLIB-MS)		Axis	Total Range (m)	Nugget	Sill1	Sill2	Sill3	Range 1 (m)	Range 2 (m)	Range 3 (m)
CU	5	ROT	90	Major	500	0.1	0.4	0.4	0.1	200	300	500
		DIPN	55	Minor	350					40	200	350
		DIPE	0	Vert	300					80	200	300
AU	5	ROT	90	Major	500	0.2	0.3	0.2	0.3	50	250	500
		DIPN	55	Minor	350					30	150	350
		DIPE	0	Vert	150					20	80	150
AG	5	ROT	90	Major	140	0.2	0.4	0.4	-	20	140	-
		DIPN	55	Minor	120					15	120	-
		DIPE	0	Vert	120					15	120	-

Table 11-12: Variogram Parameters – Island Mountain

Element	Domain	Rotation (GSLIB-MS)		Axis	Total Range (m)	Nugget	Sill1	Sill2	Sill3	Range 1 (m)	Range 2 (m)	Range 3 (m)
CU	1-6	ROT	0	Major	300	0.2	0.5	0.1	0.2	40	150	300
		DIPN	-90	Minor	150					60	100	150
		DIPE	0	Vert	120					20	80	120
	7,8	ROT	25	Major	150	0.1	0.3	0.3	0.3	50	80	150
		DIPN	0	Minor	150					30	80	150
		DIPE	-20	Vert	120					30	35	120
AU	1-6	ROT	0	Major	200	0.3	0.4	0.2	0.1	50	140	200
		DIPN	-90	Minor	150					50	80	150
		DIPE	0	Vert	100					20	50	100
	7,8	ROT	25	Major	100	0.2	0.4	0.3	0.1	50	80	100
		DIPN	0	Minor	150					40	90	150
		DIPE	-20	Vert	100					15	70	100
AG	1-6	ROT	0	Major	150	0.3	0.4	0.3	-	30	150	-
		DIPN	-90	Minor	100					20	100	-
		DIPE	0	Vert	100					20	100	-
	7,8	ROT	25	Major	150	0.1	0.6	0.3	-	50	150	-
		DIPN	0	Minor	160					30	160	-
		DIPE	-20	Vert	75					15	75	-

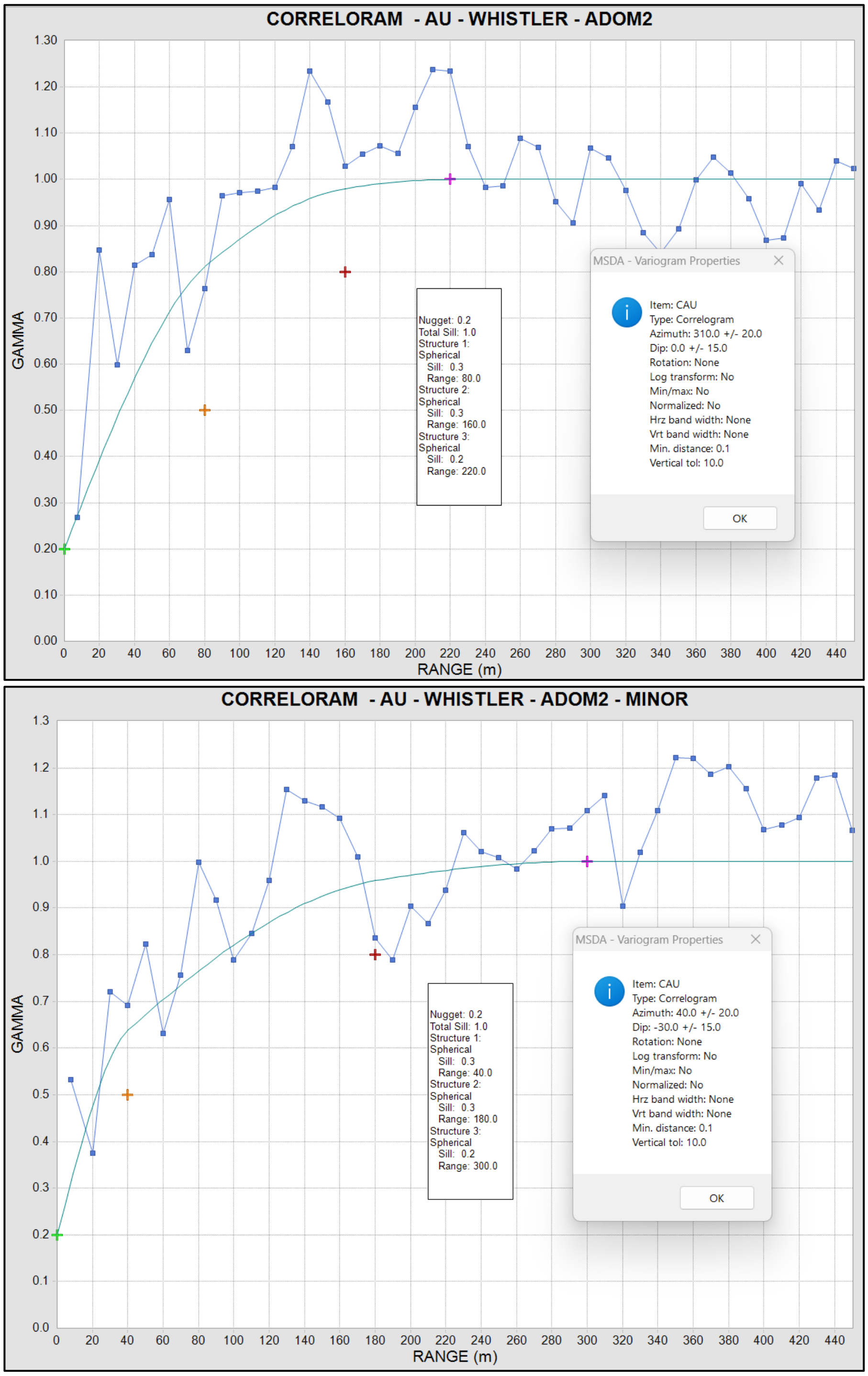
An example of the Variogram Model for Cu in Domain 1 in the major and minor axes directions is illustrated in Figure 11-11 for copper and Figure 11-12 for gold in the Whistler Deposit. Figure 11-13 is the variograms for copper at Raintree West in Domain 5, and Figure 11-14 illustrates the variogram for Island Mountain for the major and minor axes for Au.

Figure 11-11: Variogram Model for Cu Lower Grade Halo – Major and Minor Axes – Whistler Deposit



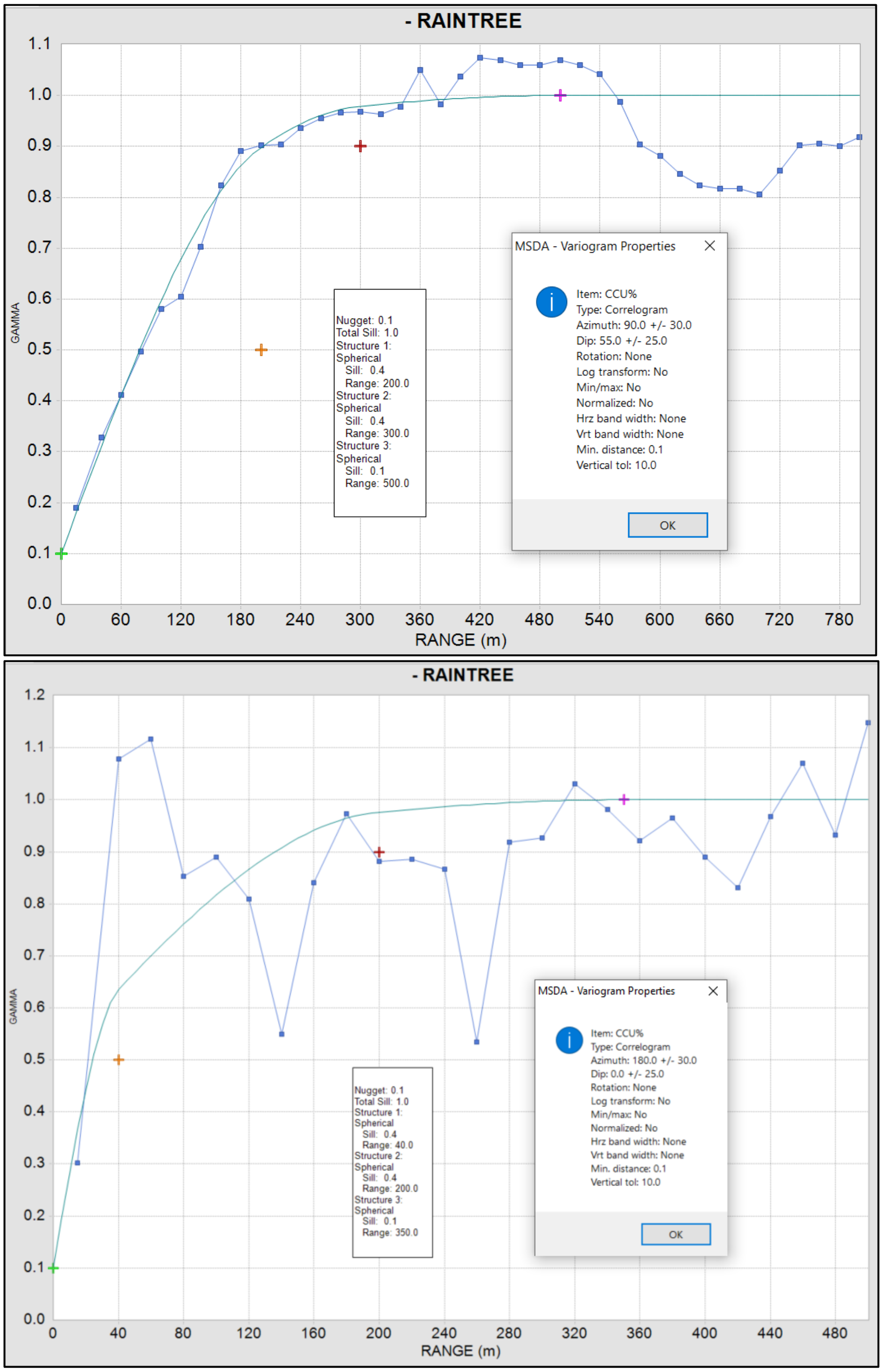
Source: MMTS, 2024

Figure 11-12: Variogram Model for Au in Domain 1 – Major and Minor Axes – Whistler Deposit



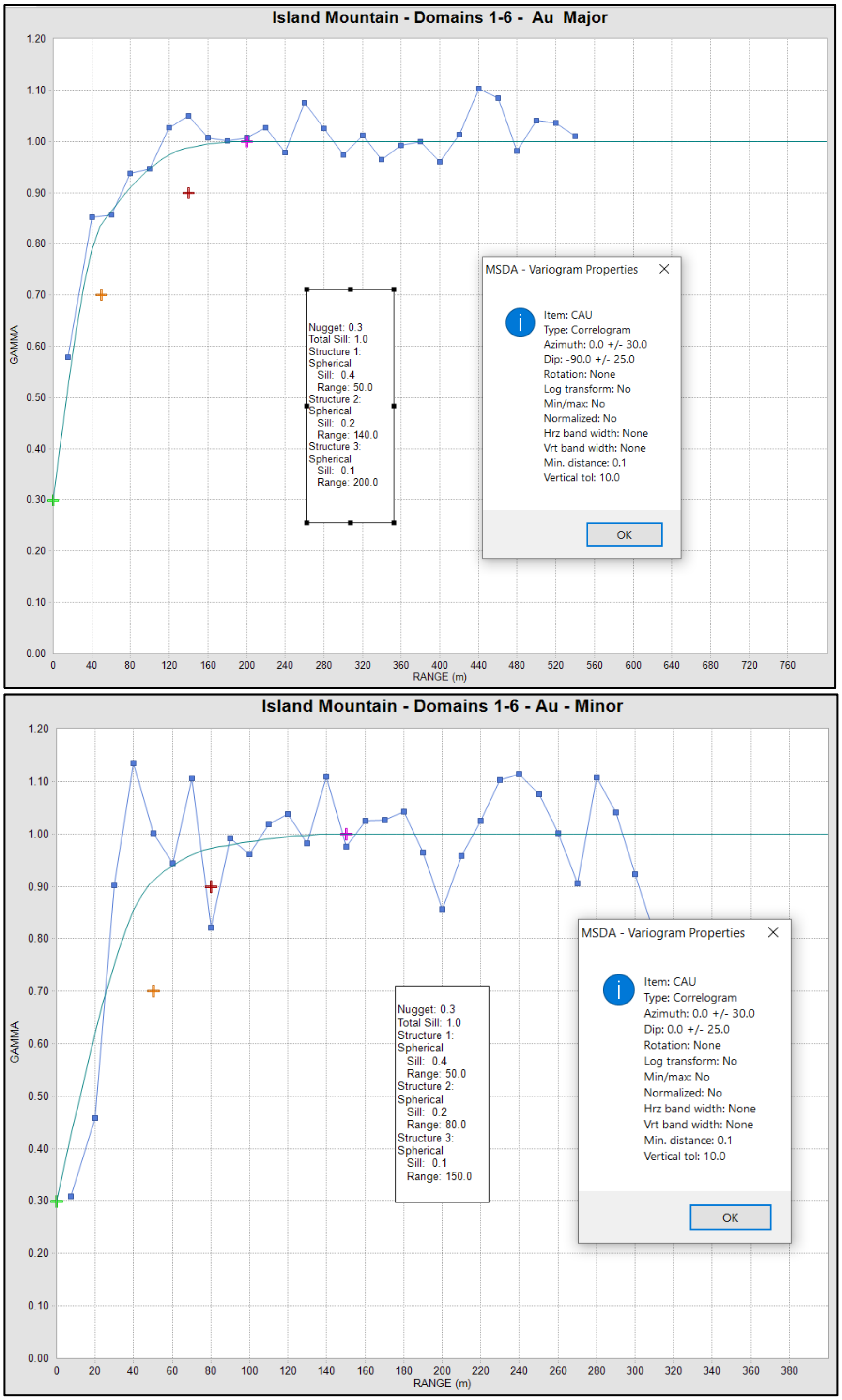
Source: MMTS, 2024

Figure 11-13: Variogram Model for Cu in Domain 5 – Major and Minor Axes – Raintree West Deposit



Source: MMTS, 2024

Figure 11-14: Variogram Model for Au in Domains 1-6 – Major and Minor Axes – Island Mountain Deposit



Source: MMTS, 2024

11.6 Block Model Interpolations

The block model limits and block size for each deposit are as given in Table 11-13.

Table 11-13: Block Model Limits

Deposit	Direction	From	To	Block size	# Blocks
Whistler	East	517,200	519,860	20	133
	North	6,870,000	6,873,000	20	150
	Elevation	-50	1,280	10	133
Raintree West	East	519,700	521,100	10	140
	North	6,871,000	6,872,000	10	100
	Elevation	-260	730	10	99
Island Mountain	East	511,500	513,600	10	210
	North	6,847,000	6,848,400	10	140
	Elevation	490	1,470	10	98

Interpolation of Au, Cu and Ag values is done by ordinary kriging (OK) in four passes based on the variogram parameters. Interpolations used hard boundaries, with composites and block codes required to match within each domain. Search parameters are summarized in Table 11-14 through Table 11-16.

Table 11-14: Search Rotation and Distances – Whistler

Element	Domain	Rot	Dist1	Dist 2	Dist3	Dist4	Dist5
CU	**1**	330	50	100	200	300	600
		0	80	160	263	350	700
		70	15	30	60	80	160
	2	60	45	90	135	180	360
		-10	25	50	75	100	200
		0	25	50	75	100	200
	3	310	50	100	200	350	700
		0	30	60	120	250	500
		0	15	30	60	280	560
AU	**1**	330	30	60	120	360	720
		0	80	160	263	350	700
		70	15	30	45	60	120
	3	310	55	73	110	220	440
		0	45	60	90	180	360
		0	55	73	110	220	440
AG	**1**	20	20	40	60	80	160
		0	20	40	60	80	160
		0	20	40	60	80	160
	3	0	30	40	60	120	240
		0	30	40	60	120	240
		0	15	20	30	60	120

Table 11-15: Search Rotation and Distances – Raintree West

Element	Domain	Rot	Dist1	Dist2	Dist3	Dist4
CU	1	90	125	250	375	500
		55	88	175	263	350
		0	75	150	225	300
AU	1	90	125	250	375	500
		55	88	175	263	350
		0	38	75	113	150
AG	1	90	35	70	105	140
		55	30	60	90	120
		0	30	60	90	120

Table 11-16: Search Rotation and Distances – Island Mountain

Element	Domain	Rot	Dist1	Dist2	Dist3	Dist4
CU	1-6	0	40	80	160	300
		-90	37.5	75	112.5	150
		0	20	40	80	120
	7,8	25	37.5	75	112.5	150
		0	30	60	112.5	150
		-20	30	60	90	120
AU	1-6	0	50	100	150	200
		-90	37.5	75	112.5	150
		0	20	40	75	100
	7,8	25	25	50	75	100
		0	37.5	75	112.5	150
		-20	15	30	60	100
AG	1-6	0	30	60	112.5	150
		-90	20	40	75	100
		0	20	40	75	100
	7,8	25	37.5	75	112.5	150
		0	30	60	120	160
		-20	15	30	56.25	75

Additional search criteria on composite selection are summarized in Table 11-17. Search criteria are used to ensure that more than one drillhole is used for all passes, and more than one quadrant is used for the first three passes, as well as to limit smoothing of grade by limiting the maximum number of composites to be used.

Table 11-17: Additional Search Criteria

Criteria	Pass 1	Pass 2	Pass 3	Pass 4
Minimum # composites	3	3	3	3
Maximum # Composites	12	12	12	12
Maximum/drillhole	2	2	2	2
Maximum/quadrant	2	2	2	na

11.7 Classification

Classification has been done in accordance with 229.1302(d)(1)(iii)(A) (Item 1302(d)(1)(iii)(A) of Regulation S-K. The Classification is based on the variogram parameters, with the required average distance to the nearest two drillholes required to be less than the distance of the range at 80% of the sill (R_{80} value) for each domain as summarized in Table 11-18.

Table 11-18: Classification Criteria

Deposit	Whistler		Raintree West		Island Mountain	
Domain	1	2	5	99	1-6	7-8
Average Distance to 2 DHs	120	120	100	100	80	80
Distance to furthest DH	170	170	na	na	na	na

11.8 Block Model Validation

11.8.1 Comparison of Tonnage and Grades

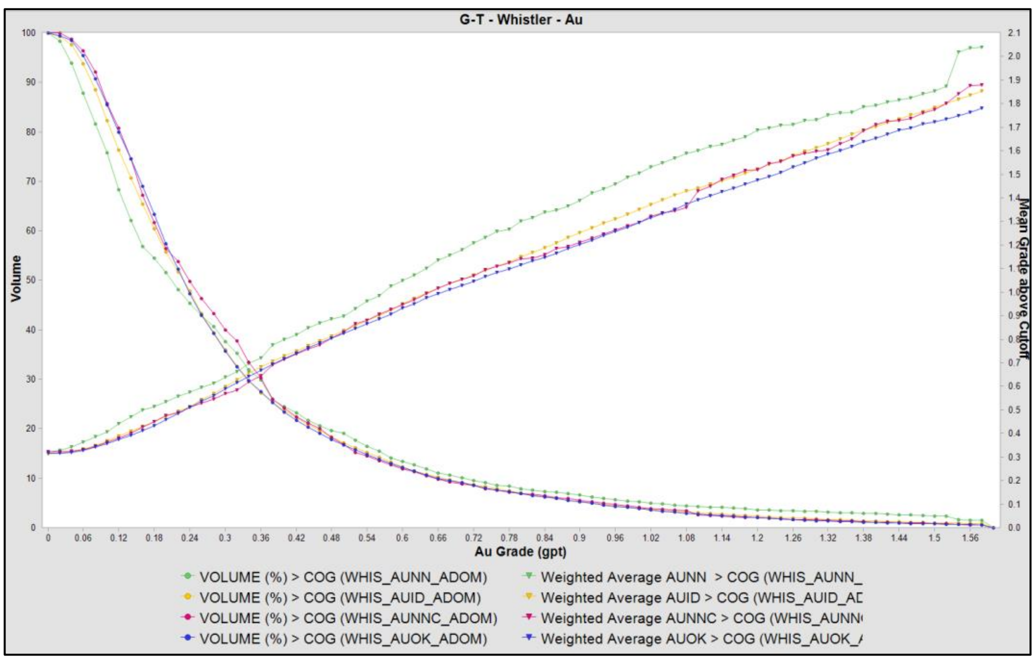
Interpolations have also been completed using a Nearest Neighbour method to essentially de-cluster the composite data for grade comparisons with the modelled grades. Table 11-19 gives a summary of the mean grades for de-clustered composites (NN interpolation), and OK grades at a 0.1% Cu and 0.1 g/t Au cutoff. The tonnage grade and metal content are variable, but generally conservative compared to the de-clustered composites. There is significant variability between the OK and NN model depending on cutoff used for the comparison.

This comparison is illustrated more succinctly in the plots of tonnage-grade curves. Cutoff grade plots (tonnage-grade curves) are constructed for each metal to check the validity of the modelling. The NN values for Au and Cu are plotted and compared to the modelled OK values for the Whistler Deposit in Figure 11-15 and Figure 11-16. For Raintree West, the tonnage-grade curves for Au and Cu are presented in Figure 11-17 and Figure 11-18, and for Island Mountain the tonnage, grade curves are presented in Figure 11-19 and Figure 11-20. The curves for Whistler and Island Mountain are within the Resource confining pit shape. For Raintree West, all blocks within modelled domains are plotted due to the underground component of the resource. In each case, the distributions show good correlation, and thus the change of support is valid.

Table 11-19: Comparison of De-clustered Composite and OK Modelled Grades

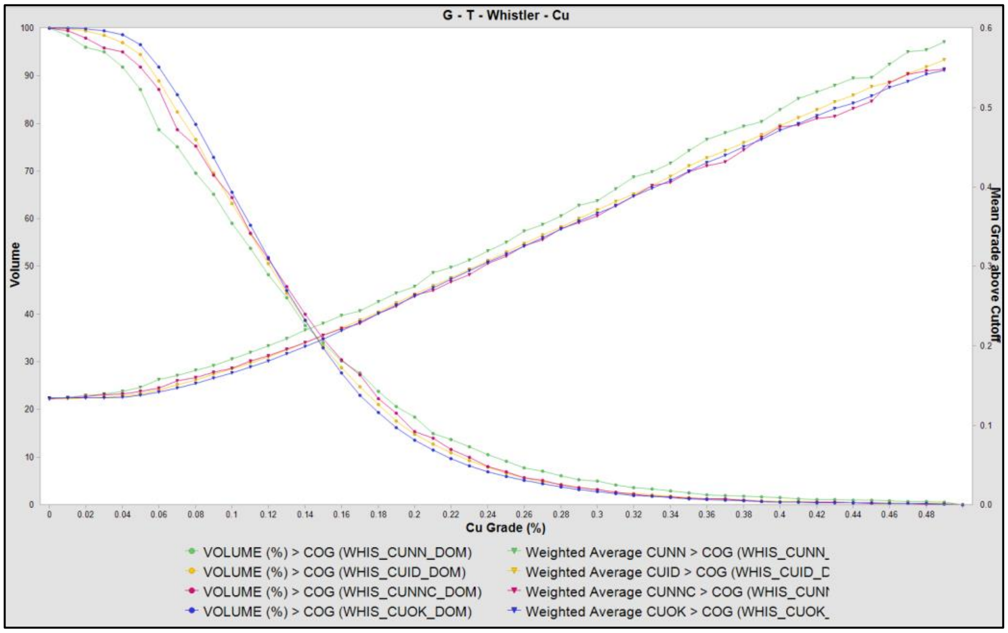
Cutoff Au (g/t)	Class	Deposit	Modelled OK			De-clustered composites (NN)			Difference (%)
			ROM Tonnage (kt)	Grade Au (g/t)	Metal (koz)	ROM Tonnage (kt)	Grade Au (g/t)	Metal (koz)	
0.1	**Cu**	Whistler	238,501	0.172	904,375	214,060	0.191	899,473	1%
		Raintree	3,981	0.136	11,910	5,013	0.154	16,986	-30%
		Island Mtn.	17,323	0.149	56,751	16,857	0.183	68,008	-17%
0.1	**Au**	Whistler	363,719	0.340	3,977	363,719	0.337	3,944	1%
		Raintree	36,156	0.389	309,990	27,803	0.476	291,517	6%
		Island Mtn.	261,880	0.312	1,803,612	189,077	0.451	1,880,778	-4%

Figure 11-15: Tonnage-Grade Curves for Au – Comparison of Interpolation Methods – Whistler



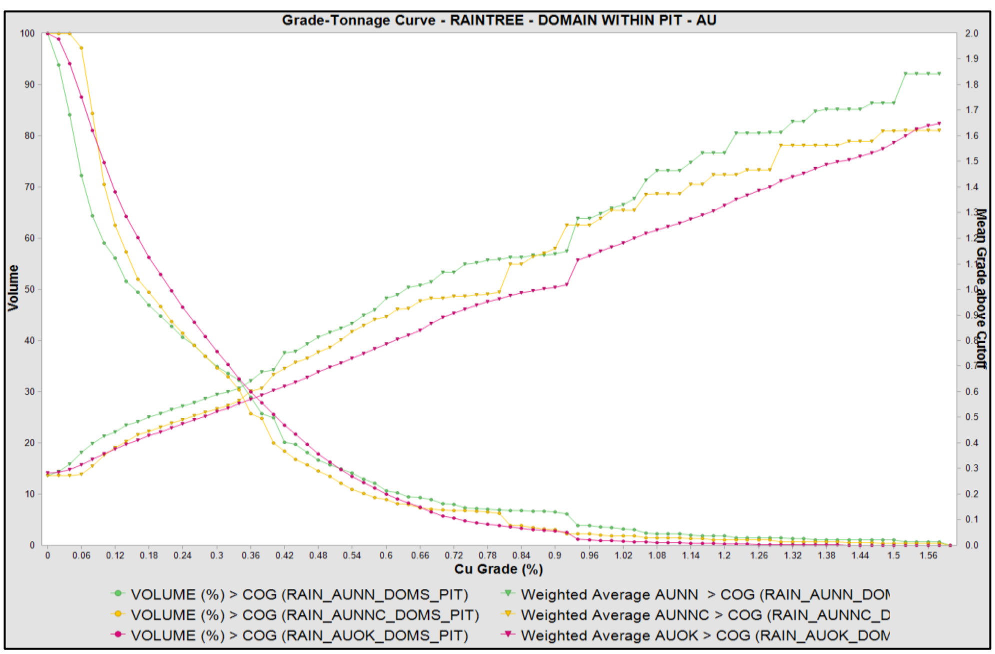
Source: MMTS, 2026

Figure 11-16: Tonnage-Grade Curves for Cu – Comparison of Interpolation Methods - Whistler



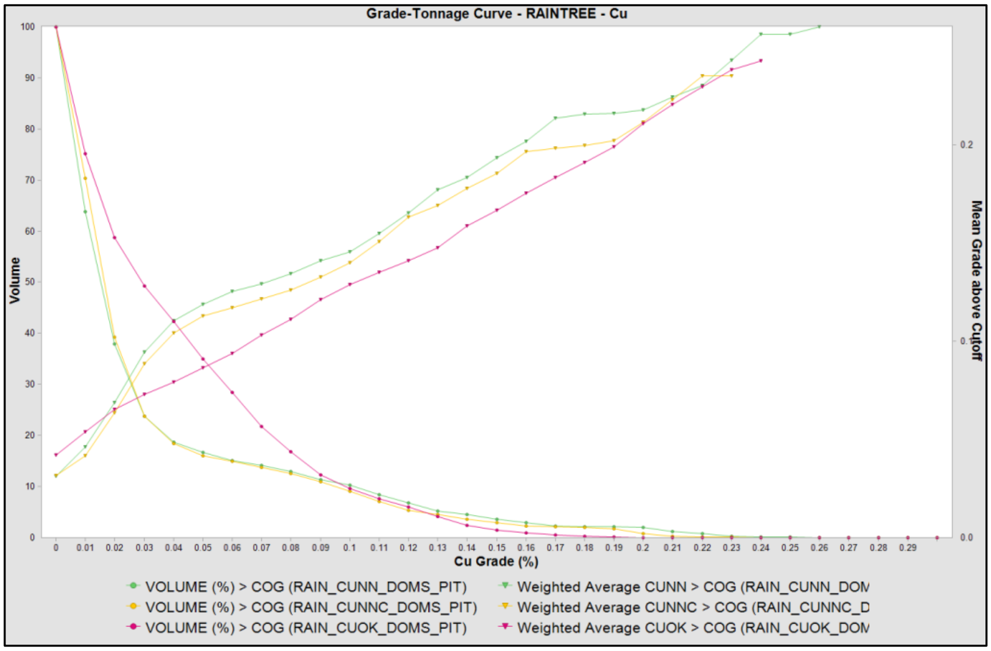
Source: MMTS, 2026

Figure 11-17: Tonnage-Grade Curves for Au – Comparison of Interpolation Methods – Raintree West



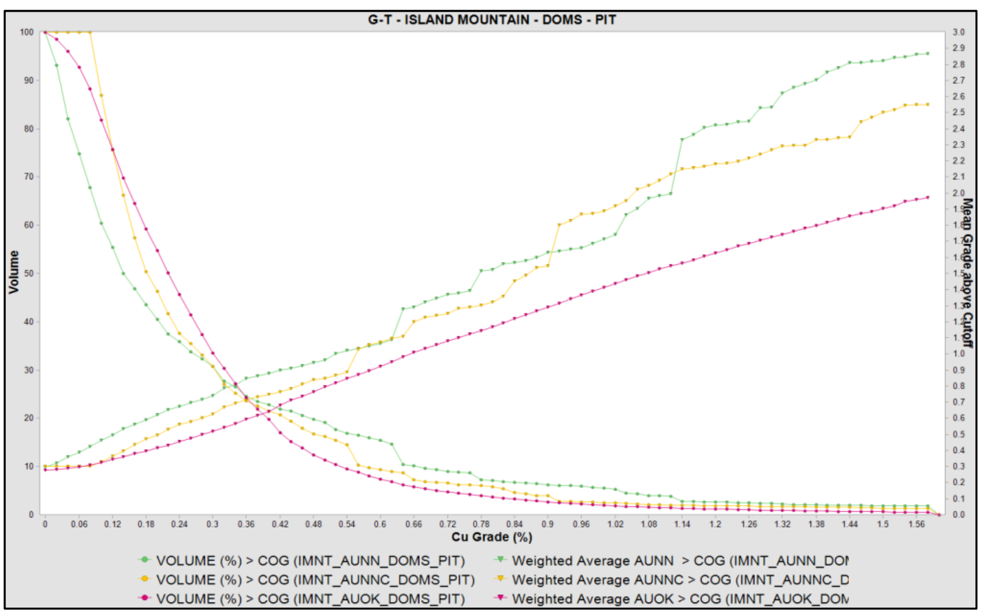
Source: MMTS, 2026

Figure 11-18: Tonnage-Grade Curves for Cu – Comparison of Interpolation Methods – Raintree West



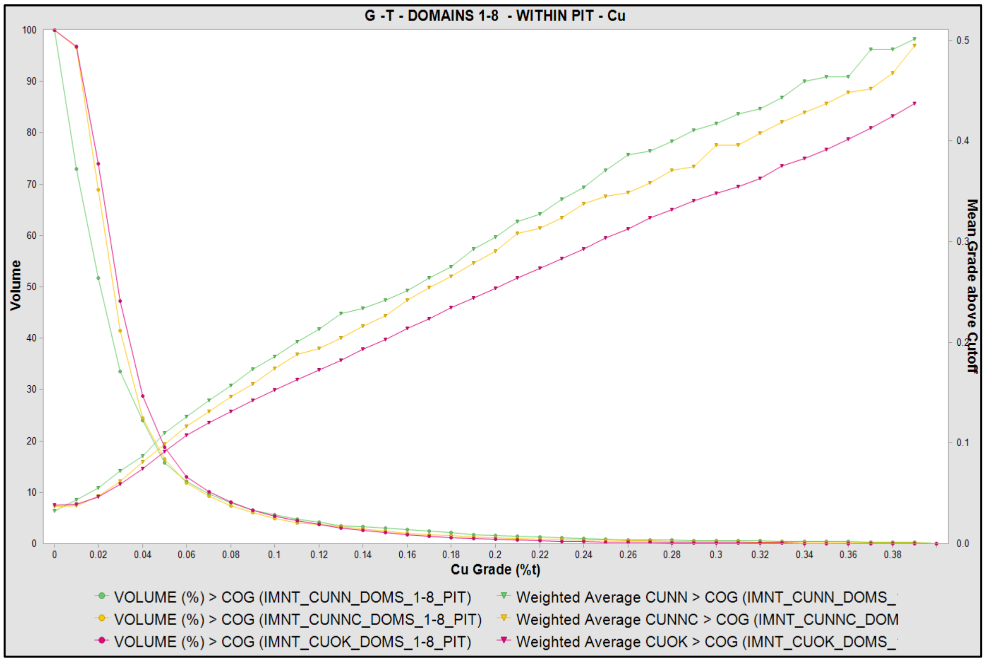
Source: MMTS, 2026

Figure 11-19: Tonnage-Grade Curves for Au – Comparison of Interpolation Methods – Island Mountain



Source: MMTS, 2026

Figure 11-20: Tonnage-Grade Curves for Cu – Comparison of Interpolation Methods - Island Mountain



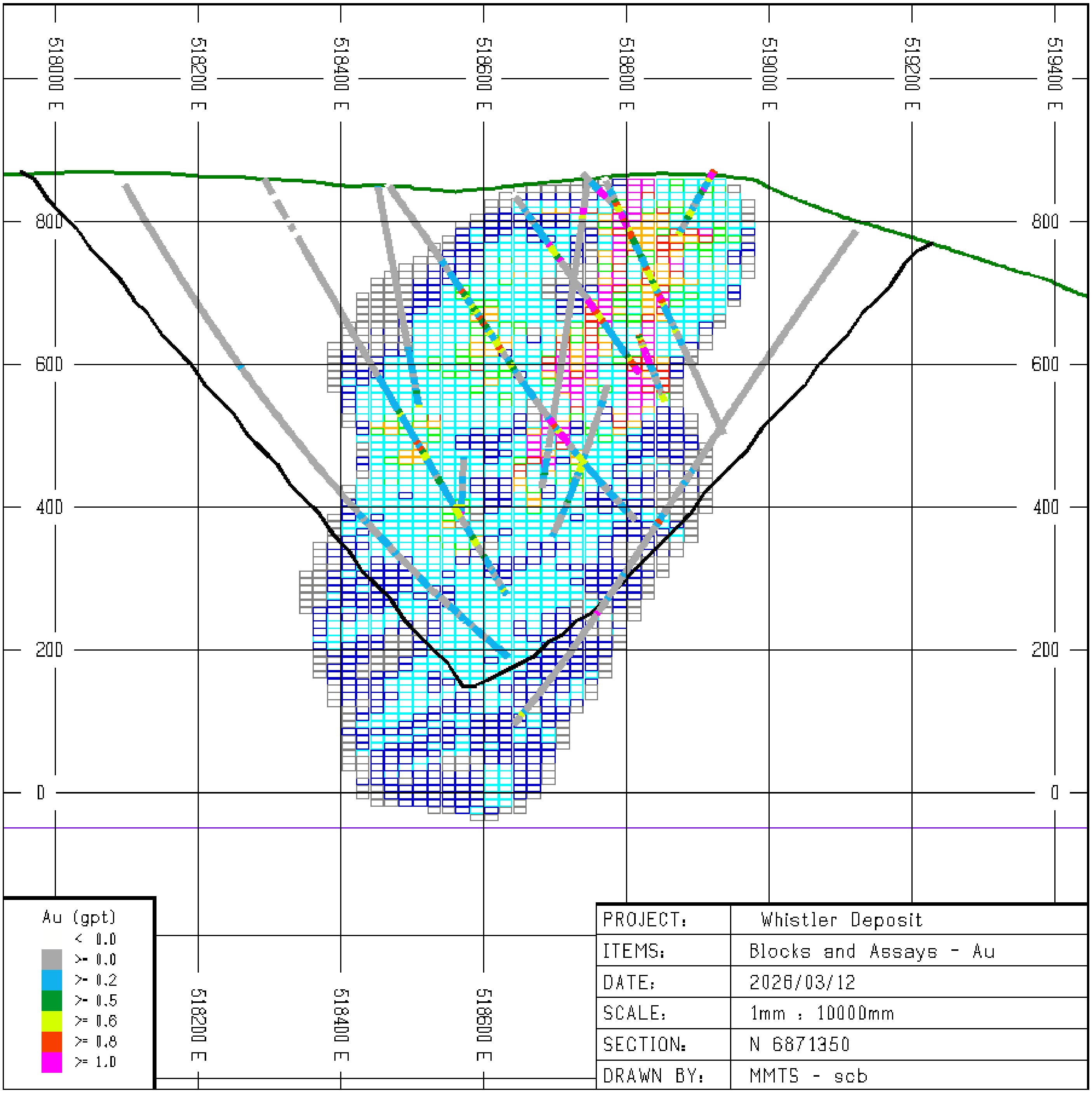
Source: MMTS, 2026

11.9 Visual Validation

A series of E-W, N-S sections (every 20 m) and plans (every 10 m) have been used to inspect the OK block model grades with the original assay data. Figure 11-21 and Figure 11-22 give examples of this comparison at Whistler for the E-W section at 6,871,330 N, for gold and copper grades respectively. Figure 11-23 and Figure 11-24 illustrate the grade comparisons at Raintree West through the center of the deposit looking southwest (SW) at an azimuth of 135 degrees. Figure 11-25 and Figure 11-26 are plots of the gold and copper grades respectively for Island Mountain through the center of the deposit at 6,847,740 N.

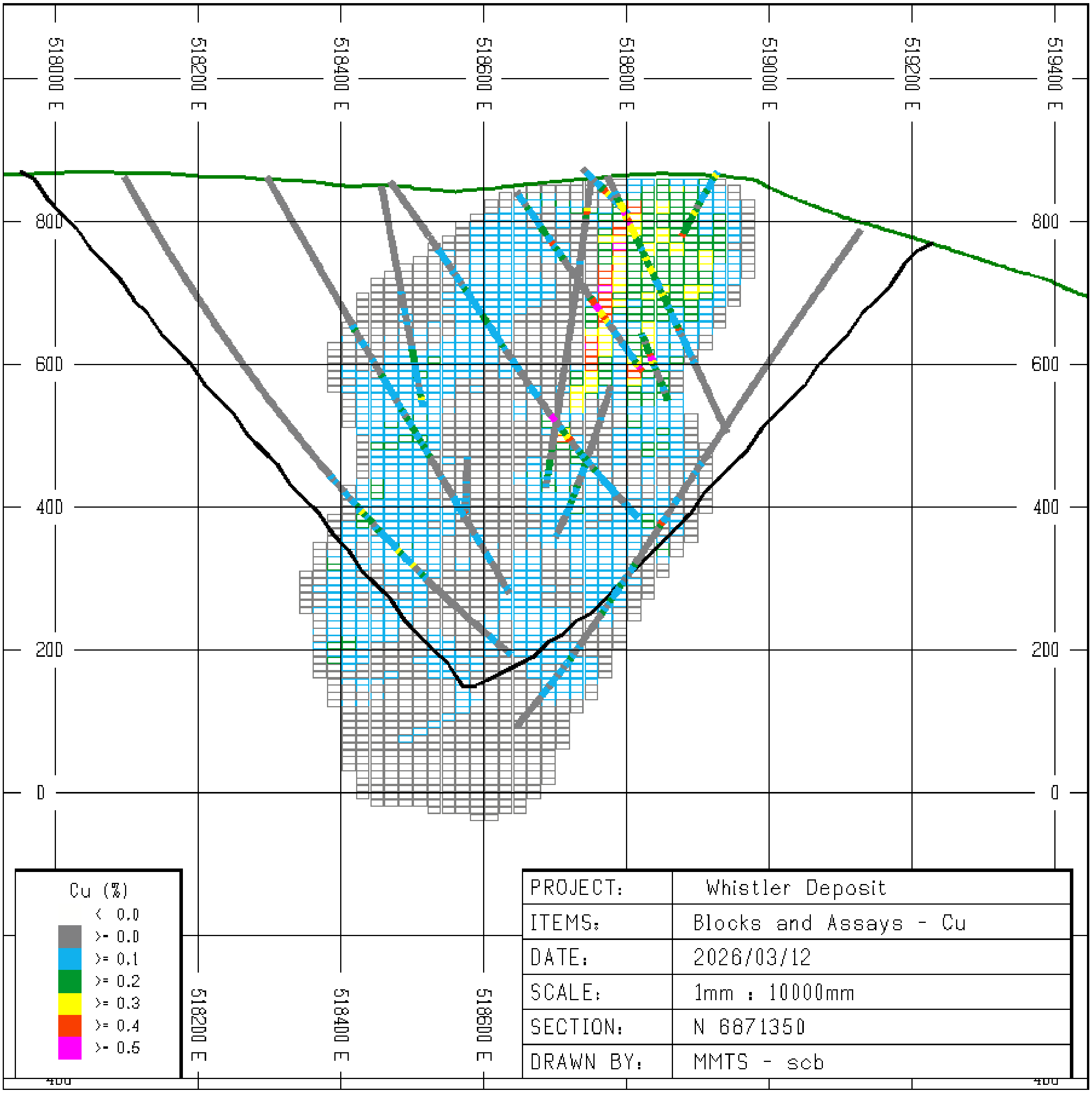
Plots throughout the model confirmed that the block model grades corresponded well with the assayed grades.

Figure 11-21: E-W Section Comparing Au Grades for Block Model and Assay Data - Whistler



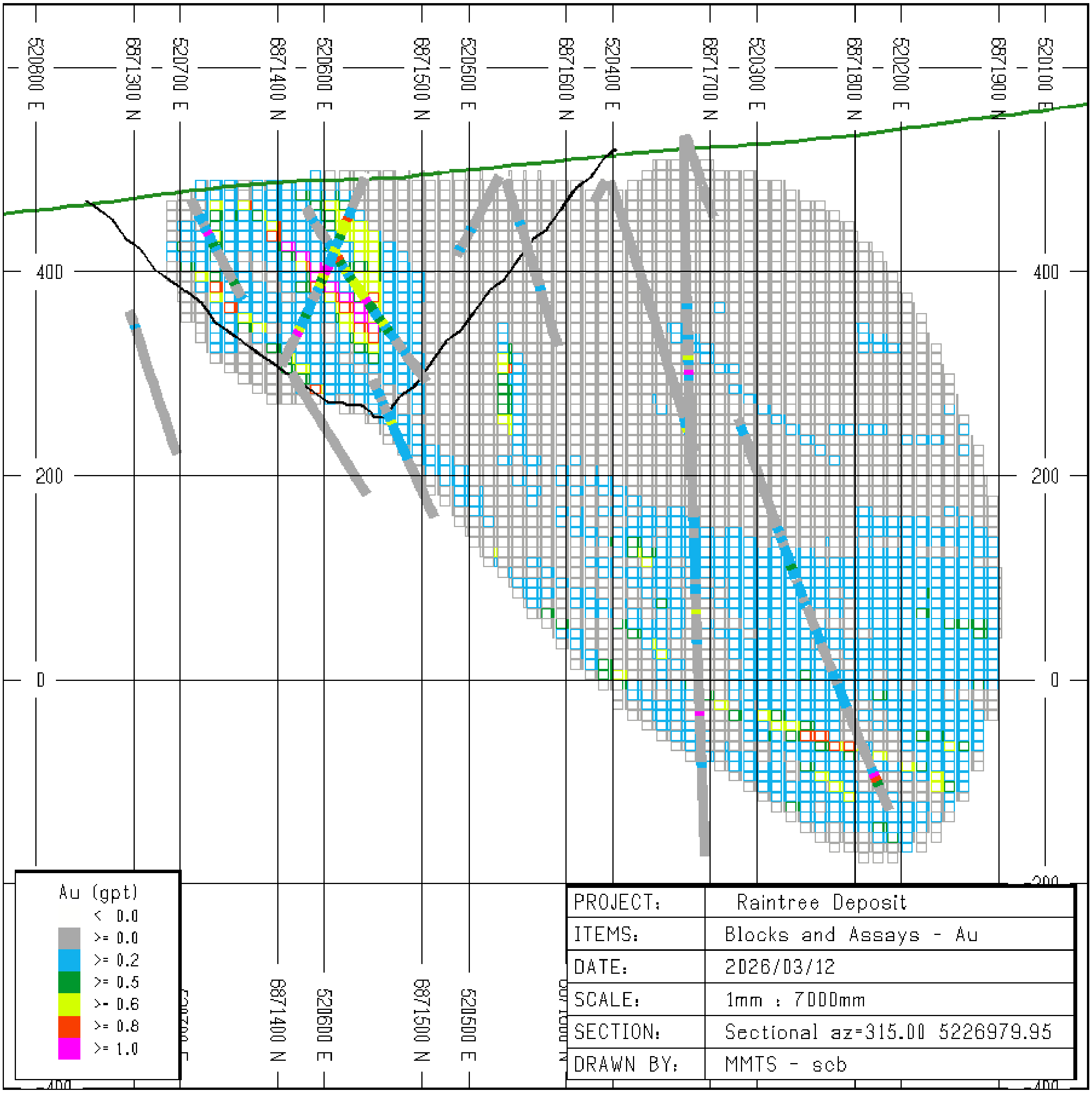
Source: MMTS, 2026

Figure 11-22: E-W Section Comparing Cu Grades for Block Model and Assay Data - Whistler



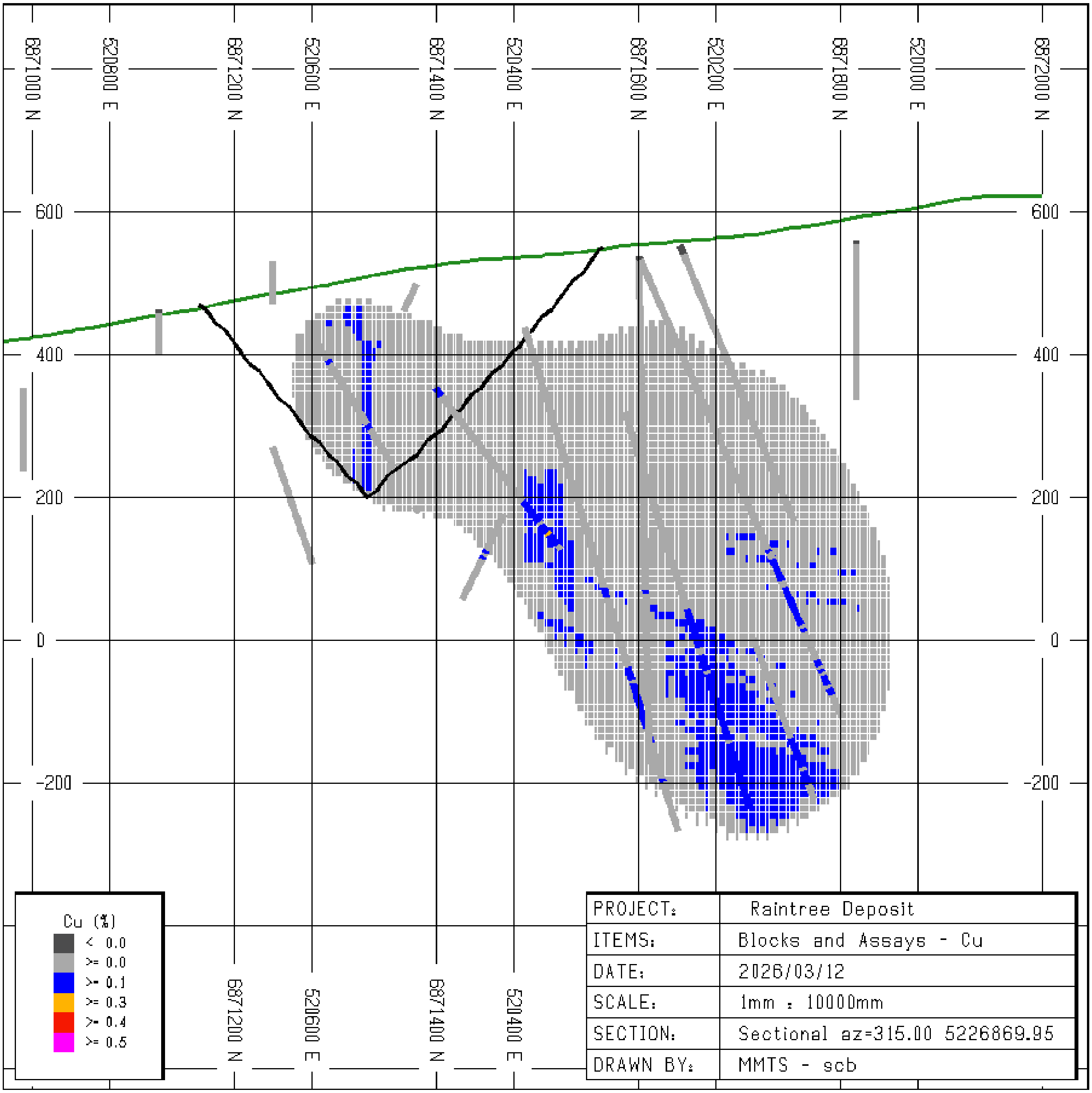
Source: MMTS, 2026

Figure 11-23: Section Looking SW - Comparing Au Grades for Block Model and Assay Data – Raintree West



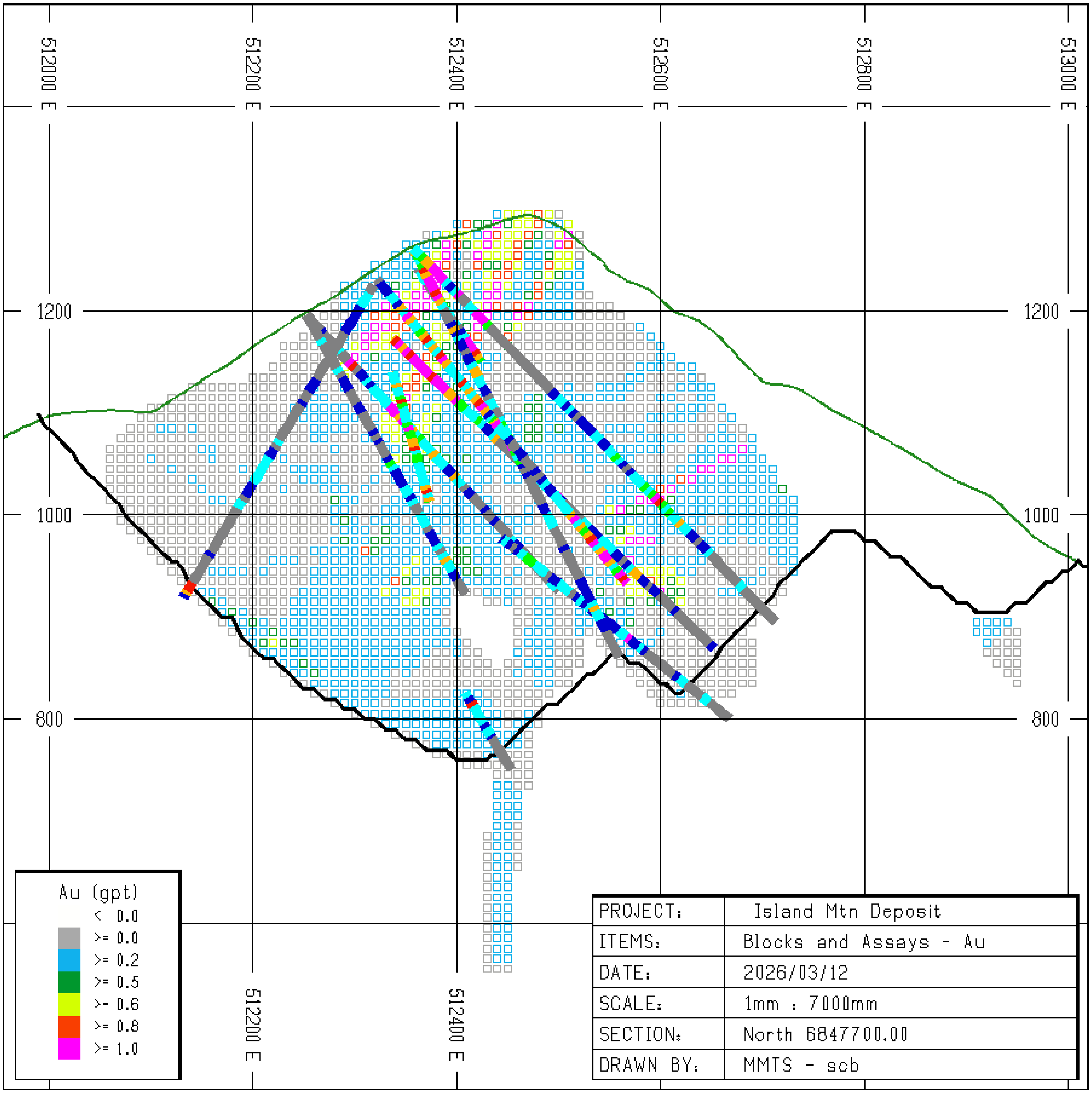
Source: MMTS, 2026

Figure 11-24: Section Looking SW - Comparing Cu Grades for Block Model and Assay Data – Raintree West



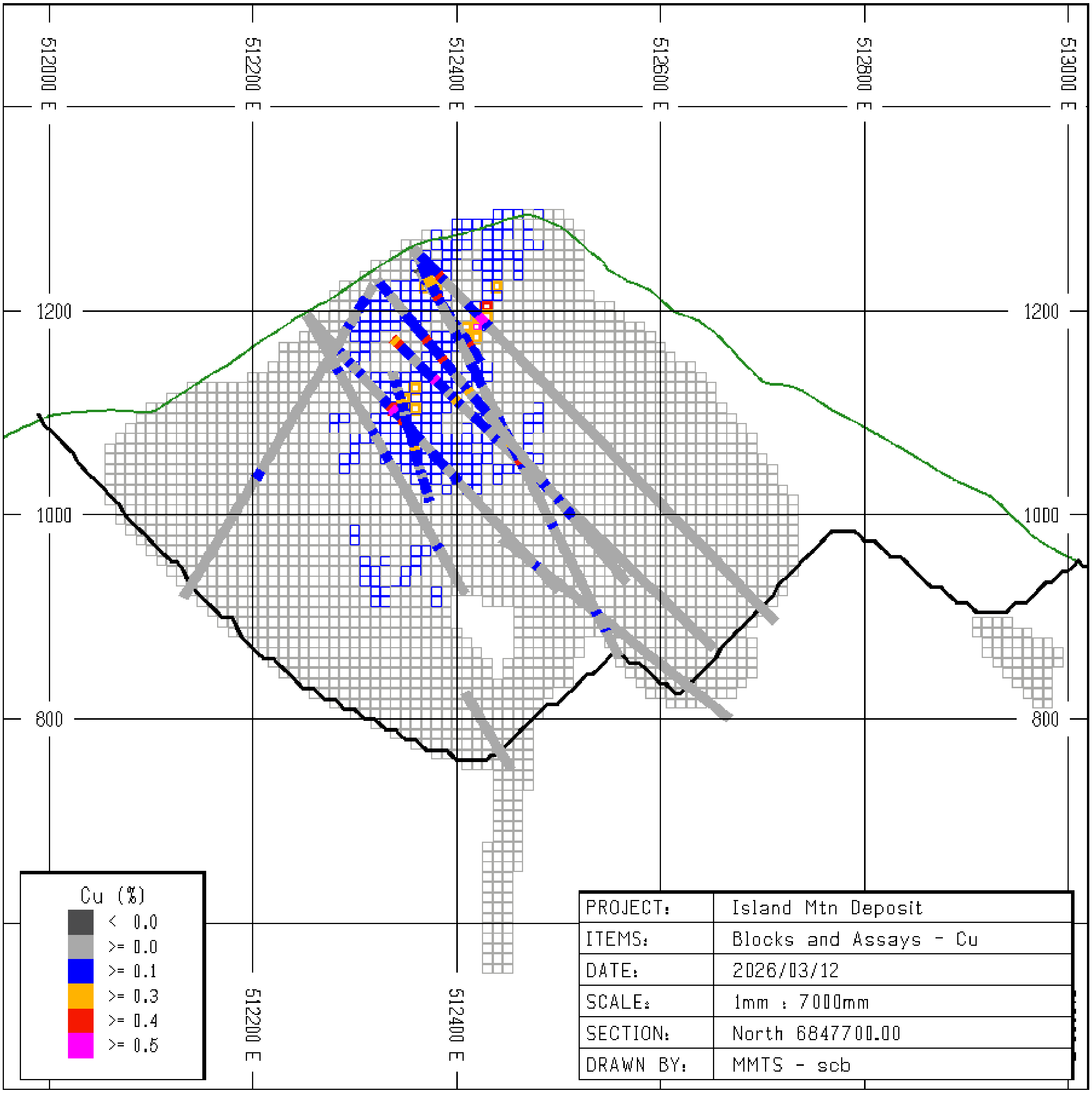
Source: MMTS, 2026

Figure 11-25: E-W Section Comparing Cu Grades for Block Model and Assay Data – Island Mountain



Source: MMTS, 2026

Figure 11-26: E-W Section Comparing Cu Grades for Block Model and Assay Data – Island Mountain



Source: MMTS, 2026

11.10 Reasonable Prospects of Eventual Economic Extraction

The resource confining pit and/or underground shapes defines a boundary for continuous mineralization with suitable grades and with a reasonable expectation that an engineered plan will produce an economic plan. The NSR calculation for both the open-pit and underground resources as well as the metallurgical recoveries are summarized in Table 11-20.

Lerchs-Grossman pits were run for each deposit using the following parameters:

- Pit slopes of 50 degrees
- Mining costs of US$2.75/t for both mineralized materials
- Processing, general and administrative costs of US$13.40/t. The cutoff value for the open pits is considered to be US$13.40/t which more than covers the Processing + G&A costs. The base-case cutoff value for the underground portion of the resource is US$40.00/t which includes Process + G&A + Underground mining costs.

The lower portion of the Raintree West deposit has been constrained by a mineable shape within "reasonable prospects of economic extraction" using a US$40.00/t cutoff, assuming the same processing costs as for the open pit, and a bulk mining scenario. Material within a cohesive shape above this cutoff has been included in the Raintree West underground resource estimate. Metal prices are based on long-term consensus, three-year trailing averages (Kitco, 2026), and are consistent with those seen to be used throughout the industry.

Table 11-20: Economic Inputs and Metallurgical Recoveries

Parameter	Value	Units
Gold Price	2,750.00	US$/oz
Copper Price	4.35	US$/lbs
Silver Price	30.00	US$/oz
Gold Payable	94.8	%
Copper payable	96.5	%
Silver Payable	88.2	%
Gold Refining	7.50	US$/oz
Copper Refining + PP	0.065	US$/lb
Silver Refining	1.00	US$/oz
Off-site	165.65	US$/WMT
Royalty	3.00	%
Net Smelter Gold Price	78.57	US$/g
Net Smelter Copper Price	3.88	US$/lb
Net Smelter Silver Price	0.77	US$/g
Gold Process Recovery	87.8	%
Copper Process Recovery	75.4	%
Silver Process recovery	49.1	%

*Indicated and Inferred resources are used for pit optimization.

The pit delineated resource is given in Table 11-2 through Table 11-4 for each deposit and for a range of NSR cutoff values with the base-case cutoff values of US$13.40/t and US$40.00/t highlighted. Process recoveries, as well as mining, processing and off-site costs have been applied in order to determine that the pit resource has a reasonable prospect of eventual economic extraction. The US$13.40/t cutoff value yields an Indicated resource of 299 Mt at 0.41 g/t Au, 0.15% Cu and 1.9 g/t Ag (0.57 g/t AuEq) for 5.41 Moz AuEq and an Inferred resource of 290.7 Mt at 0.54 g/t Au, 0.06% Cu and 1.6 g/t Ag (0.54 g/t AuEq) for a total of 4.97 Moz AuEq metal.

11.11 Statement on Prospect of Economic Extraction

The QP is of the opinion that all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

11.12 Factors That May Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Commodity price assumptions
- Metal recovery assumptions
- Mining and processing cost assumptions

There are no other known factors or issues known to the QP that materially affect the estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors.

11.13 Comparison to the Previous Resource Estimate

Table 11-21 below compares the current resource estimate to the previous, 2024 resource estimate. Changes to the resource are due to drilling at Whistler in 2024, as well as updates to the recoveries, smelter terms, mining and processing costs, cutoff values and metal prices. The increase in metal prices increased the NSR while the relative increase in gold price compared to copper and silver causes the decrease in AuEq value for the 2026 estimate.

Table 11-21: Resource Comparison to Previous Estimate

Model	Class	Cutoff Value	ROM tonnage	In situ Grades					In situ Metal			
		($/t)	(ktonnes)	NSR ($/t)	AuEqv (g/t)	Au (g/t)	Cu (%)	Ag (g/t)	AuEqv (Koz)	Au (koz)	Cu (klbs)	Ag (koz)
Current	Indicated	$13.40 for OP, $40 for UG	299,154	38.94	0.56	0.41	0.15	1.89	5,414	3,973	991,667	17,924
	Inferred		290,747	36.97	0.536	0.470	0.062	1.60	4,969	4,357	390,355	14,261
2024	Indicated	$10.50 for OP, $25 for UG	294,474	22.91	0.68	0.42	0.16	2.01	6,482	3,933	1,023,708	18,987
	Inferred		198,241	21.82	0.652	0.519	0.073	1.81	4,157	3,311	316,924	11,521
Difference (%)	Indicated	VARIES	1.6%	70.0%	-17.6%	-0.3%	-4.5%	-5.7%	-16.5%	1.0%	-3.1%	-5.6%
	Inferred		46.7%	69.4%	-17.8%	-9.6%	-15.1%	-11.6%	19.5%	31.6%	23.2%	23.8%

11.14 Risk Assessment

A description of potential risk factors is given in Table 11-22 along with either the justification for the approach taken or mitigating factors in place to reduce any risk.

Table 11-22: List of Risks and Mitigations/Justifications

#	Description	Justification/Mitigation
1	Classification Criteria	Classification based on the Range of the Variogram and therefore the variability of the mineralization within each deposit.
2	Gold and Silver Price Assumptions	Based on consensus and three-year trailing average (Kitco, 2026)
3	Capping	CPP, swath plots and grade-tonnage curves show model validates well with composite data throughout the grade distribution.
4	Processing and Mining Costs	Based on comparable projects in Alaska.

12 MINERAL RESERVE

There are no mineral reserves estimated for the Project.

13 MINING METHODS

The Whistler Deposit is amenable to conventional drill, blast, load, and haul open-pit mining methods. Open pit mine designs, a mine production schedule, and mine capital and operating costs have been developed for the Whistler Deposit at a scoping level of engineering. The Raintree West and Island Mountain deposit resources are not included in this mine plan.

13.1 Summary

The open pit is designed for approximately 16 years of operations, inclusive of one year of pre-production mining, and one year of low-grade stockpile rehandling to the mill after the open pit is exhausted. The ROM production contained within the designed open pit, summarized in Table 13-1 with a 0.19 g/t AuEq cutoff grade (NSR:US$13.40/t), forms the basis of the Whistler Mine Plan. These contents are a subset of the Indicated Mineral Resource Estimate described in Section 11; Inferred Class resources have been treated as waste.

Table 13-1: Whistler Plan ROM Production Results

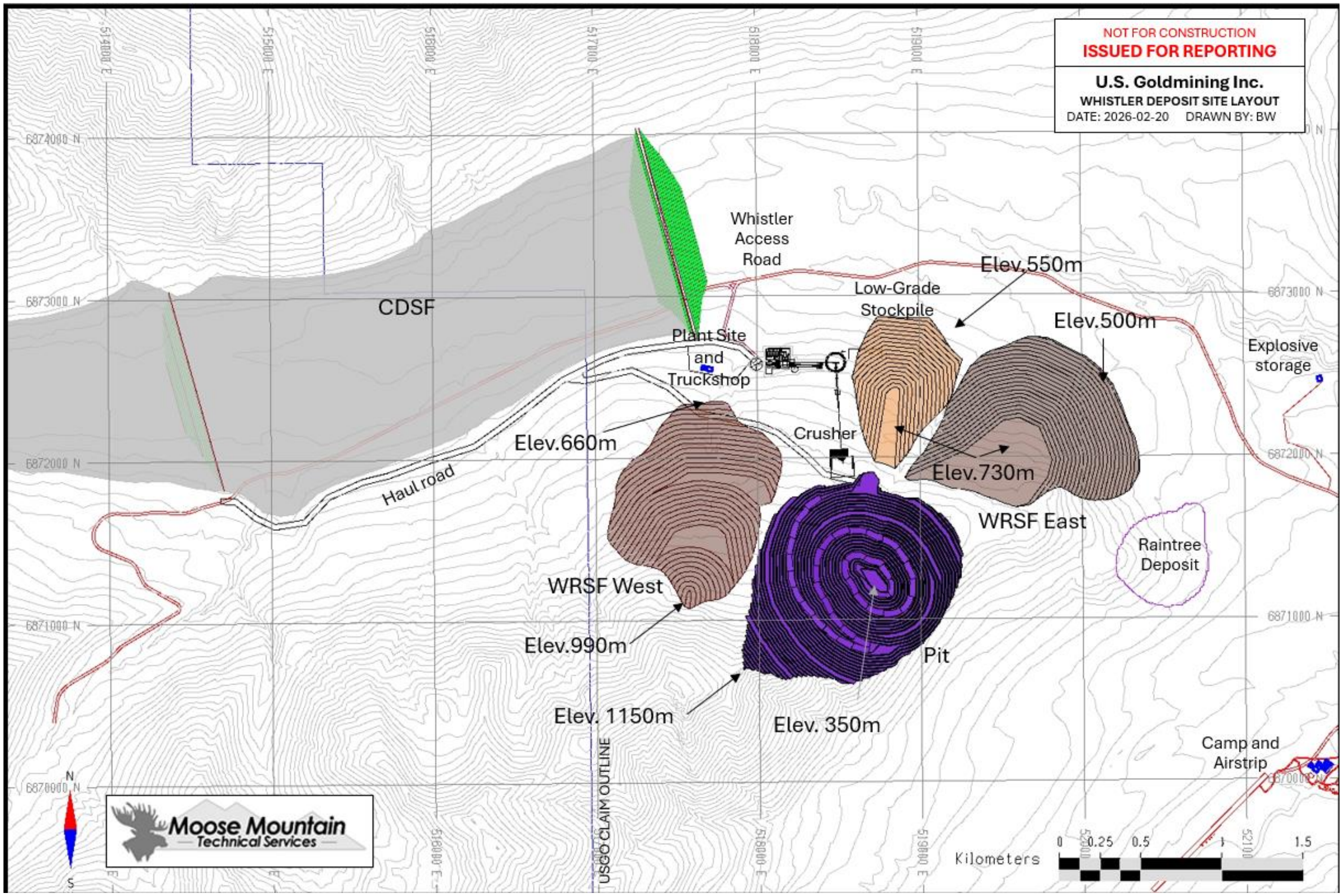
Pit Content	Amount
Mill Feed	211 Mt
Gold Grade	0.44 g/t
Copper %	0.16 %
Silver Grade	1.8 g/t
Waste Material	465 Mt
Strip Ratio	2.2

Mill feed quantities and grades include estimates of mining dilution and recovery based on 20 m x 20 m x 10 m selective block sizes and an additional 3% dilution applied to account for waste edge contracts on the outer edges of the mineralization. This dilution is balanced with an estimated 97% mining recovery.

The crusher will be fed with material from the pit and stockpile at an average feed rate of 40,000 t/d.

Figure 13-1 shows the mine layout for the pit, stockpiles, and haul roads.

Figure 13-1: Mine Layout



Source: MMTS, 2026

The open pit is split into four phases, targeting highest to lowest economic value between the pushbacks. The first phase will commence near the center of the deposit, where the highest grade of mineralized material and lowest strip ratio will be encountered and the remaining phases targeting progressively higher strip ratios and lower grades.

Mill feed will be sent to the crusher directly north of the open pit, or to the low-grade stockpile next the crusher. This low-grade facility will be reclaimed to the crusher/mill during and at the end of the mine life.

Preliminary estimates indicate that 55% of the pit waste rock is potentially acid-generating (PAG). Another 12% of pit waste rock is undefined and PAG overburden. These PAG quantities will be stored subaqueously within the Co-disposal Storage Facility (CDSF), 2 km northwest of the open pit. The remaining waste and overburden are non-potentially acid-generating (NAG) and will be stored at waste rock storage facilities (WRSF) located directly to the east and west of the pit. Suitable pit waste rock will also be hauled to the CDSF for dam construction, as needed.

13.2 Mine Design Criteria

Key mine engineering design inputs are described below. In some instances, the mine planning input is different from the final Study value. Checks have been made to ensure these differences would not materially affect the mine plan at a scoping level of engineering.

13.2.1 Mine Planning Block Model

The resource 3D block model (3DBM) described in Section 11 forms the basis of the mine planning work. Extra items are added to the resource 3DBM to carry out open-pit mine planning. Dimensions of 20 x 20 x 10 m are representative of a selective mining unit.

Inferred class resources are treated as waste. Only Indicated class resources are considered economic within the mine plan.

13.2.2 Net Smelter Return

NSR is estimated for each block and is used as a cutoff value item for break-even mill feed/waste selection. NSR is the net of off-site costs (smelting/refining, transportation, etc.) and includes on-site mill recovery. NSR is estimated using net smelter price (NSP) and process recovery as shown in the equation below. The NSP is formulated from base-case market metal prices and typical off-site transportation, smelting and refining charges.

Table 13-2: NSR Terms

	Value	Unit
Copper Price	4.35	US$/lb
Gold Price	2,750	US$/oz
Silver Price	30.00	US$/lb
Cu Payable	96.5	%
Au Payable	94.8	%
Ag Payable	88.2	%
Cu Con Smelting	65.00	US$/dmt
Cu Refining	0.065	US$/lb
Au Refining	7.50	US$/oz
Ag Refining	1.00	US$/oz
Offsites (Ocean Freight)	75.65	US$/wmt
Offsites (Trucking Freight)	90.00	US$/wmt
Royalty	3.00	%
Cu Process Recovery	75.4	%
Au Process Recovery	87.8	%
Ag Process Recovery	49.1	%

Metallurgical recoveries used for the NSR calculation are based on a preliminary modelling on the Whistler Mine Plan.

The resultant NSP for Au is US$78.57/g (US$2,444/oz), US$0.77/g for Ag (US$24/oz), and US$3.88/lb for Cu.

An NSR value is coded for each block in the model using the following formula:

$$NSR\ (\$/t) = [NSPAu\ (\$/g) * AuGrade\ (g/t) * Au\ process\ recovery\ (\%)] + [NSPCu\ (\$/lb) * CuGrade\ \% * Cu\ process\ recovery\ (\%) * 22.0462] + [NSPAg\ (\$/g) * AgGrade\ (g/t) * Ag\ Process\ Recovery\ (\%)]$$

13.2.3 Cutoff Value

The economic cutoff value is chosen as the NSR grade required to pay for processing and general and administration costs. The cutoff value calculation uses the inputs shown in Table 13-3.

Table 13-3: NSR Cutoff Value

Item	Value
Process Costs	US$11.25/t
G&A Costs	US$2.15/t
Economic Cutoff Value	US$13.40/t

13.2.4 Mining Loss and Dilution

The mineral resources are based on 20 m x 20 m x 10 m block sizes. This block sizing is also appropriate for the drill, blast, and load practices chosen for this mine plan. It is assumed that the effects on metal grades from mining selectivity are mostly built into the whole block grades.

A measurement of waste to resource block contact edges is also carried out for all mineralized blocks. Based on these measurements, an additional 3% contact dilution, at zero grades, is added to whole block measured tonnes and grade.

A 97% mining recovery is estimated to account for effects of mis-directed loads, carryback and stockpile base losses. The applied loss and dilution (at a US$13.40/t cutoff value) is:

- Loss = 3 %
- Dilution = 3%

13.2.5 Open-Pit Slopes

The pit slope criteria are based on scoping level geotechnical work done on the Whistler Deposit (KP, 2013). Geotechnical characterization was conducted based on a desktop review of the available geological and geotechnical information. A simplified geotechnical model was established, and preliminary slope stability analyses were conducted to evaluate slope recommendations for the proposed final pit.

The intact strength of all rock types was estimated to be “Medium Strong”, while the rock mass quality was characterized as “Fair”. Five steeply dipping major faults have cut through the center of the deposit. Small scale structures including joints and shear are generally shallowly dipping and/or sub horizontal inter-ramp scale geometries were determined from stability analyses of potential plane shear and wedge failure, rock mass stability, and bench geometry. Generic noncircular rock mass stability analyses were conducted for each geotechnical unit to determine slope height versus slope angle relationships for use in inter-ramp and overall slope scale designs. The kinematic stability analyses suggest that a steep bench face angle of 70° is kinematically achievable for most of the pit walls except for the west dipping walls where a 65° bench face angle would be more applicable due to the planar failure potential. Given the base-case rock mass strength parameters and pore water pressure assumptions, the overall slope stability analyses indicate that overall slope angles of 40° and 42° can be achieved for the final West and East Walls, respectively.

Pit designs are configured on 10 m bench heights, with minimum 8-m wide berms placed every second bench, or a double benching configuration. Unique bench face and inter-ramp slope inputs are based on azimuth. These slope criteria are summarized in Table 13-4.

Table 13-4: Pit Slope Parameters

Slope Zone	Bench Face Angle (°)	Inter-ramp Angle (°)	Overall Slope Angle (°)
1 - North	70	45	43
2 - West	70	45	42
3 - East	65	45	40

Source: KP, 2013

13.3 Economic Pit Limits

Economic pit limits are determined using the Pseudoflow implementation of the Lerch-Grossmann algorithm. The algorithm considers the block grades and tonnages and compares the expected costs to extract and process the block against potential revenue from processing the block (for mineralized blocks). Each block is assigned a net positive or negative value. Pit slope inputs determine which upper blocks need to be mined to extract economic blocks that are lower in the deposit. The routine uses input revenue, mining and processing costs and pit slope parameters to expand in an ever-increasing manner until the incremental tonnages of the next thin skin or pushback would generate negative economics. The total shape just prior to this 'negative pushback' is then used as the maximum economic pit limit for the selected input parameters.

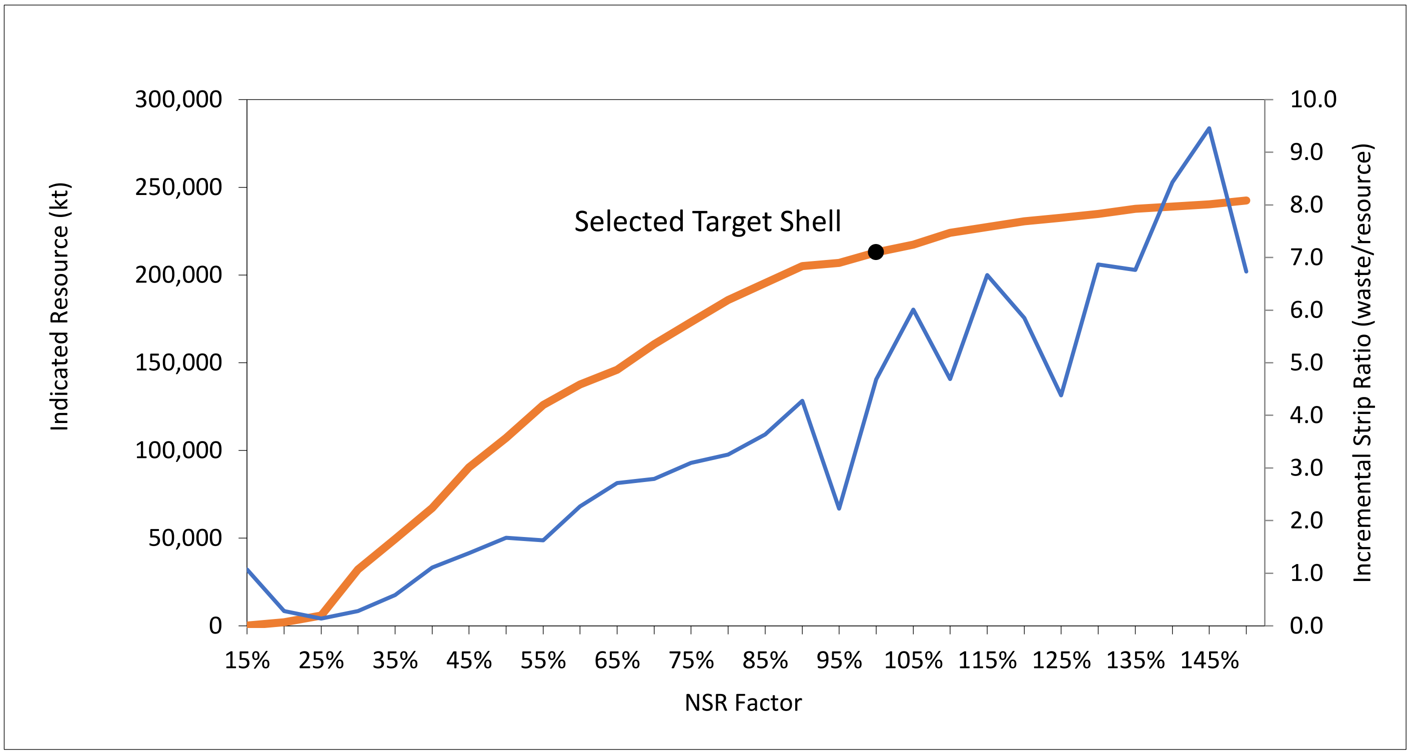
The economic pit limit is selected after evaluating a series of Lerch-Grossmann pit cases. The assessment is carried out by generating sets of Lerch-Grossmann pit shells by varying the revenue input assumptions. The inputs to the Lerch-Grossmann pit limit assessment are shown in Table 13-5.

Table 13-5: LG Pit Limit Base-case Inputs

Item	Value
Prices, costs, recoveries, exchange	Same as Table 16-2
Mining Cost	US$2.75/t
Incremental Mining Cost	US$0.03/t every 10-m drop below pit rim
Process, G&A, Site Services	US$13.40/t
Pit Slope Angle	50 degrees

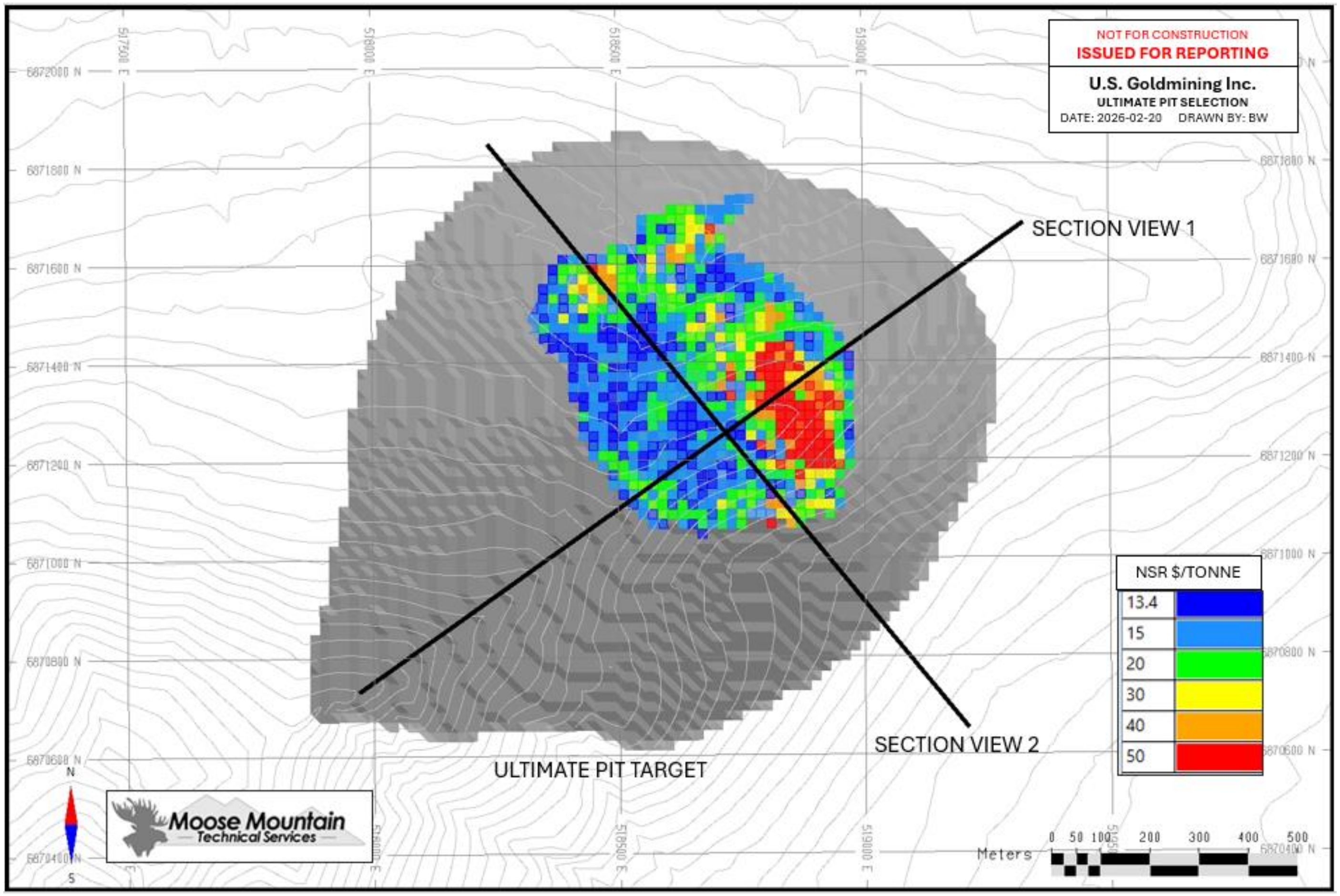
LG pit shells are generated by varying the NSR value from 15% to 150% of the base-case inputs. Figure 13-2 shows the contents of the generated Lerch-Grossmann shells. An inflection is observed at the 90% case. Another smaller inflection point is the 100% case with the pit size growing steadily after that. The contents of both the 90% and 100% pit shells were scheduled and costed and run through a preliminary financial model for the project. The 100% case shell contents generated improved discounted project economics and the 100% shell, illustrated in Figure 13-3, is selected as the ultimate pit limit target for the mine plan.

Figure 13-2: Lerch-Grossmann Pit Analysis Results



Source: MMTS, 2026

Figure 13-3: Ultimate Pit Target Selection (Section Views shown in Section 13.4)



Source: MMTS, 2026

13.4 Detailed Pit Designs

The generated Lerch-Grossmann pit shells are used to guide detailed pit phase designs.

The open pit development is designed as a conventional truck-shovel operation with 230-tonne trucks paired with both 34 m^3 and 22 m^3 shovels. The ultimate pit limit is split up into four nested phases or pushbacks to target higher economic margin material earlier in the mine life.

Other considerations for selection of interim pit phases:

- Provide enough mill feed to sustain the plant operations for at least two years.
- The pit benches should be large enough to allow an efficient area for mining and keep the vertical bench advance rate to be < 9 benches per year.

- Minimum mining width to allow an efficient area for mining is assumed to be 50 m.

The Lerch-Grossmann pit shells described in the preceding sections can typically be used as a guideline for selecting interim pit phases. Pit shells created by the optimization algorithm with lower input metal prices than the selected ultimate pit case will contain higher-grade mill feed and/or lower strip ratios.

The pit shell generated using a revenue factor of 35% is used to target a starter-pit phase. The next two pit pushbacks target the 45% and 55% case pit shells respectively. The final phase pushes out to the selected ultimate pit limits.

The pit design parameters include:

- Ramp width of 32 m, which includes allowances for double-lane traffic, berms, and ditches.
- Maximum road grades of 10%.
- No ramp in the final bench at pit bottom, assuming retreat mining.
- Second lowest bench uses single lane road width of 23 m and 12% grades since bench volumes and traffic flow will be reduced in this portion of the pit.
- Pit slopes as per Table 13-4. Bench heights of 10 m, on ly a double benching configuration.
- Initial phases (Phase 1,2 and 3) are designed at the inter-ramp angles to reduce strip ratio in early years of the project. The final pit walls are designed with the overall slope angle target.
    - Max inter-ramp height of 200 m. Ramps and wider catchment berms placed every 200 m to push out final wall.

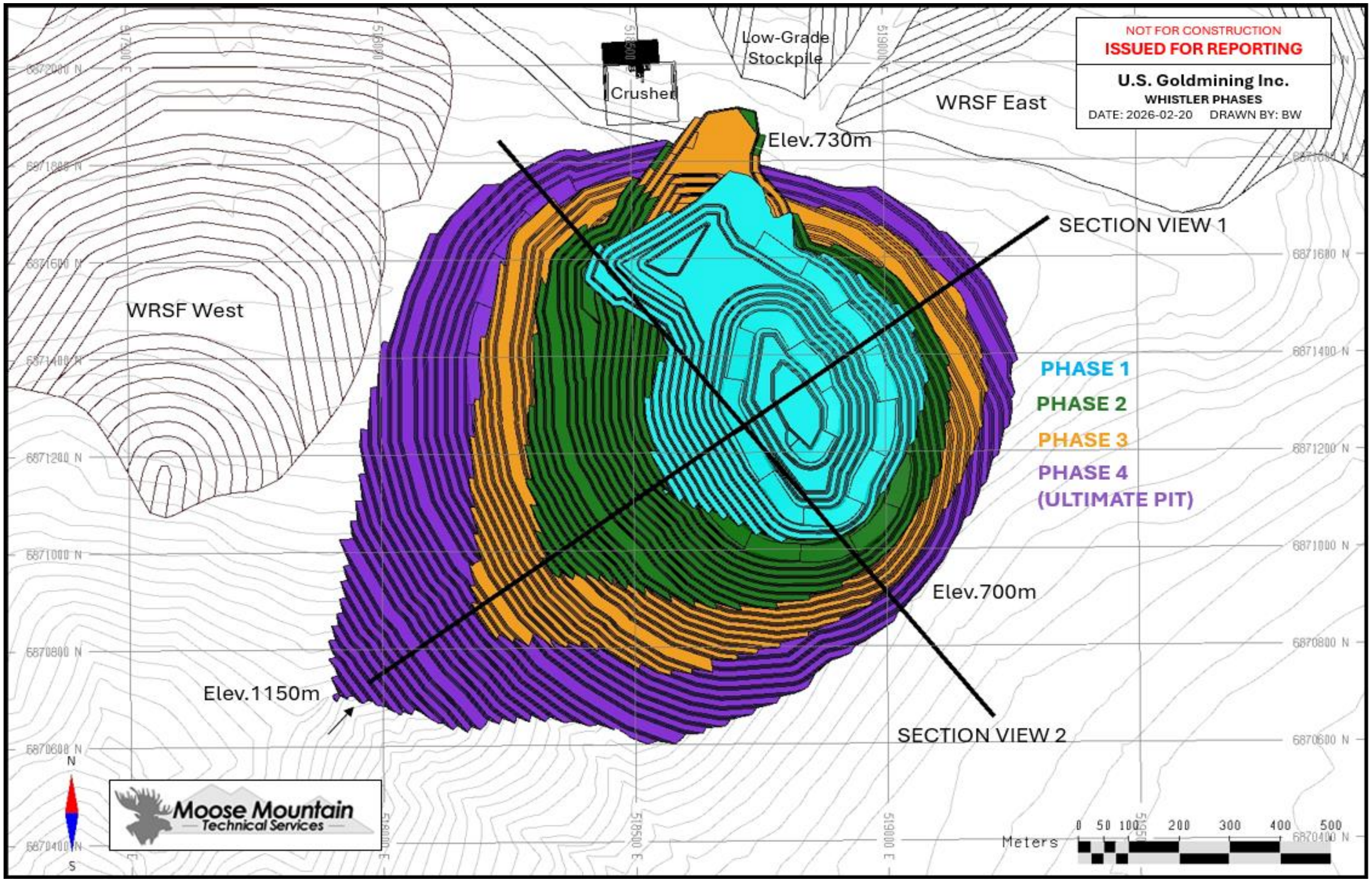
Table 13-6 shows the mill feed and waste tonnages, NSR, and strip ratio of each designed pit phase.

Table 13-6: Pit Phase Contents (US$13.40/t cutoff value)

Phase	Mill Feed (Mt)	NSR (US$/t)	Waste (Mt)	Strip Ratio (Waste/Mill Feed)	Pit Bottom (masl)
1	56.0	$50	27.8	0.5	610 m
2	66.0	$42	86.2	1.3	490 m
3	41.6	$38	109.6	2.6	410 m
4	47.7	$34	241.0	5.1	350 m
Total	**211.4**	**$42**	**464.7**	**2.2**	

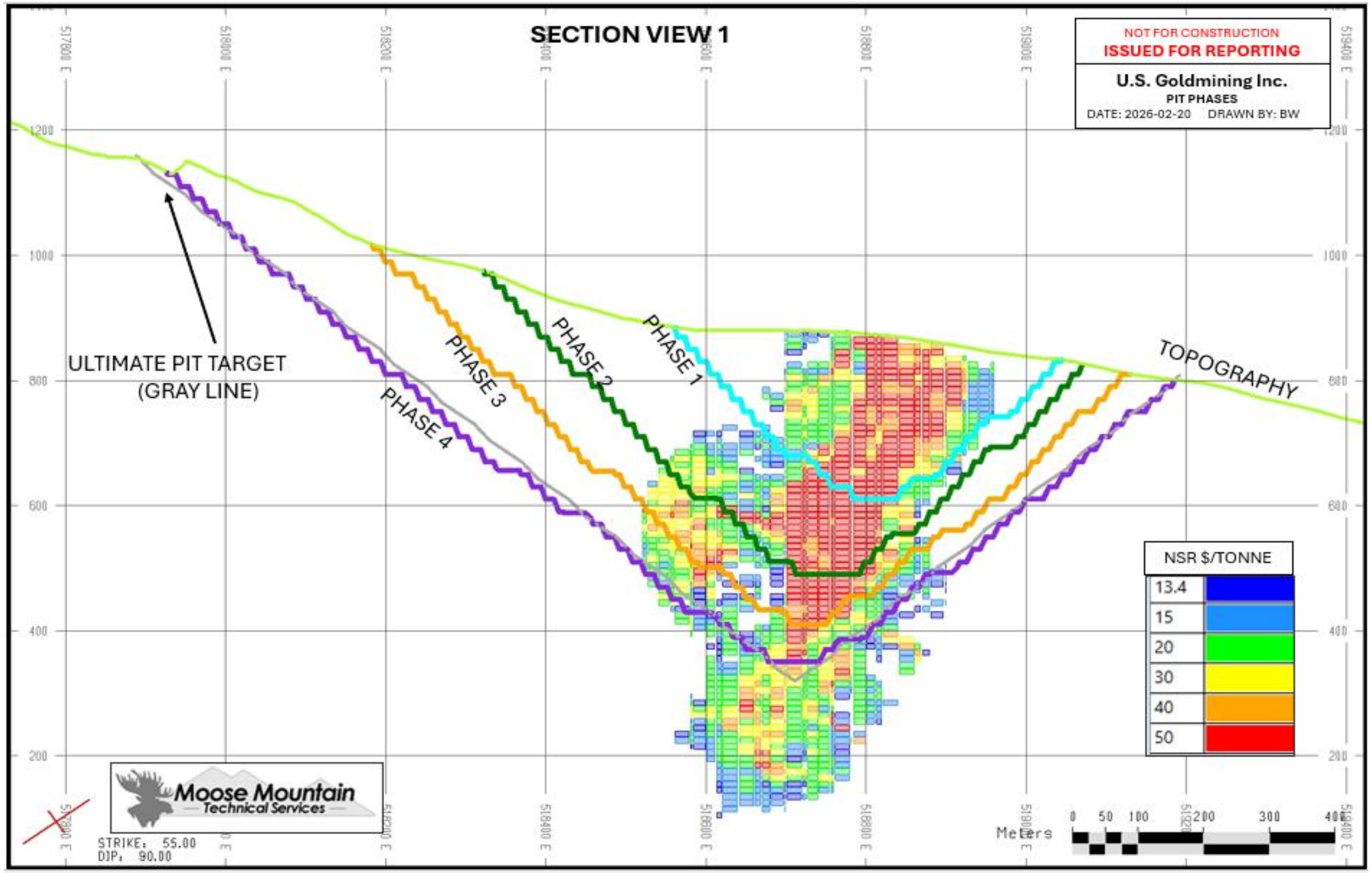
Plan view and section view of the pit and phases are shown in Figure 13-4 to Figure 13-6.

Figure 13-4: Pit Phases



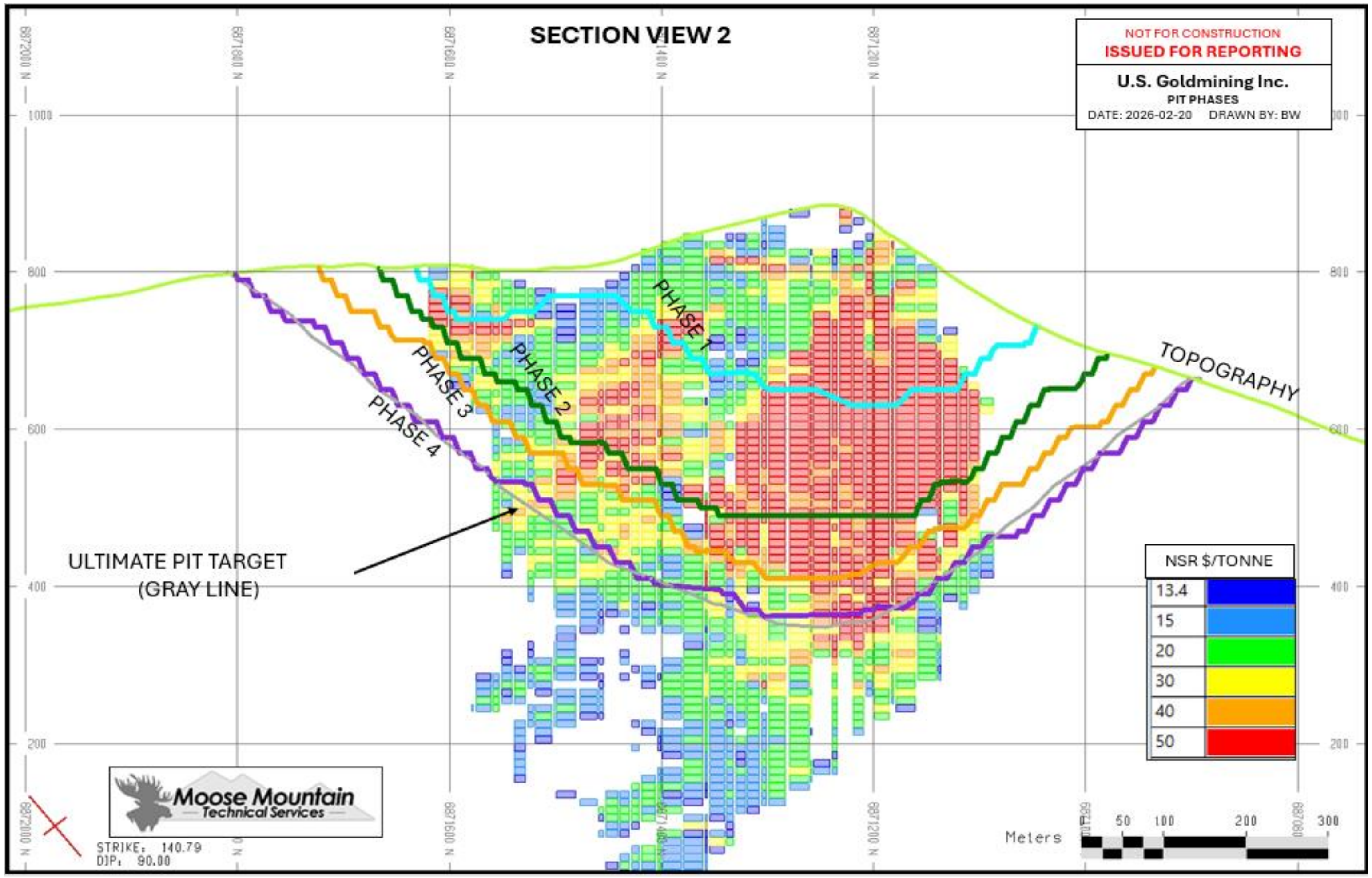
Source: MMTS, 2026

Figure 13-5: Pit Phases Section View 1



Source: MMTS, 2026

Figure 13-6: Pit Phases Section View 2



Source: MMTS, 2026

13.5 Waste Rock Management

The waste rock management plan details out the storage of all the mined material below theUS$13.40/t NSR cutoff grade (waste).

Acid rock drainage (ARD) potential criteria in the pit waste rock have been classified by sulfur, calcium and magnesium values interpolated into the 3DBM to quantify the ARD generating potential of the pit material.

The ARD classification is defined as:

- Acid Potential (AP) for block = 31.25*S
- Neutralization Potential (NP) for block = (Ca/40.08)+(Mg/24.31)*100.09*10

where S is the sulfur value in percent, Ca is the calcium value in percent, Mg is the magnesium value in percent.

Material is defined as PAG if NP/AP >2. Where no sulfur data exists, waste rock is classified as unknown and treated as PAG. All overburden has also been classified as PAG, based on the limited data available.

PAG waste rock and overburden is hauled and stored subaqueously withing the Co-disposal Storage Facility (CDSF). A capacity for 305 Mt of PAG material has been included in the CDSF design.

NAG waste rock is hauled and stored in waste rock storage piles (WRSF) located west and east of the pit. The split into two piles reduces haul times, reduces footprints, and avoids existing water bodies and proposed infrastructure. The East WRSF will be built top down by end dumping near the pit haul ramp exits. The West pile will be built top down using wrap around haul roads. The West WRSF has a capacity of 60 Mt and the East WRSF has a capacity of 105 Mt.

NAG waste rock will also be used to construct ex-pit haul roads to the crusher, the mine maintenance facilities, and the co-disposal storage facility. It will also be used to construct a majority of the CDSF's dam.

The West WRSF is built at an overall slope of 3H:1V and the East WRSF is built at an overall slope of 2.5H:1V. A swell factor of 30% from bank conditions is estimated for material placed in the WRSF.

Figure 13-1 illustrates the WRSFs and the CDSF.

13.6 Low-Grade Mill Feed Stockpile

When mill feed is mined from the pit, it will either be delivered to the crusher, the ROM stockpile located next to the crusher, or the low-grade stockpile.

A cutoff grade strategy has been employed for the production schedule, and during operations a stockpile near the crusher will be maintained to store lower-grade mill feed for later rehandling back the crusher. The low-grade stockpile has a max capacity of 32 Mt. Material will be rehandled using wheel loaders and haul trucks and delivered to the crusher using ramps built into the stockpile, no oxidation is expected over the LOM.

The stockpile is built at an overall slope of 2.5H:1V and is 150 m tall. A swell factor of 30% from bank conditions is estimated for material placed in the low-grade stockpile.

Figure 13-1 illustrates the low-grade stockpile.

13.7 Ex-pit Haul Roads

Ex-pit haul roads are designed with a maximum grade of 8% and 37 m width that accommodates dual-lane traffic and berms on both edges. Ex-pit roads are sized to accommodate 230-tonne payload class haulers. The roads will be built from blasted waste rock fill from the pits, topped with a crushed rock running surface.

Initial development to the top benches of each pit phase will be constructed from cut road internal to the pit limits. The 37-m wide road will be cut up the side of the hill, then retreat-mined out to the surrounding ex-pit haul roads.

Ex-pit haul roads have not been designed in 3D with cut and fill quantities, but clearances have been made for the selected paths of these roads.

Figure 13-1 illustrates the ex-pit haul roads.

13.8 Mine Production Schedule Results

Production requirements by scheduled period, mine operating considerations, product prices, recoveries, destination capacities, equipment performance, haul cycle times and operating costs are used to determine the optimal production schedule from the phased pit contents.

The production schedule is based on the following input parameters:

- The operations are scheduled on annual periods.
- The mineral resource and associated waste material quantities are split by pit phase and bench quantities.
- An annual mill feed rate of 14.6 Mt/a (40 kt/d) is targeted.
    - Year 1 ramp up throughput of 12.4 Mt is targeted (85% of nameplate)
- Within a given pit phase, each bench is fully mined before progressing to the next bench.
- Pit phases are mined in sequence, where the second pit phases do not mine below the first pit phases.
- Pit phase vertical progression is limited to no more than 90 m in each year (9 x 10 m benches); average annual phase progression is 70 m.
- Pre-production mining targets waste rock amounts to use for haul road and tailings facility dam construction materials.

- Resource tonnes released in excess of the mill capacity are stockpiled, including those mined in the construction phase.
- Low-grade resource is stockpiled in the early years of the mine life and rehandled to the primary crusher later in the mine life.
- Shovel and haul truck operating hour estimates are run as part of the mine schedule. Haul cycle times are simulated from all pit benches to all destinations. Total pit production is balanced on calculated hauler operating hour requirements. This strategy is used to avoid large spikes and dips in the number of haulers in the LOM schedule but leads to some variations in total tonnes mined in each period. Cycle time simulations should be refined in future engineering studies.

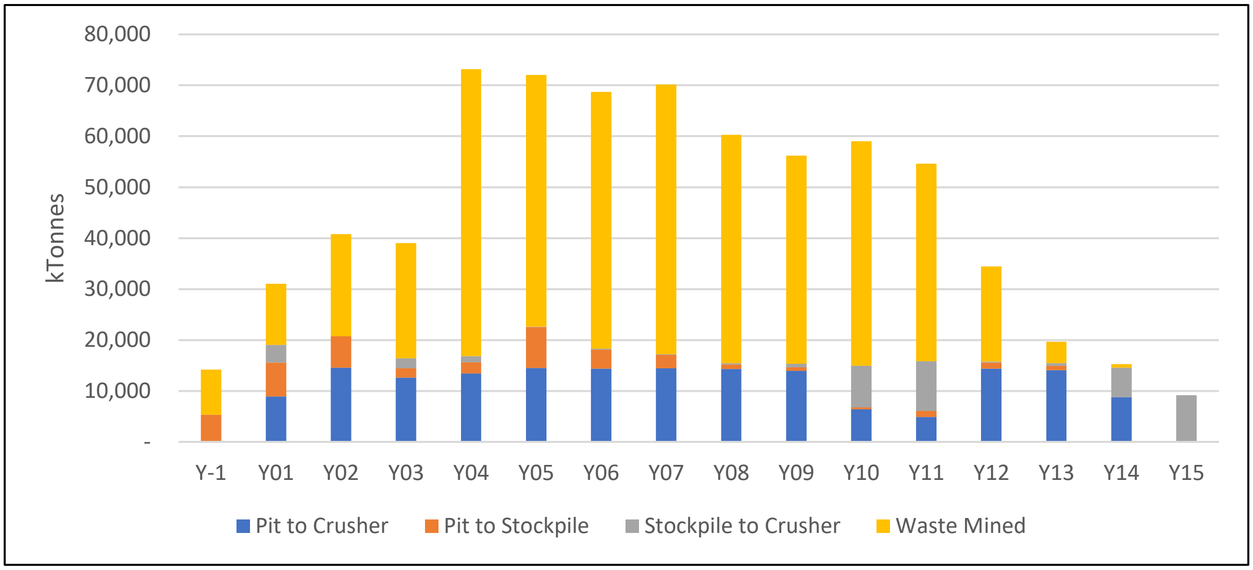
The mine production schedule is summarized in Table 13-7.

Table 13-7: Mine Production Schedule Summary

Mine Production	Year	LOM	Y-1	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14	Y15
Mill Feed	**kt**	**211,368**	**-**	**12,400**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**9,169**
Resource Mined from Pit	kt	211,368	5,335	15,565	20,724	14,418	15,655	22,533	18,092	17,098	15,199	14,645	6,753	6,108	15,561	14,920	8,761	-
NSR	US$/t	41.82	48.95	41.61	47.19	62.56	29.69	35.49	47.96	51.47	33.33	45.02	45.27	21.88	31.74	37.46	43.33	-
Resource Mined Directly to Mill	kt	169,674	-	8,900	14,592	12,593	13,451	14,538	14,389	14,461	14,333	13,919	6,400	4,900	14,370	14,067	8,761	-
NSR	US$/t	46.57	-	57.23	58.29	68.78	32.14	44.65	55.27	57.93	34.45	46.59	46.96	23.65	33.15	38.84	43.33	-
Resource Mined to Stockpile	kt	41,695	5,335	6,665	6,132	1,825	2,204	7,995	3,703	2,637	866	726	354	1,208	1,191	853	-	-
NSR	US$/t	22.48	48.95	20.75	20.77	19.65	14.77	18.82	19.59	16.07	14.75	14.85	14.69	14.67	14.81	14.69	-	-
Stockpile Retrieval to Mill	kt	41,695	-	3,500	8	2,007	1,149	62	211	139	267	681	8,200	9,700	230	533	5,839	9,169
NSR	US$/t	22.48	-	64.60	26.89	26.66	22.62	18.68	18.73	18.84	18.60	18.49	18.40	18.34	18.01	17.76	17.58	17.58
Stockpile Balance	kt		5,335	8,501	14,625	14,442	15,497	23,430	26,922	29,420	30,019	30,064	22,218	13,726	14,688	15,008	9,169	-
NSR	US$/t		48.95	20.40	20.55	19.59	18.68	18.73	18.84	18.60	18.49	18.40	18.34	18.01	17.76	17.58	17.58	-
Waste Mined	**kt**	464,671	8,830	11,964	20,055	22,584	56,345	49,467	50,408	52,902	44,801	40,855	44,091	38,827	18,672	4,199	670	-
Waste:Resource Mined Ratio		2.2	1.7	0.8	1.0	1.6	3.6	2.2	2.8	3.1	2.9	2.8	6.5	6.4	1.2	0.3	0.1	-
Cumulative Waste:Resource Ratio			1.7	1.0	1.0	1.1	1.7	1.8	2.0	2.1	2.2	2.2	2.4	2.6	2.4	2.3	2.2	2.2
Total Material Mined	kt	676,039	14,165	27,530	40,780	37,002	72,000	72,000	68,500	70,000	60,000	55,500	50,845	44,935	34,233	19,119	9,431	-
Cumulative Material Mined	kt		14,165	41,695	82,474	119,476	191,476	263,476	331,976	401,976	461,976	517,476	568,320	613,256	647,489	666,608	676,039	676,039
Total Material Moved	kt	717,734	14,165	31,030	40,788	39,009	73,149	72,062	68,711	70,139	60,267	56,181	59,045	54,635	34,463	19,652	15,270	9,169

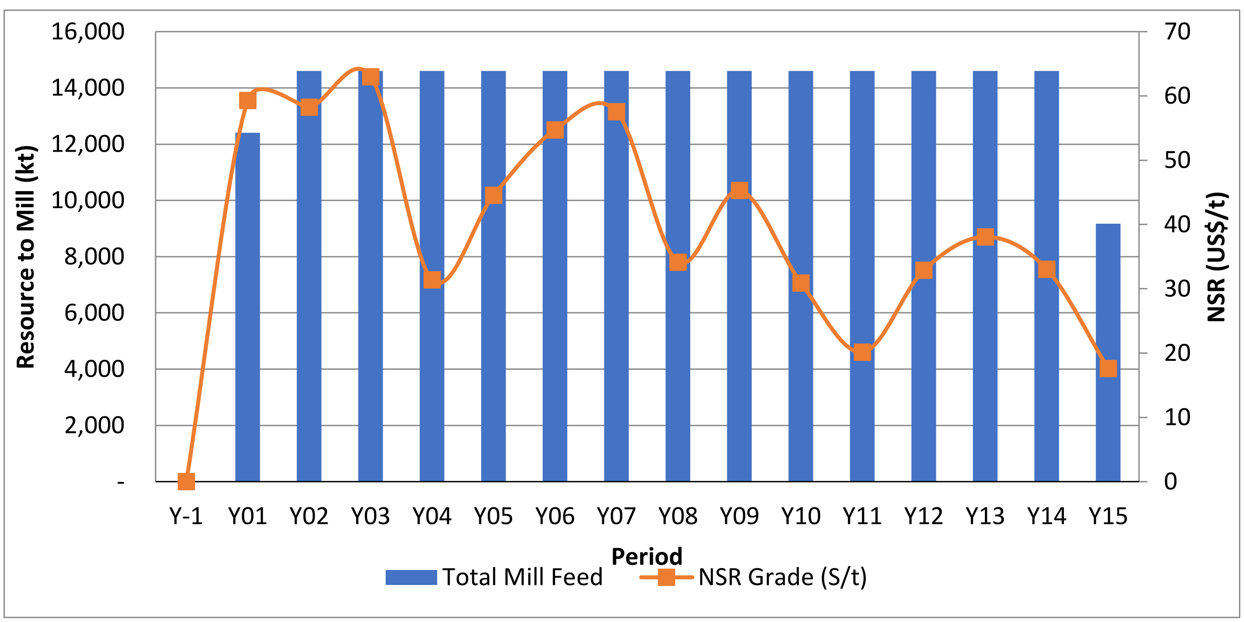
The summarized production schedule results are shown in Figure 13-7 to Figure 13-8.

Figure 13-7: Material Movement by Year



Source: MMTS, 2026

Figure 13-8: Mill Feed



Source: MMTS, 2026

13.9 Mine Operations

The mining operations are planned to be typical of similar-scale open pit operations in mountainous terrain. The mine will operate 365 days per year, 24 hours per day with two 12-hour shifts per day. An allowance of 10 days of no production has been built into the mine schedule to allow for adverse weather conditions.

In-situ rock is drilled and blasted on 10 m benches to create suitable fragmentation for efficient loading and hauling of both resource and waste rock. It is assumed that overburden material does not require drilling and blasting. Electric rotary drills are used for production drilling, with support from diesel-powered rotary drills. Tertiary diesel-powered down-the-hole (DTH) drills are also included for highwall geotechnical drilling, and for pre-shearing or buffer blasting on the pit walls.

Powder factors of 0.30 kg/t in mill feed and 0.25 kg/t in waste rock are proposed. The blasting activities are planned to fall under a contract service agreement with the explosive supplier. The supplier will provide the blasting materials and technology for the mine, including the blasting trucks. An explosive plant has not been scoped for this project, with the assumption that materials will be delivered to site from other nearby regional plants. A mixed emulsion type of explosive is assumed. The supplier mixes explosives on site. The Owner’s blasting crew takes delivery of the blasting materials, loads the holes, and performs the blasting operations. It is recommended to conduct drill penetration and blast fragmentation studies on the various Whistler rock types as part of further project engineering.

Blasthole cutting will be assayed on regular intervals, with results informing an operational grade control block model, which will influence short to medium range mine planning. Systems on-board the drills and shovels will allow for delineation of mineralization from waste at the face, based on the grade control plans from the technical services team.

All mill feed, waste rock and overburden require loading from the open pits into haul trucks. Electric hydraulic shovels are the primary loading units. Diesel-powered hydraulic shovels are included in the fleet to load mill feed and to provide flexibility to the fleet. A wheel loader is also specified for rehandling material, loading overburden, pit clean up, road construction, snow removal, or as an alternate to load trucks in the pit if periodic low shovel availability requires it. The wheel loader will also support ROM pad activities, as needed.

Fuel and electricity will both be distributed into the open pit to power the mobile mine fleet.

Mill feed, waste rock, and overburden will be hauled out of the pit and to scheduled destinations with diesel-powered rigid-frame haul trucks.

Mine pit services include:

- haul road maintenance
- shovel face maintenance
- pit floor and ramp maintenance
- stockpile and RSF maintenance

- mobile fuel and lube services
- electric cable support for shovels and drills
- ditching
- dewatering
- secondary blasting and rock breaking
- snow removal
- reclamation and environmental control
- lighting
- transporting personnel and operating supplies.

Direct mining operations and mine fleet maintenance are planned as an Owner's fleet with direct operating costs falling under mine operations. General mine expense (GME), or indirect supervision and departmental costs for mine operations, mine maintenance and mine technical services, also falls under mine operations.

The number of hourly mine operations personnel, including maintenance crews, peaks at 280 people. Due to the shift rotation, only one quarter of full personnel complement will be on shift at a given time. Approximately 40 salaried people will be required for mine operations, including the mine and maintenance supervision, mine engineering and geology.

13.10 Mining Equipment

The mine equipment descriptions are based on typical fleet contingents utilized in other large scale North American open-pit mine operations. It should be expected that equipment specifications and fleet sizes will be altered with further project engineering and optimization.

Production drilling will be carried out with 230 mm electric driven rotary drills, with support from 228 mm diesel driven rotary drills. Highwall control and depressurization drilling will be carried out with 160 mm down-the-hole (DTH) drills.

Electric powered hydraulic shovels (34 m^3 bucket) are proposed as the primary loading units to keep operating costs as low as possible. Diesel-powered hydraulic face shovels (22 m^3 bucket) are proposed to support the larger shovels, based on their ability to minimize losses and dilution along waste and mineralization contacts, and to navigate into tighter mining areas. A front-end wheel loader (24 m^3 bucket) is proposed to provide additional loading flexibility, and to provide the ability to load the crusher when required.

Rigid-frame haulers (230-tonne payload) are proposed. This truck size strikes a balance between flexibility to mine on smaller pit benches and in selective mining scenarios but are not so small that the fleet size is excessive. While the current mining plan envisions a diesel-powered haul truck fleet, it is expected that at the time of a construction decision, sufficient electric alternatives will be available for purchase. A large portion of the life-of-mine hauls are downhill loaded to the ROM pad, tailings facilities and WRSF's, which is an ideal scenario for an electric powered fleet.

Graders will be used to maintain the haul routes for the haul trucks and other equipment within the pits and on all routes to the various waste storage locations and the crusher. Rigid-frame haul trucks outfitted with a water tank (75,000 L) are included for haul road maintenance. Track dozers (450 kW and 271 kW) are included to handle waste rock at the various waste storage locations and to support the in-pit activities, including cutting roadways to access the upper benches of the open pit. Wheel dozers (370 kW) will support maintenance at the shovel faces, and along the pit floors. Front-end wheel loaders with a cable reeler attachment will support electrical distribution to the drills and shovels. Hydraulic excavators (4.5 m^3 and 2.0 m^3 bucket) are included as pit support, grade control support, and pioneer mining support. Articulated haul trucks (40 t payload) are included as a support hauler for overburden, as well as accessing smaller pit mining areas such as pit bottoms of initial bench access when diving. Custom fuel/lube trucks are included for mobile fuel/lube support. Various small mobile equipment pieces are proposed to handle all other pit service and mobile equipment maintenance functions.

Pits will be dewatered via gravity drainage out of horizontal drilled holes in the pit walls, or with conventional dewatering equipment: submersible pumps placed in-pit bottom sumps, and/or vertical pumping wells established along the pit perimeter. A nominal amount of pumping has been assumed for this pit, based on other regional large scale open pits, but it is recommended to conduct additional hydrogeologic test work and analysis to further refine this estimate in future mine planning. Pit water will be pumped to collection ponds adjacent to the pits, where it will be managed as per the overall site water management plan.

Mine fleet maintenance activities are generally performed in the maintenance facilities located near the plant site. Maintenance for the larger pieces of equipment, such as shovels and drills, is done in the field by a mobile maintenance crew.

Primary mining equipment requirements are summarized in Table 13-8. The equipment classes, as well as number of units, are preliminary scoping level estimates, and modifications in future studies should be anticipated.

Table 13-8: Primary Mining Fleet Schedule

	Y-1	Y01 – Y03	Y04 – Y11	Y12	Y13	Y14	Y15
Drilling							
Electric Rotary Drill 228 mm (9 in) holes	0	2	4	2	2	0	0
Diesel Rotary Drill 228 mm (9 in) holes	2	2	2	2	2	2	0
Loading							
Electric Hydraulic Shovel 34 m^3 bucket	0	1	2	1	1	1	1
Diesel Hydraulic Shovel 22 m^3 bucket	1	2	2	2	1	1	0
Wheel loader 24 m^3 bucket	1	1	1	1	1	1	1
Hauling							
Rigid-frame haul truck 230 t payload	8	12	22	20	12	9	4

14 PROCESSING AND RECOVERY METHODS

14.1 Overview

The Whistler Gold-Copper Project process plant is designed to treat 40,000 t/d of mineralized material from the Whistler gold-copper porphyry deposit. This plant recovers copper, gold and silver by froth flotation and leaching to produce copper concentrate and gold doré over a 14.6-year mine life.

The process flowsheet for the project is based on preliminary metallurgical laboratory testing, as discussed in Section 10, and preliminary economic modelling. The selected unit operations are conventional technologies commonly applied in copper and gold processing plants of similar throughput. The proposed flowsheet includes the following process areas and unit operations:

- Crushing – primary crushing followed by stockpiling with subsequent secondary crushing.
- Grinding – high-pressure grinding roll (HPGR) milling in closed circuit with wet screens, followed by ball milling with cyclone classification.
- Copper flotation – rougher, concentrate regrind and three-stage cleaning.
- Copper concentrate handling – concentrate thickening and filtration.
- Gold leaching – cyanide leach followed by carbon-in-pulp (CIP) adsorption.
- Gold recovery and carbon regeneration – acid washing and subsequent elution of loaded carbon using Anglo-American Research Laboratories (AARL) carbon stripping system followed by electrowinning, filtration, and smelting to produce doré.
- Cyanide destruction and tailings management – tailings cyanide destruction using the International Nickel Company (INCO) sulfur dioxide/air process, followed by tailings thickening.

The life-of-mine (LOM) average feed grades and recoveries are summarized in Table 14-1.

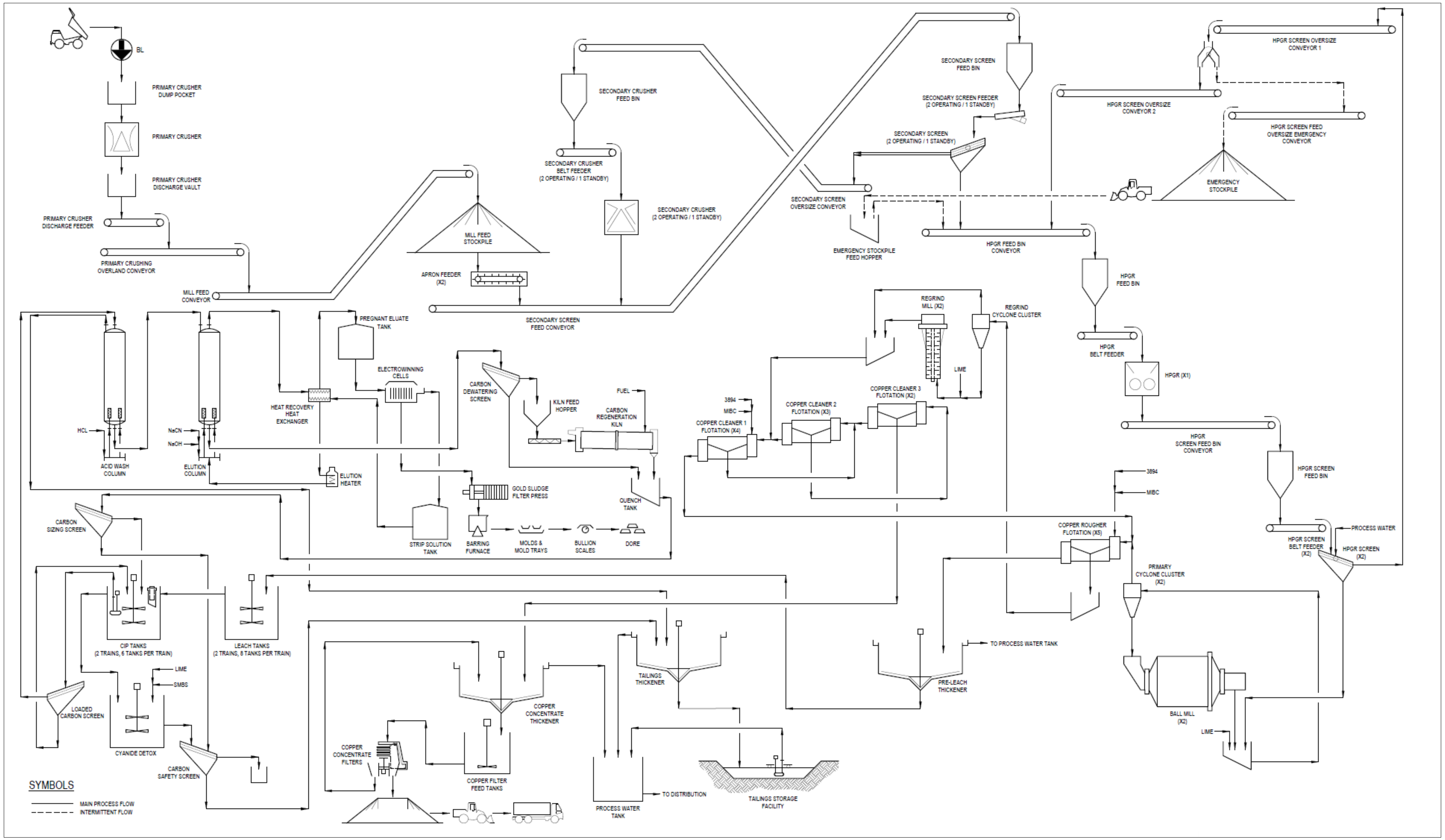
Table 14-1: LOM Average Feed Grades and Recovery

Description	Units	Value
Feed Grade, LOM average		
Copper feed grade	%	0.16
Gold feed grade	g/t	0.44
Silver feed grade	g/t	1.83
Flotation Recovery, at LOM average grade		
Copper recovery to copper concentrate	%	78.1
Gold recovery to copper concentrate	%	59.8
Silver recovery to copper concentrate	%	44.5
Gold Plant Recovery, at LOM average grade		
Gold leach extraction	%	74.9
Silver leach extraction	%	20.5
Gold soluble recovery to doré	%	95.0
Silver soluble recovery to doré	%	95.0
Average Overall Recovery, at LOM average grade		
Copper - average overall recovery over LOM	%	77.8
Gold - average overall recovery over LOM	%	88.9
Silver - average overall recovery over LOM	%	55.6

14.2 Process Flowsheet

The overall process flowsheet for the Whistler Gold-Copper project is shown in Figure 14-1.

Figure 14-1: Overall Process Flow Diagram



Source: Ausenco, 2026

14.3 Process Design Criteria

The main design parameters for the process plant are presented in Table 14-2, and the major process equipment sizing is presented in Table 14-3.

Table 14-2: Process Design Criteria

Description	Units	Value
Design Basis		
Annual throughput (dry)	Mt/a	14.6
Daily average throughput (dry)	t/d	40,000
Plant Operating Basis		
Operating days per year	d/a	365
Operating hours per day	h/d	24
Operating Availability		
Primary crushing	%	75
Secondary crushing, grinding and flotation	%	92
Gold plant	%	92
Tailings thickening	%	92
Concentrate filtration	%	83
Design Grades		
Copper (Cu)	%	0.17
Gold (Au)	g/t	0.56
Silver (Ag)	g/t	2.02
Run-of-Mine (ROM) Characteristics		
Solids specific gravity	-	2.74
Moisture content	%H_2O (wt)	3
Comminution Characteristics		
Crushing work index (CWi), 75th percentile	kWh/t	29.7
JK SMC test parameters (A x b), 25th percentile	-	23.6
Bond ball mill work index (BWi), 75th percentile	kWh/t	22.2
Abrasion index (Ai), 75th percentile	g	0.130
Crushing		
Crushing circuit feed size, F_{80}	mm	635
Crushing circuit product size, P_{80}	mm	41
Stockpile live capacity at reclaim rate	h	12
Grinding		
Grinding circuit product size, P_{80}	µm	120
Ball mill circulating load	%	350
HPGR circulating load	%	100

Description	Units	Value
Copper Flotation		
Rougher residence time, from laboratory test	min	12
Rougher residence time, scale-up factor	-	2.5
Rougher residence time, plant design	min	30
Rougher concentrate regrind product size, P_{80}	µm	15
Regrind specific energy	kWh/t	50
Cleaner 1 residence time, from laboratory test	min	5.0
Cleaner 1 residence time, scale-up factor	-	2.5
Cleaner 1 residence time, plant design	min	12.5
Cleaner 2 residence time, from laboratory test	min	3.0
Cleaner 2 residence time, scale-up factor	-	2.5
Cleaner 2 residence time, plant design	min	7.5
Cleaner 3 residence time, from laboratory test	min	2.0
Cleaner 3 residence time, scale-up factor	-	2.5
Cleaner 3 residence time, plant design	min	5.0
Copper Concentrate Handling		
Copper grade, at design feed grade	%	25.0
Concentrate thickener unit area thickening rate	$t/m^2/h$	0.2
Concentrate thickener underflow pulp density	% solids (wt/wt)	60
Specific filtration rate	$kg/m^2/h$	260
Filter cake moisture	% liquid (wt/wt)	8
Gold Leaching		
Pre-leach thickener underflow pulp density	% solids (wt/wt)	48
Pre-leach thickener unit area thickening rate	$t/m^2/h$	1.0
Leach residence time	h	48
Gold leach extraction	%	74.8
Silver lech extraction	%	20.5
Leach sodium cyanide (NaCN) addition	kg/t leach feed	0.5
Leach lime ($Ca(OH)_2$) addition	kg/t leach feed	1.4
Adsorption residence time	h	6
Gold Recovery and Carbon Regeneration		
Acid wash column	t	15
Elution column	t	15
Cyanide Destruction and Tailings Management		
Cyanide detoxification residence time	min	120
Cyanide detoxification SO_2 addition rate	g SO_2/g CNWAD	5
Cyanide detoxification pH target	-	8 - 9
Tailings thickener unit area thickening rate	$t/m^2/h$	0.7
Tailings thickener underflow pulp density	% solids (wt/wt)	60

Table 14-3: Major Equipment Sizing

Description	Specification	Size
Crushing		
Primary crusher	Gyratory crusher (Superior MK-III 62-75 or equiv.)	1 x 600 kW
Secondary crusher	Cone crusher (HP 900 or equiv.)	3 x 671 kW (2 duty/1 standby)
Grinding		
HPGR	2.40 m Dia. x 2.25 m W	1 x 8,700 kW
Ball mill (overflow discharge)	8.53 m Dia. x 12.50 m EGL	2 x 18,100 kW
Copper Flotation		
Rougher flotation cell	Tank, forced-air	5 x 500 m^3
Cleaner 1 flotation cell	Tank, forced-air	4 x 37 m^3
Cleaner 2 flotation cell	Tank, forced-air	3 x 6 m^3
Cleaner 3 flotation cell	Tank, forced-air	2 x 6 m^3
Concentrate regrind mill	High Speed Stirred mill	2 x 3,800 kW
Copper Concentrate Handling		
Copper concentrate thickener	High-rate	1 x 11 m Dia.
Copper concentrate filter	Pressure filter	1 x 60 m^2
Gold Leaching		
Pre-leach thickener	High-rate	1 x 61 m Dia.
Leach tanks	-	16 x 8,600 m^3
Adsorption tanks	-	12 x 1,400 m^3
Cyanide Destruction and Tailings Management		
Cyanide detoxification tanks	-	2 x 3,310 m^3
Tailings thickener	High-rate	1 x 73 m Dia.

14.4 Plant Design

14.4.1 Crushing

The crushing circuit consists of primary and secondary crushing stages separated by a stockpile.

14.4.1.1 Primary Crushing

The primary crushing circuit is designed for 6,570 operating hours per year, corresponding to 75% availability, at a nominal throughput of 2,222 t/h. The primary crushing facility is located near the pit boundary limits, with a rock breaker adjacent to the primary crusher for handling oversized material.

The primary crushing circuit consists of a direct-fed gyratory crusher operating in an open-circuit configuration. The product from the primary crushing circuit has a design P_{80} of 136 mm, based on the gyratory crusher operating close side setting (CSS) of 120 mm. The primary crushed material reports to the mill feed stockpile.

Run-of-mine (ROM) feed material, with a maximum lump size of 1,200 mm and 80% passing (F_{80}) 635 mm, is directly fed into the crushing circuit by 240 t haul trucks. Two haul trucks can tip directly into the dump pocket simultaneously. The open-circuit gyratory crusher (600 kW) discharges into a discharge vault fitted with a reclaim apron feeder. The apron feeder discharges onto the primary crusher discharge conveyor, which then transfers material to the stockpile feed conveyor, conveying the crushed material from the primary crushing area to the stockpile.

A metal detector is interlocked to stop the stockpile feed conveyor when tramp metal is detected, allowing the contamination to be removed without equipment damage. The stockpile feed conveyor is fitted with a weightometer to measure the mass flow rate of the crushed material reporting to the stockpile.

14.4.1.2 Stockpile and Reclaim

The stockpile and reclaim circuit are designed for 8,059 operating hours per year, corresponding to 92% availability, at a nominal reclaim rate of 1,812 t/h.

The product from the primary crusher is conveyed to a single, conical stockpile with an approximate live capacity of 21,750 t. The stockpile provides 12 hours of live capacity at the process plant nominal throughput of 1,812 t/h, ensuring continuous feed during primary crusher shutdowns. Material is reclaimed from the stockpile via a reclaim tunnel onto the secondary screen feed conveyor. The reclaim system is equipped with ventilation fans, tunnel trolley hoists, and sump pumps.

Crushed material is reclaimed from the stockpile by two variable speed apron feeders operating in a duty-standby configuration. Reclaimed material discharges onto a secondary screen feed conveyor, which transports the primary crushed material to the secondary crushing and screening circuit.

14.4.1.3 Secondary Crushing and Screening

The secondary crushing circuit is designed for 8,059 operating hours per year, corresponding to 92% availability, at a nominal throughput of 1,812 t/h.

The secondary crushing circuit consists of three cone crushers (two duty and one standby), each rated at 671 kW, operating in a closed-circuit configuration with three secondary double-deck vibrating screens (two duty and one standby). The secondary screen deck apertures are 90 mm on the top deck and 60 mm on the bottom deck. The product from the secondary crushing circuit (secondary screen undersize) has a design P_{80} of 41 mm and reports to the downstream grinding circuit.

Reclaimed material is conveyed into a secondary screen feed bin with a 5-minutes of live capacity and divided into three compartments. Each compartment is equipped with a vibrating feeder to distribute the primary crushed material onto the secondary double-deck inclined vibrating screens. Secondary screen oversized material is conveyed to and stored in a three-compartment secondary crusher feed bin with a 15-minutes of live capacity; each compartment is equipped with a belt feeder to direct material to the secondary cone crushers. Secondary crusher discharge reports back to the secondary screen feed conveyor, completing the closed crushing circuit.

14.4.2 Grinding

The grinding circuit consists of a HPGR circuit followed by a ball mill circuit, arranged in an HPGR-B configuration.

14.4.2.1 HPGR and Emergency Stockpile

The HPGR circuit is designed for 8,059 operating hours per year, corresponding to 92% availability, at a nominal throughput of 1,812 t/h.

The HPGR circuit consists of one HPGR unit rated at 8,700 kW, operating in a closed-circuit configuration with two HPGR double-deck vibrating screens. The HPGR screen deck apertures are 15 mm on the top deck and 5 mm on the bottom deck. The product from the HPGR circuit (HPGR screen undersize) has a target P_{80} of 3.25 mm and reports the downstream ball milling circuit.

The HPGR feed bin conveyor delivers secondary crushed material and material reclaimed from the HPGR emergency stockpile to the HPGR feed bin, which has 15 minutes of live capacity. A belt feeder directs material from the HPGR feed bin to the HPGR for size reduction. HPGR discharge gravitates to the HPGR screen feed bin conveyor and reports to the two-compartment HPGR screen feed bin, which has 2 hours of live capacity, prior to screening. Each bin compartment is equipped with a belt feeder to direct material to the vibrating screens. The HPGR screens are supplied with process water to improve fines separation and limit dust generation. Screen oversized material reports back to the HPGR feed bin conveyor via two transfer conveyors equipped with a divertor chute, allowing material to be optionally directed to the HPGR emergency stockpile. The emergency stockpile has a maximum capacity of 7,245 t and is reclaimed using a front-end loader, which feeds the emergency stockpile feed hopper supplying the HPGR feed bin conveyor during periods of upstream disruption.

14.4.2.2 Ball Mill

The ball mill circuit is designed for 8,059 operating hours per year, corresponding to 92% availability, at a nominal throughput of 1,812 t/h.

The ball mill circuit consists of two parallel ball mills, each rated at 18,100 kW, operating in a closed-circuit configuration with two cyclone clusters at a design circulating load of 350%. The product from the ball mill circuit (cyclone overflow) has a target P_{80} of 120 µm and reports to the downstream copper flotation circuit.

For each parallel milling line, the HPGR screen undersized material gravitates to the cyclone feed pump box, where it is combined with the ball mill discharge. The slurry is conditioned with slaked lime ($Ca(OH)_2$), and process water is added to maintain a target slurry density of 60% solids by weight prior to being pumped to the cyclone cluster. The cyclone underflow returns by gravity to the ball mill feed chute at a design solids density of 75%, while cyclone overflow reports to the copper flotation circuit.

14.4.3 Copper Flotation

The copper flotation circuit is designed for 8,059 operating hours per year, corresponding to 92% availability, at a nominal throughput of 1,812 t/h. Equipment sizing is based on flotation testwork conducted by Base Metallurgical Laboratories Ltd.

The flotation circuit consists of rougher flotation, followed by concentrate regrinding and three stages of cleaning to produce a saleable copper concentrate grading 25% Cu.

Grinding cyclone overflow is combined with recycled first cleaner stage tailings and fed to a rougher flotation bank comprising five forced-air mechanical tank cells, each with a volume of 500 m^3. Frother and collector are added to each rougher cell. Rougher concentrate is pumped to the regrind circuit, while rougher tailings report to the gold leaching circuit.

The regrind circuit consists of two parallel lines, each containing a regrind cyclone cluster operating in open circuit with a stirred grinding mill rated at 3,800 kW per mill. In each line, rougher concentrate feeds the cyclone cluster, with fine particles reporting to the cyclone overflow and coarse particles reporting to the cyclone underflow. The cyclone underflow slurry, together with added slaked lime, is pumped to the regrind mill at 40% solids for size reduction. Regrind mill discharge combines with the cyclone overflow and is pumped to the cleaner flotation circuit at a target size of P_{80} of 15 µm.

The cleaner circuit consists of three stages of cleaning. The first cleaner stage consists of four forced-air mechanical tank cells (37 m^3 per cell), while the second and third cleaner stages consist of three and two forced-air mechanical tank cells (6 m^3 per cell), respectively. Frother and collector are added at the head of each cleaner stage. Concentrate recovered from the first cleaner stage is pumped to the second cleaner stage for further upgrading, while the tailings report back to the rougher flotation. The tailings from the second cleaner stage recycle to the first cleaner stage, while the concentrate advances to the third cleaner stage. The tailings from the third cleaner stage recycle to the second cleaner stage, while the final concentrate advances to the copper concentrate handling circuit.

14.4.4 Copper Concentrate Handling

The copper concentrate handling circuit is designed for 8,059 operating hours per year, corresponding to 92% availability, with concentrate filtration operating 7,271 hours per year, or 83% availability, to accommodate batch filter press operation. The circuit is sized to accept peak copper concentrate production of approximately 15 t/h.

The copper concentrate thickening and filtration circuit consists of a single 11 m diameter high-rate thickener and one vertical filter press equipped with 10 horizontal plates, providing a total filtration area of 60 m^2.

In this circuit, concentrate from the third cleaner stage is pumped to the copper concentrate thickener with flocculant addition. The thickener overflow reports to the process water tank, while copper concentrate solids settle and are collected at an underflow density of approximately 60% solids. The thickener underflow is pumped to an agitated storage tank through a trash screen to remove oversized debris that could damage or block the pressure filter ports.

The storage tank provides 24 hours of surge capacity, allowing filter maintenance to be conducted without affecting mill throughput. The filter feed is pumped to the pressure filter to produce a concentrate cake with 8% moisture. Filtrate from the filter reports back to the copper concentrate thickener. High-pressure air for the concentrate filter is supplied by one dedicated air compressor and stored in a single air receiver for drying and pressing.

The copper concentrate cake is discharged from the filter and conveyed to a covered load-out stockpile with seven days of storage capacity and is then loaded by front-end loader into containerized highway haulage trucks for off-site transport.

14.4.5 Gold Leaching

The gold leaching circuit is designed for 8,059 operating hours per year, corresponding to 92% availability, at a nominal throughput of 1,812 t/h. Equipment sizing is based on Base Metallurgical Laboratories Ltd. leaching testwork.

The gold leaching circuit consists of a pre-leach thickener, followed by two parallel trains of leach tanks and two parallel trains of CIP adsorption tanks. The tailings from the adsorption circuit report to the cyanide destruction circuit, while loaded carbon is transferred to the gold recovery circuit.

Pre-leach thickening is performed by a single 61 m diameter high-rate thickener. The tailings from the copper rougher flotation are thickened to remove residual flotation reagents prior to feeding the leach tanks. Flocculant is added to the thickener feed stream to enhance settling. The thickener overflow reports to the process water tank, while the underflow is collected at 48% solids and pumped to the leach tanks.

The leach circuit consists of two parallel trains, each containing eight leach tanks (8,600 m^3 per tank), providing a total residence time of 48 hours. Air is sparged into the leach tanks to provide adequate oxygen for leaching. Slaked lime is added to the leach tanks to maintain the pH at the desired set point of approximately 11. Sodium cyanide (NaCN) is added to the first leach tank to maintain a target cyanide concentration of approximately 1 g/L.

The adsorption circuit consists of two parallel trains, each containing six CIP tanks (1,400 m^3 per tank), providing a total residence time of 6 hours. For each train, leach discharge is fed to the first CIP tank, while fresh or regenerated carbon is added to the last CIP tank. Carbon is advanced counter-currently to the process slurry flow via carbon slurry transfer pumps, increasing gold loading until the carbon reaches the first CIP tank. Each CIP tank is equipped with a mechanically swept carbon retention screen to retain the carbon while allowing the slurry to flow by gravity to the downstream tank. Slurry from the last CIP tank reports to the cyanide detoxification tanks. Loaded carbon is transferred from the first CIP tank to carbon elution via the loaded carbon screen using a recessed impeller pump.

14.4.6 Gold Recovery and Carbon Regeneration

The gold recovery and carbon regeneration circuit is designed for 8,059 operating hours per year, corresponding to 92% availability.

Gold recovery is performed using an AARL carbon stripping system, which consists of an acid wash, followed by a cold cyanide wash, prior to elution and subsequent carbon regeneration. The pregnant eluate containing soluble gold reports to the electrowinning circuit for gold recovery prior to filtration and smelting into final doré bars. Regenerated carbon is returned to the adsorption circuit via the carbon sizing screen.

Prior to stripping, gold-loaded carbon from the adsorption circuit is conditioned in a single acid wash column (15 t capacity) to improve elution efficiency. The acid wash is performed using a weak hydrochloric acid (HCl, 3% wt/wt) solution to remove alkaline scale and salts. The acid washed, gold-loaded carbon is then hydraulically transferred to the elution column (15 t capacity) for cold cyanide wash and elution. The cold cyanide wash is performed using a solution of sodium cyanide (NaCN, 2% wt/wt) and sodium hydroxide (NaOH, 2% wt/wt) to remove adsorbed copper. Following the cold cyanide washing, gold is stripped from carbon using a solution of NaCN (2% wt/wt) and NaOH (2% wt/wt), heated to 90°C using a heat recovery heat exchanger supplemented by an elution heater. The pregnant solution, containing gold and silver desorbed from the carbon, is then washed from the column into the eluate tank by pressurized water heated to 120°C. Pregnant solution from the eluate tank reports to the electrowinning circuit, while the barren carbon is transferred to the carbon regeneration circuit.

Following the elution cycle, the stripped carbon is hydraulically transferred from the elution column to a dewatering screen prior to regeneration. Dewatered carbon is fed to a rotary kiln operating at 750°C for carbon surface re-activation. Regenerated carbon is then cooled with water in the carbon quench tank, mixed with fresh carbon as required, and returned to the adsorption circuit.

Pregnant eluate is pumped from the pregnant eluate tank through a bank of electrowinning cells, where an electric current is applied across the cells, causing gold to deposit on the surface of the cathodes. After completion of the electrowinning cycle, the deposited gold is washed off the cathodes and dewatered using a manually operated filter press. Following filtration, the gold sludge is dried and mixed with flux before smelting in a barring furnace to produce final doré bars for sale. Barren stripped solution from the electrowinning cells reports to the strip solution tank for recycle to the elution column.

14.4.7 Cyanide Detoxification and Tailings Management

The cyanide destruction and tailings management circuit is designed for 8,059 operating hours per year, corresponding to 92% availability.

The circuit consists of two parallel cyanide detoxification tanks, followed by a tailings thickener. The dewatered tailings slurry reports to the CDSF, while recovered process water is recycled to the process water tank for reuse.

The tailings from the adsorption circuit are fed to the two parallel cyanide detoxification tanks, where the slurry remains for a total of 120 minutes. The circuit is designed to decrease weak acid dissociable cyanide (CNWAD) concentration from 200 mg/L to no more than 5.0 mg/L, with a design target of 1.0 mg/L. Cyanide destruction is accomplished using the sulfur dioxide/air method, with detoxification reaction carried out at a pH of 8–9, maintained by the addition of slaked lime. Sulfur dioxide (SO_2) is supplied in the form of sodium metabisulfite (SMBS), while the entrained copper in solution serves as a catalyst. The cyanide destruction tanks are equipped with air sparging and mechanical agitation to ensure thorough mixing. Following detoxification, the slurry is discharged to a carbon safety screen to remove any entrained carbon prior to reporting to the tailings thickener, together with acid wash column discharge.

Tailings thickening is performed using a single 73 m diameter high-rate thickener. The slurry is thickened to recover process water and reduce tailings volume prior to discharging to the CDSF. Flocculant is added to the thickener feed stream to enhance settling. The thickener overflow reports to the process water tank, while the underflow is collected at 60% solids and pumped to the CDSF.

14.5 Reagents Handling and Storage

The reagent handling system includes unloading and storage facilities, mixing and storage tanks, and feeding equipment as required for each plant reagent. Each set of compatible reagents is located in a dedicated containment area to prevent environmental contamination and inadvertent mixing of incompatible reagents. Appropriate ventilation, fire protection, safety protection, eyewash stations, and safety data sheet (SDS) stations are provided throughout the reagent handling areas. Sumps and sump pumps are provided for each containment area for spillage management and control.

The specific requirements and purpose of each reagent used on site are described in Table 14-4.

Table 14-4: Reagents Handling and Storage

Reagent	Preparation Method	Use
Quicklime	Received as powder in bulk bags or tanker trucks; slaked with raw water and transferred to a storage tank. Distributed to process plant as a slurry	pH control
Flocculant	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to thickeners throughout the plant	Settling aid
Aerophine 3894	Received in totes; dosed neat into flotation	Collector
Methyl isobutyl carbinol (MIBC)	Received in IBC totes in solution; dosed neat into flotation	Frothing aid
Sodium cyanide	Received as briquettes in bulk bags or isotainers; dissolved and transferred to a storage tank; distributed to leaching circuit	Gold lixiviant
Sodium hydroxide	Received in solution form in totes; distributed to elution circuit	pH modifier
Hydrochloric acid	Received in solution form in totes; distributed to elution circuit and eluate tanks	Carbon washing
Sodium Metabisulfite (SMBS)	Received as powder in bulk bags; dissolved and transferred to a storage tank; distributed to cyanide detoxification circuit	Cyanide destruction reagent
Activated carbon	Received in solid granular form in bulk bags; added to the carbon quench tank at start-up and as required	Gold adsorbent

14.6 Energy, Water, and Process Materials Requirements

14.6.1 Process Materials

The estimated annual consumption based on nominal usage for major plant reagents is summarised in Table 14-5.

Table 14-5: Annual Consumption for Major Reagents

Reagent	Annual Consumption (t/a)
Quicklime	28,791
Flocculant	1,164
Aerophine 3894	626
Methyl isobutyl carbinol (MIBC)	234
Sodium cyanide	8,621
Sodium hydroxide	1,170
Hydrochloric acid	719
Sodium Metabisulfite (SMBS)	10,132
Activated carbon	508
Flux additives	2

14.6.2 Water Requirements

The process plant requires raw water, fresh water, and process water to maintain the overall plant water balance. Raw water is supplied to the site as untreated source water and conditioned as required to produce fresh water for plant distribution. Process water is recovered internally and reused where possible.

14.6.2.1 Raw Water

Raw water is supplied to the site from the designated source and stored in a raw water tank. Raw water is used directly for applications that do not require conditioning such as dust control and as feed to the freshwater system.

14.6.2.2 Fresh Water

Fresh water is produced from raw water and stored in the freshwater tank, from which it is distributed to various applications across the process plant, including reagent preparation, process makeup water, and gland seal water. Total freshwater consumption is 1,150 m^3/h.

14.6.2.3 Gland Seal Water

Gland seal water is supplied from the freshwater tank and pumped to various users across the plant site. Total consumption for gland seal water is 49 m^3/h.

14.6.2.4 Process Water

Overflow from the copper concentrate thickener, pre-leach thickener and tailings thickener are collected in the process water tank and reused within the plant. Total consumption for process water is 4,088 m^3/h.

The main process water applications include HPGR wet screening, Ball milling, and copper concentrate filter wash water.

Process makeup water requirements are 971 m^3/h. This makeup water is supplied from freshwater system and is included in the total freshwater requirement of 1,150 m^3/h.

14.6.3 Air Requirements

High-pressure air at 750 kPa is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant. Compressed air filtration demand is generated by dedicated filter compressors.

14.6.4 Power Requirements

The estimated installed load for the crushing plant and process plant is 77.5 MW. The estimated nominal operating demand is 56.7 MW.

14.6.5 Personnel Requirements

The estimated personnel requirements for the process operations are 133 people. This includes 16 people for the crushing facility, 74 people for the process plant, and 43 people for maintenance.

15 INFRASTRUCTURE

15.1 Introduction

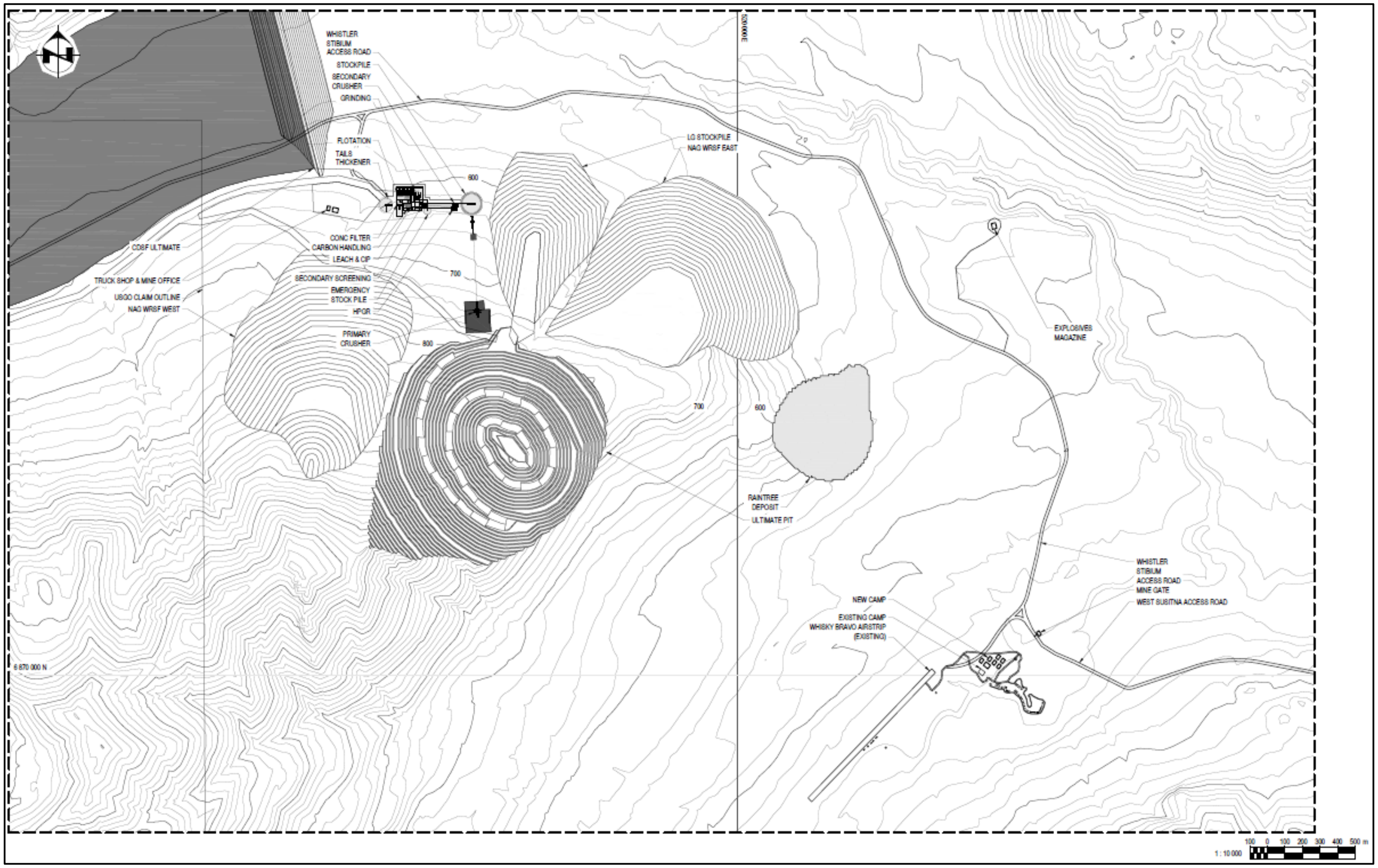
The on-site infrastructure required for the Project includes:

- Whistler Stibium Access Road
- Site roads and laydowns
- Two NAG WRSFs
- Low-grade stockpile
- ROM stockpile
- Mine office and mine dry
- Truck shop, maintenance shop, and warehouse
- Mine rescue facilities
- Diesel storage and distribution
- Power plant and site electrical distribution
- Co-disposal (tailings and PAG waste rock) storage facility (CDSF)
- Water management structures
- Explosives magazine.

The off-site infrastructure required for the Project includes:

- Whiskey Bravo airstrip (existing)
- Power transmission line from Beluga Power Plant
- West Susitna Access Road (WSAR)
- Ship loading facility upgrades at Port Mackenzie.

Figure 15-1: General Arrangement



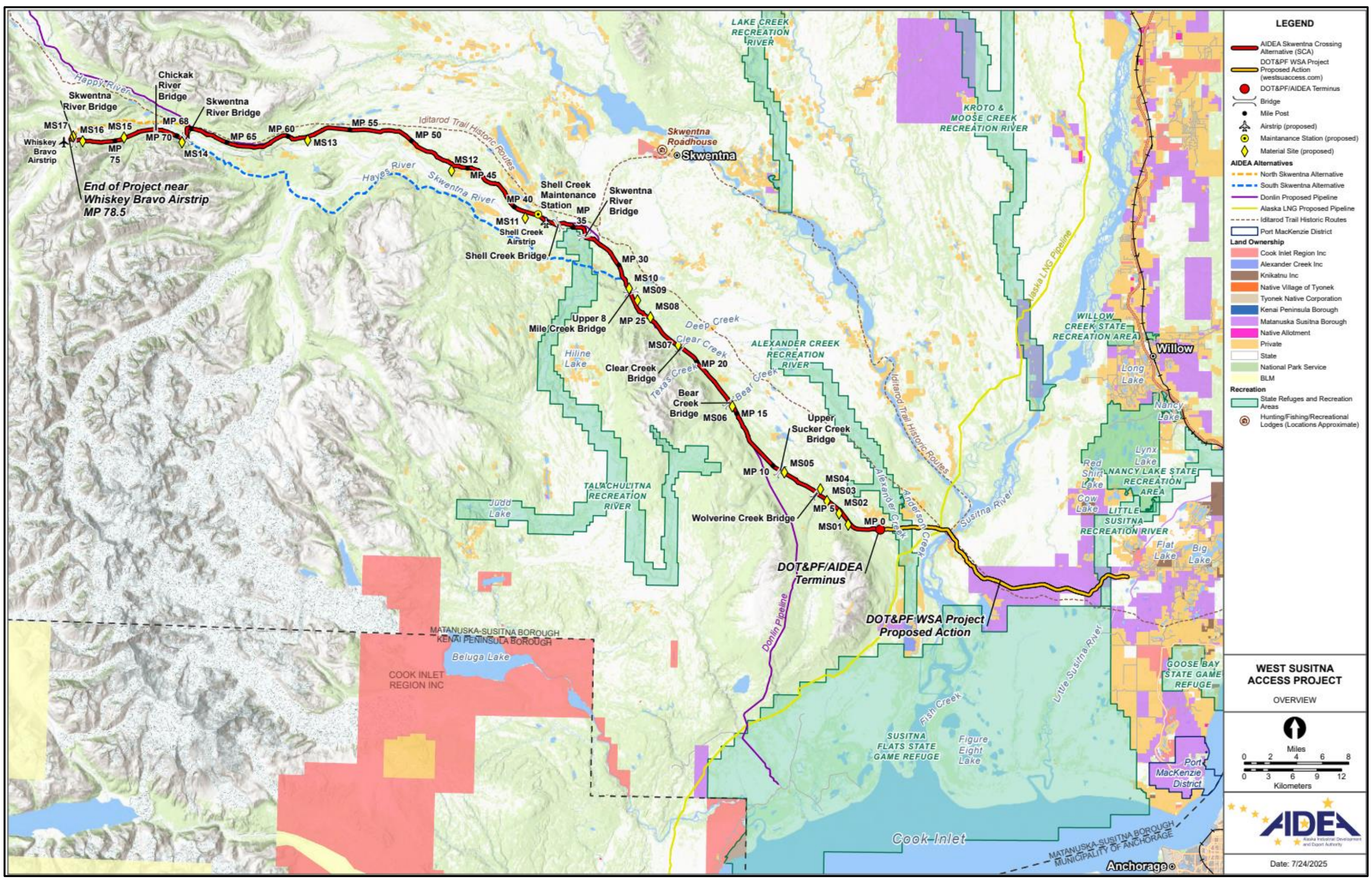
Source: Ausenco, 2026

15.2 Site Access

The Project site is currently accessible by flying into the Whisky Bravo airstrip. The existing exploration camp is located to the east of the airstrip, where pick-up trucks or side-by-side ATVs are available. The site can then be accessed via local exploration roads.

Construction of the WSAR is expected to commence in 2026 or 2027 with completion of the road expected by 2030. Once the WSAR is completed, the Whistler site will be accessible by vehicle from Anchorage or Port Mackenzie. Employees, fuel, reagents, supplies, and concentrate will then be transported via the WSAR. Figure 15-2 illustrates the WSAR route starting from the Matanuska-Susitna (Mat-Su) Borough and ending at the Whiskey Bravo airstrip.

Figure 15-2: WSAR Overview Map



Source: AIDEA, 2025

15.3 Built Infrastructure

15.3.1 On-site Roads

The Project requires multiple haul roads to be constructed. The haul roads will connect the Whistler open pit with the crusher, WRSFs, low-grade stockpile, truck shop, and CDSF. On-site access roads will also need to be constructed. The Whistler Stibium Access Road will connect the airstrip and camp to the process plant and CDSF. Access to the explosives magazine will branch off of the Whistler Stibium Access Road.

15.3.2 Process Plant Building

The process plant building will be a pre-engineered building and will house the grinding, flotation, reagent, concentrate filtration, gold plant, and load-out areas. Additional support buildings will be located on the process plant site.

15.3.3 Accommodation

A construction camp capable of accommodating 560 people will be assembled from prefabricated modules. Additional modules will be added if required. The camp will include a central complex with dining, kitchen, and recreational facilities. For operations, the construction camp will be updated to accommodate 280 operations staff.

15.3.4 Support Buildings

Additional support buildings will be required. These include the explosives magazine, primary crushing, gatehouse, e-rooms, mine office, truck shop, warehouse, and laboratory buildings.

15.4 Stockpiles

When mill feed is mined from the pit, it will either be delivered to the crusher, the ROM stockpile located next to the crusher, or the low-grade stockpile.

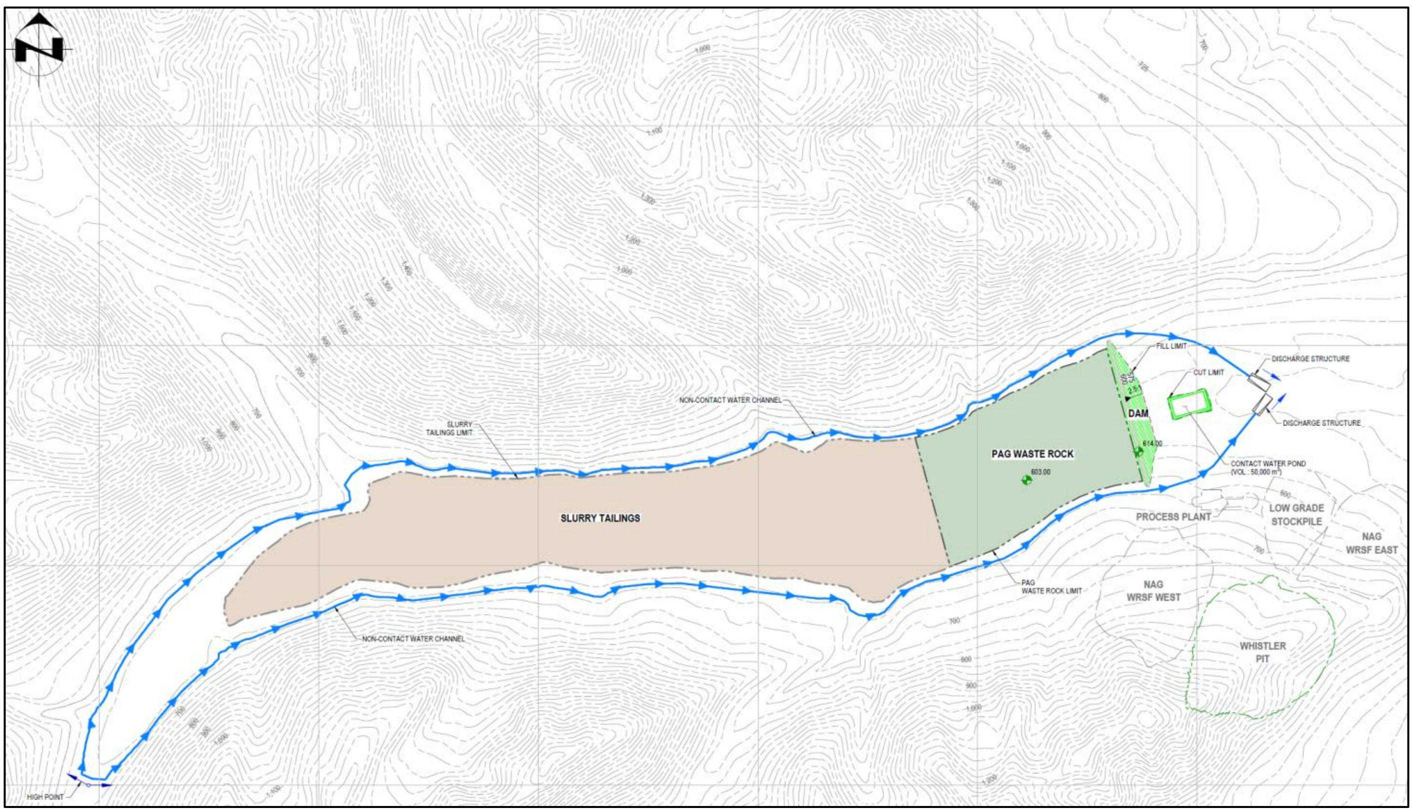
A cutoff grade strategy has been employed for the production schedule, and during operations a stockpile near the crusher will be maintained to store lower-grade mill feed for later rehandling back the crusher. The low-grade stockpile has a max capacity of 32 Mt. Material will be rehandled using wheel loaders and haul trucks and delivered to the crusher using ramps built into the stockpile, no oxidation is expected over the LOM.

The stockpile is built at an overall slope of 2.5H:1V and is 150 m tall. A swell factor of 30% from bank conditions is estimated for material placed in the low-grade stockpile.

15.5 Co-disposal Storage Facility (CDSF)

A desktop siting and waste material deposition trade-off study was conducted to evaluate potential sites and disposal methods for tailings and PAG waste rock. Several potential storage sites were identified for slurry and filtered tailings along with PAG waste rock across the site. Ultimately, it was decided to proceed with the co-placement of life-of-mine slurry tailings and PAG waste rock in the CDSF. Slurry tailings (tailings) and PAG waste rock will be permanently stored in the CDSF, located west of the open pit and process plant, while utilizing the natural topography to minimize the need for dam fill material and reduce the overall footprint. PAG waste rock will be stored subaqueously and/or covered with tailings to eliminate air contact. PAG waste rock will be placed against the embankments to also act as a buttress for the embankment. Process tailings will be placed against the PAG waste rock buttress creating beaches toward the center where the rest of PAG waste rock will be stored subaqueously and/or covered with tailings (Figure 15-3).

Figure 15-3 CDSF General Arrangement



Source: Ausenco, 2026

The primary design objectives for the CDSF are the secure confinement of tailings, subaqueous deposition, and/or tailings cover of PAG waste rock to prevent potential ARD and metal leaching (ML) along with the protection of regional groundwater and surface water during mine operations and in the long-term post-closure.

The design of the CDSF and associated water management facilities have taken into account the following:

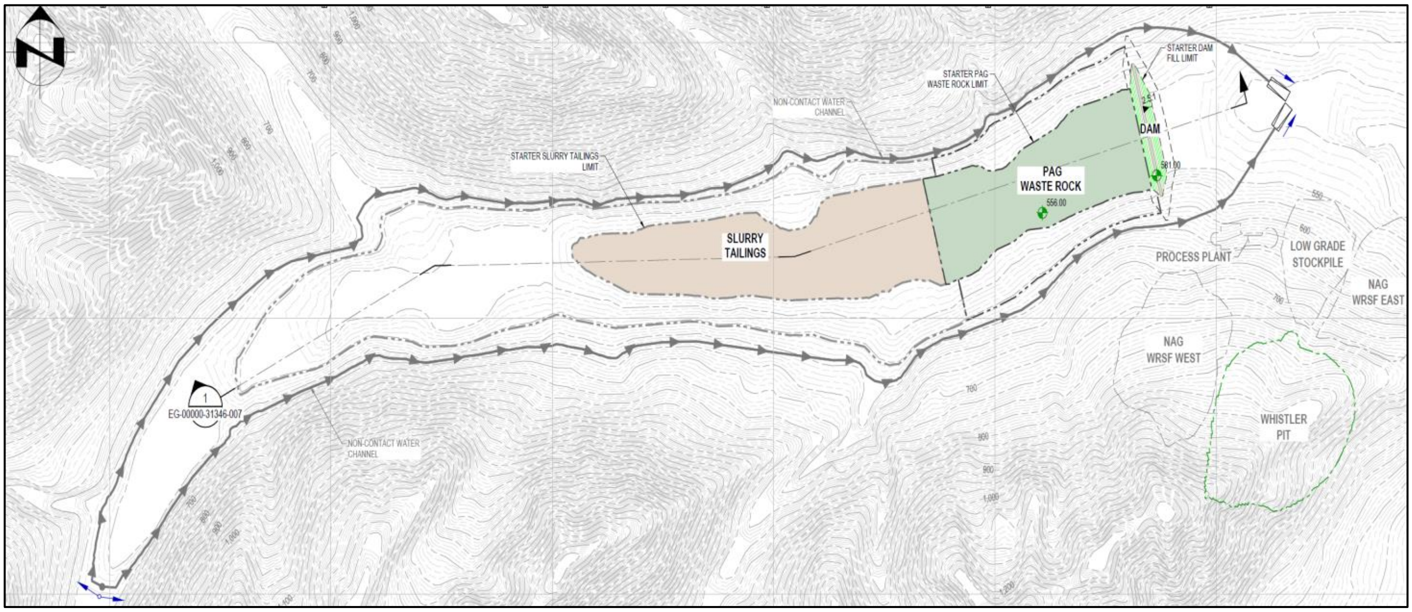
- Staged development of the facility over the life of the project
- Flexibility to accommodate operational variability in the waste rock and tailings (plant shutdowns, deposit variability, and placement during variable climate conditions)
- Control, collection, and removal of contact water from the facility during operations for reuse as process water to the maximum practical extent.

The design criteria for the CDSF consider the following requirements for slurry tailings and PAG waste rock while ensuring a minimum of 3 m of water cover and/or tailings cover over the PAG waste rock to prevent acidification:

- Tailings storage requirement: approximately 214 Mt
- PAG waste rock storage requirement: approximately 304.3 Mt
- Tailings dry density: 1.4 t/m^3
- Waste rock dry density: 2.0 t/m^3
- Tailings embankment will be built with NAG waste rock with a clay core (NAG till) at the center
- Subaqueous deposition and/or tailings cover of PAG waste rock to limit air contact
- Limiting watershed disturbance to a single catchment basin
- Limiting impacts on wildlife and fisheries resources.

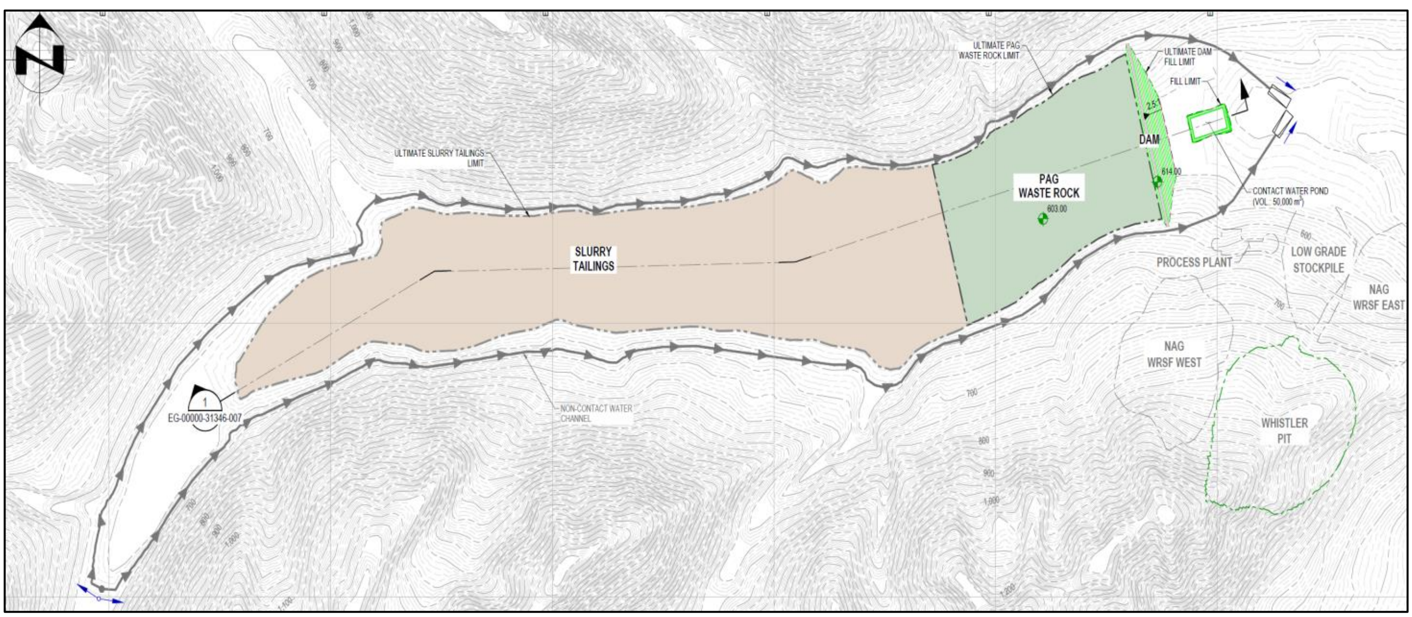
The starter CDSF has been designed to contain the first two years of tailings production and PAG waste rock of 14.9 Mt (Figure 15-4) and the ultimate CDSF has been designed to contain the LOM tailings production and PAG waste rock of 304.3 Mt (Figure 15-5).

Figure 15-4 CDSF Starter Facility



Source: Ausenco, 2026

Figure 15-5 CDSF Ultimate Facility



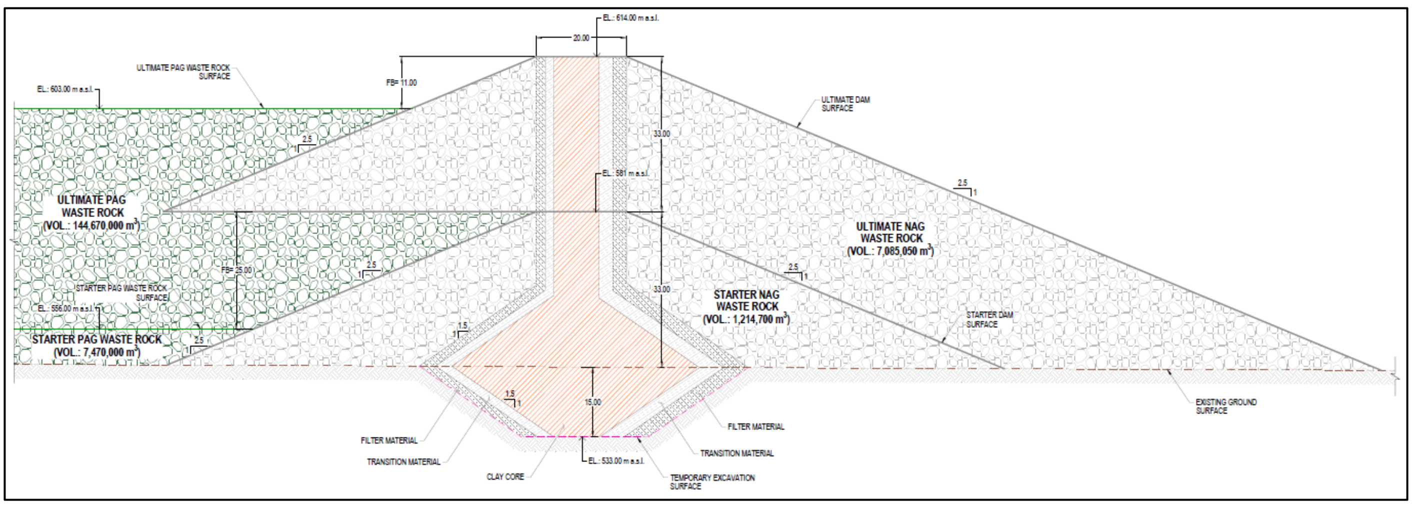
Source: Ausenco, 2026

15.5.1 CDSF Design and Construction

The CDSF footprint will be logged and cleared for foundation preparation and embankment construction. Basin preparation will include the removal of soft overburden material from low points within the topography. Soft overburden materials will be removed beneath the embankment foundation prior to fill placement. The focus of material removal is expected to be within low points. A foundation drainage network will be developed within the base of the embankment using selective placement of waste rock and dual wall HDPE pipe wrapped in a nonwoven geotextile fabric.

The starter embankment will be constructed by NAG waste rock with a clay core (NAG till) during the pre-production period using centerline raise methodology. NAG waste rock and till will be transported to the CDSF by haul trucks and placed in a borrow source stockpile for embankment construction materials. They will then be spread and compacted with dozers and compactors into 1 m lifts. The embankment will be constructed with overall 2.5:1 (H:V) slopes with a 20 m wide crest which provides a stable configuration under both static and dynamic loading based on the stability analyses (Figure 15-6). The construction will continue in the same manner until the end of the project.

Figure 15-6 Cross-section for CDSF Embankment



Source: Ausenco, 2026

The design standards for the CDSF are based on the relevant provincial, federal, and international guidelines for the construction of mining tailings storage facilities in Alaska. Dam breach assessment and dam hazard classification were excluded from the scope of work of this study. However, regulations and guidelines, such as the Inflow Design Flood (IDF) and the Earthquake Design Ground Motion (EDGM), Global Industry Standard on Tailings Management Technical (GISTM, 2020) and Bulletin—Application of Dam Safety Guidelines to Mining Dams (CDA, 2019), were used to speculate the dam hazard classification and suggested minimum target levels for some design criteria. Based on the expected area of inundation downstream of the CDSF, the consequence of a dam failure is expected to be "Very High" for the CDSF according to CDA 2007 (revised 2013). Therefore, the facility was designed following the recommended parameters in these guidelines.

To be conservative, the inflow design flood (IDF) used for the design of the CDSF during operations and post-closure is the probable maximum flood (PMF) for a “Very High” dam classification. The spillway for the CDSF is designed to accommodate the PMF. The perimeter diversion channels are designed to capture runoff above the channels for the 100-year, 24-hour storm event and safely redirect it around the CDSF. The EDGM parameters have been determined for the CDSF using the U.S. Geological Survey (USGS) National Seismic Hazard Model via the Unified Hazard Tool. The design earthquake for the CDSF is the maximum credible earthquake (MCE) for a “Very High” dam classification post-closure.

15.5.2 Stability Analysis

A section through the highest portion of the embankment was selected as the critical section. Stability of the embankment was assessed using the limit-equilibrium modelling software Slope/W, (Geostudio, 2018). Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than the minimum required values in accordance with GISTM and CDA guidelines of 1.5 FOS for static and 1.0 FOS for pseudo-static. The CSDF embankment is designed to withstand potential dynamic displacement without release of tailings during the maximum design earthquake event. The embankment stability analyses exceeded both static and pseudo-static GISTM and CDA guidelines.

15.5.3 Geotechnical Instrumentation and Monitoring

Instrumentation and monitoring will be required to assess the performance of the embankment and must be incorporated in the next phase of the study. Vibrating wire piezometers will be installed to monitor pore pressure within the CDSF, and permanent embankment fill materials, and slope inclinometers and survey monuments will be installed in the embankment to monitor slope movement and deformation.

15.5.4 CDSF Closure

The ultimate CDSF will occupy an area of approximately 822 ha in a watershed of approximately 16,047 ha. The spillway will be constructed in bedrock. The embankment will be covered with NAG waste rock and soil-vegetative cover to protect against erosion. Similarly, a waste rock cover and soil-vegetative cover will be placed over the tailings to reduce infiltration and promote surface runoff to the spillway.

15.6 Waste Storage Facilities

The waste rock management plan includes the storage of all the mined material below theUS$13.40/t NSR cutoff value (waste). Details on the management of NAG and PAG waste rock can be found in Section 13.5.

15.7 Power and Electrical

The Project requires a dedicated electrical power supply to support the process plant, on-site infrastructure, and select mining equipment. The power system is designed to deliver up to 105 MW of peak electrical load over the LOM. The average annual operating load for the process plant and on-site infrastructure is estimated at 57 MW. The average annual operating load for mining operations is 20 MW.

Power will be supplied by new transmission line. A conceptual design for the transmission line envisions a route from the Beluga Power Plant to the Project site, partially following the WSAR where possible. The Beluga Power Plant is owned and operated by the Chugach Electric Association (CEA). The unit price of electricity used for this study is US$0.08225 per kWh, which includes customer, demand, and energy rates current as of Q4 2025, as published by the CEA.

15.8 Fuel

On-site diesel storage is designed to accommodate two weeks of fuel. Weekly consumption for the mining equipment is estimated at 500,000 L with site mobile equipment estimated at 12,000 L. Diesel fuel will be delivered by truck via the WSAR.

The fuel price used for this study is US$3.79 per gallon. This is based on the monthly three-year trailing average for Rocky Mountain (PADD IV) diesel price as published by the U.S. Energy Information Administration (EIA).

15.9 Water Supply and Management

The closest weather station to the site is located in Skwentna, Alaska and is operated as part of the National Climatic Data Center network, with data available in historical climate normals for the 1981-2010 period. Table 15-1 summarizes the available climate data.

Table 15-1 Climate Normals (1981-2010) from Skwentna Alaska Station (USW00026514)

Month	Total Precipitation (mm)	Total Snowfall (mm)	Mean Temperature (C°)	Evaporation (mm)
Jan	58.7	459.7	-12.5	0.0
Feb	56.4	454.7	-9.6	0.0
Mar	25.9	243.8	-5.1	0.0
Apr	26.9	152.4	1.4	15.3
May	28.4	2.5	8.2	31.0
Jun	32.0	0.0	13.2	32.0
Jul	56.9	0.0	14.8	56.9
Aug	87.9	0.0	12.9	87.9
Sep	108.0	7.6	7.9	51.7
Oct	81.5	284.5	0.1	1.3
Nov	56.1	566.4	-8.7	0.0
Dec	88.9	858.5	-11.2	0.0
Annual	707.6	3030.2	1.0	276.1

Water for potable and process use will be sourced from noncontact and surface waters diverted away from the CDSF. The water will be diverted by water management channels, directed towards the process plant, and pumped into the freshwater tank for distribution. The estimated freshwater consumption for the process plant is 1,150 m^3/h. This includes reagent preparation, process water makeup, and gland seal water.

Contact water from the CDSF will report to a settling pond. Contact water from mining, WRSFs, and stockpiles will be intercepted in perimeter ditching or site grading and conveyed to collection ponds. Contact water will then be treated, if required, before discharging to the environment or used as makeup water. The major water management features are illustrated in the CDSF General Arrangement.

Additional work is required to evaluate the pit dewatering requirements, and the potential need to treat water from the dewatering wells prior to discharge to the environment. Noncontact water or treated contact water will be discharged to Portage Creek, a stream that originates at Stoney Glacier and flows east to the Skwentna River.

Assumed pit inflows and pit dewatering have not been considered in the context of this study.

15.9.1 Hazard Considerations

15.9.1.1 Seismic Risk

The Project is located in one of the most seismically active regions of North America, with USGS hazard maps indicating elevated ground-shaking potential across South-Central Alaska due to active crustal and subduction-zone fault systems. Updated models show higher seismic hazard levels than previous assessments, reflecting improved understanding of fault slip rates and ground-motion behavior.

<u>Mitigations:</u>

- Use updated USGS National Seismic Hazard Model (NSHM) parameters in structural design for all permanent infrastructure.
- Design critical structures for high resilience, incorporating ductile frames, base isolation or supplemental damping where appropriate.
- Conduct detailed geotechnical investigations to assess liquefaction potential and apply ground-improvement methods (densification, deep foundations, geosynthetics) where needed.
- Install real-time seismic monitoring with automated shutdown protocols for facilities such as processing plants or pipelines.

15.9.1.2 Geohazards (Landslides, Rockfall, Debris Flow)

Alaska commonly experiences geologic hazards including landslides, erosion, and ground failure, driven by steep terrain, tectonic uplift, and rapidly changing climate conditions. State surveys highlight widespread slope instability, while recent studies document over 1,000 slow-moving landslides triggered by permafrost thaw, glacier retreat, and extreme weather.

Mitigations:

- Complete detailed terrain and geohazard mapping to avoid locating infrastructure on or below unstable slopes.
- Realign access roads and corridors away from geologic structures prone to movement.
- Implement slope stabilization where required (scaling, rock bolting, retaining structures, surface drainage improvements).
- Install monitoring systems, such as GPS survey prisms, InSAR, or tilt meters, for slopes near critical facilities.
- Design drainage infrastructure to divert runoff away from potentially unstable slopes.

15.9.1.3 Cryospheric Hazards (Permafrost, Snow, Ice, Freeze–Thaw)

Alaska's snow, ice, and permafrost conditions pose risks to infrastructure, and climate warming - occurring at roughly twice the U.S. average - continues to alter freeze–thaw behavior, ground stability, and hydrology. These changes can affect foundation performance, ice-road reliability, and drainage systems. Regional permafrost mapping indicates that permafrost in the Whistler area is discontinuous and the probability of encountering near-surface permafrost in the area is 10% or less according to the USGS (USGS, 2026).

Mitigations:

- Conduct permafrost characterization using thermal modeling and boreholes (if required) prior to infrastructure placement.
- Use thermally stable foundation designs (e.g., air-gap foundations, thermosyphons, insulated pads) to limit thaw.
- Plan for variable freeze–up and thaw timing by designing all-season access roads where possible rather than relying solely on ice roads.
- Strengthen drainage control (ditching, culverts, diversion channels) to manage meltwater and prevent erosion.
- Enhance snow-management plans including snow fencing, plowing strategies, and load-bearing evaluations for roofs.

15.9.1.4 Flooding and River-Related Hazards

Flooding, erosion, and ice-jam events are recognized hazards in Alaska's river systems, with potential to affect crossings, diversions, and water-management infrastructure. Seasonal changes during snowmelt or ice breakup can create high-flow events and rapid riverbank erosion.

Mitigations:

- Set infrastructure back from riverbanks and use armoring (riprap, sheet piles) where erosion is likely.

- Implement flood-resistant water-management infrastructure, including high-capacity spillways, diversion channels, and emergency bypass routes.
- Monitor river conditions seasonally, particularly during breakup, to anticipate erosion or blockage.

15.9.1.5 Weather-Driven Hazards (Winter Storms, Heavy Snow, Extreme Precipitation)

While the region is not impacted by tropical cyclones, Alaska routinely experiences severe winter storms, high winds, heavy snowfall, and extreme rainfall events. These conditions can disrupt operations, restrict access, and trigger secondary hazards such as slope failures.

Mitigations:

- Develop severe-weather response plans, including pre-storm shutdown procedures and access restrictions.
- Design buildings and structures for high snow loads and wind loads typical of South-Central Alaska.
- Maintain all-weather access roads with adequate surfacing, ditching, and snow-storage capacity.
- Equip site with redundant power and communication systems to withstand storm-related outages.
- Use aviation risk protocols recognizing reduced visibility and icing during winter.

15.9.1.6 Avalanche Hazards

In mountainous parts of Alaska, snow and ice avalanches are common geohazards due to steep terrain and high snowfall. These events may affect access routes or infrastructure depending on local topography.

Mitigations:

- Perform avalanche hazard mapping for access routes, camps, and facilities.
- Avoid locating infrastructure in avalanche paths or utilize protective structures such as berms, deflection mounds, or snow sheds.
- Implement seasonal avalanche forecasting and worker-safety protocols.
- Restrict travel during high-risk periods and equip field teams with avalanche PPE (transceivers, probes, shovels).
- Use remote weather and snow-pack monitoring to track changing risk levels.

15.9.1.7 Climate-Driven Change

Accelerated warming, glacier retreat, and permafrost degradation increasingly influence seismic response, slope stability, and hydrologic regimes. These evolving conditions warrant adaptive engineering measures and ongoing monitoring throughout the Project life.

Mitigations:

- Incorporate long-term climate projections into water management, tailings design, closure planning, and site drainage.
- Implement adaptive engineering such as over-design factors, drainage redundancy, and thaw-resilient foundations.
- Establish long-term environmental monitoring, including permafrost temperature profiles, streamflow, slope movement, and weather trends.
- Review hazard controls periodically to adjust for evolving climate risk over the mine life.
- Use flexible operational planning to adjust haul schedules, construction timing, and workforce rotations around changing conditions.

16 MARKET STUDIES AND CONTRACTS

16.1 Market Studies

No market studies or product valuations were completed as part of this study. Market price assumptions were based on a review of public information, industry consensus, standard practice, and specific information from comparable operations.

Copper concentrates are widely traded and can be marketed directly from producer to smelter or via third-party concentrate trading entities. It is assumed that the concentrate contains negligible deleterious elements that would impact marketability.

The market for gold doré is widely traded and can be marketed domestically or internationally with significant optionality regarding the final customer. It is assumed that the doré contains negligible deleterious elements that would impact marketability.

A marketing study was not conducted to determine indicative treatment and refining terms. Marketing, refining, and transportation costs, along with payability terms, were assumed based on a review of information from comparable recent studies. The assumed payability terms and off-site costs for each metal are presented in Table 16-1.

Table 16-1: Off-Take Term Assumptions

Term	Unit	Value
Copper Maximum Payability – Cu Concentrate	%	96.50
Copper Minimum Grade Deduction – Cu Concentrate	%	1.00
Gold Payability – Cu Concentrate	%	95.0
Silver Payability – Cu Concentrate	%	95.0
Gold Payability – Doré	%	99.90
Silver Payability – Doré	%	98.00
Transport Cost – Cu Concentrate	US$/t	160.00
Treatment Cost – Cu Concentrate	US$/t	65.00
Copper Refining Cost – Cu Concentrate	US$/lb	0.065
Gold Refining Cost – Cu Concentrate	US$/oz	5.00
Silver Refining Cost – Cu Concentrate	US$/oz	0.50
Gold Refining, Transport, and Marketing Cost – Doré	US$/oz	2.50
Silver Refining, Transport, and Marketing Cost – Doré	US$/oz	0.50

The QP for this section has reviewed the market analysis and, in the QP's opinion, the data supports the assumptions in this technical report.

16.2 Commodity Price Projections

Project economics were estimated based on long-term flat metal prices of US$4.50/lb Cu, US$3,200/oz Au, and US$37.50/oz Ag, which are based on consensus forecasts from various financial institutions.

The QP notes that the pricing used in the cash flow analysis is reasonably aligned with various long-term forward-looking estimates from major international banks.

16.3 Contracts

Currently, there are no contracts for transportation or off-take of any metal products in place, but when they are negotiated, they are expected to be within industry norms. Similarly, there are no contracts currently in place for the supply of reagents, utilities, or other bulk commodities required to construct and operate the Project.

17 ENVIRONMENTAL STUDIES, PERMITTING, AND PLANS, NEGOTIATIONS, OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

U.S. GoldMining submitted an Application for Permit to Mine in Alaska (APMA) to Alaska's Department of Natural Resources (ADNR) on June 30, 2022. On September 22, 2022, the Alaska Department Natural Resources, Division of Mining, Land and Water, approved multi-year 2022-2026 Exploration and Reclamation Permit Number 2778 for Hardrock Exploration – Skwentna River – Yentna Mining District, and in addition also approved Reclamation Plan Approval Number 2778. Subsequent amendments were approved incorporating alterations to U.S. GoldMining's exploration plans in 2023 and 2024.

U.S. GoldMining commenced environmental studies in August 2022, comprising an aquatics survey completed by Owl Ridge Natural Resource Consultants Inc (Owl Ridge). The aquatics survey is summarized in a report compiled by Owl Ridge (Owl Ridge, 2024), in addition to subsequent work completed during 2023 including surface water quality sampling, additional terrestrial wildlife resources ARD and leachate potential studies, and heritage resources studies. Subsequent work completed in 2024 to date has included water quality sampling, eagle nest mapping, and on ground archaeological surveying.

U.S. GoldMining has developed a Stakeholder Engagement Plan (Parkan, 2023) which provides a comprehensive roadmap to engaging with community and native organizations, regulators and legislators, and special interest groups, to ensure broad consultation with regards to current and future activities at the Project. Currently U.S. GoldMining does not have any ongoing negotiations or agreements signed with respect to stakeholders.

17.1 Environmental Considerations

17.1.1 Baseline and Supporting Studies

17.1.1.1 Archaeology

Baseline heritage and archaeological assessments were conducted for the Project to support compliance with the Alaska Historic Preservation Act (AS 41.35.200) and requirements of the APMA. A Phase I heritage and archaeology assessment, including cultural sensitivity modeling, was completed for priority areas in 2023 to identify areas with elevated potential for undiscovered cultural resources and to minimize the risk of disturbance during exploration activities. Background research and sensitivity modeling indicated higher sensitivity at lower elevations, particularly along river corridors and benches (Owl Ridge Natural Resource Consultants, Inc., 2024a).

Field investigations were deferred in 2023 due to seasonal timing constraints, and a State of Alaska cultural resources investigation permit was obtained for post-snowmelt surveys conducted in 2024. Pedestrian surveys completed in 2024 did not identify cultural resources within the areas evaluated, including road cuts, ridge lines, and previously disturbed exploration areas. Several locations identified as moderate to high potential through sensitivity modeling were downgraded following field verification due to prior disturbance (Owl Ridge Natural Resource Consultants, Inc., 2024). Lower-elevation benches along the Skwentna River and similar high-potential areas outside the current deposits may require additional survey prior to future expansion into those areas.

17.1.1.2 Terrestrial Mammals

Small mammals, including mice, lemming, pine marten, marmot (whistle pig), fox, and beaver, are prevalent throughout the Whistler area. Larger ungulates are present at low densities, with moose observed intermittently year-round, typically numbering one to four individuals. Caribou are associated primarily with higher-elevation terrain and are not considered regular users of the Project site; no caribou have been observed on site since 2007, although seasonal regional movement remains possible (Lamborn, 2012).

Black bears are common within the Project area, with an estimated five to ten individuals present during snow-free months. They primarily use lower-elevation habitats in spring and shift to higher elevations and ridge areas later in summer as berry availability increases. Denning locations do not appear to be concentrated in any specific areas (Lamborn, 2012).

Grizzly bears are less common, with an estimated zero to five individuals present in the Whistler area. They tend to occupy lower-elevation creek bottoms and exhibit seasonal movement patterns, migrating down-valley in spring and up-valley in late summer in response to salmon availability, with hibernation typically occurring at higher elevations. Wolves are believed to transit the area occasionally, based on infrequent winter track observations. Lynx tracks observed during winter suggest a small resident population of approximately one to four individuals (Lamborn, 2012).

17.1.1.3 Aquatic Life

Exploration in the Whistler area has largely been conducted using helicopter support, which has minimized ground-based access to streams and resulted in limited historical fisheries interaction and data collection. Available observations indicate the presence of aquatic insects in freshwater streams throughout the area (Lamborn, 2012). All local streams are potentially salmon-bearing, with no barriers to fish migration.

Dolly Varden are common in clear-water creeks within the region and are inferred to migrate seasonally, likely overwintering outside the immediate area and following salmon upstream during summer months (Lamborn, 2012).

King Salmon historically spawned in Portage Creek, with a small population estimated at fewer than 1,000 individuals; however, numbers have declined and no King Salmon were observed during the 2012 field season. Silver (Coho) Salmon spawn in the Skwentna River and are present in the Happy River, though regional runs are small (fewer than 5,000 individuals), and they do not migrate into Portage Creek. Red (Sockeye) Salmon and other salmon species have not been observed within the Project area (Lamborn, 2012).

17.1.1.4 Avian Survey

Aerial surveys for bald eagle (*Haliaeetus leucocephalus*) nesting sites were conducted in the vicinity of the Project, including areas near Portage Creek and the Skwentna River, from June 21 to 24, 2023. Survey transects were developed using ArcGIS with 0.15-mile spacing, excluding high-elevation areas with low nesting potential. Surveys were flown by helicopter with two observers scanning on either side of the aircraft at speeds of approximately 35–45 knots. Observations were recorded using digital field mapping tools and photographic documentation (Rmelman, 2024).

The survey area consisted primarily of black spruce, paper birch, cottonwood, green alder, and trembling aspen forest, with alpine habitat over portions of the proposed Whistler Pit. Survey timing occurred after deciduous leaf-out, which limited visibility of lower tree sections. No active or inactive bald eagle nests or raptors were observed during the aerial survey or associated field activities. Nests of other avian species and tree deformities resembling nests (witches' brooms) were observed (Rmelman, 2024).

17.1.1.5 Wetlands and Soils

A desktop evaluation of soils, wetlands, and vegetation was conducted for the Whistler and Raintree prospect areas and surrounding Whisky Bravo camp and facilities. Available National Wetlands Inventory (NWI) mapping, color infrared imagery, soils data, and field-verified wetlands from the West Su Access Road study (HDR 2021) were reviewed. While NWI coverage in the Whistler claim block is limited and outdated, the 2024 Owl Ridge analysis applied supervised classification of high-resolution imagery and foliar cover modeling of black spruce and dwarf birch to identify potential wetland areas. This approach produced a high certainty "core wetlands" dataset and a more conservative dataset that likely overestimates wetland extent (Cartier, 2024).

Both datasets indicate that the Whistler and Raintree areas have low potential for wetlands, particularly at higher elevations and on steep slopes. These results provide a stronger foundation for high-level planning, including preliminary facility siting and estimating potential wetland impacts. However, the analysis is not sufficient for permitting and should be complemented with updated NWI mapping and ground-based wetland delineation studies, including functional assessments, as the project advances (Cartier, 2024; Owl Ridge Natural Resource Consultants Inc., 2024b).

17.1.1.6 Geochemistry

A screening-level geochemical assessment of rock assemblages at the Project was completed to evaluate acid-generating potential and the risk of acid rock drainage and metalliferous leaching under potential future development scenarios. The assessment assumes total sulfur occurs predominantly as sulfide minerals, with pyrite used as the representative sulfide, and that the available dataset is broadly representative of materials likely to be encountered during exploration-stage activities.

Based on total sulfur content, lithology, and alteration characteristics, results indicate generally low neutralization potential, consistent with alumino-silicate host rocks and minimal carbonate content. Under standard screening criteria, a substantial proportion of potential waste materials would be classified as potentially acid-generating or Uncertain -- which is typical of porphyry copper and epithermal systems. Elements of potential concern include arsenic, antimony, barium, cadmium, cobalt, copper, iron, lead, manganese, phosphorus, selenium, sulfur, and zinc, with elevated sulfate and total dissolved solids identified as potential water management considerations. Additional geochemical characterization, including formal acid–base accounting and kinetic testing, will be required as the Project advances (Logsdon & Ruckhaus, 2022).

17.1.1.7 Surface Water Quality and Hydrology

Baseline surface water quality monitoring was conducted in the Project area in October 2023 to characterize existing hydrologic and water quality conditions. Owl Ridge Natural Resource Consultants Inc. completed sampling at twelve (12) stream and river locations between October 10 and October 12, 2023, in accordance with the site-specific Field Sampling Plan and Quality Assurance/Quality Control Plan (Fulton & Ruckhaus, G. 2023).

Sampling locations were selected to represent upstream and downstream conditions relative to the Project and included tributaries and mainstem reaches of Portage Creek, Access Road Creek, Camp Creek, and the Skwentna River.

At each site, in-situ water quality parameters were measured, and grab samples were collected for laboratory analysis. Samples were handled, preserved, and transported under chain-of-custody procedures to SGS Laboratories in Anchorage, Alaska. Flow measurements were collected where site conditions permitted.

The described monitoring program provides an initial dataset with baseline surface water quality and hydrologic conditions in the Project area and will support ongoing environmental baseline characterization and future permitting activities (Fulton & Ruckhaus, G. 2023).

17.1.1.8 Subsistence and Traditional Land Use

Subsistence activities are an important component of land use in the region surrounding the Project. Alaska Native communities, regional and village corporations, and federally recognized tribes within the Upper Cook Inlet and Skwentna River areas maintain cultural and economic ties to hunting, fishing, trapping, and seasonal travel. Subsistence resources of importance include salmon, moose, small game, and freshwater fish, and access to these resources is influenced by river corridors, traditional travel routes, and seasonal patterns of use.

Stakeholder engagement and subsistence context for the Project have been documented through regional studies, agency information, and project-specific engagement planning. These efforts identify subsistence-use Indigenous and non-Indigenous communities, general harvesting areas, and considerations for minimizing potential conflicts between exploration activities and traditional land use. This information provides baseline context for future environmental evaluation and permitting and supports ongoing consultation with affected communities and regulatory agencies.

17.1.2 Environmental Monitoring

Environmental monitoring will occur throughout the mine life, as indicated by federal, state, and local regulations. Further post-closure environmental monitoring will align with the requirements for mine closure in Alaska.

Plans for PAG waste rock and tailings management are discussed in more detail in Section 18. The proposed conceptual methods of waste rock and tailings management are currently considered by regulators and standards associations as generally acceptable practices. Preliminary testwork indicates the tailings themselves to be NAG, with some additional neutralizing potential; however, additional testwork will be required.

17.1.3 Water Management

Noncontact water will be diverted away from the site to the greatest practical extent while contact water will be collected, potentially reused, and treated before being discharged to the environment. Additional work is required to evaluate the pit dewatering requirements, and the potential need to treat water from the dewatering wells prior to discharge to the environment. Noncontact water or treated contact water will be discharged to Portage Creek, a stream that originates at Stoney Glacier and flows east to the Skwentna River.

Assumed pit inflows and pit dewatering have not been considered in the context of this study.

17.2 Permitting Considerations

The Project is located on State of Alaska lands administered by the ADNR. The project area is not situated on federal lands, Alaska Native Corporation lands, or tribal trust lands. Mineral tenure is held under state mining claims, and exploration and potential development activities are subject to state regulatory oversight, including authorization through the APMA and associated environmental and reclamation approvals. The potential state, federal, and local permit requirements are illustrated in Table 17-1.

Table 17-1: Potential State, Federal, and Local Permit Requirements

Authority	Permit
Federal	
Environmental Protection Agency (EPA)	• Spill Prevention Containment and Contingency (SPCC) Plan
U.S. Army Corps of Engineers (USACE)	• CWA Section 404 Permit (wetlands dredge and fill) • River and Harbors Act (RHA) Section 10 (structures in navigable waters) • RHA Section 9 (dams and dykes in navigable waters-interstate commerce)
Bureau of Alcohol, Tobacco, and Firearms (ATF)	• License to Transport Explosives • Permit and License for Use of Explosives
Federal Aviation Administration (FAA)	• Notice of Landing Area Proposal (existing airstrip) • Notice of Controlled Firing Area for Blasting
U.S. Department of Transportation	• Hazardous Materials Registration
U.S. Fish and Wildlife Service	• Section 7 of the Endangered Species Act, Consultations requiring a Biological Assessment or Biological Opinion
State	
Department of Natural Resources (ADNR)	• Reclamation Plan Approval • Mining License • Temporary Water Use Authorizations

Authority	Permit
Department of Environmental Conservation (ADEC)	• APDES Water Discharge Permit • Alaska Multi-Sector General Permit (MSGP) for Stormwater • Stormwater Pollution Prevention Plan (part of MSGP) • Sec. 401 Water Quality Certification of the CWA Sec. 404 Permit • Integrated Waste Management Permit • Air Quality Control – Construction Permit • Air Quality Control – Title V Operating Permit • Reclamation Plan Approval • Approval to Construct and Operate a Public Water System
State Office of History and Archaeology (OHA)	• Section 106 National Historic Preservation Act Concurrence
Department of Fish and Game (ADF&G)	• Title 16 Fish Habitat and Passage Permits • Wildlife Hazing Permit
Local	
Alaska Native Corporations	• Land access agreements
Matanuska-Susitna	• Land Use or Development Permits

17.2.1 Required Environmental Permits

Extensive environmental permits will be required as the project moves forward into production. These include permits from the EPA, USACE, U.S. Fish and Wildlife Service, ADEC, and the ADF&G. These permits are listed in Table 17-1.

17.2.2 Current Exploration Permits

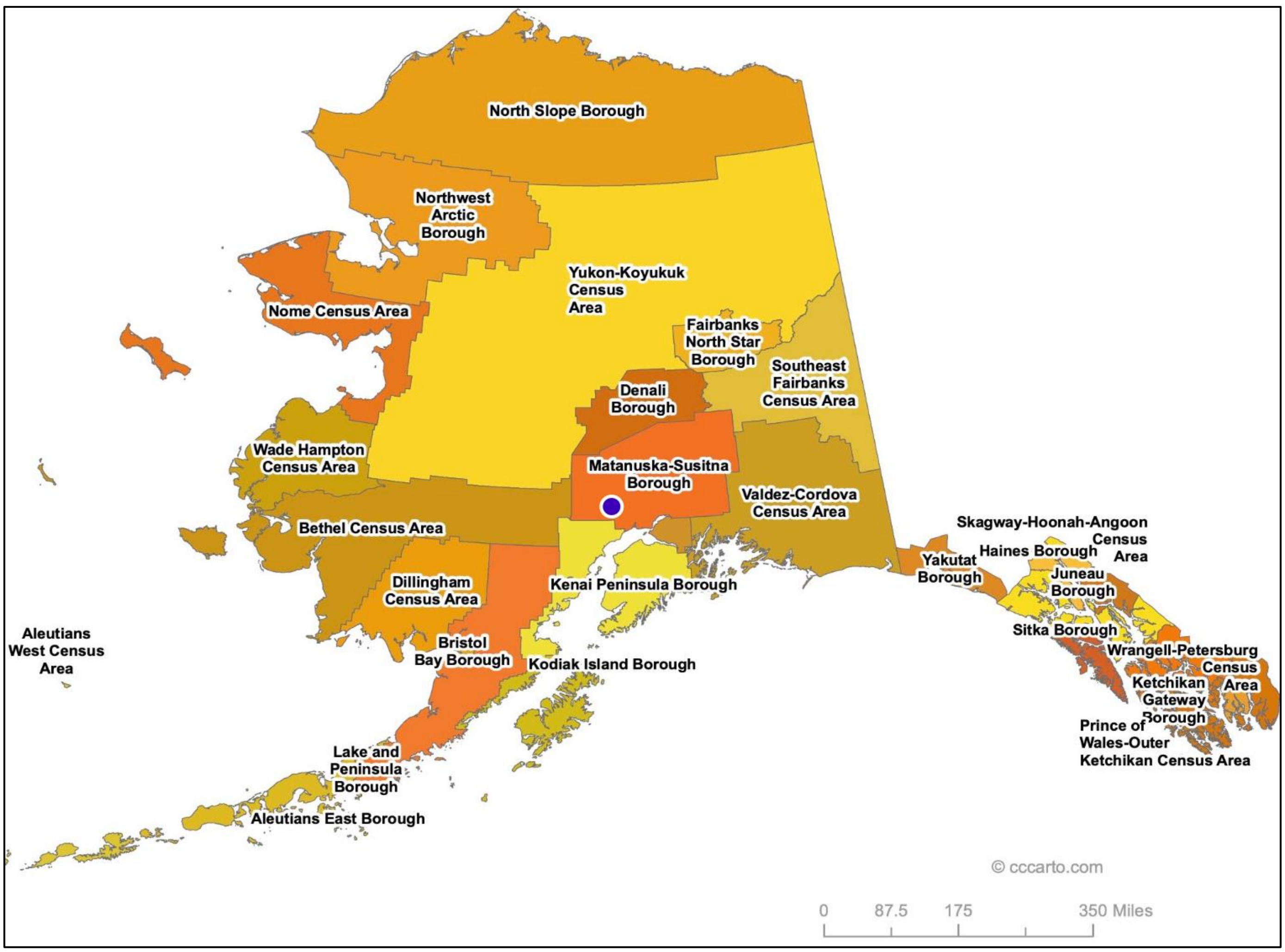
U.S. GoldMining applied for APMA to ADNR on 30 June 2022. On 22 September 2022, the Alaska Department Natural Resources, Division of Mining, Land and Water, approved multi-year 2022-2026 Exploration and Reclamation Permit Number 2778 for Hardrock Exploration – Skwentna River - Yentna Mining District, and in addition also approved Reclamation Plan Approval Number 2778. Subsequent amendments were approved incorporating alterations to U.S. GoldMining's exploration plans in 2023 and 2024.

The Project does not currently have any mining permits other than the exploration permits described above. Specific mining permits will be required from the ADNR and include reclamation plan approval, mining license, and temporary water use authorizations.

17.3 Social Considerations

The Project is located within the Matanuska-Susitna (Mat-Su) Borough, which contains some Alaska Native Claims Settlement Act (ANCSA) lands owned by the Cook Inlet Region, Inc. (CIRI). No Alaska Native Corporation or tribal lands occur within the project boundary. The broader region is traditionally and contemporarily used by Dena'ina/Tanaina Athabaskan communities, including residents of Tyonek and Skwentna, for hunting, fishing, and travel. These connections are based on customary land use within the surrounding watersheds and access corridors rather than land ownership or jurisdictional authority over the project area (Owl Ridge Natural Resource Consultants, Inc., 2023). The Alaskan boroughs and ANCSA lands are illustrated in Figure 17-1 and Figure 17-2 respectively.

Figure 17-1: Alaskan Borough Map



Note: Purple circle denotes approximate project location within Mat-Su Borough. Source: Chubb Custom Cartography, 2026

Figure 17-2: ANCSA Regional Native Corporations Map

Source: University of Anchorage, Alaska, 2026

Land use in the vicinity of the project includes active mineral exploration, guided and unguided hunting and fishing, recreational lodges, air-supported access, and historic travel routes associated with early mining activity in South-Central Alaska. These uses occur primarily outside the project footprint but are relevant to permitting and operational planning due to shared access routes, seasonal activity patterns, and regional public interest.

Stakeholder engagement has therefore focused on communities, regional Alaska Native organizations (including Cook Inlet Region, Inc. and Tyonek Native Corporation), state and federal agencies, recreational and commercial users, and regional trade organizations, with the objective of providing early project awareness, coordinating land use expectations, and supporting future permitting processes. To date, no land tenure conflicts have been identified, and no restrictions on mineral tenure or surface access are currently known that would materially affect exploration activities (Owl Ridge Natural Resource Consultants, Inc., 2023).

The Project is not located near or within any state parks or protected areas, as illustrated in Figure 17-3. The nearest parks and preserves are located approximately 50 km to the north (Donali National Park and Preserve) and 50 km to the southwest (Lake Clark National Park and Preserve).

Figure 17-3: Alaska State Parks and Protected Areas

Note: Purple circle denotes approximate project location of the Project. Source: Vidiani.com, 2026

17.4 Closure and Reclamation Planning

17.4.1 Closure and Reclamation Plans

Closure of the Project will primarily be regulated by the ADNR and Alaska Department of Environmental Conservation (ADEC) under the Alaska Reclamation Act and the Solid Waste Management Regulations. The Act requires that a reclamation and closure plan (RCP) and financial assurance (FA) be provided to the State prior to any mining activity or project development. The RCP details reclamation prescriptions which are designed to minimize or eliminate the risk of pollutants released into the environment. The RCP details conceptual means and methods used to return the site to near pre-mining conditions and protect the environment during, reclamation, and long-term site management activities.

The RCP will be prepared in parallel with mine facility designs, incorporate baseline information studies, and other operational and long-term planning efforts. Submission of the RCP is not expected to be accepted by the State until all Federal actions have been successfully approved. The proponent is required to post a bond for the amount reasonably expected to reclaim the site. Permits to operate the mine will not be granted until the RCP is approved, and the bond is secured and provided to the State of Alaska.

The project will be closed in two phases: the active closure phase and the passive post-closure phase. During the active closure phase, closure reclamation activities for the mining, process plant, infrastructure, and CDSF will take place. Environmental monitoring is assumed to be conducted during the reclamation and post-closure phases. The requirement for water treatment will be further assessed based on water balance considerations and geochemistry source term studies to be completed as the feasibility study progresses. Treatment of water utilizing passive treatment systems will be a consideration during the passive post-closure phase.

17.4.1.1 Mining

The preliminary closure strategy for the mining areas includes:

- Regrading stockpiles to overall slope angles
- Scarification of haul roads
- Placement of 0.3 m of overburden on the tops and sides of the stockpiles and selected areas of the open pit
- Placement of 0.1 m of overburden on the haul roads
- Placement of 0.3 m of topsoil on the tops and sides of stockpiles and selected areas of the open pits
- Placement of 0.1 m of topsoil on the haul roads
- Revegetation of the stockpiles and haul roads.

Mining area closure activities are based on the use of the owner operated mine fleet for completing the activities listed above, except for the revegetation. These activities are planned to occur during the active closure phase.

17.4.1.2 Process Plant and On-Site Infrastructure

In preparation for closure of the mill building, there are four areas that will be addressed:

- Removing all machinery, equipment, building, and structures
- Covering and revegetating concrete foundations
- Disposing of scrap metal
- Disposing of chemicals and reagents.

These reclamation activities are planned to occur during the active closure phase.

17.4.1.3 Co-disposal Storage Facility

Upon closure, the following activities will be initiated with regards to the CDSF:

- Placement of 2.0 m NAG waste rock material on the free crest and downstream slope of the tailings dams.
- Placement of 0.3 m of topsoil on the free crest and downstream slope of the tailings dams.
- Revegetation of the free crest and downstream slope of the tailings dams.
- Redesign of closure spillway.

These reclamation activities are planned to occur during the active closure phase.

17.4.1.4 Water Treatment

Excess water in the water management pond will be treated, if required, and discharged to local waterways during the active closure period. Treatment of water utilizing passive treatment systems will be considered during the passive post-closure period.

17.4.1.5 Environmental

Environmental monitoring will be required during the active and passive closure periods. Monitoring requirements will be based on regulatory requirements and consultation with Indigenous communities.

17.4.2 Closure Cost Estimates

losure liabilities at Whistler have identified the physical reclamation activities detailed below. The estimated closure cost is detailed in Section 18 and assumes the following:

- All disturbed areas as a result of mining activity will be regraded to promote drainage
- All buildings on site will be removed and disposed of in an approved landfill

- Water management structures will be removed, and dams breached
- An annual operating and sustaining capital allowance for site management and maintenance
- An annual allowance for environmental and regulatory compliance
- Indirect costs are included, consistent with State of Alaska policy
- The estimate assumes additive premiums and contingences to account for the level of detailed design, remoteness of the site, and other unknowns.

Mining companies may provide financial assurance using any combination of the following instruments, as long as they meet ADNR regulatory tests. Instruments approved under Alaska law include:

- Surety bond
- Letter of Credit (LOC)
- Certificate of Deposit (CD)
- Corporate guarantee, if financial tests are met
- Cash payments into the Alaska Mine Reclamation Trust Fund (AS 37.14.800)
- Any other mechanism approved by the Commissioner.

17.5 Qualified Person's Opinion on Adequacy of Environmental, Permitting, and Community Plans

In the qualified person's opinion, the environmental studies completed prior to the effective date of this TRS are not adequate to support future permitting. The commencement of comprehensive baseline studies is recommended in Section 23.

In the qualified person's opinion, the current multi-year Exploration and Reclamation Permit Number 2778 for Hardrock Exploration – Skwentna River – Yentna Mining District issued by the Alaska Department Natural Resources, Division of Mining, Land and Water, is adequate for the current exploration work being undertaken on the Project. The Company has a good understanding of the future permitting requirements. The engagement of a permitting consultant is recommended in Section 23 to support the permitting process.

In the qualified person's opinion, the community engagement completed prior to the effective date of this TRS is adequate for the conceptual stage of the Project. Further engagement with government, local groups, and communities is recommended in Section 23 to align the Project with external expectations and requirements, and to develop open and transparent relationships.

18 CAPITAL AND OPERATING COSTS

18.1 Introduction

The capital and operating costs described in this IA are based on open-pit mining operations for the Whistler Gold-Copper Project. The process plant is designed to treat 40,000 t/d of mineralized material, or 14.6 Mt/a, over a mine life of 14.6 years.

18.2 Capital Costs

18.2.1 Overview

The capital cost estimate was developed in Q4 2025 to target a level of accuracy of -30% to +50%, which aligns with an Association for the Advancement of Cost Engineering International (AACE International) Class 5 level estimate. The estimate includes mining, processing, on-site infrastructure, off-site infrastructure, project indirects, project delivery, owners' costs, and provisions. The total initial capital costs for the Project are estimated at US$1,278.6 million, including US$56.3 million of capitalized operating costs, and US$213.3 million of contingency. The LOM sustaining costs are estimated at US$381.1 million, while the closure costs are estimated at US$98.7 million. The capital cost summary is presented in Table 18-1.

Table 18-1: Capital Cost Summary

WBS	Description	Capital Cost (US$M)	Sustaining Cost (US$M)	Total Cost (US$M)
1000	Mining	39.7	319.0	358.7
2000	Crushing and Conveyance	120.4	-	120.4
3000	Process plant	354.8	-	354.8
4000	On-site Infrastructure	187.6	14.2	201.8
5000	Off-site Infrastructure	72.6	35.7	108.3
	Total Direct Costs	**775.1**	**368.9**	**1,144.0**
6000	Project Preliminaries	80.5	1.6	82.1
7000	Project Delivery	122.1	-	122.1
8000	Owner's Costs	31.3	-	31.3
	Total Indirect Costs	**233.9**	**1.6**	**235.5**
	Total Direct + Indirect Costs	**1,009.0**	**370.5**	**1,377.9**
	Contingency	213.3	10.6	223.9
	Subtotal Capital Cost	**1,222.3**	**381.1**	**1,603.4**
	Capitalized Opex	56.3	-	56.3
	Closure Costs	-	-	98.7
	Total Capital Cost	**1,278.6**	**381.1**	**1,758.4**

Note: Totals may not match due to rounding.

18.2.2 Basis of Estimate

The capital cost estimate was developed in Q4 2025 American dollars (US$). The estimate is based on budgetary quotations for equipment from recent advanced studies and execution projects, supplemented with Ausenco's in-house database, and informed by Ausenco's experience from similar operations in North America.

The following data were used as the basis of estimate:

- Mining schedules
- Engineering design by Ausenco, including but not limited to design criteria, equipment lists, and material take-offs (MTOs)
- Budgetary equipment quotes from similar recently completed advanced studies and execution projects
- Additional data such as lang factors and indirect costs from similar recently completed studies and projects.

The estimate also adhered to these parameters:

- No allowance was made for exchange rate fluctuations
- No escalation was added to the final estimate
- No growth allowance was included.

18.2.3 Mine Capital Costs

Mine capital costs have been derived from vendor quotations and operational data collected by other North American open-pit mining operations.

All mine operations site development costs— such as clear and grub, topsoil stripping, haul road construction, crush rock production, explosives storage pad preparation, and stockpile pad preparation— are capitalized. Pit dewatering and depressurization costs are estimated, which includes drilling vertical wells and horizontal holes, pump installations and maintenance.

Pre-production mine operating costs (e.g., all mine operating costs incurred before mill start-up) are capitalized and included in the capital cost estimate. Pre-production pit operating costs include grade control, drill and blast, load and haul, support, and GME (General Mine Expense) costs. GME covers the salaries and departmental overhead costs for mine operations, mine maintenance, and technical services. The Whistler mine plan includes 14.1 Mt of open-pit pre-production mining.

The mine equipment fleet purchases through year five of the project are planned as financing or lease agreements with the vendors. Down payments of 10% and monthly lease payments, at a lease rate of 8% over a 5-year term, are capitalized through the initial and sustaining periods of the project. Expansion and replacement fleet purchases made after year three of the project are assumed under a traditional capital purchase arrangement.

The initial mine fleet will consist of diesel-powered equipment, with partial electrification of the drilling and loading fleets planned in Year 1 of the project. Distribution of electricity to power the in-pit equipment is capitalized.

Equipment pricing is based on new units delivered to the mine, with all transportation, assembly and commissioning costs included. Unit prices for the fleet are from the MMTS equipment cost database, which includes data on recent vendor budgetary quotations within North America.

Estimated fleet spares and estimated initial fuel, lube, and tire inventories are capitalized.

The following items are also capitalized through the initial and sustaining periods:

- Explosives magazine
- high precision GPS and machine guidance systems
- communication radios
- mine survey gear and supplies
- geology, grade control, and mine planning software licenses
- maintenance tooling and supplies
- mine rescue gear and safety supplies
- geotechnical instrumentation
- piping for pit dewatering and culverting materials for haul roads.

Table 18-2 summarizes the Mine Area Capital Cost estimates for the Project. Of the US$96.0 million, US$39.7 million is initial capital costs and US$56.3 is capitalized operating costs. It is the QP's opinion that these estimates are reasonable for the location and planned mine development and can be used for the Study.

Table 18-2: Mining Initial Capital Costs

Description	Initial Capital (US$M)
Mine Development Costs	5.6
Captalized Mine Pre-Prodcution Operations	50.7
Initial Mine Fleet Capital	34.4
Mine Operations Infrastructure	5.3
Total Initial Mining Capital	**96.0**

Note: Totals may not match due to rounding.

18.2.4 Process Capital Costs

The selection and sizing of process equipment requirements was based on process flowsheets and process design criteria as defined in Section 17. All major equipment was sized based on the process mass balance, as dictated by the process design criteria, to develop a mechanical equipment list (MEL). The MEL was then developed through recent budgetary quotations. The remaining value of the equipment list was developed through benchmarking against recent execution projects and advanced studies.

The process plant and infrastructure engineering design was developed to a conceptual level consistent with an IA, allowing for the bulk material quantities (steel, concrete, piping, cables, instruments, etc.) to be derived for the major commodities using lang factors. Plant earthworks costs were derived from MTOs.

There are no sustaining costs associated with the process plant.

The total capital costs for the process plant are US$475.2 million. The capital cost breakdown for the process plant is summarized in Table 18-3.

Table 18-3: Process Plant Capital Cost Breakdown

WBS	Description	Initial Costs (US$M)
2100	Crushing Ancillary	80.0
2200	Crushers	18.6
2300	HPGR	21.8
3100	Ancillary	25.6
3200	Milling	139.2
3300	Flotation	19.2
3400	Regrinding	31.8
3500	Concentrate Handling	7.6
3600	Tailings Thickening	8.8
3700	Gold Plant	122.7
	Total	**475.2**

Note: Totals may not match due to rounding.

18.2.5 Infrastructure Capital Costs

18.2.5.1 On-site infrastructure

The on-site infrastructure costs consist of bulk earthworks, power switchyard and distribution, fuel storage, sewage, potable water, plant and infrastructure buildings, tailings facility and pipelines, permanent camp, and site services and mobile equipment. The total on-site infrastructure costs are estimated at US$187.6 million and are illustrated in Table 18-4.

Table 18-4: On-site Infrastructure Capital Cost Breakdown

WBS	Description	Initial Costs (US$M)
4100	Plant Bulk Earthworks	14.9
4200	Power Switchyard and Power Distribution	33.1
4300	Fuel Storage, Sewage, Potable Water	6.1
4400	Process Plant Building	13.9
4500	Infrastructure Buildings	9.2
4600	Co-disposal Storage Facility	87.1
4700	Tailings Pipeline	4.3
4800	Permanent Camp	10.6
4900	Site Services & Mobile equipment	8.4
	Total	**187.6**

Note: Totals may not match due to rounding.

18.2.5.2 Off-site infrastructure

The off-site infrastructure costs consist of water supply, power supply, and concentrate handling and port equipment. The total off-site infrastructure costs are estimated at US$72.6 million and are illustrated in Table 18-5.

Table 18-5: Off-site Infrastructure Capital Cost Breakdown

WBS	Description	Initial Costs (US$M)
5100	Water Supply	1.0
5200	Power Supply (Transmission Line)	47.1
5300	Concentrate Handling and Port Equipment	24.4
	Total	**72.6**

Note: Totals may not match due to rounding.

18.2.6 Indirect Capital Costs

Indirect costs include project preliminaries and project delivery (or EPCM). Project preliminaries include field indirects (temporary construction facilities, camp, and associated services), commissioning and operational readiness, vendor representatives, spares, and first fills. Project delivery includes engineering services and construction management. Project preliminaries are estimated at US$80.5 million and project delivery is estimated at US$122.1 million. Total indirect costs are estimated at US$202.7 million and are illustrated in Table 18-6.

Table 18-6: Indirect Capital Cost Breakdown

WBS	Description	Initial Costs (US$M)
6100	Field Indirects	49.7
6200	Commissioning and Operational Readiness	4.3
6300	Vendor Representatives	10.9
6400	Spares	4.6
6500	First Fills	10.9
	Total Project Preliminaries	**80.5**
7100	Engineering Services	63.3
7200	Construction Management	58.8
	Total Project Delivery	**122.1**
	Total Indirect Costs	**202.7**

Note: Totals may not match due to rounding.

18.2.7 Owner (Corporate) Capital Costs

Owner costs for pre-production have been estimated by factors. The estimated cost of US$31.3 million includes:

- Owner's project team and expenses
- Administration, finance, insurance and legal fees
  - Including pre-production general and administrative costs
- Environmental consultation and management
- Human resources, recruiting, and training
- Permitting and regulatory compliance activities
- Stakeholder relations
- Site security.

18.2.8 Sustaining Capital Costs

18.2.8.1 Mining Sustaining Costs

Down payments, lease payments, and purchases for the mine equipment fleet scheduled throughout the life of mine are capitalized through the sustaining periods of the project.

The sustaining costs for mining also include the cost of expanding the open-pit mining operation infrastructure, such as pit electrification and distribution, maintenance tooling, radio communications, geotechnical instrumentation, and the mobile fleet spare parts inventory. A fleet management and dispatch system is also added in the sustaining capital period of the project.

Table 18-7 summarizes the Mining Sustaining Cost estimates for the Project.

Table 18-7: Mining Sustaining Capital Costs

Description	Sustaining Capital (US$M)
Mine Fleet	308.1
Mine Operations Infrastructure	11.0
Total Sustaining Mining Capital	**319.0**

Note: Totals may not match due to rounding.

18.2.8.2 Infrastructure Sustaining Costs

The sustaining capital costs for the infrastructure are associated with the expansion of the power plant in Year 3, and the expansion of the CDSF in Years 2, 5, and 8. The total sustaining costs are US$49.8 million and are summarized in Table 18-8.

Table 18-8: Infrastructure Sustaining Capital Costs

WBS	Description	Sustaining Costs (US$M)
4200	Power Switchyard and Power Distribution	14.2
4600	Co-disposal Storage Facility	35.7
	Total Sustaining Costs	**49.8**

Note: Totals may not match due to rounding.

18.2.9 Contingency

Contingency costs account for the difference in costs between the estimated and actual cost of materials and equipment. The contingency is developed based upon the level of study and considers the level of project definition, the source or methodology of the estimates, and the expected accuracy range. It allows the capital estimate to include a provision to cover the risk from uncertainties that may arise in between the time the capital cost was developed compared to the actual costs during construction and pre-production.

The contingency for the Project has been built up by considering each individual WBS area. The total contingency included in the initial capital costs is estimated at US$213.3 million or 20.6% of total direct and indirect costs.

18.2.10 Closure and Reclamation Planning

Closure and reclamation costs for the Project include allocations for:

- Process plant and on-site infrastructure
- CDSF, WRSF and Water Management
- Mining and haul roads
- Long-term environmental monitoring.

The total closure and reclamation costs are estimated at US$98.7 million.

18.3 Operating Costs

18.3.1 Overview

The total operating costs for the Project are estimated at US$20.82/t or US$4,399.8 million over the 14.6-year mine life. These operating costs do not include pre-production operating costs. A summary of operating costs is presented in Table 18-9.

Table 18-9: Operating Cost Summary

Cost Area	LOM Total (US$M)	US$/t milled	% of Total
Mining	1,676.7	7.93	38.1
Process	2,325.8	11.00	52.9
G&A	216.1	1.02	4.9
G&A (Other) - Road Toll and Maintenance	181.2	0.86	4.1
Total	**4,399.8**	**20.82**	**100**

Note: Totals may not match due to rounding.

18.3.2 Basis of Estimate

The following was used to determine the project’s LOM process operating costs in agreement with the cost definition and estimate methodologies outlined below. This basis considers the development of a process plant designed to treat 40,000 t/d of mineralized material. Process unit operations were benchmarked against similar or comparable processing plants to ensure accuracy of cost estimates.

Assumptions made in developing the process operating cost estimate are listed below:

- Mill production is designed to treat 40,000 t/d of mineralized material.

- Process plant operating costs are calculated based on labor, power consumption, and process and maintenance consumables.
- Off-site gold refining, insurance, and transportation costs are excluded, as they are included elsewhere in the financial model.
- Labor rates were sourced from recent execution projects in the region.

Workforce will be comprised of local and regional workers.

Management and administrative staff will be on a 5/2 rotation (5 days in, 2 days out), whereas process and maintenance staff will be on a 14/14 rotation (14 days in, 14 days out).

Management and administrative staff who are not required to be on site will be based out of Anchorage or will work remotely.

- General and administration (G&A) costs were baselined against previous project experience, defined along with specific inputs from US GoldMining.
- No factor for spare parts has been applied to adjust for consumption of fewer spare parts in early years of operation.
- Grinding media consumption rates have been estimated based on the mill feed characteristics.
- Reagent consumption rates have been estimated based on the metallurgical testwork results.
    - Reagents and consumable prices were obtained via email quotes from local vendors.
- Mobile equipment cost includes for fuel, maintenance, and lease price for the equipment.
- The unit rate power cost of US$0.08225/kWh was calculated from published rates by the Chugach Electrical Association (CEA).
- The unit rate fuel cost of US$3.79/gallon based on the regional monthly 3-year trailing average published by the Energy Information Administration (EIA).

18.3.3 Mine Operating Costs

Mine operating costs are built up from first principles assuming an owner managed and operated scenario. Annual mining production tonnes are taken from the Whistler mine plan. Drilling, loading, and hauling equipment hours are calculated based on the capacities and parameters of the planned equipment fleet. These tonnes and hours also provide the basis for blasting consumables and the mine operations support fleet. Simulated hauler cycle times from source pit benches to planned destinations are utilized to inform hauler productivities

Cost inputs are from the MMTS cost database, which includes data on recent vendor budgetary quotations within North America. This includes cost and consumption rates for such inputs as fuel, lubes, explosives, tires, undercarriage, GET (ground engaging tools), drill bits/rods/strings, machine parts, machine major components, and operating and maintenance labour ratios. Labour rates for planned hourly and salaried personnel have been benchmarked to other Alaskan open-pit operations.

Estimated annual and life-of-mine unit mining costs are shown Table 18-10. It is the QP's opinion that the estimates are reasonable for the location and planned mine operation activities and can be utilized for this Study.

Table 18-10 Unit Mine Operating Costs, $/t mined & $/t milled

Operation	$/t Mined	$/t Milled	US$M
Drilling	$0.26	$0.81	$171
Blasting	$0.35	$1.11	$234
Loading	$0.29	$0.91	$191
Hauling	$1.06	$3.31	$699
Support	$0.34	$1.05	$222
Site Development	$0.02	$0.06	$12
Direct Costs - Subtotals	*$2.31*	*$7.24*	*$1,529*
Mine Operations GME	$0.11	$0.35	$74
Mine Maintenance GME	$0.04	$0.13	$28
Technical Services GME	$0.07	$0.21	$45
GME Costs - Subtotals	*$0.22*	*$0.70*	*$147*
Total Operating Cost	**$2.53**	**$7.93**	**$1,677**

Note: Totals may not match due to rounding.

Diesel price of US$3.79/gallon is used. This value includes provision for delivery and storage on site, as well as applicable state and federal taxes.

18.3.4 Process Operating Costs

The process operation cost estimate includes costs relating to the operation of the mill. The process operating costs for the mill comprise of costs associated with reagent and consumable consumption, labor, process mobile equipment, power, and maintenance. The process operating costs are US$11.00/t mined and are illustrated in Table 18-11.

Table 18-11: Process Operating Costs

Cost Area	Average Annual (US$M/a)	US$/t milled	% of Total
Reagents & Consumables	84.2	5.76	52.5
Plant Maintenance	16.6	1.13	10.3
Power	39.2	2.69	24.5
Labor	17.3	1.19	10.8
Process Plant Laboratory	3.0	0.21	1.9
Total	**160.3**	**11.00**	**100**

Note: Totals may not match due to rounding.

18.3.5 Infrastructure Operating Costs

The infrastructure costs are associated with the site access via the WSAR and associated roads. These include the toll road payments and road maintenance costs. These have been categorized as general and administrative (G&A) costs for the purpose of this study.

18.3.6 General and Administrative Operating Costs

The G&A costs for the Project are listed in Table 18-12, which also illustrates the costs estimated at US$1.88/t mined. The major components that make up the G&A costs are the WSAR annual toll payments at 25.2%, road maintenance at 20.5%, and camp and travel at 19.9% of the total G&A costs.

Table 18-12: General and Administrative Costs

Cost Area	Average Annual (US$M/a)	US$/t milled	% of Total
Site Maintenance Materials	0.1	0.01	0.4
Personnel	3.8	0.26	14.0
Human Resources	0.7	0.05	2.7
Mobile Equipment	1.5	0.10	5.5
Health and Safety	0.3	0.02	1.2
Camp and Travel	5.5	0.37	19.9
Environmental	0.4	0.03	1.5
IT & Telecommunications	0.3	0.02	1.2
Contract, Insurance & Legal Services	1.7	0.12	6.2
Administrative Costs	0.5	0.03	1.8
Toll Road Payments	6.9	0.47	25.2
Road Maintenance	5.6	0.38	20.5
Total	**27.45**	**1.88**	**100**

Note: Totals may not match due to rounding.

18.3.7 Owner (Corporate) Operating Costs

The Owner's costs for the Project are categorized as capital costs for the purpose of this study and are described in Section 18.2.7.

19 ECONOMIC ANALYSIS

19.1 Forward-Looking Information

The results of the economic analyses discussed in this section represent forward-looking information as defined under relevant securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes the following:

- Mineral resource estimates,
- Assumed commodity prices and exchange rates,
- Proposed mine production plan,
- Projected mining and process recovery rates,
- Assumptions as to mining dilution and estimated future production,
- Sustaining costs and proposed operating costs,
- Assumptions as to closure costs and closure requirements, and
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed,
- Unrecognized environmental risks,
- Unanticipated reclamation expenses,
- Unexpected variations in quantity of mineralized material, grade, or recovery rates,
- Accidents, labor disputes and other risks of the mining industry,
- Geotechnical or hydrogeological considerations during mining being different from what was assumed,
- Failure of mining methods to operate as anticipated,
- Failure of plant, equipment, or processes to operate as anticipated,
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis,

- Changes to site access, use of water for mining purposes and to time to obtain environment and other regulatory permits,
- Ability to maintain the social license to operate,
- Changes to financing interest rates, and
- Changes to applicable tax rates.

This economic analysis does not include any inferred resources.

19.2 Methodologies Used

The Project has been evaluated using a discounted cashflow (DCF) analysis based on a 5% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the midpoint of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metal prices, discount rate, head grade, recovery, total operating cost, and total capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 18 in Q4 2025 US dollars. The economic analysis has been run on a constant dollar basis with no inflation.

19.3 Financial Model Parameters

19.3.1 Assumptions

The economic analysis was performed using a copper price of US$4.50/lb, a gold price of US$3,200/oz, and a silver price of US$37.50/oz. These metal prices were based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metals price expectation over the life of the Project. No price inflation or escalation factors were considered. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- Construction and commissioning period of two years.
- Total mine life of 14.6 years.
- Cost estimates in constant Q4 2025 US dollars with no inflation or escalation factors considered.
- Results based on 100% ownership with a 3.0% net smelter return (NSR) royalty applied to the mineral reserve.
- Capital costs funded with 100% equity.

- All cash flows discounted to start of construction period using mid-period discounting convention.
- All metal products are sold in the same year that they are produced.
- Project revenue is derived from the sale of gold doré and copper concentrate, including by-product credits gold and silver.
- No contractual arrangements for refining currently exist.

19.3.2 Taxes

The Project has been evaluated on an after-tax basis to provide an approximate value of the expected economics. The tax model was compiled by Mining Tax Plan LLC ("MTP"), a firm that specializes in U.S. federal, state and foreign taxation of precious metal, non-metallic ores, coal and quarry mining located in Greenwood Village, Colorado.

MTP has prepared the U.S. federal and state income tax computations based on the Internal Revenue Code of 1986, as amended and the regulations thereunder including the computation of Alaska Mining License, Production Royalty and Matanuska-Susitna Borough Property Taxes as in effect as of February 28, 2026. Any subsequent changes of modifications to U.S. federal or state tax statutes, regulations or to the judicial and administrative interpretations thereof may impact the federal and state income tax computations. MTP has not audited or verified any of the economic or operating assumptions of the IA with economic analysis but have made inquiries to properly classified revenue, expenses and capital expenditures consistent with federal and state income tax statutes, regulations and case law.

The tax assumptions and elections in the IA with economic analysis model are made to minimize taxes payable over the life of mine and are as follows:

- The Whistler project consists of a single mine and property under Section 614.
- U.S. Goldmining will elect to treat mine development costs as incurred under 616(a) or as deferred expenses under Section 616(b).
- U.S. Goldmining will not elect out of Section 168(k) bonus depreciation unless to mitigate adverse impact on percentage depletion in excess of basis and cash taxes.
- U.S. Goldmining will elect to depreciate long-lived assets on a unit of production basis under Section 168(±)(1).
- U.S. Goldmining will have sales outside of the U.S. and is therefore eligible for §250 FDII deduction available on exported goods.
- The existing tax attributes of the Whistler Project have been based on tax returns as originally filed.
- No Section 382 ownership change will occur during construction or during the modeled life of the project which may or may not affect the utilization of U.S. tax attributes such as net operating losses and tax basis in assets in excess of fair market value.

19.3.3 Working Capital

An estimate of working capital has been incorporated into the economic analysis based on the following assumptions.

Table 19-1: Working Capital Assumptions

Description	Units	Value
Accounts Receivable	days	0
Inventory	days	30
Accounts Payable	days	30

19.3.4 Closure Costs and Salvage Value

Closure and salvage value are applied at the end of the LOM. Closure costs were estimated to be US$98.7 million with no salvage value assumed.

19.3.5 Royalties

Based on the agreements in place as of the date of this technical report and summarized in Section 3.5, a NSR royalty of 3.0% on the deposit is used for the economic analysis net of royalty negotiations and buydowns assumed to be paid prior to the start of construction as a corporate overhead expense.

19.3.6 Off-site Costs

The following off-site costs and sale terms are used for the economic analysis as defined in Section 16.2.

Table 19-2: Off-Take Term Assumptions

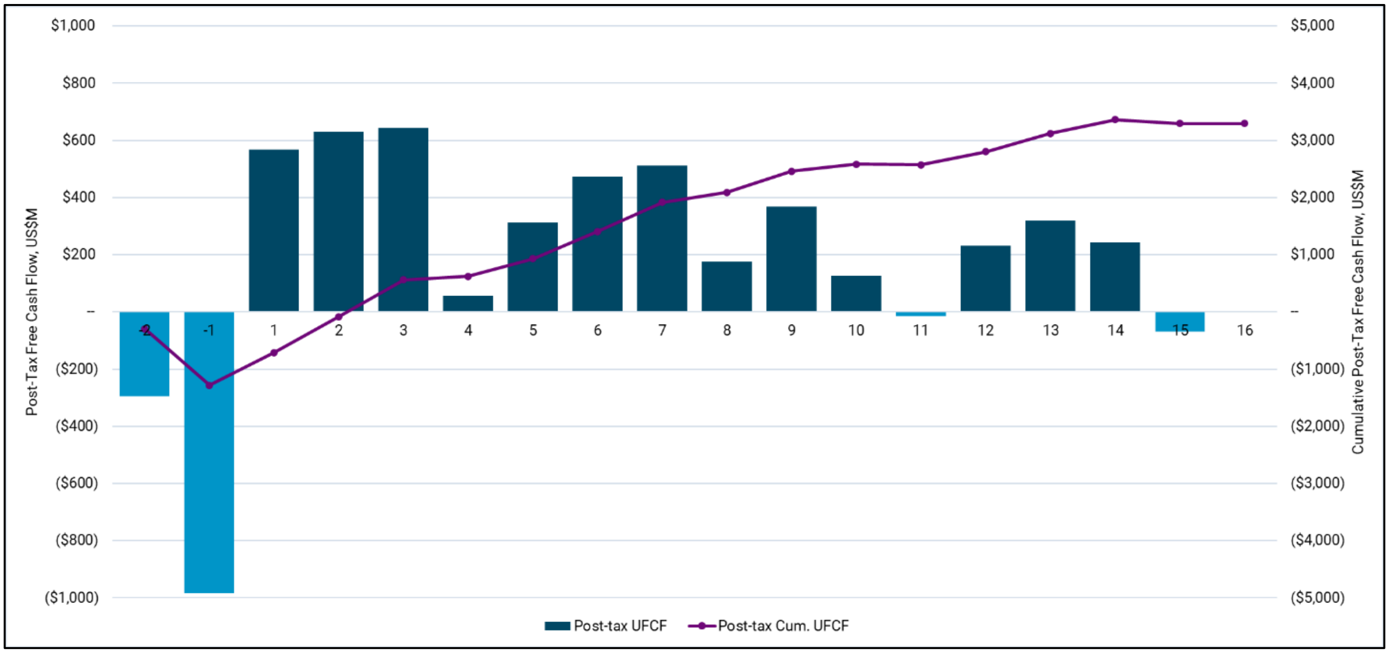
Term	Unit	Value
Copper Maximum Payability – Cu Concentrate	%	96.50
Copper Minimum Grade Deduction – Cu Concentrate	%Cu	1.00
Gold Payability – Cu Concentrate	%	95.0
Silver Payability – Cu Concentrate	%	95.0
Gold Payability – Doré	%	99.90
Silver Payability – Doré	%	98.00
Transport Cost – Cu Concentrate	US$/t	160.00
Treatment Cost – Cu Concentrate	US$/t	65.00
Copper Refining Cost – Cu Concentrate	US$/lb	0.065
Gold Refining Cost – Cu Concentrate	US$/oz	5.00
Silver Refining Cost – Cu Concentrate	US$/oz	0.50
Gold Refining, Transport, and Marketing Cost – Doré	US$/oz	2.50
Silver Refining, Transport, and Marketing Cost – Doré	US$/oz	0.50

19.4 Economic Analysis

The economic analysis is performed using a 5% discount rate. A 5% discount rate has been selected as 75% of gross revenue comes from gold, with 23% coming from copper, and 2% coming from silver. The pre-tax net present value discounted at 5% ($NPV_{5\%}$) is US$2,688 million, the IRR is 38.0%, and the payback period is 2.0 years from the start of commercial production. On a post-tax basis, the $NPV_{5\%}$ is US$2,039 million, the IRR is 33.0%, and the payback period is 2.1 years from the start of commercial production. A summary of project economics is tabulated in Table 19-3. The economic analysis is performed on an annual cashflow basis and the cashflow output is shown in

Table 19-4 and represented graphically in Figure 19-1.

Figure 19-1: Undiscounted, Unlevered, Free Cash Flow – Post-Tax



Source: Ausenco, 2026

Table 19-3: Economic Analysis Summary

Description	Units	Value
Production		
Mine life	Years	14.6
LOM Strip Ratio (Waste:Processed Material)	-	2.2
Total mined material	Mt	676.0
Total processed material	Mt	211.4
Nominal process plant rate	t/d	40,000
Gold Production		
Average gold feed grade	g/t	0.44
Average gold metallurgical recovery to final products	%	88.9
Total gold produced	koz	2,681.5
Average gold production, Year 1-3	koz/a	263.6
Average gold production, Year 1-7	koz/a	235.5

Description	Units	Value
Average gold production, LOM	koz/a	183.2
Copper Production		
Average copper feed grade	%	0.16
Average copper metallurgical recovery to final products	%	77.8
Total copper produced	Mlbs	591.6
Average copper production, Year 1-3	Mlbs /a	53.0
Average copper production, Year 1-7	Mlbs /a	47.1
Average copper production, LOM	Mlbs /a	40.4
Silver Production		
Average silver feed grade	g/t	1.83
Average silver metallurgical recovery to final products	%	55.6
Total silver produced	koz	6,903.8
Average silver production, Year 1-3	koz/a	605.1
Average silver production, Year 1-7	koz/a	528.7
Average silver production, LOM	koz/a	471.1
Gold Equivalent Production		
Average gold equivalent feed grade	g/t	0.62
Total gold equivalent produced	koz	3,594.3
Average gold equivalent production, Year 1-3	koz/a	345.2
Average gold equivalent production, Year 1-7	koz/a	308.0
Average gold equivalent production, LOM	koz/a	245.6
Operating Costs (OPEX)		
Mining unit cost	US$/t mined	2.53
Mining unit cost	US$/t milled	7.93
Process unit cost	US$/t milled	11.00
Site general and administrative (G&A) unit cost	US$/t milled	1.02
Additional G&A (access road toll and maintenance)	US$/t milled	0.86
Total operating unit cost	US$/t milled	20.82
Total cash cost, by-product basis	US$/oz Au	861
All-in sustaining cost (AISC), by-product basis	US$/oz Au	1,046
Capital Expenditures (CAPEX)		
Initial capital expenditure (including pre-strip)	US$M	1,278.6
Sustaining capital expenditure	US$M	381.1

Description	Units	Value
Closure costs	US$M	98.7
Pre-tax Economics		
$NPV_{5\%}$	US$M	2,688.2
IRR	%	38.0
Payback Period	Years	2.0
Post-tax Economics		
$NPV_{5\%}$	US$M	2,038.8
IRR	%	33.0
Payback Period	Years	2.1

Table 19-4: Cashflow Statement on an Annualized Basis

Macro Assumptions	Units	Total/Avg.	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Copper Price	US$/lb	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50
Gold Price	US$/oz	3,200	3,200	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00	3,200.00
Silver Price	US$/oz	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50	37.50
Gross Revenue	**US$M**	**11,084**	**--**	**--**	**928.6**	**1,085.2**	**1,179.2**	**576.3**	**821.9**	**1,005.8**	**1,052.0**	**622.5**	**833.5**	**556.9**	**348.4**	**603.7**	**696.1**	**587.4**	**186.5**
Off-site Charges	US$M	(308)	--	--	(23.7)	(30.8)	(28.4)	(21.2)	(21.8)	(22.9)	(23.2)	(21.7)	(20.6)	(18.2)	(15.1)	(19.4)	(17.3)	(15.5)	(7.7)
Royalties	US$M	(323)	--	--	(27.1)	(31.6)	(34.5)	(16.7)	(24.0)	(29.5)	(30.9)	(18.0)	(24.4)	(16.2)	(10.0)	(17.5)	(20.4)	(17.2)	(5.4)
Operating Expenses	US$M	(4,400)	--	--	(246)	(279)	(294)	(335)	(340)	(359)	(346)	(316)	(314)	(343)	(322)	(286)	(251)	(238)	(129)
EBITDA	**US$M**	**6,053**	**--**	**--**	**632**	**743**	**822**	**203**	**436**	**595**	**652**	**266**	**475**	**180**	**2**	**280**	**407**	**317**	**44**
Initial Capex	US$M	(1,279)	(296)	(983)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	US$M	(381)	--	--	(52.4)	(51.6)	(59.5)	(125.9)	(44.2)	(1.8)	(7.5)	(22.5)	(6.9)	(4.8)	(2.6)	(1.0)	(0.3)	--	--
Closure Capex	US$M	(99)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(98.7)
Pre-Tax Unlevered Free Cash Flow	**US$M**	**4,295**	**(296)**	**(983)**	**579**	**692**	**763**	**77**	**392**	**593**	**644**	**244**	**468**	**175**	**(1)**	**279**	**407**	**317**	**(54)**
Taxes Payable	US$M	(1,000)	--	(1)	(12)	(61)	(118)	(21)	(80)	(119)	(133)	(67)	(100)	(50)	(15)	(47)	(89)	(75)	(14)
Post-Tax Unlevered Free Cash Flow	**US$M**	**3,295**	**(296)**	**(984)**	**568**	**631**	**645**	**57**	**312**	**474**	**512**	**176**	**368**	**125**	**(16)**	**232**	**318**	**242**	**(69)**
Production Summary																			
Mineralized Material Mined	kt	211,368	--	5,335	15,565	20,724	14,418	15,655	22,533	18,092	17,098	15,199	14,645	6,753	6,108	15,561	14,920	8,761	--
Waste Mined	kt	464,671	--	8,830	11,964	20,055	22,584	56,345	49,467	50,408	52,902	44,801	40,855	44,091	38,827	18,672	4,199	670	--
Mineralized Material + Waste Total Material Mined	kt	676,039	--	14,165	27,530	40,780	37,002	72,000	72,000	68,500	70,000	60,000	55,500	50,845	44,935	34,233	19,119	9,431	--
Total Mineralized Material Processed	kt	211,368	--	--	12,400	14,600	14,600	14,600	14,600	14,600	14,600	14,600	14,600	14,600	14,600	14,600	14,600	14,600	9,169
Processing Summary																			
Mill Feed - Cu Grade	%	0.16%	--	--	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.2	0.1	0.1	0.1
Mill Feed - Au Grade	g/t	0.44	--	--	0.65	0.62	0.71	0.30	0.48	0.62	0.66	0.33	0.50	0.30	0.16	0.33	0.42	0.35	0.14
Mill Feed - Ag Grade	g/t	1.83	--	--	2.60	2.67	2.10	1.52	1.61	1.89	2.16	1.97	1.53	1.67	1.76	1.56	1.28	1.59	1.54
Total Metal Content - Cu	**mlbs**	**760.0**	**--**	**--**	**57**	**72**	**65**	**52**	**53**	**55**	**56**	**54**	**50**	**47**	**42**	**48**	**44**	**42**	**23**
Total Metal Content - Au	**koz**	**3,015**	**--**	**--**	**260**	**292**	**335**	**139**	**227**	**293**	**310**	**154**	**235**	**142**	**75**	**154**	**195**	**163**	**41**
Total Metal Content - Ag	**koz**	**12,428**	**--**	**--**	**1,036**	**1,252**	**984**	**714**	**758**	**887**	**1,013**	**927**	**718**	**784**	**824**	**732**	**601**	**746**	**454**
Average Recovery - Cu	%	77.8	--	--	80.4	82.4	83.6	79.5	79.6	79.4	78.4	78.8	78.3	75.2	70.3	77.6	74.4	70.4	65.0
Average Recovery - Au	%	88.9	--	--	88.6	89.4	89.5	89.5	89.2	88.4	87.9	88.4	89.2	88.6	88.1	89.9	89.9	88.0	88.9
Average Recovery - Ag	%	55.6	--	--	55.1	55.3	56.1%	57.1	56.6	55.6	54.8	55.0	56.7	55.2	54.0	56.1	57.3	54.7	54.0
Total Metal to Concentrate - Cu	mlbs	592	--	--	45	59	54	41	42	44	44	42	39	35	29	38	33	30	15
Total Metal to Concentrate - Au	koz	1,858	--	--	157	184	214	88	142	175	180	92	148	86	44	100	127	96	25
Total Metal to Concentrate - Ag	koz	5,568	--	--	459	557	448	334	350	397	445	409	332	348	353	333	282	327	195
Total Metal to Doré - Au	koz	824	--	--	73	76	86	36	60	84	92	44	63	40	22	38	49	48	11
Total Metal to Doré - Ag	koz	1,336	--	--	113	135	104	74	79	95	111	101	75	85	92	78	62	82	50
Total Metal Produced - Cu	mlbs	592	--	--	45.4	59.3	54.2	41.3	41.9	43.6	44.1	42.3	39.4	35.2	29.4	37.6	33.1	29.6	15.1
Total Metal Produced - Au	koz	2,682	--	--	230.4	260.7	299.8	124.2	202.6	258.8	272.1	136.4	210.1	126.0	66.4	138.3	175.4	143.8	36.5
Total Metal Produced - Ag	koz	6,904	--	--	571.0	692.0	552.3	407.9	429.2	492.8	555.4	509.5	407.4	432.9	445.0	410.7	344.6	408.2	245.0
Total Payable Copper	**mlbs**	**568**	**--**	**--**	**43.6**	**56.9**	**52.1**	**39.7**	**40.2**	**41.8**	**42.3**	**40.6**	**37.9**	**33.8**	**28.3**	**36.1**	**31.8**	**28.4**	**14.5**
Total Payable Gold	**koz**	**2,588**	**--**	**--**	**222.5**	**251.4**	**289.1**	**119.7**	**195.4**	**249.9**	**263.0**	**131.8**	**202.7**	**121.7**	**64.1**	**133.2**	**169.0**	**139.0**	**35.2**
Total Payable Silver	**koz**	**6,599**	**--**	**--**	**546**	**661**	**528**	**390**	**410**	**471**	**531**	**487**	**389**	**414**	**425**	**392**	**329**	**390**	**234**

Macro Assumptions	Units	Total/Avg.	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Total Operating Costs	**US$M**	**(4,400)**	**--**	**--**	**(246)**	**(279)**	**(294)**	**(335)**	**(340)**	**(359)**	**(346)**	**(316)**	**(314)**	**(343)**	**(322)**	**(286)**	**(251)**	**(238)**	**(129)**
Mining Costs	US$M	(1,733)	--	(56.3)	(81.1)	(91.7)	(106.3)	(147.6)	(152.7)	(171.2)	(158.6)	(128.8)	(126.1)	(155.2)	(134.0)	(98.7)	(63.3)	(50.3)	(11.0)
Processing Costs	US$M	(2,326)	--	--	(141.7)	(160.3)	(160.3)	(160.3)	(160.2)	(160.3)	(160.3)	(160.2)	(160.3)	(160.3)	(160.3)	(160.3)	(160.3)	(160.3)	(100.9)
G&A	US$M	(397)	--	--	(23.3)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(27.4)	(17.2)
Capitalized Operating Costs	US$M	56.3	--	56.3	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Unit Operating Costs	US$/t Milled	(20.8)	--	--	(19.8)	(19.1)	(20.1)	(23.0)	(23.3)	(24.6)	(23.7)	(21.7)	(21.5)	(23.5)	(22.0)	(19.6)	(17.2)	(16.3)	(14.1)
Total Off-site Charges	**US$M**	**(307.5)**	**--**	**--**	**(23.7)**	**(30.8)**	**(28.4)**	**(21.2)**	**(21.8)**	**(22.9)**	**(23.2)**	**(21.7)**	**(20.6)**	**(18.2)**	**(15.1)**	**(19.4)**	**(17.3)**	**(15.5)**	**(7.7)**
Concentrate Transport Costs	US$M	(186.7)	--	--	(14.3)	(18.7)	(17.1)	(13.0)	(13.2)	(13.8)	(13.9)	(13.3)	(12.4)	(11.1)	(9.3)	(11.9)	(10.4)	(9.3)	(4.8)
Concentrate Treatment Costs	US$M	(69.8)	--	--	(5.4)	(7.0)	(6.4)	(4.9)	(4.9)	(5.1)	(5.2)	(5.0)	(4.7)	(4.1)	(3.5)	(4.4)	(3.9)	(3.5)	(1.8)
Concentrate Refining Charges	US$M	(48.4)	--	--	(3.8)	(4.8)	(4.6)	(3.2)	(3.5)	(3.7)	(3.8)	(3.3)	(3.3)	(2.8)	(2.2)	(3.0)	(2.8)	(2.5)	(1.2)
Doré Transport and Refining Charges	US$M	(2.7)	--	--	(0.2)	(0.3)	(0.3)	(0.1)	(0.2)	(0.3)	(0.3)	(0.2)	(0.2)	(0.1)	(0.1)	(0.1)	(0.2)	(0.2)	(0.1)
NSR Royalties	**US$M**	**(323.3)**	**--**	**--**	**(27.1)**	**(31.6)**	**(34.5)**	**(16.7)**	**(24.0)**	**(29.5)**	**(30.9)**	**(18.0)**	**(24.4)**	**(16.2)**	**(10.0)**	**(17.5)**	**(20.4)**	**(17.2)**	**(5.4)**
Cash Costs (By-Product Basis)																			
C1 Cash Cost*	US$/oz Au	861	--	--	361	243	356	1,503	971	821	722	1,179	857	1,724	3,175	1,096	789	921	1,943
C3 Cash Cost**	US$/oz Au	1,046	--	--	596	448	562	2,555	1,197	829	751	1,350	892	1,763	3,215	1,104	791	921	4,746
Total Initial Capital	**US$M**	**(1,279)**	**(296)**	**(983)**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Capitalized Mining Opex	US$M	(56.3)	--	(56.3)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
1000 - Mining	US$M	(39.7)	--	(39.7)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
2000 - Process Plant	US$M	(120.4)	(30.1)	(90.3)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
3000 - Additional Process Facilities	US$M	(354.8)	(88.7)	(266.1)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
4000 - On-Site Infrastructure	US$M	(187.6)	(46.9)	(140.7)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
5000 - Off-Site Infrastructure	US$M	(72.6)	(18.1)	(54.4)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
6000 - Construction Indirects	US$M	(80.5)	(20.1)	(60.4)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
7000 - Project Delivery	US$M	(122.1)	(30.5)	(91.6)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
8000 - Owner's Costs	US$M	(31.3)	(7.8)	(23.5)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
9000 - Total Contingency Costs	US$M	(213.3)	(53.3)	(159.9)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Sustaining Capital	**US$M**	**(381.1)**	**--**	**--**	**(52.4)**	**(51.6)**	**(59.5)**	**(125.9)**	**(44.2)**	**(1.8)**	**(7.5)**	**(22.5)**	**(6.9)**	**(4.8)**	**(2.6)**	**(1.0)**	**(0.3)**	**--**	**--**
1000 - Mining	US$M	(319.0)	--	--	(51.3)	(40.5)	(44.2)	(125.0)	(28.6)	(1.6)	(7.5)	(4.7)	(6.9)	(4.8)	(2.6)	(1.0)	(0.3)	--	--
2000 - Process Plant	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
3000 - Additional Process Facilities	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
4000 - On Site Infrastructure	US$M	(49.8)	--	--	--	(8.9)	(14.2)	--	(12.5)	--	--	(14.3)	--	--	--	--	--	--	--
5000 - Off-Site Infrastructure	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
6000 - Construction Indirects	US$M	(1.6)	--	--	(0.5)	--	(0.6)	(0.5)	--	(0.1)	--	--	--	--	--	--	--	--	--
7000 - Project Delivery	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
8000 - Owner's Costs	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
9000 - Total Contingency Costs	US$M	(10.6)	--	--	(0.5)	(2.2)	(0.6)	(0.5)	(3.1)	(0.1)	--	(3.6)	--	--	--	--	--	--	--
Closure Cost and Salvage Credit	**US$M**	**(98.7)**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**(98.7)**
Total Capital Expenditures Including Salvage Value	**US$M**	**(1,758)**	**(296)**	**(983)**	**(52)**	**(52)**	**(60)**	**(126)**	**(44)**	**(2)**	**(8)**	**(22)**	**(7)**	**(5)**	**(3)**	**(1)**	**(0)**	**--**	**(99)**

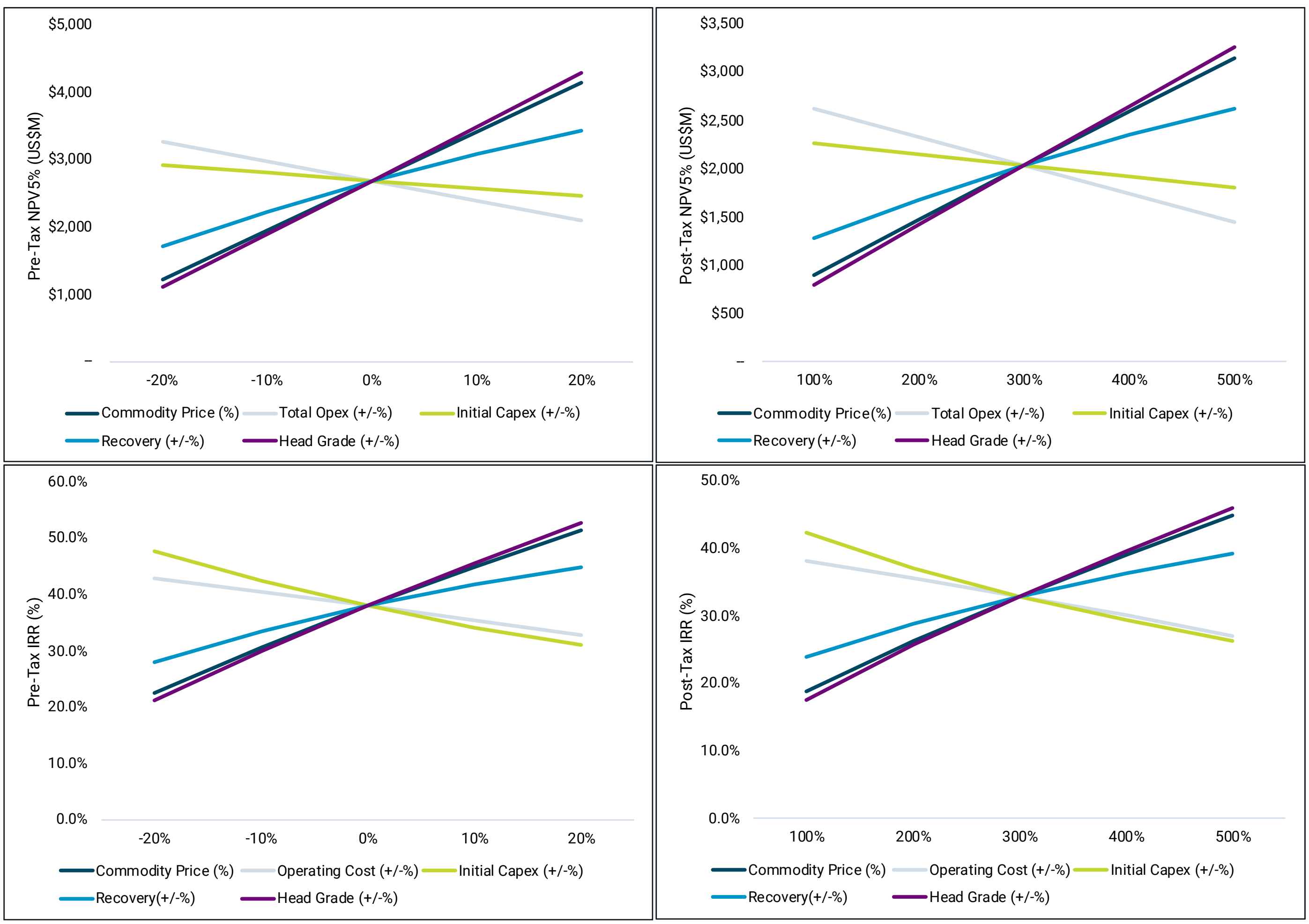
19.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base-case pre-tax and post-tax $NPV_{5\%}$ and IRR of the project using the following variables: metal prices, discount rate, total operating costs, initial capital costs, recovery, and head grade.

As shown in Figure 19-2, the sensitivity analysis reveals that the project is most sensitive to changes in head grade and commodity prices, and less sensitive to recovery, operating costs, and initial capital costs.

Table 19-5 shows the post-tax sensitivity analysis results while Table 19-6 shows pre-tax sensitivity analysis results. Table 19-7 shows the economic sensitivity to individual changes in copper and gold prices.

Figure 19-2: Sensitivity Analysis



Source: Ausenco, 2026

Table 19-5: Post-Tax Sensitivity

Post-Tax Sensitivity to Metal Price

Post-Tax $NPV_{5\%}$ (USM)Sensitivity to Discount Rate					
	Commodity Price				
Discount Rate	(20.0%)	(10.0%)	--	10.0%	20.0%
1.0%	$1,454	$2,230	$2,988	$3,742	$4,492
3.0%	$1,146	$1,815	$2,465	$3,112	$3,753
5.0%	$895	$1,475	$2,039	$2,598	$3,153
8.0%	$598	$1,076	$1,537	$1,994	$2,447
10.0%	$442	$865	$1,272	$1,676	$2,075

Post-Tax IRR (%) Sensitivity to Discount Rate					
	Commodity Price				
Discount Rate	(20.0%)	(10.0%)	--	10.0%	20.0%
1.0%	18.9%	26.3%	33.0%	39.2%	45.0%
3.0%	18.9%	26.3%	33.0%	39.2%	45.0%
5.0%	18.9%	26.3%	33.0%	39.2%	45.0%
8.0%	18.9%	26.3%	33.0%	39.2%	45.0%
10.0%	18.9%	26.3%	33.0%	39.2%	45.0%

Post-Tax $NPV_{5\%}$ (USM) Sensitivity to OPEX					
	Commodity Price				
Total OPEX	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$1,479	$2,060	$2,623	$3,183	$3,737
(10.0%)	$1,187	$1,768	$2,331	$2,890	$3,445
--	$895	$1,475	$2,039	$2,598	$3,153
10.0%	$602	$1,183	$1,747	$2,306	$2,861
20.0%	$310	$891	$1,455	$2,014	$2,569

Post-Tax IRR (%) Sensitivity to OPEX					
	Commodity Price				
Total OPEX	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	25.2%	32.0%	38.3%	44.2%	49.8%
(10.0%)	22.2%	29.3%	35.7%	41.7%	47.4%
--	18.9%	26.3%	33.0%	39.2%	45.0%
10.0%	15.2%	23.2%	30.1%	36.5%	42.5%
20.0%	10.9%	19.8%	27.1%	33.8%	40.0%

Post-Tax $NPV_{5\%}$ (USM) Sensitivity to Initial CAPEX					
	Commodity Price				
Initial CAPEX	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$1,125	$1,705	$2,269	$2,828	$3,383
(10.0%)	$1,010	$1,590	$2,154	$2,713	$3,268
--	$895	$1,475	$2,039	$2,598	$3,153
10.0%	$780	$1,360	$1,924	$2,483	$3,038
20.0%	$665	$1,245	$1,809	$2,368	$2,923

Post-Tax IRR (%) Sensitivity to Initial CAPEX					
	Commodity Price				
Initial CAPEX	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	25.9%	34.6%	42.4%	49.7%	56.6%
(10.0%)	22.0%	30.1%	37.2%	43.9%	50.3%
--	18.9%	26.3%	33.0%	39.2%	45.0%
10.0%	16.2%	23.2%	29.4%	35.2%	40.6%
20.0%	13.9%	20.5%	26.3%	31.8%	36.9%

Post-Tax $NPV_{5\%}$ (USM) Sensitivity to Mill Recovery					
	Commodity Price				
Mill Recovery	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$256	$781	$1,282	$1,768	$2,253
(10.0%)	$598	$1,155	$1,682	$2,209	$2,725
--	$895	$1,475	$2,039	$2,598	$3,153
10.0%	$1,155	$1,755	$2,354	$2,940	$3,537
20.0%	$1,369	$1,999	$2,624	$3,243	$3,869

Post-Tax IRR (%) Sensitivity to Mill Recovery					
	Commodity Price				
Mill Recovery	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	9.5%	17.3%	24.0%	29.9%	35.5%
(10.0%)	14.7%	22.3%	28.9%	34.9%	40.6%
--	18.9%	26.3%	33.0%	39.2%	45.0%
10.0%	22.2%	29.6%	36.4%	42.7%	48.8%
20.0%	24.9%	32.4%	39.3%	45.8%	52.0%

Post-Tax $NPV_{5\%}$ (USM) Sensitivity to Head Grade					
	Commodity Price				
Head Grade	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	($198)	$323	$791	$1,245	$1,683
(10.0%)	$375	$908	$1,423	$1,923	$2,422
--	$895	$1,475	$2,039	$2,598	$3,153
10.0%	$1,401	$2,030	$2,654	$3,274	$3,901
20.0%	$1,898	$2,588	$3,272	$3,963	$4,649

Post-Tax IRR (%) Sensitivity to Head Grade					
	Commodity Price				
Head Grade	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	0.7%	10.6%	17.6%	23.6%	29.1%
(10.0%)	11.4%	19.1%	25.8%	31.7%	37.4%
--	18.9%	26.3%	33.0%	39.2%	45.0%
10.0%	25.3%	32.8%	39.7%	46.1%	52.4%
20.0%	31.2%	38.9%	46.1%	52.9%	59.5%

Table 19-6: Pre-Tax Sensitivity

Pre-Tax Sensitivity to Metal Price

Discount Rate	Pre-Tax NPV$_{5\%}$ (USM)Sensitivity to Discount Rate – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%	Discount Rate	Pre-Tax IRR (%) Sensitivity to Discount Rate – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%
1.0%	$1,921	$2,911	$3,902	$4,892	$5,883	1.0%	22.5%	30.6%	38.0%	45.0%	51.6%
3.0%	$1,539	$2,386	$3,232	$4,079	$4,925	3.0%	22.5%	30.6%	38.0%	45.0%	51.6%
5.0%	$1,228	$1,958	$2,688	$3,418	$4,148	5.0%	22.5%	30.6%	38.0%	45.0%	51.6%
8.0%	$863	$1,457	$2,050	$2,644	$3,238	8.0%	22.5%	30.6%	38.0%	45.0%	51.6%
10.0%	$671	$1,193	$1,715	$2,237	$2,759	10.0%	22.5%	30.6%	38.0%	45.0%	51.6%

Total OPEX	Pre-Tax NPV$_{5\%}$ (USM) Sensitivity to OPEX – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%	Total OPEX	Pre-Tax IRR (%) Sensitivity to OPEX – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$1,813	$2,543	$3,273	$4,003	$4,733	(20.0%)	28.3%	35.8%	42.8%	49.5%	56.0%
(10.0%)	$1,520	$2,250	$2,980	$3,710	$4,440	(10.0%)	25.5%	33.2%	40.4%	47.3%	53.8%
--	$1,228	$1,958	$2,688	$3,418	$4,148	--	22.5%	30.6%	38.0%	45.0%	51.6%
10.0%	$936	$1,666	$2,396	$3,126	$3,856	10.0%	19.2%	27.8%	35.5%	42.6%	49.3%
20.0%	$644	$1,374	$2,104	$2,834	$3,564	20.0%	15.6%	24.8%	32.8%	40.2%	47.1%

Initial CAPEX	Pre-Tax NPV$_{5\%}$ (USM) Sensitivity to Initial CAPEX – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%	Initial CAPEX	Pre-Tax IRR (%) Sensitivity to Initial CAPEX – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$1,458	$2,188	$2,918	$3,648	$4,378	(20.0%)	29.9%	39.2%	47.8%	55.9%	63.7%
(10.0%)	$1,343	$2,073	$2,803	$3,533	$4,263	(10.0%)	25.8%	34.5%	42.4%	49.9%	57.1%
--	$1,228	$1,958	$2,688	$3,418	$4,148	--	22.5%	30.6%	38.0%	45.0%	51.6%
10.0%	$1,113	$1,843	$2,573	$3,303	$4,033	10.0%	19.7%	27.3%	34.2%	40.8%	47.0%
20.0%	$998	$1,728	$2,458	$3,188	$3,918	20.0%	17.3%	24.5%	31.1%	37.2%	43.0%

Mill Recovery	Pre-Tax NPV$_{5\%}$ (USM) Sensitivity to Mill Recovery – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%	Mill Recovery	Pre-Tax IRR (%) Sensitivity to Mill Recovery – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$461	$1,090	$1,719	$2,348	$2,978	(20.0%)	12.6%	20.9%	28.1%	34.7%	40.9%
(10.0%)	$867	$1,550	$2,232	$2,914	$3,597	(10.0%)	18.1%	26.2%	33.4%	40.2%	46.6%
--	$1,228	$1,958	$2,688	$3,418	$4,148	--	22.5%	30.6%	38.0%	45.0%	51.6%
10.0%	$1,544	$2,316	$3,088	$3,860	$4,632	10.0%	26.1%	34.3%	41.8%	49.0%	55.8%
20.0%	$1,815	$2,623	$3,431	$4,239	$5,048	20.0%	29.0%	37.3%	45.0%	52.3%	59.3%

Head Grade	Pre-Tax NPV$_{5\%}$ (USM) Sensitivity to Head Grade – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%	Head Grade	Pre-Tax IRR (%) Sensitivity to Head Grade – Commodity Price (20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	($28)	$537	$1,103	$1,669	$2,235	(20.0%)	4.4%	13.8%	21.2%	27.7%	33.7%
(10.0%)	$599	$1,247	$1,894	$2,542	$3,190	(10.0%)	14.6%	22.8%	30.0%	36.7%	42.9%
--	$1,228	$1,958	$2,688	$3,418	$4,148	--	22.5%	30.6%	38.0%	45.0%	51.6%
10.0%	$1,861	$2,673	$3,486	$4,298	$5,111	10.0%	29.5%	37.8%	45.5%	52.8%	59.8%
20.0%	$2,497	$3,393	$4,288	$5,183	$6,079	20.0%	36.0%	44.6%	52.7%	60.4%	67.8%

Table 19-7: Economic Sensitivity to Specific Metal Prices

Post-Tax NPV$_{5\%}$ (USM)Sensitivity to Metal Price

Gold Price		Copper Price				
		(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	$920	$1,059	$1,194	$1,323	$1,455
	(10.0%)	$1,356	$1,488	$1,616	$1,746	$1,877
	--	$1,779	$1,909	$2,039	$2,169	$2,299
	10.0%	$2,201	$2,331	$2,459	$2,586	$2,714
	20.0%	$2,616	$2,743	$2,870	$2,997	$3,128

Post-Tax IRR (%) Sensitivity to Metal Price

Gold Price		Copper Price				
		(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	19.2%	21.0%	22.7%	24.3%	25.9%
	(10.0%)	24.9%	26.5%	28.0%	29.5%	31.0%
	--	30.1%	31.5%	33.0%	34.4%	35.8%
	10.0%	34.9%	36.3%	37.7%	39.0%	40.4%
	20.0%	39.6%	40.9%	42.2%	43.5%	44.8%

Pre-Tax NPV$_{5\%}$ (USM) Sensitivity to Metal Price

Gold Price		Copper Price				
		(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	$1,260	$1,427	$1,594	$1,760	$1,927
	(10.0%)	$1,808	$1,974	$2,141	$2,308	$2,474
	--	$2,355	$2,522	$2,688	$2,855	$3,022
	10.0%	$2,902	$3,069	$3,236	$3,402	$3,569
	20.0%	$3,449	$3,616	$3,783	$3,950	$4,116

Pre-Tax IRR (%) Sensitivity to Metal Price

Gold Price		Copper Price				
		(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	22.9%	24.7%	26.6%	28.3%	30.1%
	(10.0%)	29.0%	30.7%	32.4%	34.1%	35.7%
	--	34.8%	36.4%	38.0%	39.6%	41.1%
	10.0%	40.2%	41.8%	43.3%	44.8%	46.3%
	20.0%	45.4%	46.9%	48.4%	49.9%	51.3%

20 ADJACENT PROPERTIES

No information on adjacent properties has been included in this Report.

21 OTHER RELEVANT DATA AND INFORMATION

No other relevant data or information has been included in this Report.

22 INTERPRETATION AND CONCLUSIONS

22.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

22.2 Analytical Data Collection in Support of Mineral Resource Estimation

The procedures documented by Kennecott, Geoinformatics and Kiska and U.S. GoldMining for sampling, analysis and security are deemed adequate. Analysis of the QA/QC samples indicates the laboratory results are of sufficient quality for resource estimation.

The amount of data fully supported by certificate and QA/QC is 92% in Whistler, 100% in Raintree West, and 100% in Island Mountain, which is typical or better than similar projects with most of the drilling completed before 2010. Inconsistencies detected during validation of the assay database are minimal. Measurements made during the site visit and previous reports indicate a collar survey is to be considered.

22.3 Metallurgical Testing

The metallurgical studies of the Whistler deposit and adjacent deposits have been performed from 2004 to 2026. Historic testwork conducted prior to 2023 includes material from additional deposits and is excluded from the current metallurgical interpretation.

Testwork performed on sample materials obtained from the exploration campaigns under the direction of U.S. GoldMining conducted in 2023 and 2024, which were focused exclusively on the Whistler deposit, is considered the most appropriate for estimation of the metallurgical results. To the extent known, the metallurgical samples are representative of the styles and types of mineralization and the mineral deposit as a whole.

The scope and quantity of metallurgical testwork completed are appropriate for the level of this study and define the metallurgical response of the resource materials to the selected process. The recovery model used in the financial analysis is based on the metallurgical testing completed to date.

22.4 Mineral Resource Estimate

In the opinion of the QP the block model resource estimate and resource classification reported herein are a reasonable representation of the global gold, copper and silver mineral resources found in the Whistler, Raintree West, and Island Mountain deposits. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

22.5 Mining Methods

A reasonable open-pit mine plan, including pit and stockpile designs, a mine production schedule, mine fleet selection, and mine capital and operating cost estimates have been developed for the Whistler Deposit.

Pit layouts and mine operations are typical of other mountainous open-pit metal mine operations in North America, and the unit operations within the developed mine operating plan are proven to be effective for these other operations.

The total estimated LOM run-of-mine mill feed quantities, based on a US$13.40/t NSR cutoff value and treating all inferred class resources as waste, are 211 Mt at 0.44 g/t Au, 0.16% Cu, and 1.83 g/t Ag, with an associated waste mining quantity of 465 Mt. These quantities will be mined over a 1-year construction period, a 14-year pit to mill operating period, and a 1-year period of low-grade stockpile rehandling to the mill.

The mine plan supports the cash flow model and financials developed for this Study.

22.6 Recovery Plan

Recovery is planned using a crushing, high pressure grinding and flotation circuit in conjunction with a flotation tailings leach circuit to produce a copper concentrate product with precious metal by-products and a doré product. The process plant is designed to achieve a 40,000 t/d average throughput with an overall mill availability of 92%.

The process plant will produce a copper concentrate that is projected to contain a payable gold and silver by-product and negligible deleterious elements that would impact salability or the viability of economic extraction.

The selected flowsheet aligns with conventional industry practice. Comminution, flotation, precious metal extraction, recovery of payable metals, destruction of free cyanide, and tailings management are achieved using established processes that are commonly applied to similar projects with no reliance on novel or unproven technologies. The flowsheet was developed based on the latest test work results and financial evaluations.

22.7 Infrastructure

The infrastructure design is based on reasonable assumptions and is suitable for this level of study. The Company has engaged with local governments and businesses to establish the inputs used to support the Project.

The Project is currently accessible by flying into the Whiskey Bravo airstrip. Once the WSAR is completed in 2030, the Project will be accessible by road from Anchorage or Port Mackenzie. Employees, fuel, reagents, supplies, and concentrate will then be transported via the WSAR.

The on-site roads, process plant building, support buildings, and stockpiles have been developed at a conceptual level consistent with a PEA.

The CDSF has been designed in an area that has not been previously studied. As a result, no geotechnical data is currently available for the region. The design standards for the CDSF are based on the relevant state, federal, and international guidelines for the construction of mining tailings storage facilities in Alaska.

22.8 Markets and Contracts

No market studies or product valuations were completed as part of this study. Market price assumptions were based on a review of public information, industry consensus, standard practice, and specific information from comparable operations.

Copper concentrates are widely traded and can be marketed directly from producer to smelter or via third-party concentrate trading entities. It is assumed that the concentrate contains negligible deleterious elements that would impact marketability.

The market for gold doré is widely traded and can be marketed domestically or internationally with significant optionality regarding the final customer. It is assumed that the doré contains negligible deleterious elements that would impact marketability.

Marketing, refining, and transportation costs, along with payability terms, were assumed based on a review of information from comparable recent studies.

22.9 Environmental, Permitting and Social Considerations

The Company holds a multi-year Exploration and Reclamation permit, has commenced environmental studies, and has developed a Stakeholder Engagement Plan and an approved Reclamation Plan. Potential state, federal, and local permit requirements have been identified. A conceptual closure plan has been developed.

The Project is located within the Matanuska-Susitna (Mat-Su) Borough, which contains some Alaska Native Claims Settlement Act (ANCSA) lands owned by the Cook Inlet Region, Inc. (CIRI). No Alaska Native Corporation or tribal lands occur within the project boundary. To date, no land tenure conflicts have been identified, and no restrictions on mineral tenure or surface access are currently known that would materially affect exploration activities (Owl Ridge Natural Resource Consultants, Inc., 2023).

22.10 Capital Cost Estimate

The capital cost estimate was developed in Q4 2025 to target a level of accuracy of -30 to +50%, which aligns with an Association for the Advancement of Cost Engineering International (AACE International) Class 5 level estimate. The estimate includes mining, processing, on-site infrastructure, off-site infrastructure, project indirects, project delivery, owners' costs, and provisions. The total initial capital costs for the Project are estimated at US$1,278.6 million, including US$56.3 million of capitalized operating costs, and US$213.3 million of contingency. The LOM sustaining costs are estimated at US$381.1 million, while the closure costs are estimated at US$98.7 million.

22.11 Operating Cost Estimate

The total operating costs for the Project are estimated at US$20.82/t or US$4,399.8 million over the 14.6-year mine life. These operating costs do not include pre-production operating costs. This estimate considers the development of a process plant designed to treat 40,000 t/d of mineralized material. Process unit operations were benchmarked against similar or comparable processing plants to ensure accuracy of cost estimates.

22.12 Economic Analysis

An economic model was developed to estimate the project's annual pre-tax and post-tax cash flows, sensitivities, and net present value results using a 5% discount rate. Based on the assumptions and parameters, the IA with economic analysis shows positive post-tax economics of US$2,038.8 million $NPV_{5\%}$ and 33.0% post-tax IRR.

A sensitivity analysis was conducted on the base-case pre-tax and post-tax NPV and IRR of the project using the following variables: metal prices, discount rate, operating costs, initial capex, metal recovery, and head grade. The Project is most sensitive to changes in commodity price and head grade, and less sensitive to changes in operating costs, recovery, and initial capital costs.

The disclosure of mineral resources used in this IA are preliminary in nature. No mineral reserves are included in this IA, and there is no certainty that this economic assessment will be carried out.

22.13 Risks and Opportunities

22.13.1 Risks

22.13.1.1 Sample Preparation, Analysis and Security

The drill core is stored in wood boxes subject to weathering on site, they are beginning to fall apart. An opportunity exists to protect these samples from further weathering by moving them or building dry storage. The risk of continued decay is that the historic core may no longer be available to future potential owners for review and verification.

22.13.1.2 Metallurgical Testing and Processing and Recovery Methods

The flowsheet was developed based on the 2025 testwork data available at the time of the reporting. Due to the limited volume of testwork in the current dataset, the following additional work would further de-risk the process design and metallurgical recovery projections:

- Additional comminution testwork to confirm or optimize the selection of crushing and grinding equipment.
- Additional metal recovery testwork, including flotation and gravity concentration, to optimize the process flowsheet and further reduce uncertainty in the performance projections.

- Additional precious metal and copper deportment or diagnostic testing to confirm and optimize the recovery projections.
- Additional pyrite concentration testwork and pyrite leach testwork to confirm the value of whole tailings leaching.
- Additional leach testwork to confirm the viability of the leaching operating conditions and the precious metal extraction estimates.
- Cyanide detoxification testwork to confirm the reagent addition rates and reagent consumption requirements.

22.13.1.3 Mineral Resource Estimate

Risk in the geologic interpretations relating to the continuity of the mineralization exist and can be mitigated by additional geologic modelling for use in controlling the block model interpolations. A description of additional potential risk factors concerning the resource estimate is given in Table 11-22 along with either the justification for the approach taken or mitigating factors in place to reduce any risk. Opportunities to increase the confidence in the resource through infill drilling and to expand the resource from step-out and exploration drilling are discussed in the recommendations section below.

22.13.1.4 Mining Methods

Risks to the Whistler Mine Plan defined mill feed quantities, metal grades, associated waste rock quantities and the estimated costs to exploit include changes to the following factors and assumptions:

- Metal prices
- Interpretations of mineralization geometry and continuity in mineralization zones
- Geotechnical and hydrogeological assumptions
- Geochemical assumptions for mined waste materials
- Ability of the mining operation to meet the annual production rate and anticipated grade control standards and recoveries
- Ability of the milling operation to meet the annual production rate and recoveries
- Operating cost assumptions and cost creep
- Ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social license to operate
- Ability to access capital for project financing.

22.13.1.5 Infrastructure

22.13.1.5.1 Hazard Considerations

Risks including seismic risk, geohazards, cryospheric hazards, flooding and river-related hazards, avalanche hazards, and climate-driven changes are discussed in Section 15.9.1.

22.13.1.5.2 On-site Infrastructure

Risks related to on-site infrastructure include:

- Lack of geotechnical information throughout the project site could impact the stability and locations of the proposed infrastructure designs, including the stockpiles and WRSFs.
- Building costs at the IA-level are based on benchmark costs and do not reflect actual structurally engineered designs. These costs could increase with a more detailed design that considers specific site conditions design criteria.

22.13.1.5.3 Off-site Infrastructure

Risks related to off-site infrastructure include:

- Certain assumptions have been made regarding the routing, capital cost, and operation of the power transmission line. These assumptions could change, the transmission line may require a different route option or configuration, which could result in increased costs or a delayed timeline.
- The Project assumes that the WSAR will be constructed on-time, on-budget, and to the current proposed design and routing. A change in any of these areas may impact the project.
- Early-stage discussions with Port Mackenzie regarding the ship loading facility for concentrate have been positive. However, no contracts or agreements exist between U.S. GoldMining and Port Mackenzie, and the port may not be available for use as planned in the future.

22.13.1.5.4 Geotechnical

The project has proposed a CDSF in an area that has not been previously studied. As a result, no geotechnical data is currently available for the region. The key risks associated with the CDSF location and study area are as follows:

- Ground conditions, geological containment, and slope stability within the proposed CDSF footprint are unknown, as no geotechnical investigation program has been completed to date.
- There is potential for increased project costs if geotechnical or hydrogeological conditions differ from the assumptions used in this study. Any deviations may affect capital costs, sustaining capital, and operating costs associated with the CDSF.

22.13.1.5.5 CDSF

- The majority of waste rock is assumed to be PAG, and requires subaqueous deposition which adds sustaining and operational costs to the project.
- The CDSF is primarily located outside the property boundaries. Although this has been discussed and reviewed with the current mineral claims holder, any changes to these agreements could potentially affect the project's cost or schedule.
- Very limited public information was available about the wetlands at the project site at the time of this study. The CDSF location and design may need to be revised when more data becomes available, which could potentially change the overall project costs.

22.13.1.6 Environmental Studies, Permitting and Plans, Negotiations, or Agreements with Local Individuals or Groups

22.13.1.6.1 Environmental Studies

Risks related to environmental studies are:

- Multi-year and seasonal baseline environmental studies are limited or have not been completed to date. Similarly, socio-economic studies to support future permtting and community engagement are also limited.
- There have been no comprehensive multi-year Salmon studies in the area. Salmon are an important species in Alaska, and the project may face certain challenges to design water management structures that consider Salmon.
- There is no site weather station established and site-specific meteorological conditions are unknown.
- The requirement for water treatment will be further assessed based on water balance considerations and geochemistry source term studies to be completed as the feasibility study progresses. Those studies need to progress to better identify potential risks and increased costs.

22.13.1.6.2 Permitting

Risks related to permitting are:

- Permitting requirements, timelines, and impediments have not yet been determined.

22.13.1.6.3 Social Considerations

Risks related to social considerations are:

- From the QPs understanding, limited social engagement has been completed.
- The QP has not verified the status of native corporation lands, tribal lands, or other local land use permissions or ownership.

22.13.1.6.4 Closure

Risks related to closure are:

- Limited information indicates a high probability of ARD/ML, and the reclamation / closure plan and the soil covers for waste rock and tailings may may be insufficient as engineered covers may be required, as well as long-term water treatment, resulting in an underestimate of closure costs.

22.13.1.7 Economic Analysis

The risks or uncertainties that could reasonably be expected to affect the reliability or confidence in the projected economic outcomes are:

- Geological and resource uncertainty.
- Metallurgical and processing uncertainty.
- Mining and geotechnical uncertainty.
- Infrastructure assumptions.
- Capital and operating cost uncertainties.
- Commodity price and market risks.
- Environmental, permitting, and regulatory risks.
- Social and community considerations.
- Political and jurisdictional risk.
- Project schedule assumptions.

22.13.2 Opportunities

22.13.2.1 Metallurgical Testing and Processing and Recovery Methods

Should additional metallurgical testwork become available, opportunities exist to optimize the flowsheet to improve recovery, reduce operating costs, and further de-risk the project. Further phases of development include the following engineering trade-off studies:

- Optimum grind size and regrind size selection.
- Opportunities to reduce material reporting to the cyanide leach circuit based on classification or relative density separation techniques.
- Optimization of flotation, leaching, and adsorption operating conditions.
- Pre-flotation gravity separation to reduce material reporting to the cyanide leach circuit.

22.13.2.2 Mineral Resource Estimate

There are potential additional resource estimates which are not currently considered in the IA with economic analysis:

- **Island Mountain Resource**, located 15 miles south of Whistler deposit contains open-pit Inferred Mineral Resources of 187 Mt at 0.42 g/t AuEq for 2.54 Moz AuEq.
- **Raintree West Resource**, located <1 mile east of Whistler deposit, contains open pit Indicated resources of 10 Mt at 0.52 g/t AuEq for 156 koz AuEq, and Inferred open pit resources of 18.8 Mt at 0.55 g/t AuEq for 289 koz AuEq. Underground Indicated resources are 4.6 Mt at 0.85 g/t AuEq for 127 koz AuEq, and Inferred resources of 80 Mt at 0.80 g/t AuEq for 2.06 Moz AuEq.

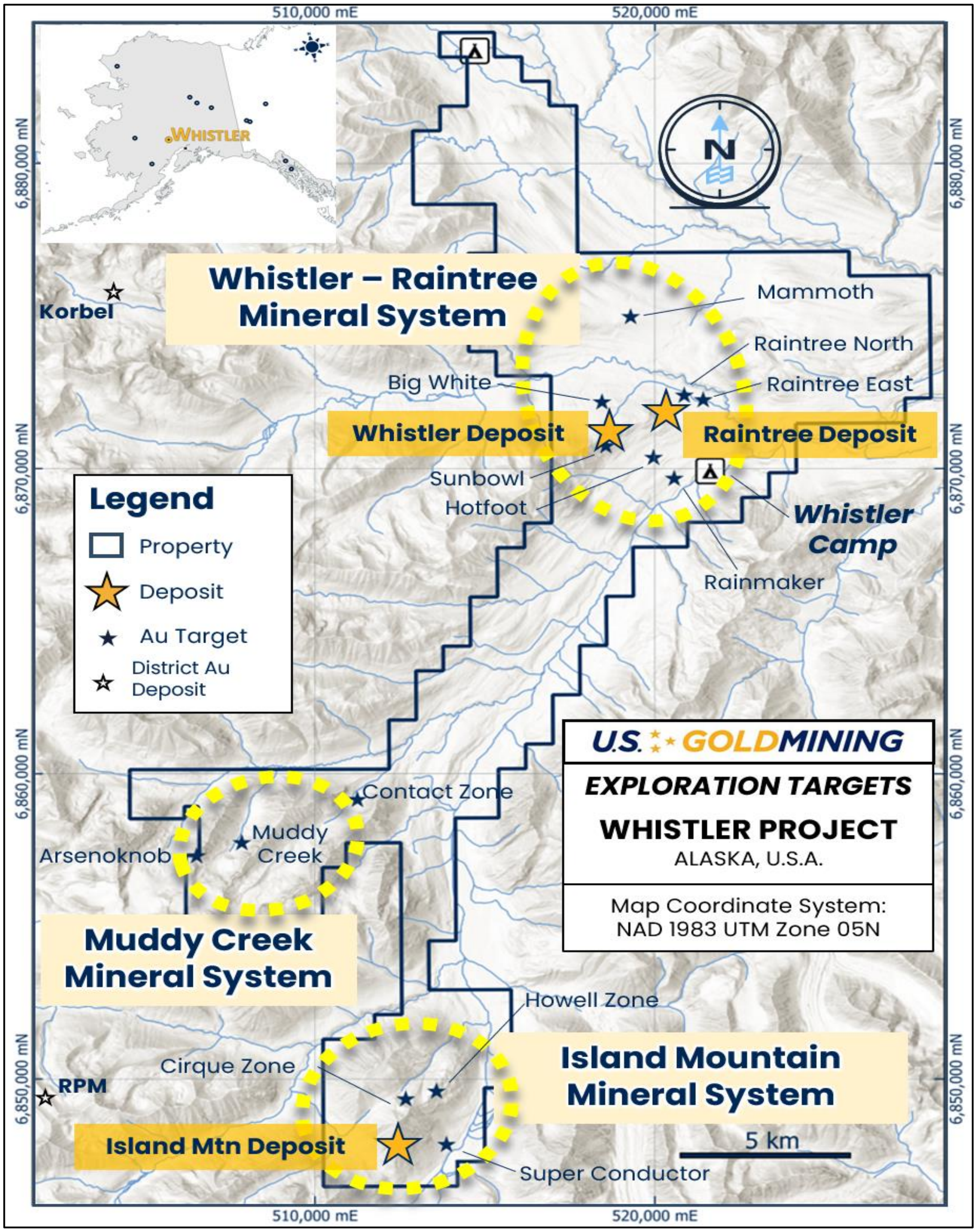
There is additional future exploration expansion potential of Whistler Deposit including 'wingspan' exploration potential remains around the edges of the current mineral resource:

- Along strike to the north associated with under-drilled Induced Polarization (IP) chargeability anomalies.
- To the south of the currently delineated Whistler Deposit on the southern margin of a post-mineral dyke.
- Additional infill drilling could also potentially provide greater confidence in connecting zones of higher-grade mineralization.

There is also additional Whistler Property-scale exploration as illustrated in Figure 22-1 includes three broad mineral systems are identified within the Whistler Property:

- Whistler – Raintree mineral system, or 'Whistler Orbit', comprises a classic porphyry cluster over an area of approximately 5 km x 5 km with potential for additional gold ± copper porphyry-style mineralization to be discovered.
- Island Mountain mineral system – encompasses the known Island Mountain deposit plus several additional porphyry or intrusion related gold targets over an area of mapped intrusive rocks with diameter of approximately 3 km.
- Muddy Creek mineral system – a large gold-in-soil geochemical footprint with an intrusion-related gold geochemical signature over an area of approximately 6 km x 4 km.

Figure 22-1: The Whistler Project



Notes: The Whistler Project contains three gold ± copper ± silver mineral systems: Whistler – Raintree which contains the existing Whistler and Raintree deposits, Island Mountain which contains the namesake gold deposit and several additional undrilled targets, and Muddy Creek which contains potential for discovery of an intrusive-related gold system. Source: U.S. GoldMining, 2026

22.13.2.3 Infrastructure

22.13.2.3.1 Geotechnical

- Opportunity to conduct a more robust geochemical analysis to define the NAG/PAG spilt. This study can help reduce the PAG crushing and handling costs, the tailings handling and storage costs, and subsequently the overall size of the CDSF, and associated potential impacts.

22.13.2.3.2 CDSF

- Look at potential opportunity to optimize infilling of the void space in the PAG waste rock with tailings to reduce the storage capacity requirements of the CDSF.

22.13.2.4 Environmental Studies, Permitting and Agreements with Local Individuals or Groups

22.13.2.4.1 Environmental Studies

Opportunities related to environmental studies are:

- Building on current information, commence well-informed, technologically advanced environmental, socio-economic, and cultural baseline studies to support the Project throughout permitting, development, operations and closure phases.
- There are opportunities to work closely with ongoing geological and geotechnical drilling program teams to work cooperatively and collect required geochemical samples and establish a hydrogeological monitoring and testing program that will be required for future feasibility and permitting phases of the project. Building on the current geological model and available geochemical data, design and commence robust geochemical monitoring and testing programs to inform water management and treatment requirements during operations and closure.

22.13.2.4.2 Permitting

Opportunities related to permitting are:

- Ongoing engagement with permitting consultant to support the permitting process and associated regulatory requirements.

22.13.2.4.3 Local Individuals or Groups

Opportunities related to local individuals or groups involve:

- Engage with local groups to learn more about historical and current land and resource use in the area, especially concerning the harvesting of Salmon, and how the Project can work to minimize impacts to harvestors and to support Salmon habitat.
- Engage governments and groups early and often, being transparent with the plans for the Project, and opening two-way discussions on how to build a better project and support the local Indigenous and non-Indigenous communities.

23 RECOMMENDATIONS

23.1 Introduction

The study highlights positive economic results based on a conceptual level of design. Further exploration work is recommended to evaluate the Project’s mineral potential. Additional field work, laboratory testwork, and analysis are required prior to advancing to a PFS. It is also recommended that the Company initiates environmental baseline studies along with ramping up engagement with permitting consultants, government, local groups, local communities, and regional infrastructure owners. The recommended work is estimated to cost US$68.7 million and is summarized in Table 23-1.

Table 23-1: Recommended Work Programs

PFS Program Component	Total Cost (US$M)
Exploration and Resource Drilling	20.0
Metallurgical Testwork	1.5
Mining Drilling and Testwork	7.6
Process Engineering and Testwork	1.5
Infrastructure Hazard Mitigation	3.8
Infrastructure Geotechnical Program	6.0
Environmental Baseline Programs	15.0
Permitting Consultant	3.0
Government and Community Engagement	3.0
Geochemical Program	2.5
Groundwater and Surface Water Investigations	2.0
Wetlands Mapping and Delineation	0.3
PFS-level Engineering Study	2.5
Total	**68.7**

23.2 Exploration and Resource

23.2.1 Whistler

At the Whistler Deposit, recommendations include:

- Ongoing development of the geologic model to provide a better understanding of how the three stages of intrusion relate to the mineralization geometry and continuity. This would involve utilizing the logging with the current knowledge of the multi-element assay values and hyperspectral data. Solids modelling of each intrusive phase and alteration facies could be constructed from this interpretation.
- A better understanding of the current known faults could be an opportunity for increasing the resource at Whistler, especially in the northern section of the deposit where the Rover Fault system dissects and complicates the geometry of lithological contacts, alteration and mineralization facies.
- Additional deep drilling to investigate the potential for the High-Grade Core, or other potential zones of high-grade mineralization, to extend to depth. Noting that the Raintree West Underground Resource lies at a deeper vertical horizon compared with the Whistler deposit, there is potential for Whistler mineralization to extend well beyond the base of the currently estimated open pit mineral resource.

23.2.2 Raintree West

For the Raintree West deposit, the following recommendations are made:

- Infill and step-out drilling to the north and south of the current deposit to potentially upgrade the classification of the current resource estimate and to potentially increase the resource. Specifically shallow holes (200 to 250 m) dipping east on sections 6,871,350 N and 6,871,400 N and 6,871,500 N should be drilled to increase the confidence in near-surface mineralization.
- In concert with the new drilling, the previous drill core should be relogged and a robust geological model/domains should be constructed for future resource estimates.
- Further specific gravity measurements should be collected from current and future drillholes.
- Metallurgical testing should be conducted on Raintree West samples.

23.2.3 Island Mountain

For the Island Mountain deposit, the following recommendations are made:

- Infill and step-out drilling to the north and south of the deposit. This drilling should be done to potentially upgrade the classification of the current resource estimate and to potentially increase the resource. Drilling should aim to link the mineralized breccias drilled north of the resource area, with the main breccia complex. Deep drilling under the breccia complex is also warranted to potentially locate the causative, and potentially mineralized, intrusive driving the brecciation.

23.2.4 Exploration Program and Budget

The recommended exploration program is divided into two streams: known deposit expansion and delineation, and exploration for discovery of new porphyry deposits.

1) Known deposit expansion and delineation – Whistler, Raintree West, and Island Mountain – up to 10,000 m core drilling to test the wingspan expansion of the existing deposits and upgrade resource classification.
   a) Follow-up drilling of the Whistler deposit (approximately 5,000 m) should target lateral and depth expansion opportunities to grow the mineral resource estimate (MRE), especially within the existing MRE constraining pit shell where waste can potentially be converted to mineralization with additional drilling.
   b) Grid drilling program and step-out drilling at Raintree West (2,500 m). Any significant mineralized intercepts from this phase of step-out drilling should be sent for metallurgical testing with particular focus on the impact of the relatively high lead-zinc concentrations.
   c) Approximately 2,500 m of diamond drilling to infill and expand mineralization at the Breccia Zone at Island Mountain. Mineralization is open to south and north, and undrilled breccia bodies occur for 700 m to the north of the Breccia Zone.
2) Exploration for discovery of new porphyry deposits.
   a) Focused on the Whistler Orbit and Island Mountain areas which represent classic ‘porphyry clusters’.
   b) Review of previous soil/till geochemical sampling coupled with surficial geological mapping, and further surface mapping and sampling, plus “top-of-bedrock” relogging and resampling of existing core holes in the Whistler Orbit area, and potentially additional top-of-bedrock grid drilling program in the Whistler Orbit area. The grid drilling program would penetrate the glacial cover and drill approximately 25 m into bedrock to obtain geological and geochemical data. Drilling on 200-m centers from fifty holes (1,250 m) would cover the most prospective areas in the Whistler area.
   c) This data, in conjunction with the existing airborne magnetic data and 3D IP data, would considerably enhance exploration targeting for deeper drill testing of high priority targets: 2500 m at Whistler and 2500 m at Island Mountain.
   d) Compilation work to rank and prioritize other exploration targets on the project area (Muddy Creek, Snow Ridge, Puntilla, Round Mountain, Howell Zone, Super Conductor), with the aim to test one or more of these targets with drilling (1,500 m).

23.3 Sample Preparation, Analysis and Security

To ensure and further improve data quality, MMTS recommends that:

- Three suitable CRMs that are made of porphyry copper material and represent expected low, medium, high mineralization grades for Au, Cu, and Ag be sourced and included in any future drilling. None of the CRMs used before 2023 were certified for silver.

- Future drilling should continue using coarse/crush blank material (quartz or limestone).
- Individual failed samples need to be identified in a timely manner and the neighboring primary assays samples be re-assayed if warranted. If this was indeed done in the past, the database has not been correctly maintained as the provided re-assay certificates do not cover all failures. The number of failures does not appear to be of material significance currently. Future programs should adhere to standardized control procedures.

The estimated cost for these recommendations is US$150,000. These costs are integrated in the all-in exploration costs of US$20.0 million.

23.4 Metallurgical Testwork

Based on the current study, it is recommended that the project advance to the next stage of PFS engineering evaluation. Metallurgical testwork completed to date has demonstrated that the proposed process flowsheet is technically viable and provides a reasonable basis for the metallurgical recovery assumptions used in the study. Additional metallurgical testwork is recommended as the project advances to the PFS stage to further increase confidence in metallurgical performance assumptions and to provide the engineering parameters required for the PFS process plant design. The recommended program should include additional variability flotation testing across representative mineralized material domains, further locked-cycle testing to confirm metallurgical performance, comminution testing to support crushing and grinding circuit design, direct cyanide leaching of the flotation tailings without a pyrite flotation stage, and environmental evaluation. Drill samples are to be provided from the exploration drilling budget, or from other core from previous recent drill programs.

The estimated cost for the metallurgical testwork is up to US$1.5 million including testwork management.

23.5 Mining Methods

MMTS recommends the following testwork and analysis to advance the Whistler project to Prefeasibility level engineering designs:

- Open pit geotechnical and hydrogeological testwork and analysis, including:
  - Geomechanical logging.
  - In-situ permeability testing.
  - Piezometer installation.
  - Groundwater monitoring.
  - Laboratory rock strength tests will be conducted on representative core samples for intact rock and joint properties. Additional surface mapping and televiewer borehole surveys are also recommended to collect additional structural data for the rock mass.
  - Refining a geotechnical model that incorporates site-specific geotechnical and hydrogeological data into the existing geological and structural model, to establish a 3D geotechnical model for further pit slope design.

   - Develop a hydrogeological model for pre-mining conditions. The model will provide key input for slope stability analyses and pit dewatering design.
   - Further slope stability analyses should be conducted based on the additional geomechanical data and creation of the geotechnical model. Both kinematic and rock mass stability analyses should be performed to determine appropriate pit slope geometries for designated design sectors at various stages.
- Waste rock geochemical characterization, with targeted updates to source terms and PAG definition.
- Geotechnical analysis of the foundations identified for the WRSF's.
- A site study of mountain operation avalanche risks and potential risk mitigations.
- Condemnation drilling of the footprints identified for the WRSF's and site infrastructure.
- Drill and blast testing to be carried out by drilling vendors and local explosives suppliers by analysing local rock types and conditions to assess the achievable drill penetration rates, optimal explosives mix and target powder factor for use in this operation.
- Updating of all mine planning work to incorporate results from other recommended studies; including optimization studies for pit limits and mine scheduling.
- Mine operational and cost trade-off studies examining contractor vs. owner equipment fleets, cost comparisons of various equipment class sizes, and utilization of electrically driven mine equipment (including trolley or battery systems for haulers) over diesel driven units.

Geotechnical drilling and hydrogeological pumping tests = US$3.0 million.

Geotechnical and hydrogeological lab work and engineering for PFS-level design = US$1.0 million.

Geochemistry fieldwork, lab work, and analysis program = US$0.8 million.

Site avalanche risk study = US$0.3 million.

Condemnation drilling = US$2.0 million.

Mining Engineering and Analysis = US$0.5 million.

The estimated cost for these recommendations is US$7.6 million.

23.6 Processing and Recovery Methods

In next stage of study, the process plant design should be further refined through advancement of the process flowsheet, mass balance, and design criteria. This includes confirmation of the comminution circuit configuration and operating parameters for the HPGR and downstream grinding circuit, refinement of the flotation circuit configuration and residence time requirements for copper and gold recovery, and optimization of the downstream cyanide leach circuit for gold recovery from flotation tailings. The design of the cyanide leaching and carbon adsorption circuit should also be advanced to confirm leach residence time, carbon inventory, detoxification requirements, and integration with the overall plant water balance.

Completion of this work during the next stage will allow refinement of the process plant design, improve confidence in the operating and capital cost estimates, and reduce technical and execution risks associated with the proposed processing and recovery methods.

The estimated cost for these recommendations is US$1.5 million including process engineering design work.

23.7 Infrastructure

23.7.1 On-site and Off-site Infrastructure

There are several recommendations with regards to infrastructure that should be undertaken. These are related to the transmission line, power source, access road, and ship loading facility.

Transmission line and power source:

- Continued engagement with AIDEA and Nova Minerals is recommended to confirm proposed transmission line routing and sharing of powerline costs.
- Continued engagement with Chugach Electrical Association is recommended to confirm availability of power, potential sharing of powerline capital costs, and confirmation and potential reduction of the rate schedule over the LOM.
- Continued engagement with alternative power source providers including the proposed Donlin gas pipeline and the proposed Terra Energy coal power plant is recommended.

Access road:

- Continued engagement with AIDEA and state government is recommended to support the WSAR timeline and cost structure.

Ship loading facility:

- Continued engagement with Port Mackenzie is recommended to align any port upgrades, construction, and/or expansion with the timeline of the Whistler Project.

The estimated cost for these recommendations is US$100,000 per year, for a total of US$300,000 for a three year period.

Hazard Considerations:

- Complete detailed terrain and geohazard mapping to avoid locating infrastructure on or below unstable slopes.
- Conduct permafrost characterization using thermal modeling and boreholes (if required) prior to infrastructure placement.

- Develop severe-weather response plans, including pre-storm shutdown procedures and access restrictions.
- Perform avalanche hazard mapping for access routes, camps, and facilities.

The estimated cost of these hazard mitigation recommendations are US$3.5 million.

The total estimated cost for these infrastrucure-related recommendations is US$3.8 million.

23.7.2 Geotechnical

Due to the conceptual nature of this study and the limited information available at the time of writing, several assumptions have been made regarding the layout, MTOs, and construction of the proposed CDSF. Geotechnical properties of construction materials will be required to support slope stability analyses and other geotechnical assessments needed to confirm that the CDSF can be constructed as designed. Additionally, a tailings and PAG waste rock deposition plan will be required, which may result in adjustments to the conceptual staging to accommodate the required storage capacities.

Additional studies and data collection will be required to advance project development beyond the conceptual stage. Some—though not necessarily all—of the current data gaps that should be addressed in future studies include the following:

- Geological and geotechnical site investigations, including drilling, in-situ testing, and laboratory testing for the infrastructure, process plant, WRSF, and CDSF, to characterize subsurface soil and rock conditions, construction material properties, and existing groundwater levels.
- Seepage, stability, and deformation analyses for the CDSF, based on geotechnical field and laboratory programs, geochemical analyses, seismic hazard studies, and hydrological assessments.
- Additional geotechnical testing of anticipated tailings, waste rock, and construction materials, including embankment filter materials, underdrain materials, and potential geomembrane liners within the embankments.
- Development of site-specific geohazard and seismic hazard studies to refine design parameters and support long-term risk assessment.
- Collection of hydrological information through site-specific climate and hydrological studies to support the design of ponds, diversion channels, spillways, development of the CDSF water balance and tying in with the site-wide water balance.
- Collection of hydrogeological information from desktop studies and site investigations to improve understanding of subsurface flow regimes and their implications for facility design and performance.

As additional information is obtained, assumptions made in this study can be verified or updated to advance the project to the next level of design. The cost of implementing the above recommendations is estimated at US$5.0 million including drilling and test pitting equipment, and geotechnical engineering design.

An additional robust geotechnical site investigation program should be expanded to include the process plant area, crusher area, and stockpile areas, in addition to proposed road areas. The estimated cost for these recommendations is US$1.0 million.

23.8 Environmental Studies, Permitting and Plans, Negotiations, or Agreements with Local Individuals or Groups

Recommendations related to environmental studies, permitting and plans, negotiations, or agreements with local individuals or groups are summarized below:

- Commence the design and implementation of a robust multi-year and seasonal baseline environmental studies program that aligns with BLM, DEC, EPA, and NEPA EIS guidelines. The estimated cost for these recommendations is up to US$5.0 million per year; recommended to continue over a period of three years, for a total cost of up to US$15.0 million.

- Engage a professional permitting consultant to support permitting efforts throughout the exploration and development phases. The estimated cost for these recommendations is US$1.0 million per year; recommended to continue over a period of 3 years, for a total cost of US$3.0 million.

- Engage with government, local groups, and communities to align the Project with external expectations and requirements, and to develop open and transparent relationships to help drive project success. The estimated cost for these recommendations is US$1.0 million per year; recommended to continue over a period of 3 years, for a total cost of US$3.0 million.

- Start a comprehensive geochemical testing program which includes waste rock and mill feed characterization, formal acid–base accounting, kinetic testwork, tailings geochemical work, water-rock interaction modelling, geochemical source term modelling, freeze-thaw geochemical testwork, and associated data management, QA/QC, interpretations and reporting to support PFS documentation and integration with EIS baseline studies. The estimated cost for these recommendations is US$2.5 million.

- Start detailed groundwater and surface water investigations including water quality and water geochemistry programs to support an integrated water balance model and site-specific water quality objectives. The total cost for these recommendations is US$2.0 million.

- Complete updated NWI mapping and ground-based wetland delineation studies, including functional assessments, as the project advances. The total cost for these recommendations is US$0.25 million.

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25 RELIANCE ON INFORMATION PROVIDED BY THE REGISTRANT

25.1 Legal Matters Outside the Expertise of the Qualified Person

The QPs have relied upon other expert reports that provided information regarding mineral rights, surface rights, property agreements, royalties, and taxation contained within this Report. The QPs considered it reasonable to rely on U.S. GoldMining for this information because U.S. GoldMining has retained experienced third-party consultants to manage this information.

25.1.1 Mineral Tenure and Surface Rights

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area or underlying property agreements. The QPs have fully relied on information supplied by U.S. GoldMining, through its parent, GoldMining, experts retained by GoldMining for this information through the following documents:

- Letter from Stoel Rives, LLP dated December 8, 2025, and titled: Limited Title Review for Alaska State Mining Claims.

This title information is used in Sections 1.3.1, 3.1, 3.2, and 3.3 of the report.

25.1.2 Royalties and Encumbrances

The QPs have not reviewed the royalty agreements nor independently verified the legal status of the royalties and other potential incumbrances. The QPs have fully relied on information supplied by U.S. GoldMining, for this information through the following documents. This information was provided as a series of documents from U.S. GoldMining:

- Funding Agreement and Net Smelter Returns Royalty Agreement dated January 11, 2021.
- Notice confirming Osisko Mining (USA) Inc. Is the royalty holder.
- Notice of Address Change & Assignment of Buy-back Right - Osisko Mining (USA) Inc.
- Royalty Holder Notice to U.S. GoldMining (Whistler Royalty), Nevada Select Royalty Inc.

This title information is used in Sections 1.3.2, 3.1, 3.5, 19 and 22.12 of the report.

25.2 Environmental Matters Outside the Expertise of the Qualified Person

The Ausenco QPs have not independently reviewed the current status or future prospects of mineral explorations and mining permits held or required by U.S. GoldMining for work on the Project. The Ausenco QPs have fully relied on information supplied by U.S. GoldMining.

This permitting information is used in Sections 1.15, 17 and 22.9, of the report.

25.3 Macroeconomic Trends, Data and Assumptions, and Interest Rates

The Ausenco QPs have not independently reviewed the taxation information. The Ausenco QPs have fully relied on information supplied by U.S. GoldMining on behalf of their third-party consultant Mining Tax Plan LLC, for information related to taxation.

This expert information is used in support of the sub-section on tax information and the tax inputs to the financial model that provides the post-tax financial analysis in Sections 1.17, 19 and 22.12 of the report.

APPENDIX A – MINERAL CLAIMS

[MTRS = Meridian, Township, Range, Section; Q = ¼ Section; Q-Q = ¼ ¼ Section]

No.	ADL Serial Number	Claim Name	Claim Owner	MTRS Q or Q-Q
1	633446	PORT 2151*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 NE4NW4
2	633447	PORT 2152*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 NW4NE4
3	633448	PORT 2153*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 NE4NE4
4	633449	PORT 2251*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 SE4SW4
5	633450	PORT 2252*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 SW4SE4
6	633451	PORT 2253*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 SE4SE4
7	633452	PORT 2351*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 NE4SW4
8	633453	PORT 2352*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 NW4SE4
9	633454	PORT 2353*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 NE4SE4
10	633455	PORT 2354*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 NW4SW4
11	633456	PORT 2355*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 NE4SW4
12	633457	PORT 2454*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SW4NW4
13	633458	PORT 2455*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SE4NW4
14	633459	PORT 2456*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SW4NE4
15	633460	PORT 2457*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SE4NE4
16	633461	PORT 2458*	U.S. GoldMining Inc.	S 22N 18W Sec. 21 SW4NW4
17	633462	PORT 2459*	U.S. GoldMining Inc.	S 22N 18W Sec. 21 SE4NW4
18	633463	PORT 2555*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 NE4NW4
19	633464	PORT 2556*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 NW4NE4
20	633465	PORT 2557*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 NE4NE4
21	633466	PORT 2558*	U.S. GoldMining Inc.	S 22N 18W Sec. 21 NW4NW4
22	633467	PORT 2559*	U.S. GoldMining Inc.	S 22N 18W Sec. 21 NE4NW4
23	633468	PORT 2655*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 SE4SW4
24	633469	PORT 2656*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 SW4SE4
25	633470	PORT 2657*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 SE4SE4
26	641182	WHISPER 105*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 NW4SW4
27	641183	WHISPER 106*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 SW4SW4
28	641184	WHISPER 107*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 NE4SW4
29	641185	WHISPER 108*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 NW4SE4
30	641186	WHISPER 109*	U.S. GoldMining Inc.	S 22N 18W Sec. 17 NE4SE4
31	641187	WHISPER 120*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 NW4NW4
32	641188	WHISPER 127*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 NW4SW4
33	641189	WHISPER 128*	U.S. GoldMining Inc.	S 22N 18W Sec. 19 SW4SW4
34	641190	WHISPER 129*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SW4SW4
35	641191	WHISPER 130*	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SE4SW4

No.	ADL Serial Number	Claim Name	Claim Owner	MTRS Q or Q-Q
36	641192	WHISPER 139*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 NW4NW4
37	641193	WHISPER 140*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 SW4NW4
38	641194	WHISPER 141*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 SE4NW4
39	641195	WHISPER 142*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 SW4NE4
40	641196	WHISPER 143*	U.S. GoldMining Inc.	S 22N 18W Sec. 30 SE4NE4
41	641197	WHISPER 1	U.S. GoldMining Inc.	S 23N 19W Sec. 23 NW4
42	641198	WHISPER 2	U.S. GoldMining Inc.	S 23N 19W Sec. 23 NE4
43	641199	WHISPER 3	U.S. GoldMining Inc.	S 23N 19W Sec. 24 NW4
44	641201	WHISPER 9	U.S. GoldMining Inc.	S 23N 19W Sec. 23 SW4
45	641202	WHISPER 10	U.S. GoldMining Inc.	S 23N 19W Sec. 23 SE4
46	641203	WHISPER 11	U.S. GoldMining Inc.	S 23N 19W Sec. 24 SW4
47	641204	WHISPER 12	U.S. GoldMining Inc.	S 23N 19W Sec. 24 SE4
48	641206	WHISPER 17	U.S. GoldMining Inc.	S 23N 19W Sec. 26 NW4
49	641207	WHISPER 18	U.S. GoldMining Inc.	S 23N 19W Sec. 26 NE4
50	641208	WHISPER 19	U.S. GoldMining Inc.	S 23N 19W Sec. 25 NW4
51	641209	WHISPER 20	U.S. GoldMining Inc.	S 23N 19W Sec. 25 NE4
52	641212	WHISPER 27	U.S. GoldMining Inc.	S 23N 19W Sec. 26 SW4
53	641213	WHISPER 28	U.S. GoldMining Inc.	S 23N 19W Sec. 26 SE4
54	641214	WHISPER 29	U.S. GoldMining Inc.	S 23N 19W Sec. 25 SW4
55	641215	WHISPER 30	U.S. GoldMining Inc.	S 23N 19W Sec. 25 SE4
56	641218	WHISPER 37	U.S. GoldMining Inc.	S 23N 19W Sec. 35 NW4
57	641219	WHISPER 38	U.S. GoldMining Inc.	S 23N 19W Sec. 35 NE4
58	641220	WHISPER 39	U.S. GoldMining Inc.	S 23N 19W Sec. 36 NW4
59	641221	WHISPER 40	U.S. GoldMining Inc.	S 23N 19W Sec. 36 NE4
60	641227	WHISPER 48	U.S. GoldMining Inc.	S 23N 19W Sec. 35 SE4
61	641228	WHISPER 49	U.S. GoldMining Inc.	S 23N 19W Sec. 36 SW4
62	641229	WHISPER 50	U.S. GoldMining Inc.	S 23N 19W Sec. 36 SE4
63	641241	WHISPER 63	U.S. GoldMining Inc.	S 22N 18W Sec. 6 SW4
64	641242	WHISPER 64	U.S. GoldMining Inc.	S 22N 18W Sec. 6 SE4
65	641247	WHISPER 69	U.S. GoldMining Inc.	S 22N 18W Sec. 7 NW4
66	641248	WHISPER 70	U.S. GoldMining Inc.	S 22N 18W Sec. 7 NE4
67	641249	WHISPER 71	U.S. GoldMining Inc.	S 22N 18W Sec. 8 NW4
68	641250	WHISPER 72	U.S. GoldMining Inc.	S 22N 18W Sec. 8 NE4
69	641251	WHISPER 73	U.S. GoldMining Inc.	S 22N 18W Sec. 9 NW4
70	641252	WHISPER 74	U.S. GoldMining Inc.	S 22N 18W Sec. 9 NE4
71	641257	WHISPER 79	U.S. GoldMining Inc.	S 22N 18W Sec. 7 SW4
72	641258	WHISPER 80	U.S. GoldMining Inc.	S 22N 18W Sec. 7 SE4
73	641259	WHISPER 81	U.S. GoldMining Inc.	S 22N 18W Sec. 8 SW4
74	641260	WHISPER 82	U.S. GoldMining Inc.	S 22N 18W Sec. 8 SE4
75	641261	WHISPER 83	U.S. GoldMining Inc.	S 22N 18W Sec. 9 SW4
76	641262	WHISPER 84	U.S. GoldMining Inc.	S 22N 18W Sec. 9 SE4

No.	ADL Serial Number	Claim Name	Claim Owner	MTRS Q or Q-Q
77	641263	WHISPER 85	U.S. GoldMining Inc.	S 22N 18W Sec. 10 SW4
78	641267	WHISPER 89	U.S. GoldMining Inc.	S 22N 19W Sec. 13 NW4
79	641268	WHISPER 90	U.S. GoldMining Inc.	S 22N 19W Sec. 13 NE4
80	641269	WHISPER 91	U.S. GoldMining Inc.	S 22N 18W Sec. 18 NW4
81	641270	WHISPER 92	U.S. GoldMining Inc.	S 22N 18W Sec. 18 NE4
82	641271	WHISPER 93	U.S. GoldMining Inc.	S 22N 18W Sec. 17 NW4
83	641272	WHISPER 94	U.S. GoldMining Inc.	S 22N 18W Sec. 17 NE4
84	641273	WHISPER 95	U.S. GoldMining Inc.	S 22N 18W Sec. 16 NW4
85	641274	WHISPER 96	U.S. GoldMining Inc.	S 22N 18W Sec. 16 NE4
86	641275	WHISPER 181	U.S. GoldMining Inc.	S 22N 19W Sec. 12 SE4
87	641276	WHISPER 97	U.S. GoldMining Inc.	S 22N 18W Sec. 15 NW4
88	641280	WHISPER 101	U.S. GoldMining Inc.	S 22N 19W Sec. 13 SW4
89	641281	WHISPER 102	U.S. GoldMining Inc.	S 22N 19W Sec. 13 SE4
90	641282	WHISPER 103	U.S. GoldMining Inc.	S 22N 18W Sec. 18 SW4
91	641283	WHISPER 104	U.S. GoldMining Inc.	S 22N 18W Sec. 18 SE4
92	641284	WHISPER 110	U.S. GoldMining Inc.	S 22N 18W Sec. 16 SW4
93	641285	WHISPER 111	U.S. GoldMining Inc.	S 22N 18W Sec. 16 SE4
94	641286	WHISPER 112	U.S. GoldMining Inc.	S 22N 18W Sec. 15 SW4
95	641287	WHISPER 113	U.S. GoldMining Inc.	S 22N 18W Sec. 15 SE4
96	641291	WHISPER 117	U.S. GoldMining Inc.	S 22N 19W Sec. 24 NE4
97	641292	WHISPER 118	U.S. GoldMining Inc.	S 22N 18W Sec. 19 NW4
98	641293	WHISPER 119	U.S. GoldMining Inc.	S 22N 18W Sec. 19 NE4
99	641294	WHISPER 121	U.S. GoldMining Inc.	S 22N 18W Sec. 21 NE4
100	641295	WHISPER 122	U.S. GoldMining Inc.	S 22N 18W Sec. 22 NW4
101	641296	WHISPER 123	U.S. GoldMining Inc.	S 22N 18W Sec. 22 NE4
102	641299	WHISPER 126	U.S. GoldMining Inc.	S 22N 19W Sec. 24 SE4
103	641300	WHISPER 131	U.S. GoldMining Inc.	S 22N 18W Sec. 20 SE4
104	641301	WHISPER 132	U.S. GoldMining Inc.	S 22N 18W Sec. 21 SW4
105	641302	WHISPER 133	U.S. GoldMining Inc.	S 22N 18W Sec. 21 SE4
106	641303	WHISPER 134	U.S. GoldMining Inc.	S 22N 18W Sec. 22 SW4
107	641304	WHISPER 135	U.S. GoldMining Inc.	S 22N 18W Sec. 22 SE4
108	641305	WHISPER 138	U.S. GoldMining Inc.	S 22N 19W Sec. 25 NE4
109	641306	WHISPER 144	U.S. GoldMining Inc.	S 22N 18W Sec. 29 NW4
110	641307	WHISPER 145	U.S. GoldMining Inc.	S 22N 18W Sec. 29 NE4
111	641308	WHISPER 146	U.S. GoldMining Inc.	S 22N 19W Sec. 25 SE4
112	641309	WHISPER 147	U.S. GoldMining Inc.	S 22N 18W Sec. 30 SW4
113	641310	WHISPER 148	U.S. GoldMining Inc.	S 22N 18W Sec. 30 SE4
114	641311	WHISPER 149	U.S. GoldMining Inc.	S 22N 18W Sec. 29 SW4
115	641312	WHISPER 150	U.S. GoldMining Inc.	S 22N 18W Sec. 29 SE4
116	641313	WHISPER 151	U.S. GoldMining Inc.	S 22N 18W Sec. 28 SW4
117	641314	WHISPER 152	U.S. GoldMining Inc.	S 22N 18W Sec. 28 NW4

No.	ADL Serial Number	Claim Name	Claim Owner	MTRS Q or Q-Q
118	641315	WHISPER 153	U.S. GoldMining Inc.	S 22N 18W Sec. 28 SE4
119	641316	WHISPER 154	U.S. GoldMining Inc.	S 22N 18W Sec. 28 NE4
120	641317	WHISPER 155	U.S. GoldMining Inc.	S 22N 18W Sec. 27 SW4
121	641318	WHISPER 156	U.S. GoldMining Inc.	S 22N 18W Sec. 27 NW4
122	641319	WHISPER 182	U.S. GoldMining Inc.	S 22N 18W Sec. 31 NW4
123	641320	WHISPER 157	U.S. GoldMining Inc.	S 22N 18W Sec. 27 SE4
124	641321	WHISPER 158	U.S. GoldMining Inc.	S 22N 18W Sec. 27 NE4
125	641322	WHISPER 159	U.S. GoldMining Inc.	S 22N 18W Sec. 31 NE4
126	641323	WHISPER 160	U.S. GoldMining Inc.	S 22N 18W Sec. 32 NW4
127	641324	WHISPER 161	U.S. GoldMining Inc.	S 22N 18W Sec. 32 NE4
128	641325	WHISPER 162	U.S. GoldMining Inc.	S 22N 18W Sec. 33 NW4
129	641326	WHISPER 163	U.S. GoldMining Inc.	S 22N 18W Sec. 33 NE4
130	641327	WHISPER 164	U.S. GoldMining Inc.	S 22N 18W Sec. 34 NW4
131	641329	WHISPER 166	U.S. GoldMining Inc.	S 22N 18W Sec. 31 SE4
132	641330	WHISPER 167	U.S. GoldMining Inc.	S 22N 18W Sec. 32 SW4
133	641331	WHISPER 168	U.S. GoldMining Inc.	S 22N 18W Sec. 32 SE4
134	641332	WHISPER 169	U.S. GoldMining Inc.	S 22N 18W Sec. 33 SW4
135	641333	WHISPER 170	U.S. GoldMining Inc.	S 22N 18W Sec. 33 SE4
136	641334	WHISPER 171	U.S. GoldMining Inc.	S 21N 18W Sec. 5 NW4
137	641335	WHISPER 172	U.S. GoldMining Inc.	S 21N 18W Sec. 5 NE4
138	641337	WHISPER 174	U.S. GoldMining Inc.	S 22N 19W Sec. 1 NW4
139	641338	WHISPER 175	U.S. GoldMining Inc.	S 22N 19W Sec. 1 NE4
140	641339	WHISPER 176	U.S. GoldMining Inc.	S 22N 19W Sec. 1 SW4
141	641340	WHISPER 177	U.S. GoldMining Inc.	S 22N 19W Sec. 1 SE4
142	641341	WHISPER 178	U.S. GoldMining Inc.	S 22N 19W Sec. 12 NW4
143	641342	WHISPER 179	U.S. GoldMining Inc.	S 22N 19W Sec. 12 NE4
144	641343	WHISPER 180	U.S. GoldMining Inc.	S 22N 19W Sec. 12 SW4
145	644845	WHISPER 183	U.S. GoldMining Inc.	S 23N 19W Sec. 14 NW4
146	644846	WHISPER 185	U.S. GoldMining Inc.	S 23N 19W Sec. 14 SW4
147	644847	WHISPER 186	U.S. GoldMining Inc.	S 23N 19W Sec. 14 SE4
148	644848	WHISPER 187	U.S. GoldMining Inc.	S 23N 19W Sec. 15 NE4
149	645698	IM 1	U.S. GoldMining Inc.	S 19N 19W Sec. 6 SW4
150	645699	IM 2	U.S. GoldMining Inc.	S 19N 19W Sec. 6 SE4
151	645700	IM 3	U.S. GoldMining Inc.	S 19N 19W Sec. 5 SW4
152	645701	IM 4	U.S. GoldMining Inc.	S 19N 19W Sec. 5 SE4
153	645702	IM 5	U.S. GoldMining Inc.	S 19N 19W Sec. 4 SW4
154	645703	IM 10	U.S. GoldMining Inc.	S 19N 19W Sec. 6 NW4
155	645704	IM 11	U.S. GoldMining Inc.	S 19N 19W Sec. 6 NE4
156	645705	IM 12	U.S. GoldMining Inc.	S 19N 19W Sec. 5 NW4
157	645706	IM 13	U.S. GoldMining Inc.	S 19N 19W Sec. 5 NE4
158	645707	IM 14	U.S. GoldMining Inc.	S 19N 19W Sec. 4 NW4

No.	ADL Serial Number	Claim Name	Claim Owner	MTRS Q or Q-Q
159	645708	IM 15	U.S. GoldMining Inc.	S 19N 19W Sec. 4 NE4
160	645709	IM 19	U.S. GoldMining Inc.	S 20N 19W Sec. 31 SW4
161	645710	IM 20	U.S. GoldMining Inc.	S 20N 19W Sec. 31 SE4
162	645711	IM 21	U.S. GoldMining Inc.	S 20N 19W Sec. 32 SW4
163	645712	IM 22	U.S. GoldMining Inc.	S 20N 19W Sec. 32 SE4
164	645713	IM 23	U.S. GoldMining Inc.	S 20N 19W Sec. 33 SW4
165	645714	IM 24	U.S. GoldMining Inc.	S 20N 19W Sec. 33 SE4
166	645715	IM 28	U.S. GoldMining Inc.	S 20N 19W Sec. 31 NW4
167	645716	IM 29	U.S. GoldMining Inc.	S 20N 19W Sec. 31 NE4
168	645717	IM 30	U.S. GoldMining Inc.	S 20N 19W Sec. 32 NW4
169	645718	IM 31	U.S. GoldMining Inc.	S 20N 19W Sec. 32 NE4
170	645719	IM 32	U.S. GoldMining Inc.	S 20N 19W Sec. 33 NW4
171	645720	IM 33	U.S. GoldMining Inc.	S 20N 19W Sec. 33 NE4
172	645721	IM 34	U.S. GoldMining Inc.	S 20N 19W Sec. 34 NW4
173	645723	IM 37	U.S. GoldMining Inc.	S 20N 19W Sec. 29 SW4
174	645724	IM 38	U.S. GoldMining Inc.	S 20N 19W Sec. 29 SE4
175	645725	IM 39	U.S. GoldMining Inc.	S 20N 19W Sec. 28 SW4
176	645726	IM 40	U.S. GoldMining Inc.	S 20N 19W Sec. 28 SE4
177	645727	IM 41	U.S. GoldMining Inc.	S 20N 19W Sec. 27 SW4
178	645729	IM 44	U.S. GoldMining Inc.	S 20N 19W Sec. 29 NW4
179	645730	IM 45	U.S. GoldMining Inc.	S 20N 19W Sec. 29 NE4
180	645731	IM 46	U.S. GoldMining Inc.	S 20N 19W Sec. 28 NW4
181	645732	IM 47	U.S. GoldMining Inc.	S 20N 19W Sec. 28 NE4
182	645733	IM 48	U.S. GoldMining Inc.	S 20N 19W Sec. 27 NW4
183	645736	IM 52	U.S. GoldMining Inc.	S 20N 19W Sec. 20 SE4
184	645737	IM 53	U.S. GoldMining Inc.	S 20N 19W Sec. 22 SW4
185	645740	IM 57	U.S. GoldMining Inc.	S 20N 19W Sec. 20 NE4
186	646059	IM 6	U.S. GoldMining Inc.	S 20N 19W Sec. 30 SW4
187	646060	IM 7	U.S. GoldMining Inc.	S 20N 19W Sec. 30 SE4
188	646074	IM 61	U.S. GoldMining Inc.	S 19N 19W Sec. 7 NW4
189	646075	IM 62	U.S. GoldMining Inc.	S 19N 19W Sec. 7 NE4
190	646076	IM 63	U.S. GoldMining Inc.	S 19N 19W Sec. 8 NW4
191	646077	IM 64	U.S. GoldMining Inc.	S 19N 19W Sec. 8 NE4
192	646078	IM 65	U.S. GoldMining Inc.	S 19N 19W Sec. 9 NW4
193	646325	WHISPER 428	U.S. GoldMining Inc.	S 22N 18W Sec. 31 SW4
194	646327	WHISPER 430	U.S. GoldMining Inc.	S 21N 18W Sec. 6 NW4
195	646328	WHISPER 431	U.S. GoldMining Inc.	S 21N 18W Sec. 6 NE4
196	646330	WHISPER 433	U.S. GoldMining Inc.	S 21N 18W Sec. 6 SW4
197	646331	WHISPER 434	U.S. GoldMining Inc.	S 21N 18W Sec. 6 SE4
198	646338	WHISPER 441	U.S. GoldMining Inc.	S 21N 18W Sec. 7 NW4
199	646339	WHISPER 442	U.S. GoldMining Inc.	S 21N 18W Sec. 7 NE4

No.	ADL Serial Number	Claim Name	Claim Owner	MTRS Q or Q-Q
200	646343	WHISPER 446	U.S. GoldMining Inc.	S 21N 19W Sec. 12 SE4
201	646344	WHISPER 447	U.S. GoldMining Inc.	S 21N 18W Sec. 7 SW4
202	646350	WHISPER 453	U.S. GoldMining Inc.	S 21N 19W Sec. 13 NE4
203	646351	WHISPER 454	U.S. GoldMining Inc.	S 21N 18W Sec. 18 NW4
204	646355	WHISPER 458	U.S. GoldMining Inc.	S 21N 19W Sec. 13 SW4
205	646356	WHISPER 459	U.S. GoldMining Inc.	S 21N 19W Sec. 13 SE4
206	646764	IM 71	U.S. GoldMining Inc.	S 20N 19W Sec. 6 NE4
207	646765	IM 72	U.S. GoldMining Inc.	S 20N 19W Sec. 5 NW4
208	646766	IM 73	U.S. GoldMining Inc.	S 20N 19W Sec. 5 NE4
209	646767	IM 74	U.S. GoldMining Inc.	S 20N 19W Sec. 4 NW4
210	646774	IM 81	U.S. GoldMining Inc.	S 20N 19W Sec. 5 SE4
211	646775	IM 82	U.S. GoldMining Inc.	S 20N 19W Sec. 4 SW4
212	646783	IM 90	U.S. GoldMining Inc.	S 20N 19W Sec. 8 NE4
213	646784	IM 91	U.S. GoldMining Inc.	S 20N 19W Sec. 9 NW4
214	646792	IM 99	U.S. GoldMining Inc.	S 20N 19W Sec. 8 SE4
215	646793	IM 100	U.S. GoldMining Inc.	S 20N 19W Sec. 9 SW4
216	646801	IM 108	U.S. GoldMining Inc.	S 20N 19W Sec. 17 NE4
217	646802	IM 109	U.S. GoldMining Inc.	S 20N 19W Sec. 16 NW4
218	646810	IM 117	U.S. GoldMining Inc.	S 20N 19W Sec. 17 SE4
219	646819	IM 126	U.S. GoldMining Inc.	S 20N 19W Sec. 21 SW4
220	646820	IM 127	U.S. GoldMining Inc.	S 20N 19W Sec. 21 SE4
221	646824	WHISPER 464	U.S. GoldMining Inc.	S 23N 19W Sec. 27 NW4
222	646825	WHISPER 465	U.S. GoldMining Inc.	S 23N 19W Sec. 27 SW4
223	646826	WHISPER 466	U.S. GoldMining Inc.	S 23N 19W Sec. 34 NW4
224	646839	WHISPER 479	U.S. GoldMining Inc.	S 23N 19W Sec. 22 SE4
225	646840	WHISPER 480	U.S. GoldMining Inc.	S 23N 19W Sec. 27 NE4
226	646841	WHISPER 481	U.S. GoldMining Inc.	S 23N 19W Sec. 27 SE4
227	646842	WHISPER 482	U.S. GoldMining Inc.	S 23N 19W Sec. 34 NE4
228	646855	WHISPER 495	U.S. GoldMining Inc.	S 22N 19W Sec. 2 SW4
229	646856	WHISPER 496	U.S. GoldMining Inc.	S 22N 19W Sec. 11 NW4
230	646857	WHISPER 497	U.S. GoldMining Inc.	S 22N 19W Sec. 11 SW4
231	646858	WHISPER 498	U.S. GoldMining Inc.	S 22N 19W Sec. 14 NW4
232	646864	WHISPER 504	U.S. GoldMining Inc.	S 22N 19W Sec. 2 NE4
233	646865	WHISPER 505	U.S. GoldMining Inc.	S 22N 19W Sec. 2 SE4
234	646866	WHISPER 506	U.S. GoldMining Inc.	S 22N 19W Sec. 11 NE4
235	646867	WHISPER 507	U.S. GoldMining Inc.	S 22N 19W Sec. 11 SE4
236	646868	WHISPER 508	U.S. GoldMining Inc.	S 22N 19W Sec. 14 NE4
237	646869	WHISPER 509	U.S. GoldMining Inc.	S 22N 19W Sec. 14 SE4
238	646927	WHISPER 567	U.S. GoldMining Inc.	S 21N 19W Sec. 24 NW4
239	646928	WHISPER 568	U.S. GoldMining Inc.	S 21N 19W Sec. 24 NE4
240	646934	WHISPER 574	U.S. GoldMining Inc.	S 21N 19W Sec. 23 SE4

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241	646935	WHISPER 575	U.S. GoldMining Inc.	S 21N 19W Sec. 24 SW4
242	646942	WHISPER 582	U.S. GoldMining Inc.	S 21N 19W Sec. 26 NW4
243	646943	WHISPER 583	U.S. GoldMining Inc.	S 21N 19W Sec. 26 NE4
244	646944	WHISPER 584	U.S. GoldMining Inc.	S 21N 19W Sec. 25 NW4
245	646952	WHISPER 592	U.S. GoldMining Inc.	S 21N 19W Sec. 26 SW4
246	646953	WHISPER 593	U.S. GoldMining Inc.	S 21N 19W Sec. 26 SE4
247	646958	WHISPER 598	U.S. GoldMining Inc.	S 21N 19W Sec. 33 NW4
248	646959	WHISPER 599	U.S. GoldMining Inc.	S 21N 19W Sec. 33 NE4
249	646960	WHISPER 600	U.S. GoldMining Inc.	S 21N 19W Sec. 34 NW4
250	646961	WHISPER 601	U.S. GoldMining Inc.	S 21N 19W Sec. 34 NE4
251	646962	WHISPER 602	U.S. GoldMining Inc.	S 21N 19W Sec. 35 NW4
252	646968	WHISPER 608	U.S. GoldMining Inc.	S 21N 19W Sec. 33 SW4
253	646969	WHISPER 609	U.S. GoldMining Inc.	S 21N 19W Sec. 33 SE4
254	646970	WHISPER 610	U.S. GoldMining Inc.	S 21N 19W Sec. 34 SW4
255	646971	WHISPER 611	U.S. GoldMining Inc.	S 21N 19W Sec. 34 SE4
256	646972	WHISPER 612	U.S. GoldMining Inc.	S 21N 19W Sec. 35 SW4
257	650959	MUD 1	U.S. GoldMining Inc.	S 21N 19W Sec. 32 NE4
258	650960	MUD 2	U.S. GoldMining Inc.	S 21N 19W Sec. 32 NW4
259	650961	MUD 3	U.S. GoldMining Inc.	S 21N 19W Sec. 31 NE4
260	650962	MUD 4	U.S. GoldMining Inc.	S 21N 19W Sec. 31 NW4
261	650963	MUD 5	U.S. GoldMining Inc.	S 21N 20W Sec. 36 NE4
262	650964	MUD 6	U.S. GoldMining Inc.	S 21N 20W Sec. 36 NW4
263	650965	MUD 7	U.S. GoldMining Inc.	S 21N 20W Sec. 35 NE4
264	650966	MUD 8	U.S. GoldMining Inc.	S 21N 20W Sec. 35 NW4
265	650967	MUD 9*	U.S. GoldMining Inc.	S 21N 20W Sec. 34 NE4NE4
266	650968	MUD 10*	U.S. GoldMining Inc.	S 21N 20W Sec. 34 SE4NE4
267	650969	MUD 11*	U.S. GoldMining Inc.	S 21N 20W Sec. 34 NE4SE4
268	650970	MUD 12*	U.S. GoldMining Inc.	S 21N 20W Sec. 34 SE4SE4
269	650971	MUD 13	U.S. GoldMining Inc.	S 21N 20W Sec. 35 SW4
270	650972	MUD 14*	U.S. GoldMining Inc.	S 21N 20W Sec. 35 NW4SE4
271	650973	MUD 15*	U.S. GoldMining Inc.	S 21N 20W Sec. 35 NE4SE4
272	650974	MUD 16*	U.S. GoldMining Inc.	S 21N 20W Sec. 35 SW4SE4
273	650975	MUD 17	U.S. GoldMining Inc.	S 21N 20W Sec. 36 SW4
274	650976	MUD 18	U.S. GoldMining Inc.	S 21N 20W Sec. 36 SE4
275	650977	MUD 19	U.S. GoldMining Inc.	S 21N 19W Sec. 31 SW4
276	650978	MUD 20	U.S. GoldMining Inc.	S 21N 19W Sec. 31 SE4
277	650979	MUD 21	U.S. GoldMining Inc.	S 21N 19W Sec. 32 SW4
278	650980	MUD 22	U.S. GoldMining Inc.	S 21N 19W Sec. 32 SE4
279	650981	MUD 23	U.S. GoldMining Inc.	S 20N 19W Sec. 6 NW4
280	650982	MUD 24	U.S. GoldMining Inc.	S 20N 20W Sec. 1 NE4
281	650983	MUD 25	U.S. GoldMining Inc.	S 20N 20W Sec. 1 NW4

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282	650984	MUD 26	U.S. GoldMining Inc.	S 20N 20W Sec. 2 NE4
283	650985	MUD 27	U.S. GoldMining Inc.	S 20N 20W Sec. 2 NW4
284	650986	MUD 28*	U.S. GoldMining Inc.	S 20N 20W Sec. 3 NE4NE4
285	650987	MUD 29*	U.S. GoldMining Inc.	S 20N 20W Sec. 3 SE4NE4
286	650988	MUD 30*	U.S. GoldMining Inc.	S 20N 20W Sec. 3 NE4SE4
287	650989	MUD 31*	U.S. GoldMining Inc.	S 20N 20W Sec. 3 SE4SE4
288	650990	MUD 32	U.S. GoldMining Inc.	S 20N 20W Sec. 2 SW4
289	650991	MUD 33	U.S. GoldMining Inc.	S 20N 20W Sec. 2 SE4
290	650992	MUD 34	U.S. GoldMining Inc.	S 20N 20W Sec. 1 SW4
291	650993	MUD 35	U.S. GoldMining Inc.	S 20N 20W Sec. 1 SE4
292	650994	MUD 36	U.S. GoldMining Inc.	S 20N 19W Sec. 6 SW4
293	650995	MUD 37	U.S. GoldMining Inc.	S 20N 20W Sec. 11 NE4
294	650996	MUD 38	U.S. GoldMining Inc.	S 20N 20W Sec. 11 NW4
295	650997	MUD 39	U.S. GoldMining Inc.	S 20N 20W Sec. 10 NE4
296	650998	MUD 40*	U.S. GoldMining Inc.	S 20N 20W Sec. 3 SW4SE4
297	650999	MUD 41	U.S. GoldMining Inc.	S 20N 20W Sec. 10 SE4
298	651000	MUD 42	U.S. GoldMining Inc.	S 20N 20W Sec. 11 SW4
299	651001	MUD 43	U.S. GoldMining Inc.	S 20N 20W Sec. 11 SE4
300	656421	MUD 44	U.S. GoldMining Inc.	S 20N 20W Sec. 12 NW4
301	656422	MUD 45	U.S. GoldMining Inc.	S 20N 20W Sec. 12 NE4
302	656423	MUD 46	U.S. GoldMining Inc.	S 20N 20W Sec. 12 SW4
303	656424	MUD 47	U.S. GoldMining Inc.	S 20N 20W Sec. 12 SE4
304	667695	BT049	U.S. GoldMining Inc.	S 19N 19W Sec. 4 SE4
305	738137	WHI 001	U.S. GoldMining Inc.	S 22N 18W Sec. 5 SW4
306	738138	WHI 002	U.S. GoldMining Inc.	S 22N 18W Sec. 5 SE4
307	738139	WHI 003	U.S. GoldMining Inc.	S 22N 18W Sec. 4 SW4
308	738140	WHI 004	U.S. GoldMining Inc.	S 22N 18W Sec. 4 SE4
309	738141	WHI 005	U.S. GoldMining Inc.	S 22N 18W Sec. 3 SW4
310	738142	WHI 006	U.S. GoldMining Inc.	S 22N 18W Sec. 3 SE4
311	738143	WHI 007	U.S. GoldMining Inc.	S 22N 18W Sec. 2 SW4
312	738144	WHI 008	U.S. GoldMining Inc.	S 22N 18W Sec. 2 SE4
313	738145	WHI 009	U.S. GoldMining Inc.	S 22N 18W Sec. 10 NW4
314	738146	WHI 010	U.S. GoldMining Inc.	S 22N 18W Sec. 10 NE4
315	738147	WHI 011	U.S. GoldMining Inc.	S 22N 18W Sec. 11 NW4
316	738148	WHI 012	U.S. GoldMining Inc.	S 22N 18W Sec. 11 NE4
317	738149	WHI 013	U.S. GoldMining Inc.	S 22N 18W Sec. 12 NW4
318	738150	WHI 014	U.S. GoldMining Inc.	S 22N 18W Sec. 12 NE4
319	738151	WHI 015	U.S. GoldMining Inc.	S 22N 17W Sec. 7 NW4
320	738152	WHI 016	U.S. GoldMining Inc.	S 22N 18W Sec. 10 SE4
321	738153	WHI 017	U.S. GoldMining Inc.	S 22N 18W Sec. 11 SW4
322	738154	WHI 018	U.S. GoldMining Inc.	S 22N 18W Sec. 11 SE4

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323	738155	WHI 019	U.S. GoldMining Inc.	S 22N 18W Sec. 12 SW4
324	738156	WHI 020	U.S. GoldMining Inc.	S 22N 18W Sec. 12 SE4
325	738157	WHI 021	U.S. GoldMining Inc.	S 22N 17W Sec. 7 SW4
326	738158	WHI 022	U.S. GoldMining Inc.	S 22N 18W Sec. 15 NE4
327	738159	WHI 023	U.S. GoldMining Inc.	S 22N 18W Sec. 14 NW4
328	738160	WHI 024	U.S. GoldMining Inc.	S 22N 18W Sec. 14 NE4
329	738161	WHI 025	U.S. GoldMining Inc.	S 22N 18W Sec. 13 NW4
330	738162	WHI 026	U.S. GoldMining Inc.	S 22N 18W Sec. 13 NE4
331	738163	WHI 027	U.S. GoldMining Inc.	S 22N 17W Sec. 18 NW4
332	738164	WHI 028	U.S. GoldMining Inc.	S 22N 18W Sec. 14 SW4
333	738165	WHI 029	U.S. GoldMining Inc.	S 22N 18W Sec. 14 SE4
334	738166	WHI 030	U.S. GoldMining Inc.	S 22N 18W Sec. 13 SW4
335	738167	WHI 031	U.S. GoldMining Inc.	S 22N 18W Sec. 13 SE4
336	738168	WHI 032	U.S. GoldMining Inc.	S 22N 17W Sec. 18 SW4
337	738169	WHI 033	U.S. GoldMining Inc.	S 22N 18W Sec. 23 NW4
338	738170	WHI 034	U.S. GoldMining Inc.	S 22N 18W Sec. 23 NE4
339	738171	WHI 035	U.S. GoldMining Inc.	S 22N 18W Sec. 24 NW4
340	738172	WHI 036	U.S. GoldMining Inc.	S 22N 18W Sec. 24 NE4
341	738173	WHI 037	U.S. GoldMining Inc.	S 22N 17W Sec. 19 NW4
342	738174	WHI 038	U.S. GoldMining Inc.	S 22N 18W Sec. 23 SW4
343	738175	WHI 039	U.S. GoldMining Inc.	S 22N 18W Sec. 23 SE4
344	738176	WHI 040	U.S. GoldMining Inc.	S 22N 18W Sec. 24 SW4
345	738177	WHI 041	U.S. GoldMining Inc.	S 22N 18W Sec. 24 SE4
346	738178	WHI 042	U.S. GoldMining Inc.	S 22N 19W Sec. 36 NE4
347	738179	WHI 043	U.S. GoldMining Inc.	S 22N 19W Sec. 36 SE4
348	738180	WHI 044	U.S. GoldMining Inc.	S 21N 19W Sec. 1 NW4
349	738181	WHI 045	U.S. GoldMining Inc.	S 21N 19W Sec. 1 NE4
350	738182	WHI 046	U.S. GoldMining Inc.	S 21N 19W Sec. 2 SE4
351	738183	WHI 047	U.S. GoldMining Inc.	S 21N 19W Sec. 1 SW4
352	738184	WHI 048	U.S. GoldMining Inc.	S 21N 19W Sec. 1 SE4
353	738185	WHI 049	U.S. GoldMining Inc.	S 21N 19W Sec. 11 NE4
354	738186	WHI 050	U.S. GoldMining Inc.	S 21N 19W Sec. 12 NW4
355	738187	WHI 051	U.S. GoldMining Inc.	S 21N 19W Sec. 12 NE4
356	738188	WHI 052	U.S. GoldMining Inc.	S 21N 19W Sec. 11 SW4
357	738189	WHI 053	U.S. GoldMining Inc.	S 21N 19W Sec. 11 SE4
358	738190	WHI 054	U.S. GoldMining Inc.	S 21N 19W Sec. 12 SW4
359	738191	WHI 055	U.S. GoldMining Inc.	S 21N 19W Sec. 14 NW4
360	738192	WHI 056	U.S. GoldMining Inc.	S 21N 19W Sec. 14 NE4
361	738193	WHI 057	U.S. GoldMining Inc.	S 21N 19W Sec. 13 NW4
362	738194	WHI 058	U.S. GoldMining Inc.	S 21N 19W Sec. 15 SE4
363	738195	WHI 059	U.S. GoldMining Inc.	S 21N 19W Sec. 14 SW4

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364	738196	WHI 060	U.S. GoldMining Inc.	S 21N 19W Sec. 14 SE4
365	738197	WHI 061	U.S. GoldMining Inc.	S 21N 19W Sec. 22 NW4
366	738198	WHI 062	U.S. GoldMining Inc.	S 21N 19W Sec. 22 NE4
367	738199	WHI 063	U.S. GoldMining Inc.	S 21N 19W Sec. 23 NW4
368	738200	WHI 064	U.S. GoldMining Inc.	S 21N 19W Sec. 23 NE4
369	738201	WHI 065	U.S. GoldMining Inc.	S 21N 19W Sec. 22 SW4
370	738202	WHI 066	U.S. GoldMining Inc.	S 21N 19W Sec. 22 SE4
371	738203	WHI 067	U.S. GoldMining Inc.	S 21N 19W Sec. 23 SW4
372	738204	WHI 068	U.S. GoldMining Inc.	S 21N 19W Sec. 28 NE4
373	738205	WHI 069	U.S. GoldMining Inc.	S 21N 19W Sec. 27 NW4
374	738206	WHI 070	U.S. GoldMining Inc.	S 21N 19W Sec. 27 NE4
375	738207	WHI 071	U.S. GoldMining Inc.	S 21N 19W Sec. 28 SE4
376	738208	WHI 072	U.S. GoldMining Inc.	S 21N 19W Sec. 27 SW4
377	738209	WHI 073	U.S. GoldMining Inc.	S 21N 19W Sec. 27 SE4