M3-PN230328
Effective Date:
June 1, 2024
Issue Date:
November 21, 2024
Revision 0

ARCHITECTURE
ENGINEERING
CONSTRUCTION MANAGEMENT

Black Pine Project

NI 43-101 Technical Report

Cassia County, Idaho

Matthew Sletten, P.E.
Benjamin Bermudez, P.E.
Todd Carstensen, RM-SME
Richard DeLong, M.S., P.G., MMSA
Nicholas Rocco, Ph.D., P.E.
Gary L. Simmons, MMSA
Valerie Wilson, M.Sc. P.Geo.

Prepared For:

Libertygold

DATE AND SIGNATURES PAGE

The effective date of this report is June 1, 2024. The issue date of this report is November 21, 2024. See Appendix A, Pre-Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

BLACK PINE
FORM 43-101F1 TECHNICAL REPORT

TABLE OF CONTENTS

LIST OF FIGURES AND ILLUSTRATIONS

LIST OF TABLES

LIST OF APPENDICES

1 EXECUTIVE SUMMARY

This Technical Report ("Technical Report") has been prepared by M3 Engineering and Technology Corporation ("M3") with Liberty Gold Corp. ("Liberty Gold" or "LGD") in accordance with the National Instrument 43-101F1 Standards of Disclosures for Mineral Projects ("NI 43-101"). The Technical Report presents an updated Mineral Resource estimate and the Black Pine preliminary feasibility study ("PFS"), which incorporates new design-work, scheduling, and projected costs, in support of first-time disclosure of Mineral Reserve estimates in the Black Pine gold deposits. The Mineral Resource estimate presented in this Technical Report supersedes all prior resource estimates for the Project (as defined herein).

Liberty Gold is listed in the Toronto Stock Exchange (XTSE:LGD) and holds its interest in the Black Pine Project through its wholly owned subsidiary, Pilot Gold (USA) Inc., a Delaware, USA Corporation.

Liberty Gold holds a 100% ownership interest in the federal lode claims that host the Mineral Resource at the Black Pine property. The property originally acquired from Western Pacific Resources Corp. (Western Pacific) through an agreement dated June 15, 2016. Under this agreement Western Pacific received $800,000 in cash, a 0.5% net smelter royalty (NSR), and 300,000 common shares of Liberty Gold. Portions of the property staked subsequent to the purchase are not subject to the NSR.

1.1 Principal Findings

The key project parameters and findings are presented in Table 1-1, including a summary of the project size, productions, capital and operating costs, metal prices, and financial indicators.

Table 1-1: Key Project Parameters & Findings

Mine Life	17 Years
Mine Type	Open Pit
Process Description	ROM heap leach Gold recovery by ADR plant & Refinery, three carbon column trains
Total Mineral Reserve Estimate[1]	299.4 M tonnes
Average Grade	0.323 gpt Au
Contained Gold	3.110 M oz Au
Average Recovery	70.4% Au
Payable Gold	2.189 M oz Au
Average Annual Tonnes Mined	42 M tonnes
Strip Ratio	1.32:1
Process (ROM) Throughput (tonnes/day)	50,000
Capital Expenditures	
Initial Capital Expenditures	$326.6 M
Sustaining Capital Expenditures	$219.8 M
Unit Operating Costs	
Average Life of Mine ("LOM") Mining Costs	$6.50 / ore tonne
Average LOM Process Costs	$1.80 / ore tonne
Average LOM G & A Costs	$0.73 / ore tonne
Average LOM Refining Costs	$0.07 / ore tonne
Average LOM Total Operating Costs	$9.11 / ore tonne
Cash Costs	$1,250 / oz Au
All in Sustaining Costs ("AISC") – See Note	$1,381 / oz Au

[1]See Table 1-3 and the notes hereto.

Note: AISC includes: Sustaining Capital (Mining, Process, and Owner's Costs), plus Reclamation & Closure Costs and Idaho Mine License Tax

Financial Indicators	Spot Price (Au)	Base +300	Base Case	Base -150	Base -300
Gold Price (per troy oz)	$2,600	$2,300	$2,000	$1,850	$1,700
Pre-tax Cash Flow, $M	$2,349.6	$1,694.7	$1,039.8	$712.3	$384.8
Pre-tax NPV (5%) in $M	$1,572.6	$1,114.3	$655.9	$426.7	$197.5
Pre-tax Internal Rate of Return (IRR)	67.0%	52.2%	34.5%	25.5%	15.5%
Pre-tax Payback (Years)	1.5	1.8	3.1	3.7	4.3
After-tax Cash Flow, $M	$1,919.0	$1,394.2	$871.0	$605.4	$342.1
After-tax NPV (5%) in $M	$1,294.3	$922.2	$550.2	$360.6	$172.0
After-tax IRR	62.0%	47.2%	31.8%	23.6%	14.6%
After-tax Payback (Years)	1.5	1.9	3.3	3.8	4.3

1.2 Property Description

The Black Pine Project is located in Cassia and Oneida counties, Idaho, approximately 18 mi (29 km) northwest of the town of Snowville, Utah, the nearest substantial community, and 8.1 mi (13 km) north-northeast of Curlew Junction, the intersection of Utah State Highways 30 and 42. The approximate geographic center of the Black Pine property is 42.082°N latitude and 113.047°W longitude.

The climate in the Project area and the surrounding region is of the continental, intermontane type.

The Black Pine property straddles the eastern margin of the northerly-trending Black Pine Mountains. Elevations within the property range from a low of 5,413 ft (1,650 m) along the eastern edge, to a maximum of approximately 8,005.2 ft (2,440 m) in the western part of the property. The topography is moderately steep over much of the area.

1.3 Land Tenure

The Black Pine property consists of a contiguous block of 679 unpatented federal lode mining claims covering 12,792.6 acres (5,177 ha), one lease of State of Idaho mineral rights over a one square mile section (260 ha), a majority interest in 911 acres (386.7 ha) of private mineral rights, and 139.4 acres (56.4 ha) of private property, all located in the State of Idaho within either Cassia or Oneida counties. The combined properties occupy an area of 14,485 acres (5,862 ha).

Liberty Gold is the 100% owner of all unpatented federal lode claims that comprise the majority of the Black Pine property, having purchased 345 of the unpatented claims from Western Pacific through an agreement dated June 15, 2016.

Annual claim-maintenance fees are the only federal payments related to unpatented mining claims, and these fees have been paid in full through September 1, 2025. County recording fees are also required annually, as well as a lease payment to the State of Idaho. Liberty Gold's annual claim holding costs were US$123,359 in 2024.

A 0.5% NSR royalty to Wheaton Precious Metals (Cayman) Co.("WPM") Covers all claims, leases, private properties, and mineral rights. The royalty held by WPM is subject to a 50% buyback for $3.6M, the cost of which has been included in the cash flow analysis in this PFS. The Idaho State Minerals Lease is subject to a minimum annual royalty of $1,000 for Years 1 through 5 and $2,500 for Years 6 through 20. Additionally, production is subject to a 5% Net Smelter Return Royalty payable to the State of Idaho. Production of metallic minerals from the private mineral rights lands described above will be subject to a 0.25% NSR. Mineral production from the entire property is subject to the Idaho Mine License Tax, equivalent to 1.0% of "ores mined or extracted, and royalties received from mining".

1.4 Existing Infrastructure

Services are readily available in nearby towns, including Snowville and Tremonton, Utah, and Burley, Idaho. Skilled labor and experienced contractors can be sourced from Salt Lake City, Utah, and Elko, Nevada. Grid electrical power is available from a transformer on a major power line about 10 km southeast of the Project, with a 25 kV distribution line extending to the eastern property boundary. Liberty Gold received a positive initial system impact study from Idaho Power Distribution Company on the supply of up to 10 megawatts of electrical power along the distribution line, which is managed by Raft River Rural Electric Co-op Inc.

Water for exploration drilling needs is available from several wells on BLM land and private land immediately east of the property.

1.5 Exploration and Mining History

Numerous prospects and small mines in the Black Pine mountains exploited base- and precious-metal deposits commencing in the late 1800s and extending into the early 1900s, when minor amounts of zinc, silver, and mercury were produced. Gold was discovered in the late 1930s or early 1940s at the Tallman mercury mine, located within the current Black Pine Project, and a small open pit was operated at Tallman from 1949 to 1955 with total production reported to be 98,884 tonnes (109,000 tons) with an average gold grade of 5.14 g/t Au.

From 1963 through mid-1990, Newmont Mining, Kerr Addison Mines Ltd, Gold Resources Inc. (Gold Resources), Permian Exploration Account, ASARCO, Pioneer Nuclear Inc., Pegasus Gold Corp. (Pegasus), Inspiration Resource Corp., and Noranda Exploration, Inc. (Noranda) explored various portions of the Black Pine property. During this period, extensive soil-sample geochemical grids were completed, and a total of 218,934 ft (66,731 m) are known to have been drilled in 775 drill holes. Approximately 99% of the historical holes and meters drilled were completed using reverse-circulation (RC) and, for some uncertain but small number of holes, conventional-rotary methods. A total of eight holes were drilled using diamond-core (core) methods.

In 1986 through 1989, Noranda completed 536 of the holes mentioned above and discovered and delineated several zones of disseminated, sedimentary-rock-hosted gold mineralization. Noranda then produced a feasibility study in 1990 prior to selling the property to Pegasus in June 1990. Pegasus put the property into production in late 1991 as an open-pit run-of-mine ("ROM") heap-leach operation that closed in 1997. During this period, Pegasus also drilled 1,080 RC holes and 18 core holes, for an aggregate total of 385,830.1 ft (117,601 m).

Approximately 24.0 million tonnes (26.5 million tons) of waste rock and 28.1 million tonnes (31 million tons) of ore were mined by Pegasus between 1991 and 1997, with 434,800 ounces of gold produced at an average gold recovery of 65%. The heap-leach pad was rinsed and reclaimed after production ceased.

The property was idle from 1999 to 2009. Western Pacific acquired the property by staking in 2009, carried out geophysical surveys, and drilled 35 RC holes for a total of 23,678 ft (7,217 m) prior to vending the property to Liberty Gold in 2016.

Since acquiring the Project, Liberty Gold has undertaken extensive data compilation and analysis.

1.6 Geology and Mineralization

As presently understood, the Black Pine property geology is comprised of a lower structural plate that includes the Devonian Jefferson Formation and Mississippian Manning Canyon Shale, a middle plate characterized by Pennsylvanian carbonate rocks of the Oquirrh Group, and an upper plate predominantly consisting of Permian siltstones and sandstones of the Oquirrh Group. The lithologic contact between the lower plate and middle plate is sheared and brecciated, and middle plate units are complexly structurally interleaved. Middle plate strata are considerably more deformed than strata in the upper and lower plates.

The middle plate, which hosts the gold mineralization of interest, has a structural thickness ranging from approximately 656.2 to 1,640 ft (200 to 500 m). At least two major deformational events are evident, manifested by Mesozoic thrust faults and tight to open folds, overprinted by Cenozoic, low- to high-angle normal faults. Gold is distributed throughout the middle structural plate, with higher-grade mineralization occurring within favorable stratigraphic units, such as calcareous siltstones, as well as in and adjacent to breccia bodies and along variously orientated low- to high-angle brittle faults.

The Black Pine gold mineralization can be best classified as sedimentary rock-hosted, Carlin-style mineralization.

Three-dimensional modelling by Liberty Gold, utilizing surface mapping and drill data, envisions relatively flat faults separating the lower and middle plates, with a structurally thickened middle plate centered on the outcropping area of mineralization and diminishing in thickness to the north and south. The distribution of higher-grade gold mineralization is controlled to a large extent by favorable stratigraphy as well as a series of north- to northwest-striking listric normal faults that bound the east side of an overthickened zone of massive limestone and dolostone.

1.7 Drilling

Liberty Gold carried out drilling activities in 2017 and 2019 through 2023, totaling 788,914 ft (240,461 m) in 1,016 holes, including 33 core holes (8,287 m; 27,188 ft) and 972 RC holes (232,174 m; 761,726 ft). Liberty Gold drilling focused on confirming and expanding the known smaller, near-surface satellite areas of mineralization (primarily E, F, M, and Back Range), expansion and infill of the Discovery and Rangefront areas, and testing of reconnaissance targets, including Bobcat and South Rangefront.

1.8 Sampling, Sample Preparation, Analysis and Security

While documentation with respect to drilling and assaying methods and protocols is not available for all holes, all of the historical operators were reputable, well-known mining/exploration companies, and the independent laboratories used to analyze the drill samples of the historical operators prior to the historical open-pit mining operation at the Black Pine Project were also widely known and commonly used by the exploration and mining industry at the time. There is ample evidence that these companies and their chosen commercial laboratories followed accepted industry practices with respect to sample preparation, analytical procedures, and security.

The sample preparation, analysis, and security protocols of Liberty Gold, as well as their quality assurance and quality control (QA/QC) program, are consistent with current industry standards, and no material issues were identified through analysis of their QA/QC results.

1.9 Data Verification

The historical drill hole data has undergone extensive verification. This verification included checking of the database values using historical records, and statistical analyses.

The resource estimation was guided to a significant extent by Liberty Gold's lithological and structural (geological) models. SLR is of the opinion that Liberty Gold's geological model is of high quality and provided critical support to the estimation of the Project Mineral Resources.

The QP is of the opinion that the Black Pine data as a whole are acceptable for the purposes used in this report.

1.10 Metallurgical Testing and Mineral Processing

1.10.1 Historical Metallurgical Testing (1974-1998)

A significant number of historical reports that document metallurgical testing completed prior to the historical mining operations that began in 1991 were reviewed. Production records from the Pegasus operation indicate that the average gold recovery by ROM heap leaching from 1991 through 1998 was 64.1%. The highest annual average recovery reported was 80% in 1993, and the lowest was 54% in 1994.

1.10.2 Liberty Gold Testing (2018-2024)

Liberty Gold initiated metallurgical testing in 2019. Five metallurgical test programs were subsequently completed and a Phase 5a (columns) and 5b programs are planned in 2024-25. These programs include:

1. 2019 Bulk Sample Program – six backhoe bulk samples were excavated from five historic Pegasus Gold open pits and one surface resource area. The samples underwent geo-metallurgical characterization and bottle-roll and column-leach testing.
2. 2019 Large-diameter PQ Core Program (Phase 1) – Liberty Gold drilled six large-diameter PQ core holes in the Discovery and Rangefront zones in late 2019. A total of 29 metallurgical composites were selected for geo-metallurgical characterization, bottle-roll and column-leach testing.
3. 2020 Large-diameter PQ Core Program (Phase 2) – In late 2020, Liberty Gold drilled nine additional PQ core holes in the Discovery and CD zones. A total of 45 metallurgical composites were selected for a testing program similar to that described in item 2 above.
4. 2021 Low Grade Composite Program (Phase 3) - In late 2021, Liberty Gold directed KCA to perform bottle-roll and column-leach testing on 15 composites, composed from Phase 1 and Phase 2 low-grade of PQ core with average head assays between 0.1 and 0.2 g/t Au.
5. 2022 Rangefront Zone Column Test Program (Phase 4a) – In late 2021, Liberty Gold drilled four PQ core holes in the Rangefront Zone. A total of 24 metallurgical composites were selected for a testing program like those described above.
6. 2022 Discovery, E Zone, CD, F and M Zone (Phase 4b, 4c) - In early 2023 Liberty Gold identified testing of 36 metallurgical composites for testing similar to those described above.
7. 2023 Large-diameter PQ Core Program (Phase 5a) - In 2023 Liberty Gold drilled 11 PQ core holes in the Backrange and J zone satellite areas and infilled metallurgical gaps in M Zone, Discovery Zone and F Zone. 25 out of the 47 composites were prioritized to be complete in time for this report. An additional 22 composites have been collected but have not been submitted to KCA labs for testing at the time of writing.

Metallurgical characterization from the above programs identified the following lithologic units as potential unique metallurgical recovery domains, where the numbers in parentheses indicate the number of samples that have been tested.

- PPos – sandstone, quartzite, and siltstones (11)
- Pola – limestone and sandy limestone (44)
- Polb – siltstone, sandy limestone, and dolomite (30)
- Polc – siltstone, limestone, sandstone, and dolomite (37)
- Pold – limestone, dolomite, sandstone, and quartzite (27)
- Pols – limestone, sandstone, and quartzite (23)
- PMmx – limestone, siltstone, quartzite (3)
- PMmc – shale, limestone, and quartzite (0)– lower plate, underlies most gold mineralization. It was not tested as it is generally carbonaceous and not mineralized.

Recovery equations have been assigned to each of the lithologic units, with separate equations for low and high-grade domains (Met 1, 2 or 3).

The QP is of the opinion that samples tested are sufficiently representative to support the conclusions summarized herein. Metallurgical testing is ongoing and is designed in part to continue to evaluate all types and styles of mineralization.

1.11 Mineral Resources

The Mineral Resource is based on 1,755 historical RC holes and 18 historic diamond core holes, as well as 966 RC and 31 core holes drilled by Liberty Gold. The historical holes at the Black Pine Project were primarily drilled from the mid-1980s to the late 1990s by Noranda and Pegasus Mining.

Mineral Resources at the Black Pine Project were modelled and estimated by:

- Developing a geological model in Leapfrog Geo reflecting low-angle fault control and stratigraphic control of mineralization hosted in receptive carbonate host rocks.
- Evaluating the drill data statistically.
- Interpreting medium (0.1 g/t Au) and high-grade (0.3 g/t Au for Rangefront and 0.5 g.t Au for all other areas) gold-domains using Leapfrog Geo. Within these domains, low grade material (below 0.1 g/t Au) was captured using indicator shells.
- Compositing data to 10 feet (3.048 m) within the gold domains.
- Coding a block model comprised of 10 x 10 x 5 (x, y, z) meter blocks and sub-blocked to 2.5 x 2.5 x 1.25-meter blocks to the domains.
- Analyzing the modelled mineralization geostatistically to aid in the establishment of estimation and classification parameters.
- Interpolating gold grades using inverse distance cubed (ID^3) and a three-pass interpolation strategy into the model blocks using the mineral domain coding to explicitly constrain the gold grade estimations.
- Evaluating, statistically and visually, the resulting model in detail prior to finalizing the mineral resource estimation.

The Black Pine Deposit Mineral Resource has been constrained by optimized pit shells created using a gold price of US$2,000/ounce and pit slopes ranging from 45 to 47 degrees in eight sectors defined by geotechnical studies. Additional inputs for the pit-optimizations include: Mining - $2.35/tonne mined; heap leaching - $1.83/tonne processed; and G&A cost of $0.80/tonne processed at an assumed 18.25 million tonnes (Mt) per year processing rate. Gold recoveries are based on equations derived from metallurgical data and vary by grade and rock unit. A 0.5% net smelter return royalty was also applied.

The in-pit resources were further constrained by the application of a cut-off grade of 0.10 g Au/t to all model blocks lying within the optimized pit shells. The portions of blocks coded as containing carbonaceous material were assigned a recovery of 0% and thus excluded from the resource estimate.

The Black Pine Mineral Resources were classified as Indicated or Inferred based on drill hole spacing, confidence in the geological interpretation, supporting data (historical versus Liberty Gold data) and interpreted continuity of mineralization. There are no Measured Mineral Resources at the Black Pine Project. Indicated Mineral Resources were defined where drill hole spacing of 164 ft to 197 ft (50 m to 60 m) was achieved. The drill holes spacing for Indicated classification is supported by experimental variogram ranges. All remaining blocks contained within the wireframe model and estimated within the block model were limited to an Inferred classification. The backfill and waste dumps, as well as low grade domains that contain mineralization above the cut-off grade were classified as Inferred Mineral Resources.

The total estimated Mineral Resources at the Project are presented in Table 1-2. The Mineral Resources comprise 402.6 million tonnes (Mt) at an average grade of 0.32 g/t Au containing 4.16 million ounces (Moz) in the Indicated category and 97.7 Mt at an average grade of 0.23 g/t Au containing 0.71 Moz Au in the Inferred Mineral Resources category. There are no Measured Mineral Resources.

Table 1-2: Summary of Mineral Resources for the Black Pine Project - June 1, 2024

Classification	Tonnage Mt	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	402.6	0.32	4,163
Inferred	97.7	0.23	712

Notes:

1. Mineral Resources were prepared by SLR Consulting (Canada) Inc. (SLR) and has an effective date of June 1, 2024. The Qualified Person responsible as defined under NI 43-101 for the Mineral Resource estimate is SLR Principal Resource Geologist, Valerie Wilson, P.Geo, who is independent of Liberty Gold
2. CIM Standards (as defined herein) definitions were followed for Mineral Resources.
3. Bulk density is variable by rock type.
4. Mineral Resources are reported within conceptual open pits estimated at a gold cut-off grade of 0.10 g/t, using the PFS pit slope parameters, a long-term gold price of US$2,000 per ounce and the PFS variable gold leach recovery model derived from extensive metallurgical studies.
5. All gold mineralized material falling outside the conceptual open pits and carbonaceous material is considered waste rock and is excluded from resource classification.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Mineral Resources are inclusive of Mineral Reserves.
8. Rounding may result in apparent discrepancies between tonnes, grades, and contained gold content.

The QPs are not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors not discussed in this report that could materially affect the potential development of the Black Pine Project Mineral Resources as of the Effective Date (as defined herein) of this Technical Report.

1.12 Mineral Reserve Estimate

The Black Pine Gold Project is planned to be an open pit operation using conventional mining equipment. The Mineral Reserve estimate is reported on the mine designs and mine plans generated by AGP Mining Consultants, Inc. (AGP).

The Mineral Reserves consist of Indicated blocks above a cut-off of 0.10 Au g/t and contained within the ultimate pit designs. No Measured Mineral Resources are contained in the Black Pine deposits, so there are no Proven Reserves. Pits were designed in accordance with geotechnical recommendations, and based on economic calculations using metal prices, costs, and recoveries. The Black Pine open pits include the larger Discovery, Tallman, and Rangefront pits and the smaller CD, E, F, J, M, and Backrange pits. Mineral Reserves are estimated from a single resource model containing the nine pit areas.

The Mineral Reserves for the Black Pine are listed in Table 1-3 with the gold grade (Au) estimates based on the mine diluted grades in the block model.

Table 1-3: Proven and Probable Reserves – Black Pine Gold Project

Reserve Class	Tonnes (Mt)	Au (g/t)	Contained Ounces (Mozs)
Proven	0.0	-	-
Probable	299.4	0.323	3.11
Total Capacity	299.4	0.323	3.11

Notes:
1. The Mineral Reserve estimate was prepared by AGP Mining Consultants Inc., Barrie ON, Canada ("AGP") and has an effective date of June 1, 2024. The Qualified Person responsible as defined under NI 43-101 for the Mineral Reserve estimate is Todd Carstensen RM-SME, Principal Mine Engineer and independent of Liberty Gold
2. Mineral Reserves reported are consistent with the CIM Standards.
3. Mineral Reserves are reported to have a cut-off grade of 0.10 Au g/t.
4. The cut-off grade is based on a gold price of US$1,650 Au oz.
5. Metallurgical recovery of gold is based on a variable gold leach recovery model derived from extensive metallurgical studies.
6. All carbonaceous materials have been treated as waste. Overall leach recovery averages 70.4%.
7. Mine dilution was estimated based on a 1.0 m skin applied to ore to waste contacts.
8. Units are metric tonnes, metric grams & troy ounces; "Au" = gold.
9. The estimate of Mineral Reserves may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves. Areas of uncertainty that may materially impact Mineral Reserve estimation include:

- Commodity price and exchange rate assumptions;
- Capital and operating cost estimates;
- Geotechnical slope designs for pit walls;
- Mining selectivity near the ore contacts; and
- Gold leach recovery.

1.13 Mining Methods

The Black Pine Project consists of three large deposits and several smaller deposits that are planned to be developed using conventional open pit mining methods. The leach process facility will be located at lower elevations on the east side of the property where the topography begins to flatten out. Ore from the pits will be hauled to the leach facility and end dumped on to the pad. Underground mining was not considered for this study.

Open pit designs are configured on 10 m bench heights, with catch berms placed every two benches, or double benching. Bench Face Angles (BFAs), inter-ramp angles (IRAs) and bench widths were provided by Knight Piésold Ltd. (KP). Pit restrictions include a 50-meter offset from the historic heap leach facility and the Tallman historic tailings.

Two mining fleets are planned at Black Pine. The larger mining fleet includes 22 m^3 diesel hydraulic shovels and 144 tonne rigid body trucks. The smaller fleet has 6.7 m^3 diesel hydraulic excavators and 64 tonne rigid body trucks. Additional loading support will be provided by 11.5 m^3 front end wheel loaders. The small pits, pioneering cuts, and mining areas with narrow bench widths and limited mining space will be mined with the smaller mining equipment. The 64 tonne trucks with decreased ramp width will also reduce the construction and maintenance cost on the surface roads that transverse the steep terrain leading to the pits located at the top of the Black Pine mountains. Blasthole drilling will be completed with diesel down the hole hammer (DTH) drills with 165 mm bits.

There are two main waste rock facilities (WRF). The Discovery WRF is a valley fill located north of the Discovery pit and the Rangefront WRF is located southwest of the Rangefront pit near the CD pit. A small WRF is located near

Backrange and CD and M pits are backfilled with waste rock. Based on geochemical characterization to date, no specific waste rock management procedures are envisioned.

There are four stockpile areas, two located between the Rangefront and C/D pits and two northwest of the M Zone pit. Different stockpiles are established for the Met 1 and the Met 2-3 material.

Mine dilution is applied in the study based on a 1.0 m skin applied to ore to waste contacts Grade control will be done using the blast hole sampling method.

All pits are considered dry and will not require dewatering.

1.14 Recovery Methods

The process selected for recovery of gold from the Black Pine deposit is a conventional ROM heap leach. Ore will be mined by standard open pit mining methods from multiple pits. The ore will be truck-stacked on the heap as ROM ore directly, without crushing. Lime will be added directly to the haul trucks for pH control.

The stacking rate will be in accordance with the mine plan. The ROM ore placement is equivalent to a Life of Mine (LOM) average of 50,000 tonnes per day.

Gold in the stacked ore will be leached with a dilute cyanide solution using a drip irrigation system at application rates in the range of 8,000-10,000 gallons per minute. The leached gold will be recovered from solution using a carbon adsorption circuit. The gold will be stripped from carbon using a desorption process, followed by electrowinning to produce a precipitate sludge. The precipitate sludge will be processed using a retort oven for drying and mercury recovery, and then refined in a melting furnace to produce gold doré bars.

1.15 Infrastructure – Heap Leach and Water Management

The heap leach facility (HLF) was sited at the eastern extent of the Project, designed in four phases to contain the estimated 299 million dry metric tonnes of leachable material: a Starter (Years 1 and 2), Phase 2 (Years 3 through 7), Phase 3 (Years 8 through 12), and Phase 4 (Year 13 through 18). The HLF phasing was developed to not only accommodate the mine plan schedule, but also to improve recoveries by generally separating distinct metallurgical material types.

The HLF solution containment system includes a composite liner composed of HDPE geomembrane and a compacted soil liner, and an overlying solution collection system. Captured solution will be conveyed to process solution tanks.

The heap was designed at 3H:1V side slopes to accommodate the closure footprint; actual operational slopes will include benching and a steeper inter-bench face slopes. The maximum heap height will be 330 ft (100 m), as measured from the pad grading to the top of the heap. Run-of-mine leachable material will be truck-stacked in nominal 30-foot (9 m) lifts.

Contact water will be routed to the Event Pond located at the topographic low at the eastern extent of the HLF. The Event Pond will be double-lined with a HDPE geomembrane liner, and will include a leak detection and pumpback system. The Event Pond will also serve as the overflow containment for the adjacent ADR plant (connected with a lined channel) and the pregnant solution tank situated on a tank shelf within the pond.

Surface water diversions will route non-contact water away from the HLF, conveying clean runoff water from native areas downstream to the planned discharge points. Diversions are also planned around the open pits, development rock storage areas, and stockpiles. Runoff from the development rock storage areas will report to sediment control ponds prior to discharging downstream to prevent sediment-laden water from being discharged off-site.

1.16 HLF Design and Operational Safety

The HLF and associated structures have been designed to meet regulatory requirements and industry-accepted standards and practices, suitable for a PFS-level design. Additional investigations, evaluations, and analyses will be required at subsequent design phases to confirm assumptions and reduce the risk of encountering unforeseen conditions during construction or operation.

During construction, a rigorous Construction Quality Assurance ("CQA") program will be implemented to ensure the construction materials meet or exceed specified values that are key to HLF performance. Materials not meeting the specifications will either not be used in construction or approved after confirming the deviations will not negatively impact facility performance.

A robust Operations, Maintenance, and Safety ("OMS") manual will be a key component to ensure operations and monitoring controls are in place for the lifecycle of the structure. The OMS manual will include instrumentation and monitoring to provide early warning systems which will allow Liberty Gold to monitor conditions at the HLF and provide recommendations if values trend toward thresholds for potentially unsuitable levels. The Capital Cost Estimate ("CAPEX") includes preliminary instrumentation and monitoring systems.

In addition, final designs and significant design criteria or concept changes will be reviewed by a qualified third party.

1.17 Environmental Studies, Permitting and Social or Community Impact

The Black Pine Mine Project is located on National Forest System ("NFS") lands administered by the United States Forest Service (USFS), public lands administered by the Bureau of Land Management ("BLM"), State of Idaho mineral title, and private lands controlled by Pilot Gold (USA) Inc. The mineral rights on the NFS lands, public lands, and state mineral title are controlled by Liberty Gold.

Liberty Gold has contracted qualified third parties to perform environmental baseline data collection and review the adequacy of existing environmental baseline reports and data. This baseline data collection is ongoing through 2024. Additionally, EAs were completed, and Plans of Operations approved in 1988, 1991, and 1993 for the Black Pine Mine by the previous operator for the site.

Since the Project occurs on both NFS lands and public lands, use of these lands is subject to multiple USFS and BLM regulatory programs. The USFS will require a Plan of Operations under 36 Code of Federal Regulations ("CFR") 228, and the BLM will require a Plan of Operations under 43 CFR 3809. In addition, the BLM will require a Mining Plan under 43 CFR 3500 for those activities on the acquired lands at that NFS lands. For activities on the BLM-managed acquired lands and those lands with private or State of Idaho mineral estate, the BLM will require a Plan of Development under 43 CFR 2800. It is anticipated that all these federal permitting requirements can be addressed in a single Mine Plan of Operations ("MPO") permit application to the USFS and the BLM.

The Idaho Department Land ("IDL") is responsible for implementation of the Idaho Joint Review Process, which was established to coordinate and facilitate the overall mine permitting process in the state. There are a number of permits, licenses and authorizations required from the State of Idaho to operate a mine which the Project will require including permits addressing air, water, fuel storage, cyanide use, waste management and sewer systems.

According to its environmental experts, Stantec Consulting Services Inc. (Brown, 2016), Liberty Gold is liable only for disturbance incurred as part of Liberty Gold's exploration activities, or if Liberty Gold causes disturbance to the historical leach pad or other designated areas.

1.18 Capital and Operating Costs

Capital and operating costs were estimated for the pre-feasibility study by AGP (mine development) and M3 (process plant, site development, power transmission and distribution, and ancillaries), and NewFields (heap leach facility). Table 1-4 shows the estimated capital costs for the Project. This includes $326.6 million in Year -1 and $219.8 million for sustaining capital. Total capital costs are estimated at $546.3 million.

Table 1-4: Capital Cost Summary

Category	Units	Initial	Sustaining	Total
Site General (Earthworks)	K USD	$11,785	-	$11,785
Process Plant (ADR, Refinery, Reagents)	K USD	$47,741	$9,474	$57,215
Power Systems	K USD	$4,253	-	$4,253
ADR Bldg. & Ancil. (Warehouse, Maint, Admin, Fuel)	K USD	$22,924	-	$22,924
Freight (Process Plant)	K USD	$3,408	-	$3,408
Sub-Total Direct Cost (Process Plant & Support)	K USD	$90,111	$9,474	$99,585
Construction Support (inc. Mobilization)	K USD	$2,703	-	$2,703
Engineering, Procurement, & Const. Mgmt.	K USD	$15,500	-	$15,500
Vendor Support	K USD	$1,618	-	$1,618
Spare Parts (Capital, Commissioning)	K USD	$1,578	-	$1,578
First Fills (Process Plant)	K USD	$480	-	$480
Contingency (Process Plant)	K USD	$22,398	-	$22,398
Owner's Cost	K USD	$9,200	$10,625	$19,825
Sub-Total Indirect Cost (Process Plant & Support)	K USD	$53,477	$10,625	$64,102
Heap Leach Facility Direct Cost (NewFields)	K USD	$47,515	$107,791	$155,306
Heap Leach Facility Indirect Cost (NewFields)	K USD	$14,755	$35,467	$50,222
Sub-Total Heap Leach Facility	K USD	$62,269	$143,258	$205,528
Mine Capital Equipment (AGP)	K USD	$31,411	$56,398	$87,809
Mine Preproduction Costs (AGP)	K USD	$89,291	-	$89,291
Sub-Total Mine Capital	K USD	$120,702	$56,398	$177,100
TOTAL CAPITAL COST	K USD	$326,560	$219,755	$546,315

Table 1-5 shows the estimated operating costs for the LOM project. Operating costs were estimated at $2.726 billion for the LOM. This is $9.11 per tonne processed or $1,245 per ounce of gold produced.

Table 1-5: Operating Cost Summary

		Operating Cost Ratio	
Category	K USD	$ / tonne	$ / Au oz
Mining Costs	$1,945,536	$6.50	$889
Process Plant	$538,322	$1.80	$246
G&A	$219,950	$0.73	$100
Refining	$21,908	$0.07	$10
TOTAL OPERATING COST	$2,725,716	$ 9.11	$1,245

1.19 Economic Analysis

The economic analysis in this study includes a pre-feasibility study-compliant modeling of the annual cash flows based on projected production volume, sales revenue, initial capital, operating cost, and sustaining capital with resulting evaluation of key economic indicators such as internal rate of return (IRR), net present value (NPV), and payback period (time in years to recapture the initial capital investment) for the Project. The sales revenue is based on the production of gold in doré bullion. The estimates of the capital expenditures and site production costs have been developed specifically for this Project and have been presented in Section 21 of this Technical Report.

Ore will be processed by cyanide heap leaching as ROM and recovered via an ADR facility as described in Section 17 of this Technical Report. Overall production over the life-of-mine is summarized in Table 1-6.

Table 1-6: Life of Mine Process Statistics

Total Ore (kt)	299,363
Gold Grade (gpt)	0.323
Contained Gold (kozs)	3,110
Gold Recovery %	70.4%
Recovered Gold (kozs)	2,191

Annual revenue is determined by applying metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life-of-mine production without escalation or hedging. Gold bullion revenue is based on the gross value of the payable metals sold before refining and transportation charges. Gold metal pricing is based on the long-term consensus price estimates of several metals price analysts; the base case gold price utilized in the economic assessment is $2,000 per troy ounce.

A detailed financial model developed in compliance with PFS requirements. This model has captured all the parameters of the mine production volume, annual sales revenue, and all the associated costs. This model was used to calculate the economics of the Project, as well as for the sensitivity analysis. A 5-year straight line depreciation calculation was applied for initial and sustaining capital and a 10-year straight line depreciation was used for development costs which consists primarily of the heap leach expansion costs. The financial model reflects Liberty Gold's intention to buy down certain existing royalties prior to production. Idaho Mine license tax of 1%, Idaho Corporate Income Tax of 5.695% and Federal corporate tax of 21% are calculated in the model.

The economic analyses for the Project are summarized in Table 1-7 below. The NPV calculations have been conducted per the Year-End discounting method.

Table 1-7: Key Economic Indicators

Indicators	Before-Tax	After-Tax
LOM Cash Flow ($000)	$1,039,763	$871,018
NPV @ 5% ($000)	$655,883	$550,207
NPV @ 10% ($000)	$423,897	$352,440
IRR	34.5%	31.8%
Payback (years)	3.1	3.3

1.20 Conclusions

The authors of this Technical Report believe that Black Pine is a project of merit and warrants advancing the study to detailed engineering and ultimately project construction.

The authors have reviewed the Project data, including the drill-hole database and available metallurgical information, and have visited the Project site. The authors believe that the data provided by Liberty Gold, as well as the geological interpretations that have been derived from the data, are generally an accurate and reasonable representation of the

Black Pine property. Based on the positive results of this PFS, the Project should continue on a path to a production decision.

Results of historical metallurgical tests and those commissioned by Liberty Gold indicate there are multiple metallurgical material types within the various gold deposits. Due to the multiple material types and the dependence of gold recoveries on head grades, numerous different gold ROM recovery equations are used to project the processing and gold production estimates presented in this Technical Report.

The process selected for recovery of gold and silver from the Black Pine mineralized material is a conventional heap-leach recovery circuit. The material will be mined by standard open-pit mining methods and trucked from each deposit to a centralized area of heap-leach pads and processing facilities.

This technical report indicates an average gold production over the estimated 17-year LOM of about 129,000 ounces per year, with peak production in Year 5 of 231,000 ounces of gold. Cash costs are estimated to be $1,250 per ounce of gold, and AISC are estimated to be $1,381 per ounce of gold. The resulting after-tax cash flow is $871.0 million, for an after-tax NPV (5%) of $550.2 million and an estimated payback period of 3.3 years.

This project continues to exhibit excellent potential for significant returns and based on the information noted above is well suited to advance into the next phases of permitting and detailed engineering ultimately progressing towards production.

1.21 Recommendations

The authors of this technical report believe that Black Pine is a project of merit and warrants significant additional investment with the following recommendations.

- Additional drill programs to expand and upgrade the mineral resource estimate
- Optimization and alternatives analysis for the proposed mining methods
- Additional metallurgical test work to advance the geo-metallurgical model
- More advanced geotechnical investigation for the HLF area
- Refined modelling of ore type management within the HLF, and
- Further hydrological and hydrogeological investigations

The estimated total cost of the recommended work program sufficient to advance the Project to a construction decision, over an approximately three-year period, is summarized in the Table 1-8 below.

Table 1-8: Estimated Total Costs

Activity	Cost
Exploration	$9,900,000
Testwork	
Metallurgy	$800,000
Hydrogeology & Water	$850,000
Permitting	$550,000
Geotechnical	$1,000,000
Engineering and Feasibility Study	$3,500,000
Total	$16,600,000

2 INTRODUCTION

M3 has prepared this technical report on the Black Pine Project ("Black Pine", the "Black Pine Mine", or the "Project"), located in Cassia and Oneida counties, Idaho, for Liberty Gold. The purpose of this report is to disclose the results of an updated Mineral Resource estimate. This report, with an effective date of June 1, 2024 (the "Effective Date"), conforms to National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects. The Mineral Resource estimate presented in this Technical Report supersedes all prior resource estimates for the Project.

Liberty Gold is listed on the Toronto Stock Exchange (XTSE:LGD) and holds its interest in the Black Pine Project through its wholly owned subsidiary, Pilot Gold (USA) Inc., a Delaware, USA Corporation.

Liberty Gold is the 100% owner of federal lode claims hosting the resource at the Black Pine property, having originally purchased the property from Western Pacific Resources Corp. (Western Pacific) through an agreement dated June 15, 2016.

The Black Pine Project was the site of open-pit mining and cyanide heap-leach processing from 1991 to 1998.

2.1 Sources of Information

Ms. Valerie Wilson is a Qualified Person (QP) under NI 43-101 and is responsible for Sections 6 to 12 and 14, as well as subsections 1.3-1.9, 1.11, 24, 25.1, and 26.1. Section 13 was prepared by Consulting Metallurgist Gary L. Simmons, MMSA, a QP with respect to metallurgy.

Ms. Wilson visited the Black Pine property along with Mr. Pete Shabestari and Liberty Gold staff on April 11, 2024. This site visit included geologic overviews of the Project, detailed inspections of exposures in most of the historical open pits, reviews of Liberty Gold reverse-circulation rotary ("RC") chips and drill core, and the verification of drill hole locations. The visit also included detailed discussions of the evolving geologic understanding of the project and associated mineralization.

Sections 6 to 12, and 14, and relevant subsections of this Technical Report have been prepared by or under the supervision of Valerie Wilson. Ms. Wilson is a QP under NI 43-101 and has no affiliation with Liberty Gold except that of independent consultant/client relationship. Ms. Wilson was assisted by Sarah Conolly, P. Geo, SLR Senior Geologist, Lorraine Tam P. Geo, SLR Consultant Resource Geologist, Aline Romagna, GIT, SLR Geologist in Training, April Barrios, Senior Resource Geologist for Liberty Gold, and Will Lepore, Principal Geologist for Liberty Gold in the preparation of wireframe models and resource estimations and in the data verification and validation procedures and analysis.

Discussions were held with the following Liberty Gold personnel:

- Moira Smith, Ph.D., P.Geo., Corporate Technical Advisor for Liberty Gold
- Jon Gilligan, B.Sc., Ph.D., Chief Operating Officer for Liberty Gold
- Peter Shabestari, B.Sc., C.P.G., Vice President of Exploration for Liberty Gold
- April Barrios, B.Sc., P.Geo., Senior Resource Geologist for Liberty Gold
- Will Lepore, M.Sc., P.Geo., Principal Geologist for Liberty Gold

Section 1.10, 13, 25.2, and 26.3 were prepared by Consulting Metallurgist Gary L. Simmons, MMSA, a QP with respect to metallurgy. Mr. Simmons has no affiliation with Liberty Gold except that of independent consultant/client relationship.

As of the Effective Date of this report, Mr. Simmons visited the Black Pine Project site on June 3, 2019, October 18 and 19, 2019, June 22 and 23, 2020, May 3, 2021, and August 1-2, 2023. In respective chronology, during these site visits Mr. Simmons toured the Project and reviewed relevant geology, the historical pits, and historical Noranda metallurgical sample locations; collected bulk samples for metallurgical testing; revisited the historical open pits, visited some drill sites, and discussed metallurgical aspects of the Project with the on-site geologists; reviewed the core logging and sampling and truth checked cyanide solubility models and, in the company of other Liberty Gold staff and consultants, determined potential heap leach and crusher locations for the purposes of the ongoing economic assessment.

Sections 1.1, 1.2, 1.20, 1.21, 2, 3, 4, 5, 18.1-18.4, 18.11-18.16, 19, 23, 25.5, 25.6, 26.11, and 27 of this Technical Report have been prepared by or under the supervision of Matthew Sletten. Mr. Matthew Sletten is a QP under NI 43-101 and has no affiliation with Liberty Gold except that of independent consultant/client relationship. Mr. Sletten conducted a site visit to the Black Pine property on October 31, 2023, and May 8, 2024.

Sections 1.12, 1.13, 15, 16, 21.4, 21.7, 25.3, 25.4, and 26.2 of this Technical Report have been prepared by or under the supervision of Todd Carstensen. Mr. Carstensen is a QP under NI 43-101 and has no affiliation with Liberty Gold except that of independent consultant/client relationship.

Mr. Carstensen conducted a site visit to the Black Pine property on October 31, 2023. While on site, Mr. Carstensen viewed the general topography, property boundaries, existing pits, historic waste and backfill storage areas, existing road network, and spent time reviewing drill core in the core shed. Additionally, the proposed infrastructure locations including the waste storage areas, mine facilities, access roads, and the plant and heap leach pad locations were reviewed.

Sections 1.14, 1.18, 1.19, 17, 21.1, 21.2, 21.5, 21.6, 22, and 26.10 of this Technical Report were prepared by or under the supervision of Benjamin Bermudez. Mr. Bermudez is a QP under NI 43-101 and has no affiliation with Liberty Gold except that of independent consultant/client relationship. Mr. Bermudez has not visited the Black Pine Project property.

Sections 1.15, 1.16, 18.5-18.10, 21.3, and 26.4-26.9 of this Technical Report were prepared by or under the supervision of Nicholas Rocco. Mr. Nicholas Rocco is a QP under NI 43-101 and has no affiliation with Liberty Gold, except that of independent consultant/client relationship. Mr. Rocco visited the Black Pine Project Property on April 17, 2024.

Meetings were held on site with the various team members including Black Pine personnel responsible for geology, mining, environmental activities and other team members for processing and infrastructure.

Table 2-1 presents a summary of the QP report preparation and responsibilities in this Technical Report.

Table 2-1: Summary of QP Report Preparation and Responsibilities
(Liberty Gold Corp. – Black Pine Project)

Qualified Person	Title & Position	Site Visit Date	Sections of Responsibility
Matthew Sletten, P.E.	Professional Engineer and General Manager, M3 Engineering and Technology Corporation	October 31, 2023 and May 8, 2024	1.1, 1.2, 1.20, 1.21, 2, 3, 4, 5, 18.1-18.4, 18.11-18.16, 19, 23, 25.5, 25.6, 26.11, and 27
Benjamin Bermudez, P.E.	Chemical/Process Engineer, M3 Engineering and Technology Corporation	No visit to site	1.14, 1.18, 1.19, 17, 21.1, 21.2, 21.5, 21.6, 22, and 26.10
Todd Carstensen, RM-SME	Principal Mine Engineer, AGP Mining Consultants Inc.	October 31, 2023	1.12, 1.13, 15, 16, 21.4, 21.7, 25.3, 25.4, and 26.2
Richard DeLong	Senior Technical Advisor, Westland Engineering and Environmental Sciences	No visit to site	1.17, 20, and 26.10
Nicholas Rocco	Principal Engineer, NewFields MDTS	April 17, 2024	1.15, 1.16, 18.5-18.10, 21.3, and 26.4-26.9
Gary L. Simmons, MMSA	Consultant, GL Simmons Consulting LLC	June 3, 2019, October 18-19, 2019, June 22-23, 2020, May 3, 2023, and August 1-2, 2023	1.10, 13, 25.2, and 26.3
Valerie Wilson M.Sc. P.Geo.	Principal Resource Geologist, SLR Consulting (Canada) Ltd.	No visit to site	1.3-1.9, 1.11, 6, 7, 8, 9, 10, 11, 12, 14, 24, 25.1, and 26.1

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 27 References. Of particular note, this report incorporates information and descriptions drawn from a report for Liberty Gold by Gustin et al. (2021) entitled "Updated Technical Report and Resource Estimate for the Black Pine Gold Project, Cassia County, Idaho, USA" with an effective date of June 20, 2021.

The Effective Date of this technical report is June 1, 2024.

2.2 Units and List of Abbreviations

In this report, measurements are generally reported in metric units unless original Imperial units are deemed to be best reported as-is. Where information was originally reported in Imperial units and converted to metric for the purposes of this report, conversions have been made according to the formulas shown below.

Currency, units of measure, and conversion factors used in this report are listed below:

Linear Measure

- 1 centimeter = 0.3937 inch
- 1 meter = 3.2808 feet = 1.0936 yard
- 1 kilometer = 0.6214 mile

Area Measure

- 1 hectare = 2.471 acres = 0.0039 square mile

Capacity Measure (liquid)

- 1 liter = 0.2642 US gallons

Weight

- 1 tonne = 1.1023 tons = 2,205 pounds
- 1 kilogram = 2.205 pounds

Unless otherwise indicated, all references to dollars ($) in this report refer to currency of the United States.

mg	microgram	kVA	kilovolt-amperes
a	annum	kW	kilowatt
A	ampere	kWh	kilowatt-hour
bbl	barrels	L	liter
Btu	British thermal units	lb	pound
°C	degree Celsius	L/s	liters per second
C$	Canadian dollars	m	meter
cal	calorie	M	mega (million); molar
cfm	cubic feet per minute	m^2	square meter
cm	centimeter	m^3	cubic meter
cm^2	square centimeter	mi	mile
d	day	min	minute
dia	diameter	mm	micrometer
dmt	dry metric tonne	mm	millimeter
dwt	dead-weight ton	mph	miles per hour
°F	degree Fahrenheit	MVA	megavolt-amperes
ft	foot	MW	megawatt
ft^2	square foot	MWh	megawatt-hour
ft^3	cubic foot	oz	Troy ounce (31.1035g)
ft/s	foot per second	oz/st, opt	ounce per short ton
g	gram	ppb	part per billion
G	giga (billion)	ppm	part per million
Gal	Imperial gallon	psia	pound per square inch absolute
g/L	gram per liter	psig	pound per square inch gauge
Gpm	Imperial gallons per minute	RL	relative elevation
g/t	gram per tonne	s	second
gr/ft^3	grain per cubic foot	st	short ton
gr/m^3	grain per cubic meter	stpa	short ton per year
ha	hectare	stpd	short ton per day
hp	horsepower	t	metric tonne
hr	hour	tpa	metric tonne per year
Hz	hertz	tpd	metric tonne per day
in.	inch	US$	United States dollar
in^2	square inch	USg	United States gallon
J	joule	USgpm	US gallon per minute
k	kilo (thousand)	V	volt
kcal	kilocalorie	W	watt
kg	kilogram	wmt	wet metric tonne
km	kilometer	wt%	weight percent
km^2	square kilometer	yd^3	cubic yard
km/h	kilometer per hour	yr	Year
kPa	kilopascal		

Table 2-2: Acronyms and Abbreviations

Abbreviation	Description
AA	atomic absorption spectrometry
ABA	acid-base accounting
Ag	silver
ARD	Acid Rock Drainage
Au	gold
Au_{CN}	cyanide-soluble gold
Au_{FA}	gold analysis by fire assay, total gold content
BLM	Bureau of Land Management
BMP	Best management practices
Calc, calc	calculated
CAPEX	Capital Expenditure
CFR	Code of Federal Regulations
cm	centimeters
core	diamond core-drilling method
˚C	degrees Celsius
dmt	dry metric tonne
dwt	dead-weight ton
Ext	extracted
°F	degrees Fahrenheit
FeOx	iron (oxyhydr)oxides
ft	foot or feet
ft^2, sf	square feet
gal	gallon(s)
g	gram
gpl	grams per liter
GPM, gpm	gallons per minute
g/t	grams per metric tonne
Ha	hectares
hd	head
HLF	Heap Leach Facility
HP	horsepower
Hr., hr., hrs	hour, hours
ICP	inductively-coupled plasma-emission spectrometric method
ICP-AES	inductively-coupled plasma atomic emission
ICP-MS	inductively-coupled plasma-emission and mass spectrometry
IDEQ	Idaho Department of Environmental Quality
IDL	Idaho Department Land
in	inch or inches
kg	kilograms
km	kilometers
kW	kilowatts
kWh/m^3	kilowatt-hours per cubic meter
kWh/yr	kilowatt-hours per year
l	liter (L in metallurgical use)
LOM	Life of Mine

Abbreviation	Description
lb or lbs.	Pounds
MIS	Management Indicator Species
MPO	Mine Plan of Operation
MTO	Material take-off
ML	Metal leaching
masl	meters above sea level
m	Meters
MWMT	meteoric-water mobility tests
mi	mile or miles
mm	millimeters
μm	micron or 10^{-6} meters
NFS	National Forest Services
NSR	net smelter return
Opt, oz/ton	troy ounce per short ton
org	Organic
oz	troy ounce
P_{80}	the theoretical square screen-opening, through which 80 weight percent of the particles will pass.
ppm	parts per million
ppb	parts per billion
QA/QC	quality assurance and quality control
RC	reverse-circulation drilling method
RQD	rock-quality designation
SO_4	Sulfate
st	Imperial short ton (2,000 pounds)
t	metric tonne or tonnes
tot	Total
VQO	Visual quality objective
wt%	weight percent
yr	year

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by M3 Engineering and Technology for Liberty Gold. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to M3 at the time of preparation of this Technical Report.
- Assumptions, conditions, and qualifications as set forth in this Technical Report.
- Data, reports, and other infrastructure supplied by Liberty Gold and other third-party sources.

For the purpose of this Technical Report, M3 has relied on ownership information provided by Liberty Gold. With respect to land title for unpatented mining claims, the client has relied on an opinion by Erwin and Thompson, LLP dated June 9, 2016; updates by Erwin Thompson Faillers dated: December 12, 2017, January 26, 2018, October 2, 2018, and September 10, 2019; and an opinion by Parsons Behle and Latimer dated March 25, 2022. For the private surface lands owned by Liberty Gold, a title report dated April 29, 2022, was provided by Parsons, Behle and Latimer. For private mineral rights, an opinion was provided by Parsons, Behle and Latimer dated April 22, 2022. All opinions found all properties in good standing. References are provided in Section 27 of this report. These opinions are relied on in Section 4.2 and the Summary of this Technical Report. M3 has not researched property title or mineral rights for Liberty Gold and expresses no opinion as to the ownership status of the property.

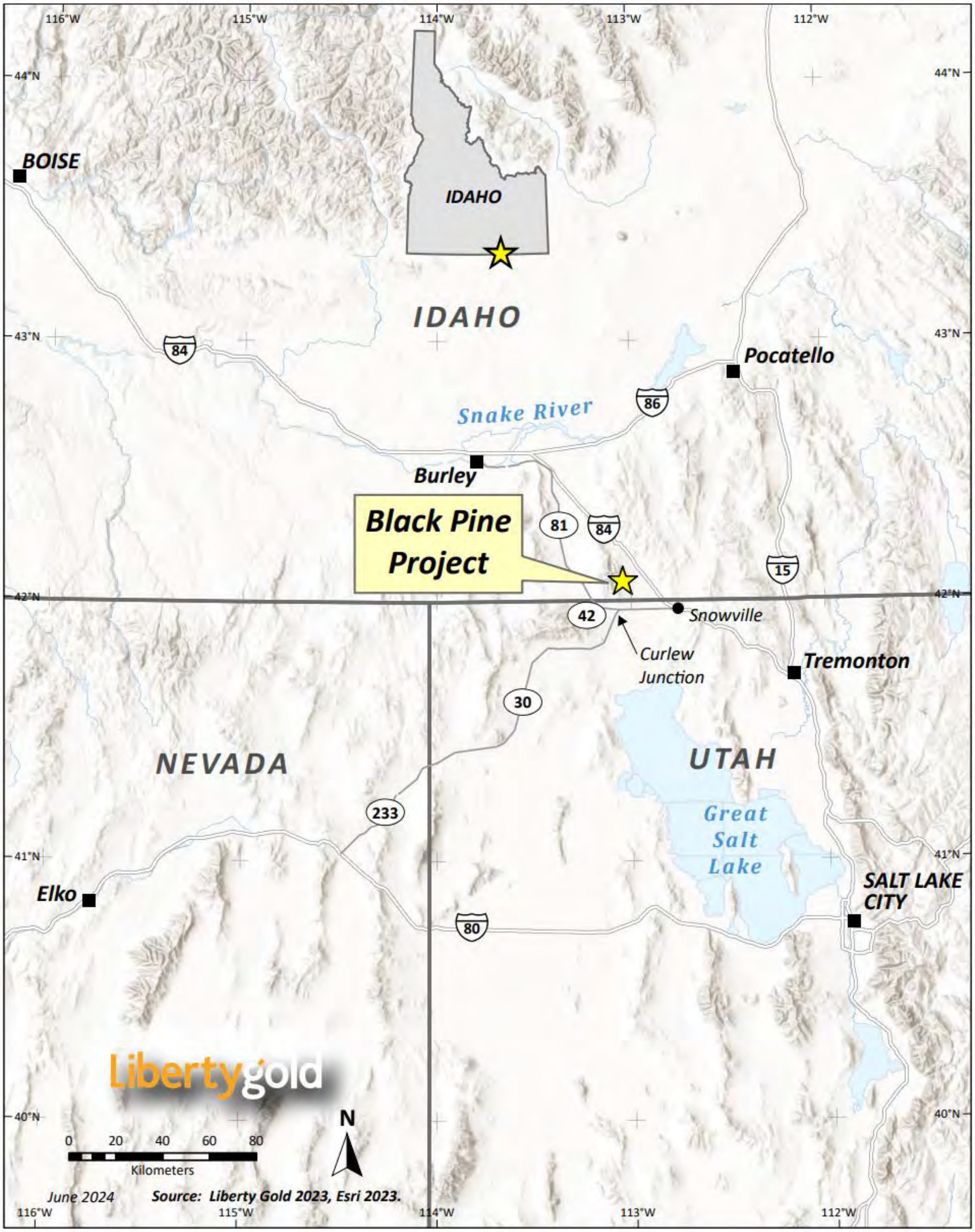
M3 has relied on information regarding potential environmental liabilities or concerns at the Project provided in a Stantec Consulting Services Inc. ("Stantec") report prepared by Brown (2016).

Liberty Gold provided guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from Black Pine.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Black Pine Project is located in Cassia and Oneida counties, Idaho, approximately 18 miles (29 km) northwest of the town of Snowville, Utah, the nearest substantial community, and 13 km north-northeast of Curlew Junction, the intersection of Utah State Highways 30 and 42 (Figure 4-1). The approximate geographic center of the Black Pine property is 42.082°N latitude and 113.047°W longitude.



Source: Liberty Gold, 2024

Figure 4-1: Location Map

4.2 Land Tenure

The combined area of the Black Pine property occupies an area of 14,485 acres (5,862 hectares), including unpatented federal mining claims, an Idaho State Minerals Lease, the Liberty Ranch private property, and private mineral rights.

4.2.1 Unpatented Federal Mining Claims

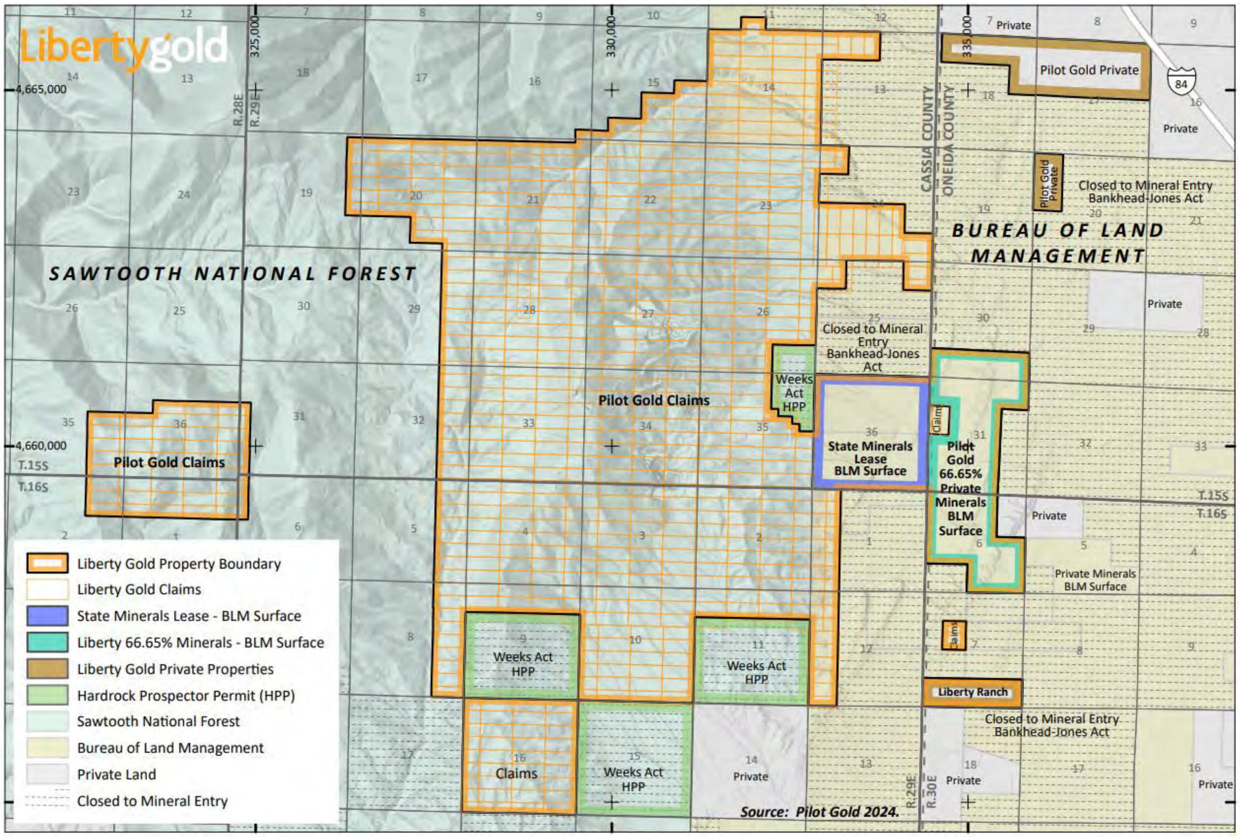
The portion of the Black Pine property on which the Mineral Resource lies consists of a largely contiguous block of 679 unpatented Federal mining claims within Cassia and Oneida counties, Idaho (Figure 4-2). The claims occupy a combined area of 12,792.6 acres (5,177 ha) as of the Effective Date of this report. The unpatented claims lie in portions of or all of Sections 11, 13-16, 19-29, and 31-35 of T15S, R29E; Sections 1-6, 8-12, and 16 of T16S, R29E; Sections 35 and 36 of T15S, R28E; Sections 1 and 2, T16S, R28E; Section 31, T15S, R30E; and Section 7, T16S, R30E, Boise Meridian.

Liberty Gold is the 100% owner of all unpatented federal lode and placer claims that comprise the majority of the Black Pine property, having purchased 345 of the unpatented lode claims from Western Pacific Resources Corp. ("Western Pacific") through an agreement dated June 15, 2016. Under this agreement, Western Pacific received $800,000 in cash, a 0.5% net smelter royalty ("NSR") on production from the 345 unpatented claims, and 300,000 common shares of Liberty Gold. Western Pacific subsequently assigned the 0.5% NSR to Deer Trail Mining Company, LLC. Liberty Gold expanded the property by staking 334 unpatented claims between 2016 and 2024, including 11 placer claims over the historic heap leach facility, for a total of 679 claims.

The irregular pattern of claims in the eastern part of the property reflects federal lands that are not open to Mineral Location by staking (discussed further below).

The unpatented claims are monumented with 4 in by 4 in (10-cm by 10-cm) wooden posts bearing metal tags so as to meet Idaho State regulations. The claim map, valid as of the Effective Date of this report, is presented in Figure 4-2.

Ownership of the unpatented mining claims is in the name of the holder (locator), subject to the paramount title of the United States of America. The majority of the claims are under the administration of the U.S. Forest Service ("USFS"). A total of 106 claims in the eastern portion of the property lie partly or entirely within lands administered by the U.S. Bureau of Land Management (BLM). Under the Mining Law of 1872, which governs the location of unpatented mining claims on Federal lands, the locator has the right to explore, develop, and mine minerals on unpatented mining claims without payments of production royalties to the U.S. government, subject to the surface-management regulation of the USFS and the BLM. In recent years, there have been unsuccessful efforts in the U.S. Congress to amend the 1872 Mining Law to include, among other items, a provision for production royalties payable to the U.S. government. Annual claim-maintenance fees are the only Federal payments related to unpatented mining claims, and these fees have been paid in full through September 1, 2025. County recording fees are also required annually. Liberty Gold's annual holding costs for the Black Pine unpatented mining claims, exclusive of lease fees, were $103,015 in 2023 and $123,359 in 2024 with the addition of new claims (Table). The unpatented claims do not expire as long as the Federal and county fees are paid.



Source: Liberty Gold, 2024

Figure 4-2: Property Map

Table 4-1: Annual Land Holding Costs for the Black Pine Property (Liberty Gold Corp. – Black Pine Project)

Annual Fee Type	2023 (US$)	2024 (US$)
Unpatented BLM Claim Fees	102,630	122,940
County Recording Fees	385	419
Total Annual Claim Fees	103,015	123,359
Section 36 Mineral Lease	3,037	3,098
Private Property Taxes	889	889 (estimated)
Total Land Holding Costs	106,941	127,346

4.2.2 Lands Closed to Mineral Entry

Some BLM-administered lands to the east and south of the Black Pine Project are closed to locatable mineral entry under the Bankhead-Jones Farm Tenant Act of 1937, Pub. L. No. 75-10, 50 Stat. 522 (codified as amended at 7 U.S.C. §§ 1010-1013a (2006)). Past operators have obtained Hardrock Prospector Permits allowing for gold and silver exploration activities on Bankhead-Jones Act lands. However, recent rulings by the U.S. Government have restricted mining activities on these lands to energy minerals (e.g., uranium) on Bankhead-Jones Act lands. Whether or not a path exists for Liberty Gold to explore for gold or locate infrastructure on these lands, which are not part of Liberty Gold's landholdings, is under investigation.

Some USFS-administered lands are closed to locatable mineral entry under the Weeks Act of 1911 (36 Stat. 961). Past operators have obtained Hardrock Prospector Permits and Special Use Permits from the USFS for gold and silver exploration and surface use. In May 2023 Liberty Gold applied for a Hardrock Prospector Permit on these lands, with approval expected in September 2024. These lands are in Sections 26 and 35 of T15S R29E and Sections 9, 11, and 15 of T16S R29E, covering approximately 1,762 acres (713 ha).

4.2.3 Idaho State Mineral Lease

Effective November 18, 2021, Pilot Gold was granted a 20-year lease by the Idaho State Board of Land Commissioners on metallic mineral rights in Section 36, T15S R29E in Cassia County, totaling 642 acres (260 ha). The surface is owned and administered by the BLM.

4.2.4 Liberty Ranch

In February 2021, Pilot Gold purchased 139.4 acres (56.4 ha) in Section 7, T15S R30E, immediately southeast of the Project area, including water and mineral rights. Power and water were provided to the site, with construction of a core storage facility and it is presently being used as a base for exploration activities.

4.2.5 Private Mineral Rights

In February and August 2022, Pilot Gold purchased a 66.65% controlling interest in 911 acres (386.7 ha) of mineral rights located in Section 6 of T16S R30E, and Sections 30 and 31 of T15S R30E to the east of the Idaho State mineral rights in Section 36. The BLM owns and administers the surface rights.

4.2.6 Other Private Properties

In March and June 2023 Liberty Gold purchased an additional 884 acres (89.5 ha) of private land northeast of the Project for habitat conservation and mitigation projects. The properties are in Sections 17, 18, and 20 of T15S R30E and Sections 7 and 8 of T14S R31E in Oneida County.

4.3 Agreements and encumbrances

Liberty Gold obtained its interest in the Black Pine property by means of an agreement with Western Pacific dated June 15, 2016. Under this agreement, Western Pacific received consideration of $800,000 in cash, a grant of a 0.5% NSR, and 300,000 common shares of Liberty Gold. As a result of this transaction, Liberty Gold is the 100% owner of the Black Pine property.

Western Pacific assigned the 0.5% NSR to Deer Trail Mining Company, LLC. This royalty applied to production from the original 345 claims obtained by Liberty Gold from Western Pacific. In September 2023 Liberty Gold bought back the Western Pacific/Deer Trail Mining Company, LLC royalty, and assigned a new 0.5% NSR royalty to Wheaton Precious Metals (Cayman) Co. This new royalty covers all claims, leases, private properties, and mineral rights owned by Liberty Gold as of September 8, 2023, as well as subsequently acquired properties, covering approximately 17,131 acres (6,933 ha).

Section 36 Idaho State Minerals Lease is subject to a minimum annual royalty of $1,000 for Years 1 through 5 and $2,500 for Years 6 through 20. Production is subject to a 5% Net Smelter Return Royalty payable to the State of Idaho.

Production of metallic minerals from the private mineral rights lands described above will be subject to a 0.25% NSR.

Mineral production from the entire property is subject to the Idaho Mine License Tax, equivalent to 1.0% of the value of “ores mined or extracted, and royalties received from mining”.

Surface rights for access, exploration, and mining of the unpatented claims are fully held by Liberty Gold under the Mining Law of 1872, subject to surface-use regulations under applicable Federal and State environmental law (see Section 4.2.10).

4.4 Environmental Liabilities

Liberty Gold retained Stantec to review information regarding potential environmental liabilities or concerns, the results of which are documented in a report by Brown (2016). According to Stantec, Liberty Gold is liable only for disturbance incurred as part of Liberty Gold's exploration activities, or if Liberty Gold causes disturbance of the historical leach pad or other designated areas.

The historical heap-leach pad, which lies partially within the Black Pine property, was reclaimed prior to Liberty Gold's acquisition of the property (Figure 4-3). Pegasus Gold Corp. (Pegasus) stopped adding cyanide solution to the heap-leach pad in 1998. Since then, the USFS has been capturing runoff water at the base of the heap leach in buried concrete vaults, treating it with zero-valent iron, and delivering the treated water to a 100.1-acre (40.5-hectare) land-application area downhill from the leach pad. The exact volume of water produced is unknown but estimated to be 7.5 million gallons annually. Water is sampled two to four times during the land-application period and soils are analyzed periodically. The heap leach has ongoing issues with cyanide and elevated levels of nitrate and arsenic. The USFS provides annual water-quality monitoring reports to the Idaho Department of Environmental Quality (IDEQ, (http://www.deq.idaho.gov/). The heap leach and land-application area are fenced off. A local farmer monitors the land application system in the summers.

The USFS and IDEQ hold a $1.5 million bond from Pegasus, and the interest on this bond covers the cost of the ongoing water-monitoring program. This bond is expected to cover any future issues with the previous operations.

Tailings from the 1950s Virmyra Mine (Tallman pit) were left exposed to the elements for many years, contaminating downhill soils with arsenic. In 1994-1995, Pegasus excavated the soils and placed them in a repository on top of the old tailings. The repository was subsequently buried under a minimum of six feet of waste rock. Current mining plans to not disturb the repository.

Source: Liberty Gold, 2017

Figure 4-3: View of Reclaimed Black Pine Mine Heap-Leach Pad, Looking East

4.5 Environmental and Permitting

All exploration work on unpatented claims between June 2011 and February 2019 was permitted under a Plan of Operations ("PoO") approved by the USFS, as described below. This PoO (#2011-030938-B) was granted to Western Pacific by the USFS on June 2, 2011, and subsequently amended on May 30, 2012. A cash bond totaling $67,300 was posted with the USFS to cover potential reclamation costs. PoO 2011-030938-B was transferred to Liberty Gold in 2016 and assigned a new number (#2016-063179), and the bond amount was increased to $206,400. PoO #2016-063179 authorizes 33.12 acres of disturbance (13.4 ha).

A new Plan of Operations (#2017-072046) was submitted to the USFS on May 11, 2017, and approved on February 12, 2019. The new PoO allows for construction of roads and up to 370 drill sites, 29.8 mi (47.9 km) of drill roads, and 141.1 acres (57.1 ha) of disturbance within a 2.82 mi^2 (7.3 km^2) area surrounding the historical mined pits and two satellite areas to the northwest and southeast.

In February 2020, a modification to the PoO adding up to 154 drill sites, 15.3 mi (24.6 km) of drill roads, and 50.7 acres (20.5 ha) of disturbance within a 1.81 mi^2 (4.7 km^2) area was submitted to the USFS and BLM. Approval was granted in March 2021, allowing access to lower elevation areas along the eastern range front. This PoO revision also grants access to the historic Black Pine Mine Well and use of public roads on BLM-administered land (case number IDI-039132). Total permitted access includes up to 596 drill sites, 56.6 mi (91.1 km) of drill roads, and 225 acres (91.0 ha) of disturbance within a 4.6 $miles^2$ (11.9 km^2) area.

An additional PoO was submitted to the BLM in September 2021 and approved in September 2022 (case numbers IDI-039411 and IDI-039412). This approval is additional to the existing PoO issued by the USFS for current exploration activities at Black Pine over a surface area of 4.8 mi^2 (12.4 km^2), totaling 35.1 acres (14.2 ha) of disturbance over 16.7 mi (18.8 km) of new and existing roads and 117 drill pads, in the Section 36 lease area, unpatented lode claims, and the private mineral lands located to the east of the Black Pine resource area.

In July 2022, Liberty Gold received an approved Notice of Intent (NOI) from the USFS permitting a drill site on the high priority regional Gully Target, located approximately 1.24 mi (2 km) north of the current area of operations.

In May 2023, a new modification to the PoO was submitted to the USFS. Proposed disturbance includes 245 new drill sites, 32.4 mi (52.5 km) of drill roads, and 22.8 mi^2 (59 km^2) of disturbance over an area of 4.4 mi^2 (11.4 km^2) surrounding the existing PoO area. Approval is expected in June 2024.

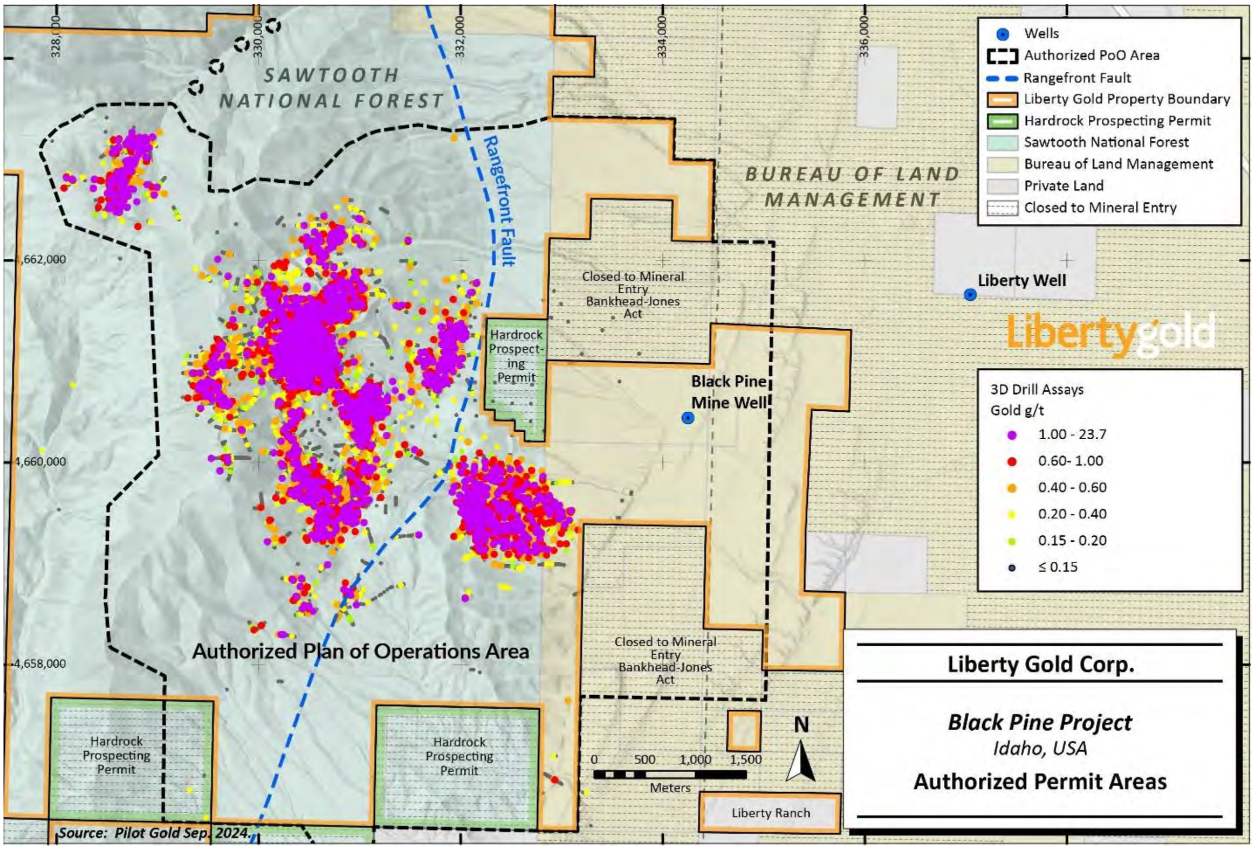
In October 2023 Liberty Gold received a Categorical Exclusion from the USFS for characterization of the historic heap leach facility with sonic drilling.

With the expected approval of the May 2023 modification and Hardrock Prospector Permit in the second half of 2024, combined authorized exploration activity under the USFS and BLM PoOs include 977 drill sites, 100.7 mi (162 km) of drill roads, and 405.7 acres (164.2 ha) of disturbance, over an area of 15.7 mi^2 (40.6 km^2).

Permit areas are summarized in Figure 4-4.

As of December, 2023, there were 158 open drill sites and 239 reclaimed drill sites, 21.7 mi (35.0 km) of open drill roads, 13.6 mi (21.9 km) of reclaimed drill roads, and 84 acres (34.0 ha) of open disturbance and 56.1 acres (22.7 ha) of reclaimed disturbance (total 56.7 ha).

There are no unique biological or cultural issues currently identified within the Project area. Mitigation/avoidance procedures for resources such as greater sage-grouse mating periods, mule-deer winter range, sensitive plant species, and introduction of noxious-weed species are stipulated in the PoO. At present, USFS land drilling is restricted to the months between July 1 to February 28 within 2.0 mi (3.1 km) of active sage-grouse leks to account for sage-grouse mating periods. BLM land drilling is restricted to certain times of day from November 1 to July 31 within 1.9 mi (3 km) of active sage-grouse leks. In the south and west, drilling is restricted to the months between May 1 and January 31 to protect mule deer winter habitat. However, there are no restrictions for other areas, which comprise most of the mineralized zones and targets. Two reclamation bonds totaling $4,485,400 cover Liberty Gold's permitted disturbance on USFS-administered claim areas. The BLM holds three reclamation bonds totaling $603,151 for activities on BLM-administered lands and USFS HPP areas. The Idaho Department of Lands holds a $5,000 bond for activities on the state minerals lease. Together the bonds total $5,093,551.



Source: Liberty Gold, 2024

Figure 4-4: Permit Areas

4.6 Water Rights

Several water wells are located immediately east of the property on BLM land. In accordance with Idaho Code 42-202A, Liberty Gold was granted temporary, 5 acre-foot per annum ("afa") water rights by the Idaho Department of Water Resources ("IDWR") in 2019 through 2021. Water was taken from a nearby BLM stock well (Black Pine Well No. 1) and was used for drilling and dust suppression. In April 2020, an additional 50 afa was leased from a local farmer through the Idaho State water bank. The point of diversion for this water was transferred to the historic Black Pine Mine Well. The use of water for mining or exploration is considered a beneficial use approved by IDWR.

The water needs of the historical mining were being met through a single production well known as the Black Pine Mine Well, which was licensed for 868.5 afa in two water rights. Access to the well was granted to Liberty Gold as part of the PoO amendment in March 2021, allowing Liberty Gold to refurbish and start pumping from the well. In November 2022, Pilot Gold USA successfully purchased these two rights out of the Bankruptcy and transferred title to their name.

In 2021 and 2022, Liberty Gold secured two separate lease-option agreements for an aggregate 2,194.5 afa of agricultural water from the nearest farm owners, granting water rights to be used during mining activities.

Liberty Gold also purchased the rights to an additional 140 afa of water rights through its acquisition of the private ranch described in Section 4.2.3.

In total, Liberty Gold has secured, through purchase and lease agreements, access to over 3,200 afa of water rights, sufficient for any future large-scale mining operation envisioned at Black Pine.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The information summarized in this section is derived from publicly available sources, as cited.

5.1 Access to Property

The Black Pine Project is located approximately 6 mi (10 km) west of U.S. Interstate Highway 84 (I-84) and access is available from I-84 and Utah State Highway 30 via improved gravel roads (County Road 36,000W and County Road 9,000S), jointly maintained by Oneida County and Liberty Gold. These connect with Forest Route 201, a USFS-maintained gravel road, for 2.5 mi (4 km) to the property entrance. The property can also be accessed from the north on I-84 via County Road 38,000W, an improved gravel road.

There are a number of locked gates within the property. Permission to enter and keys must be obtained from Liberty Gold or the USFS.

A number of major population centers with commercial air service are located in the region surrounding the Black Pine Project. The cities of Twin Falls, Idaho, and Salt Lake City, Utah, are located about 109 mi (175 km) to the northwest and 118 mi (190 km) to the southeast, respectively. Elko, Nevada is located approximately 186 (300 km) southwest of the Project, and Boise, Idaho is 211 mi (340 km) to the northwest. Figure 4-1 shows the location of the Project and Figure 5-1 shows the access routes to the Project from surrounding cities.

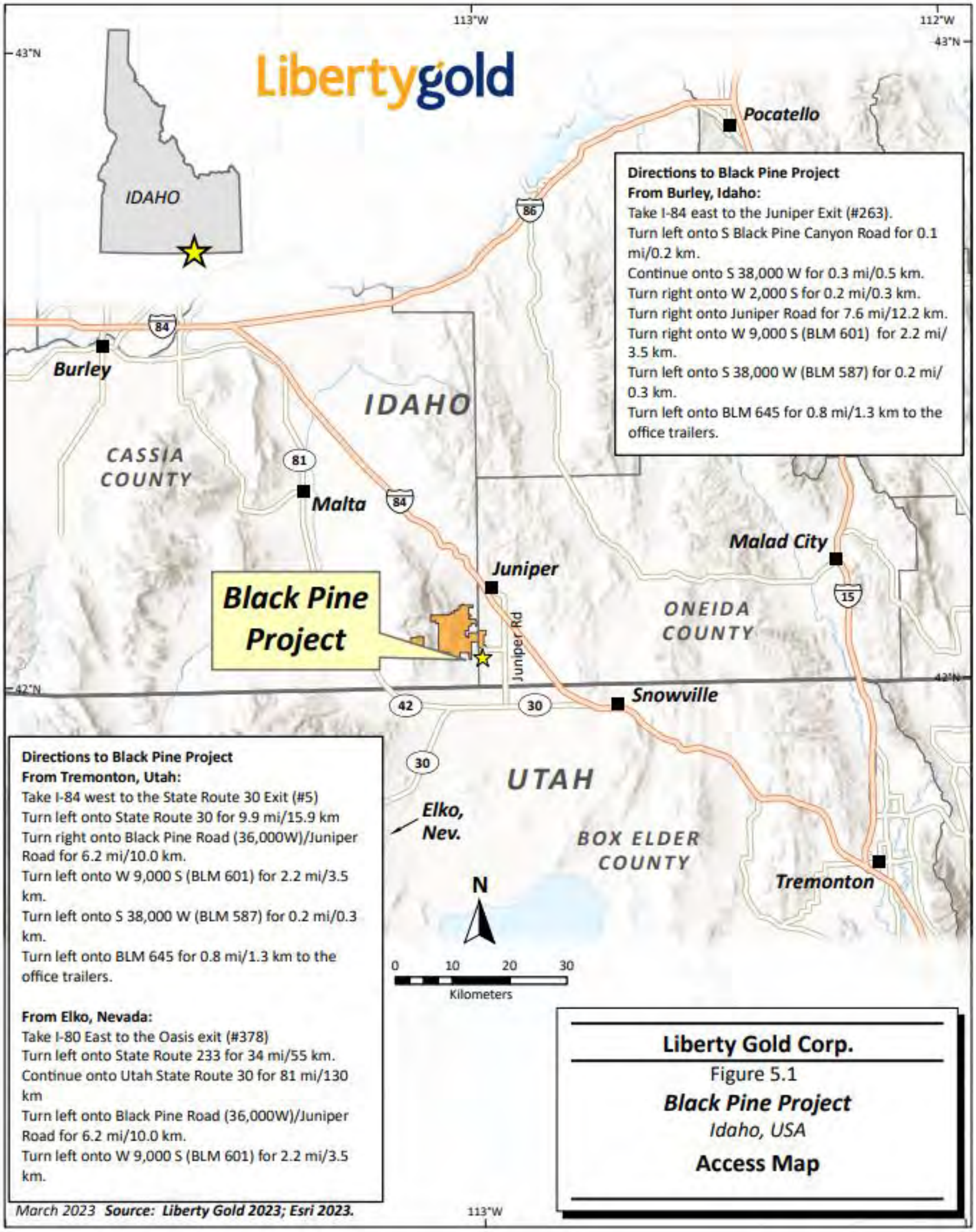
5.2 Climate

The climate in the Project area and the surrounding region is of the continental, intermontane type. Temperatures and precipitation can vary widely from the high-desert valleys directly east of the property to the crest of the Black Pine Mountains on the west side of the property. Annual precipitation is approximately 10 in (25 cm) at the base of the range, with significant variations dependent on elevation. Summer temperatures in the valleys commonly range from 5°C to over 40°C. Winter temperatures generally vary from -10°C to 10°C, but they can occasionally drop to -20°C. Winter snow at higher elevations can impede access from mid-November through late April unless snow removal equipment is deployed. Mining can be conducted year-round, but exploration activities can be impacted by winter snowstorms.

Liberty Gold installed two weather stations and an air quality monitoring station in 2021-22 to collect baseline data on climate and air quality.

5.3 Physiography

The Black Pine property straddles the eastern margin of the northerly-trending Black Pine Mountains. Elevations within the property range from a low of 5,413 ft (1,650 m) along the eastern edge, to a maximum of approximately 8,005 ft (2,440 m) in the western part of the property. The topography is moderately steep over much of the area. There are no perennial streams; all watersheds in the property eventually drain into the Great Salt Lake basin, located to the south. Vegetation in the lower elevations of the Project area consists mainly of grasses and sagebrush. In the higher elevations, increased moisture allows juniper, piñon, mountain mahogany, and locally on steep, north-facing slopes, spruce to grow. Wildfires in 2006-07 denuded much of the range and only a few scattered patches of trees remain within the property.



Source: Liberty Gold, 2024

Figure 5-1: Access Map

5.4 Local Resources and Infrastructure

The small agricultural community of Snowville, Utah, is the nearest town to the Project, about 19 mi (30 km) to the southeast. Basic lodging, fuel, and some supplies are available. Burley, Idaho, the Cassia County seat, is located 50 mi (80 km) to the northwest, and Tremonton, Utah, is located an equal distance to the southeast. Both are full-service communities with availability of food, lodging, fuel, banking, telecommunications, and other project needs. Heavy equipment and operators are available from numerous local contractors. Drilling, engineering, and heavy-equipment services are available in Salt Lake City, Utah, and Elko, Nevada, as is skilled labor for mining and construction.

Grid electrical power is available from a transformer on a major power line about 6.2 mi (10 km) southeast of the Project, with a 25 kV distribution line extending to the eastern property boundary. Liberty Gold received a positive initial system impact study from Idaho Power Distribution Company on the supply of up to 10 megawatts of electrical power along the distribution line, which is managed by Raft River Rural Electric Co-op Inc. Further studies are on-going to refine transmission bottlenecks, system design constraints and cost estimates.

Water for exploration drilling needs is available from several wells on BLM and private land immediately east of the property. The Black Pine Mine Well, used during the historical mining operations, was drilled in the southeast quarter of the northeast quarter of Section 36, with water encountered in alluvium just below 5,000 feet (1,524 m) in elevation, or approximately 185 feet below the ground surface (USDA Forest Service, 1993).

6 HISTORY

The information summarized in this section was originally extracted and modified from Hefner et al. (1991), Shaddrick (2013), and unpublished company files, as well as other sources as cited, and it is largely unmodified from that presented in Gustin et al (2021).

6.1 Exploration History

The Black Pine Mountains were first explored in the 1880s (Sawyer et al., 1997). Numerous prospects and small mines exploited base- and precious-metal deposits through the late 1800s and early 1900s, when minor amounts of zinc, silver, and mercury were produced. Gold was discovered in the late 1930s or early 1940s at the Tallman mercury mine, located on the current Black Pine Project. The Virmyra Gold Mining Company operated a small open pit from 1949 to 1955 in the Tallman area (Prochnau, 1985). Total production was reported to be 98,884 tonnes (120,000 tons) with an average gold grade of 5.14 g Au/t (Hefner et al, 1991).

Modern exploration of the Black Pine Project area began in the 1960s. Relatively little information is currently available concerning exploration work done in the 1960s to 1981. Much of what is known of that period is based on a summary in Threlkeld (1983) and archival material as follows:

- 1963 - 1964: Newmont Mining ("Newmont") carried out geologic mapping and surface geochemical sampling, which culminated in the drilling of 17 holes. Newmont terminated their involvement with the property in 1964 at approximately the same time as the Carlin deposit was discovered in Nevada.
- 1974 - 1975: Newmont reacquired the property and drilled 20 holes. At least three of the holes encountered gold grades >1.71 g/t Au. Newmont also carried out soil geochemical surveys, as well as induced potential ("IP") and ground magnetic surveys over the Tallman mercury mine area. The geophysical work was done on NW-SE lines and detected a broad area of IP chargeability highs beneath the Tallman Pit area. Newmont terminated their second involvement with the property in 1975.
- 1975: Kerr Addison Mines Ltd. collected rock samples from unknown locations on the property and submitted them for copper, zinc, and gold analyses.
- 1974 - 1976: Gold Resources Inc. ("Gold Resources") and Permian Exploration Account ("Permian") held claims over a portion of the property and collected numerous rock and soil samples. Liberty Gold has historical records that indicate Gold Resources drilled 16 holes during this time period. Kermadex also staked claims and carried out soil sampling in the region during this time, but little else is known of their work or results.
- 1977 - 1978: ASARCO leased the property from Gold Resources and Permian and carried out grid-based soil sampling, geological mapping, and geophysical surveys. The geophysics consisted of ground-based gravity, VLF, and IP surveys on two lines. A shallow conductor attributed to either disseminated sulfides or graphitic material was detected with the IP and VLF, but the gravity response was minimal (Paterson, 1979). ASARCO drilled 34 "percussion" holes before terminating their interest in 1978. No data is available for the 34 holes drilled by Asarco.
- 1979 - 1981: Pioneer Nuclear Inc. ("Pioneer") acquired the property in 1979. Pioneer carried out soil sampling and drilled 23 holes in 1979, of which 13 holes encountered gold grades greater than 0.51 g Au/t. In 1980 and 1981, Pioneer drilled five holes in and around the historical Tallman pit.
- 1983 - 1986: Permian and Pegasus formed a joint venture and drilled 88 holes at the property during 1983, and an additional 36 RC holes and one diamond core ("core") hole in 1984. Pegasus re-assayed samples from selected Pioneer holes in 1985 and defined the Tallman and Tallman NE gold deposits to a significant extent with their drilling.
- 1986: Inspiration Resource Corp. ("Inspiration") took soil samples across several lines of existing soil grids. This work was likely completed as due-diligence confirmation sampling, as there doesn't appear to have been a joint venture agreement between Inspiration and Permian.

- 1986 - 1990: In 1986, Noranda Exploration, Inc. ("Noranda") acquired the property from Permian. Over four years, Noranda carried out an extensive exploration and drilling program, including soil and rock sampling, detailed geological mapping, and stratigraphic studies. In 1987, Noranda contracted TerraSense Inc. to complete an airborne magnetic survey over a significant portion of the Black Pine Mountains that included the current property. These data have not been digitized or fully interpreted by Liberty Gold. Noranda drilled a total of 532 RC and conventional rotary holes, as well as four core holes for metallurgical testing samples.

On the basis of this work, Noranda discovered most of the gold zones that were later mined by Pegasus. Noranda produced a feasibility study in early 1990 and sold the property to Pegasus in June 1990.

- 1990 - 1998: Pegasus put the property into production in late 1991 as an open-pit heap-leach operation. Pegasus did not build the mine as designed in the Noranda feasibility study, however, choosing to load the leach pads with ROM mineralized material instead of crushing it. Pegasus drilled 1,082 RC holes and 17 core holes from 1990 through 1997. Soil samples were collected from grids along the southern range front and north of Mineral Gulch, and an extensive rock-sampling program was carried out. Three-dimensional deposit models were created based on the domains of exploration drill hole and blast-hole assays, without taking detailed geology into account. Mining ceased in late 1997, and the last gold was recovered from the heap in 1998. The USFS seized the reclamation bond and reclaimed the property.
- 1999 – 2009: The property was idle from 1999 to 2009. Western Pacific acquired the property by staking claims in 2009 and 2010.
- 2010 - 2012: Western Pacific contracted 82 line-km of gravity and 20 line-km of ground magnetic surveys and drilled a total of 38 RC holes. This was followed by an aeromagnetic survey in 2012 of 1,842 line-km flown by EDCON-PRJ, Inc. and interpreted by Fritz Geophysics (Fritz, 2012).

6.2 Historical Geological Mapping

The regional to district-scale geology of the Black Pine Project area is illustrated by the 1:50,000 scale U.S. Geological Survey (USGS) map of the Strevell 15-Minute Quadrangle, Cassia County, Idaho by Smith (1982). Noranda geologists and consultants produced the most comprehensive geological map of the Black Pine property (Ohlin, 1988). Later mapping by Pegasus did not appear to improve upon the Noranda maps, even with the additional exposures afforded by the open pits.

Pit-geology maps generated by Willis (2011) for Western Pacific were imported into the Liberty Gold database and draped onto topography using Leapfrog software, which allowed Liberty Gold to conclude that the 2011 pit maps correlate well with down-hole lithology data.

Liberty Gold possesses numerous scanned geological maps from historical operators, which have either been integrated into the database or superseded by Liberty Gold mapping and modelling.

6.3 Historical Soil Sample and stream Sediment Data

Liberty Gold has compiled and digitized geochemical data from 12,623 soil samples collected and analyzed by at least three historical operators, including Noranda, Pegasus, and Western Pacific (Figure 6-1). Relatively little is known about the soil-sampling methods used by operators prior to 2011. Soil sampling was primarily grid-based, with sample spacings ranging from approximately 15 m to 120 m. There are also numerous scanned maps of earlier soil grids of limited extents from nearly every historical operator, including Gold Resources, Newmont, Kermadex, Inspiration, Permian, Asarco, Pegasus (pre-Noranda), and Pioneer. Where possible, these have been geospatially registered and digitized by Liberty Gold. Comparison of the Pegasus-era soil data with scanned maps of Noranda-era compiled soil samples indicates that Pegasus collected up to 2900 soil samples to the north of Mineral Gulch, south of the I Pit and Rangefront zone.

Western Pacific contracted two soil surveys. The northern grid (1,175 samples) was collected by Rangefront Consulting LLC of Elko, Nevada, and the southern grid (1,300 samples) was collected by North American Exploration of Salt Lake City, Utah. The work was done under the supervision of Western Pacific's qualified person. Samples were collected on 50 m by 50 m grids with locations established by handheld Global Positioning System ("GPS") units. Soil material was taken from the "B" horizon, where present, and omitted in areas of exposed rock.

The soil samples compiled by Liberty Gold delineate a strong gold-in-soil anomaly, with 4,986 samples that assayed in excess of 0.050 g Au/t and 3,205 samples that assayed greater than 0.100 g Au/t. These samples principally form a broad, diamond-shaped anomaly over the historical mine area, of approximately four km north-south by about three km in an east-west direction (Figure 6-1). It is clear that soil geochemistry played a critical part in determining historical exploration targets, owing to the excellent correlation between elevated gold-in-soils and the locations of historical deposits and pits, as well as its correlation with historical drill targets. Significant portions of the historical gold-in-soil anomalies have not been adequately drill tested at Black Pine and are high-priority drill targets for Liberty Gold.

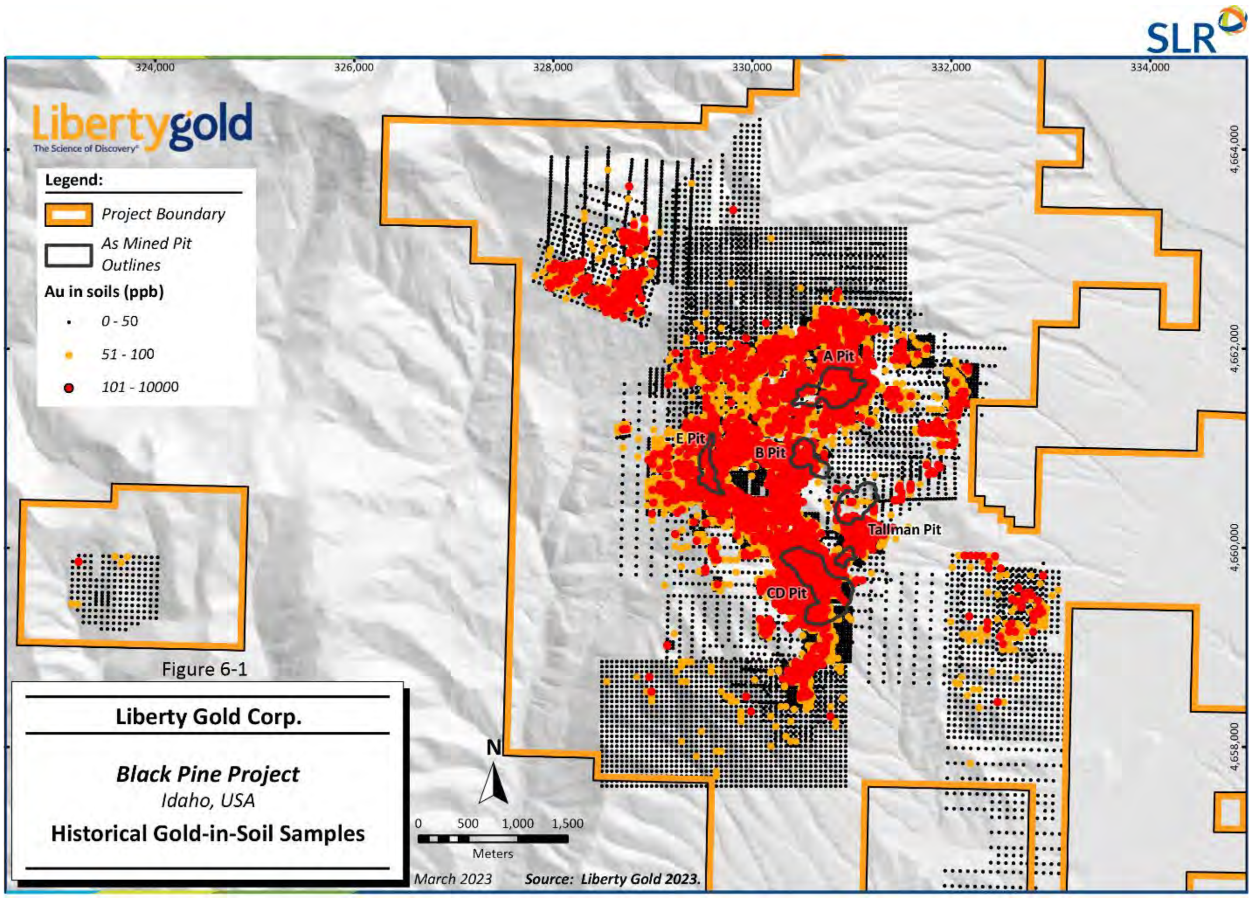


Figure 6-1: Historical Gold-in-Soil Samples

Stream-sediment surveys were carried out by previous operators across the broader Black Pine Mountains as part of a regional exploration effort. This data is not presently being used by Liberty Gold, as it has largely been superseded by soil and rock data.

6.4 Historical Rock-Chip Geochemistry

A large number of historical surface rock samples have been taken over the course of exploration of the Black Pine property. A historical electronic database with 5,202 samples across the Black Pine Mountains was recovered from

Pegasus' project archives, including 4,516 that were taken within the current property boundary. Of these, 59 are lacking location information. Liberty Gold has scanned and digitized all maps that could be georeferenced to validate the historical rock sample locations.

Western Pacific collected 251 rock-chip samples, primarily focused on existing pits and road cuts, that were intended to verify and expand the known mineralization indicated by the historical exploration and mining data (Shaddrick, 2013). A total of 250 of these samples are located within the current property boundary.

Of the 4,516 historical samples in the digital database taken within Liberty Gold's current property boundary, 1,344 returned gold values in excess of 0.1 g Au/t, 168 were in excess of 1.0 g Au/t, and 19 samples assayed greater than 5.0 g Au/t. The presently compiled historical gold results from rock samples are shown in Figure 6-2.

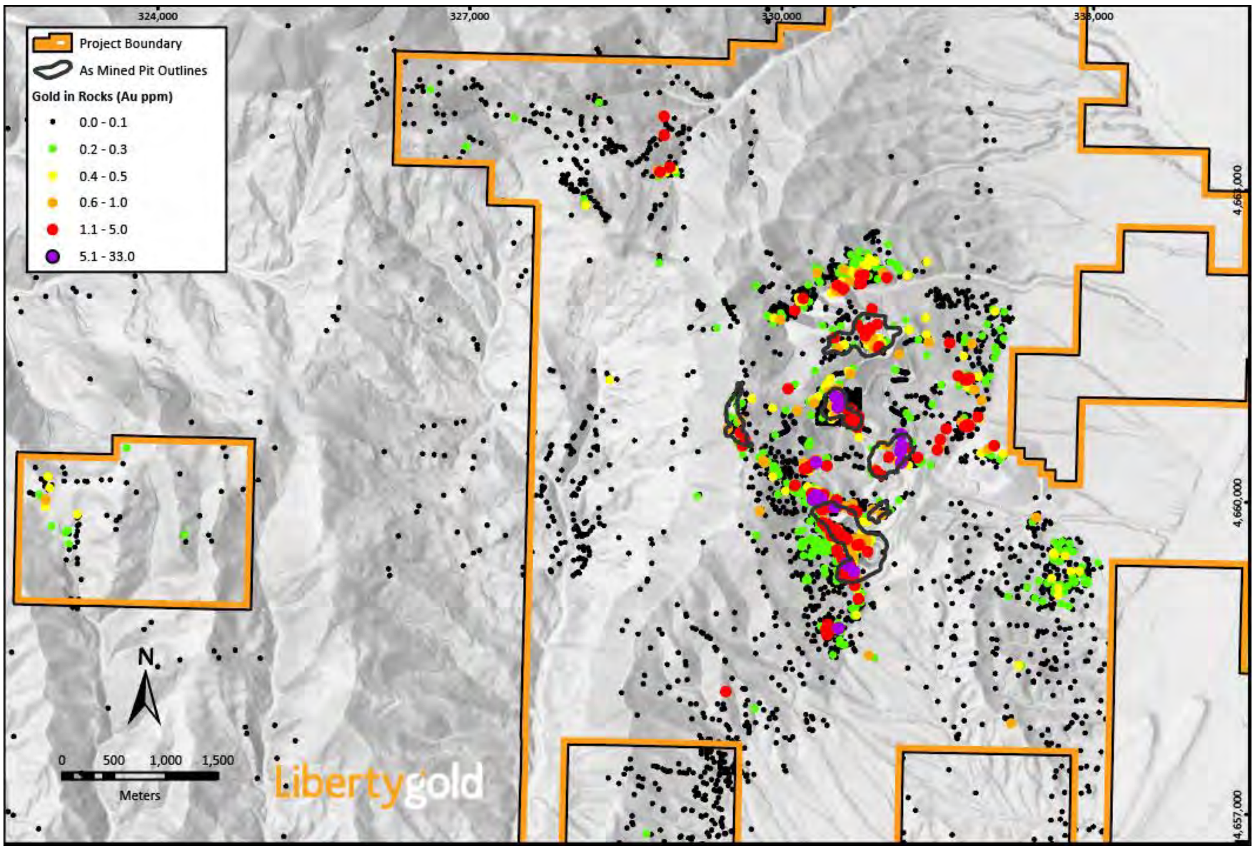


Figure 6-2: Historical Gold-in-Rock Samples

6.5 Historical Drilling

This section summarizes the drilling carried out in the Black Pine property by historical operators. The information presented in this section of the Technical Report is derived from multiple sources, as cited.

6.5.1 Summary

Liberty Gold has compiled information for a total of 192,248 m drilled in 1,887 holes at Black Pine, not including ASARCO holes for which no data were found (Table 6-1). Approximately 99% of the holes and meters were drilled using conventional rotary and RC methods, and 26 of the holes were drilled using diamond-core methods. Other than the core holes, many of the historical holes lack explicit designation as to the type of drilling method, specifically conventional rotary versus RC. In many cases, these are assumed to be RC holes, but it is likely that some are conventional-rotary holes, especially the older holes. There is no assay data currently available for 34 conventional rotary or RC holes drilled by ASARCO in 1977.

Table 6-1: Summary of Black Pine Project Historical Drilling

Company	Year	RC/Rotary Holes		Core Holes		Total	
		No.	Meters	No.	Meters	No.	Meters
Newmont	1964, 1974	37	3,119	-	-	37	3,119
Gold Resources	1974-1976	13	1,083	3	135	16	1,218
Pioneer Nuclear	1979-1981	28	2,458	-	-	28	2,458
PEA/Pegasus	1983-1985	123	8,245	1	76	124	8,321
Noranda	1986-1989	532	51,366	4	245	536	51,611
Pegasus	1990-1997	1,082	116,447	16	1154	1,098	117,601
Western Pacific	2011-2012	38	7,920	-	-	38	7,920
Total[1]		1,853	190,638	24	1,610	1,877	192,248

(Liberty Gold Corp. – Black Pine Project)
Notes:
1. 34 holes drilled by ASARCO in 1977 excluded from the tally as there is no assay data currently available.

Most historical holes were drilled vertically, or within 10° of vertical. Roughly one-third have been drilled as angled holes, including 676 holes drilled at angles shallower than -75°. The geometry of gold mineralization at Black Pine varies considerably but is generally gently dipping with some areas of more steeply dipping mineralization. Historical operators appear to have generally designed drill holes to intersect mineralization as obliquely as possible.

Figure 6-3 shows the locations of historical drill hole collars within the Black Pine property.

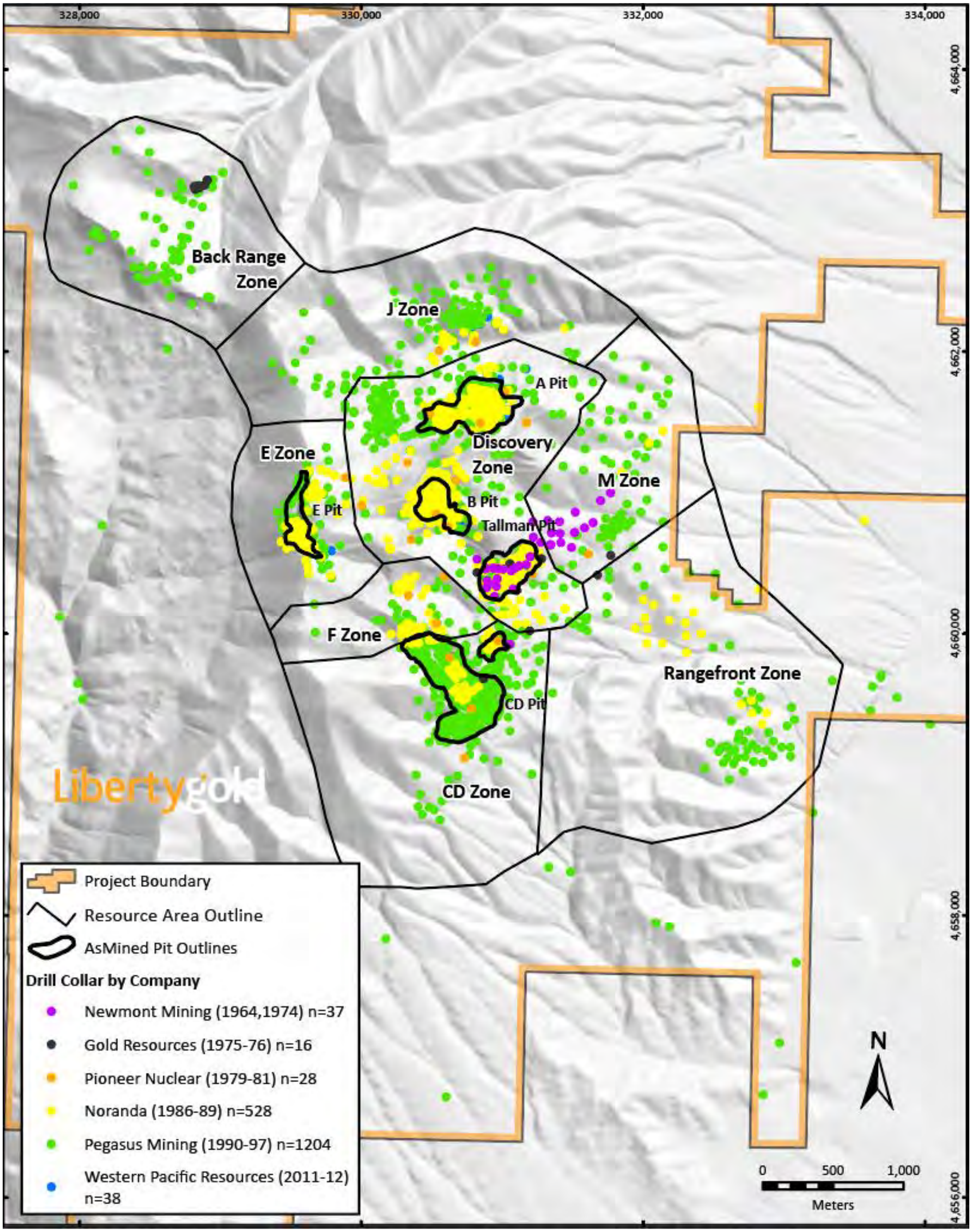


Figure 6-3: Historical Drill Holes

The QP is not aware of the details regarding the drilling contractors, drilling methods, sampling procedures, collar-survey methods, and types of drill rigs utilized in the historical Black Pine drilling programs other than those summarized below.

The historical drilling discovered and defined gold mineralization that was eventually mined from seven historical open pits. These pits produced approximately 435,000 ounces of recovered gold from a little more than 30 Mt of ore between 1991 and 1997. The pits lie within mineralized zones of various sizes, and only a portion of each mineralized zone was mined (Figure 6-4). Table 6-2 summarizes the size, average drilled grade, highest-grade drill hole assay, and best gold intersection in terms of grade multiplied by thickness from each of the mined mineralized zones.

Table 6-2: Summary of Mined Gold zones and Drill Highlights
(Liberty Gold Corp. – Black Pine Project)

Gold Zone	Length (ft)	Width (ft)	Depth (ft)	Avg Mined Grade (g/t Au)	Highest-Grade Drill Assay (g/t Au over 1.5 m)	Highest Grade x Thickness Intercept[1]
E	575	100	75	1.5	46.7	19.81 m @ 16.09 g/t Au
B	350	300	100	1.38	38.26	73.16 m @ 3.24 g/t Au
A	650	350	100	0.6	8.57	96.0 m @ 1.03 g/t Au
Tallman	350	200	120	0.9	11.31	50.3 m @ 1.76 g/t Au
C/D	800	250	100	0.58	25.27	103.6 m @ 0.83 g/t Au

Notes:
1. Intervals reported at 0.2 g Au/t cut-off.
2. Width is intercept width and may not portray true thickness

Several additional zones were identified by the drilling but were not mined, such as the A Basin, J Anomaly, and Rangefront Anomaly (Figure 6-4).

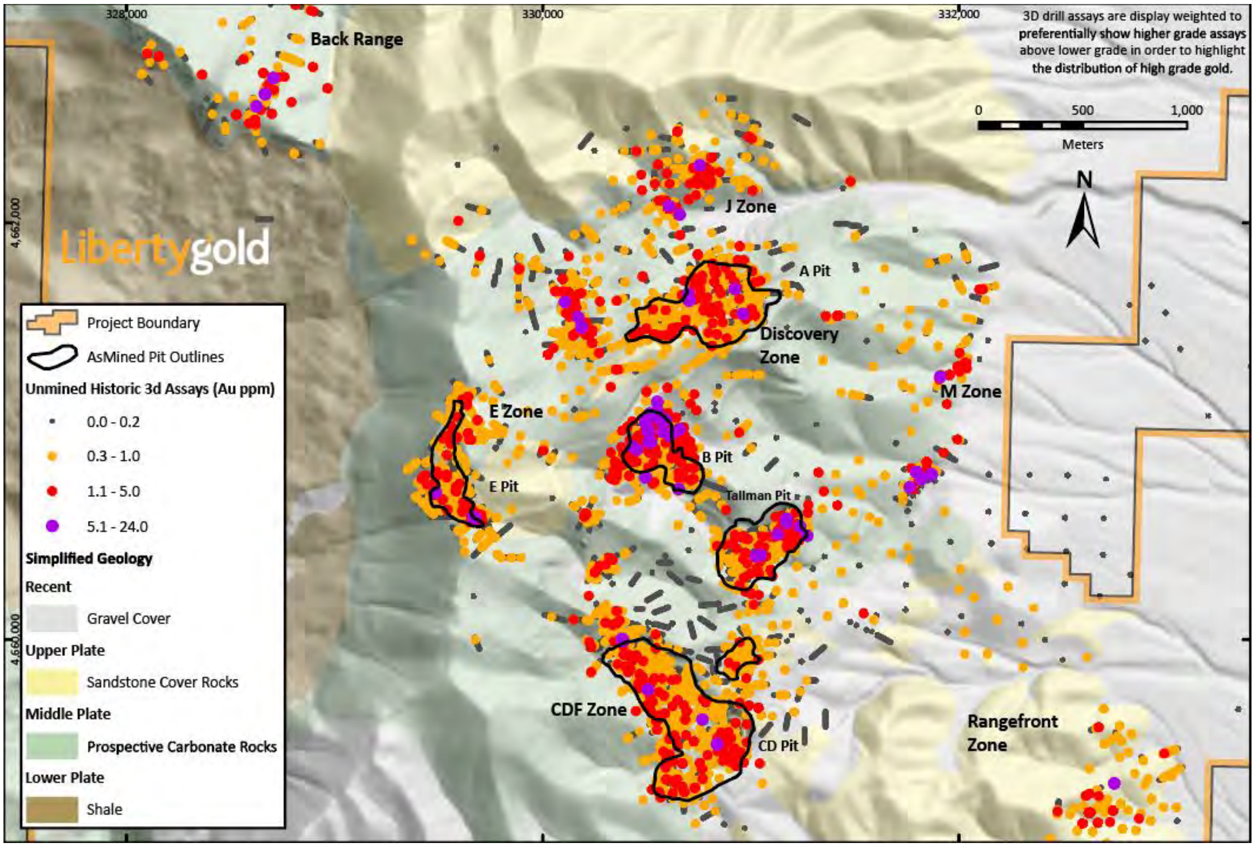


Figure 6-4: Historical Pits and Unmined Gold in Drill Intervals

6.5.2 Newmont 1964 and 1974

Newmont drilled 17 rotary holes of uncertain type (RC or conventional rotary) in the area of the historical Tallman mine in 1964 and collected drill-chip samples over 1.52-metre (5-foot) intervals. Drilling was carried out by Sprague and Henwood, Inc. with 14.3-centimetre tricone and 12.1-centimetre hammer bits. Some or all of these holes were drilled with a truck-mounted Portadrill No. 753. Drilling was done both wet and dry; difficult drilling conditions were commonly noted. Newmont concluded the results of the drilling program “did not indicate sufficient strength of mineralization to encourage us to look further for an orebody” (Hardie, 1964) and terminated its interest in the property.

Newmont reacquired the property and in 1974 drilled 20 rotary and/or RC percussion holes. Eklund Drilling (“Eklund”) of Elko, Nevada was the contractor, and samples were collected over 1.52-metre intervals. Drilling was carried out to the northeast of the historical Tallman mine, exploring for a possible extension or offset of mineralization. An unknown quantity of drill hole collar locations, but greater than five, were later surveyed by Desert West Land Surveys at the direction of Noranda.

6.5.3 Gold Resources and Permian 1974-1976

A total of 13 RC or rotary holes and three core holes, for a total of 1,218 m, were drilled by Gold Resources and Permian. Udy Core Drilling of Leadore, Idaho carried out the core drilling and Drilling Services International carried out the RC drilling for Gold Resources and Permian, but no other information is available.

6.5.4 ASARCO 1977

ASARCO drilled 34 “percussion” holes, mostly at the E Zone in the area of the top of Black Pine Cone Peak, with several holes west of Anomaly A and into A Basin in the Discovery Zone. ASARCO abandoned their interest in the property in 1978. No other information is available.

6.5.5 Pioneer Nuclear 1979-1981

Pioneer Nuclear drilled 28 RC drill holes for a total of 2,458 m in 1979 through 1981, of which 13 holes intersected gold grades more than 0.55 g Au/t in at least one sample interval. Samples were collected variably on 1.52 to 3.05 m intervals. An unknown number of collar locations were later surveyed by Desert West Land Surveys in the direction of Noranda.

6.5.6 Permian and Pegasus 1983-1986

The Permian Pegasus joint venture drilled 88 holes in 1983. At least some of the holes were sampled over 6.1 m intervals. In 1984, 35 RC holes were drilled and sampled at 1.52 m intervals. One core hole was also drilled. Drilling was carried out principally in the Discovery Zone, defining gold mineralization that would later be mined in the B and B Extension pits. Drill-hole collar locations were surveyed by plane table by Ron Willden, with an unknown quantity of collar locations later surveyed by Desert West Land Surveys of Burley, Idaho, at the direction of Noranda.

6.5.7 Noranda 1986-1990

Noranda drilled 536 RC holes and four metallurgical core holes over four years, for a total of 51,611 m. Typically, one or two truck- or track-mounted RC drill rigs were utilized, depending on access road conditions. Some holes were drilled dry. and others were drilled with water injection.

Boyles Bros. Drilling Company of Salt Lake City was the drilling contractor in 1987 for PQ-size core drilling. Eklund was the contractor for most of the 1988 drilling, with some RC drilling by Hard Rock Mineral Drilling Company of Fort Collins, Colorado at the end of the year. Dateline Drilling Inc. (“Dateline”) of Missoula, Montana provided some RC drilling in early 1989, followed by Modern International Inc. of Elko, Nevada, who used a track-mounted RC rig used for most of their 1989 drilling.

The locations of Noranda’s holes, as well as some holes drilled earlier by other operators, were surveyed by Desert West Land Surveys of Burley, Idaho using a Lazer Theodolite survey instrument; Grey Eagles Surveys also surveyed some collars.

6.5.8 Pegasus 1990-1997

Pegasus drilled 51 holes in 1990, 88 holes in 1991, 237 holes in 1992, 284 holes in 1993, 240 holes in 1994, 103 holes in 1995, 73 holes in 1996, and six holes in 1997, for a total of 116,448 m. All were drilled with RC methods, except for 16 core holes. Samples were collected over 1.52 m intervals and assayed at the Black Pine mine laboratory. Little information is available about drill contractors used by Pegasus. Dateline and Hackworth Drilling Inc. of Elko, Nevada drilled the RC holes in 1992 and 1993. In 1995, O’Keefe Drilling Company of Elko, Nevada drilled wet RC holes using 14-centimeter (cm) hammer bits, and 13.7 cm tricone bits.

6.5.9 Western Pacific 2011-2012

Western Pacific drilled 38 RC holes in two campaigns, for a total of 7,920 m. Drill logs and RC chip trays are available for holes 1 to 31, but logs for holes 32 to 35 are missing from the data files. Holes 36, 37, and 38 were not logged and no assay data for these holes are available. After completion of the holes, the collars were marked with stamped brass tags fastened onto a steel wire, and their locations were surveyed by an unknown method.

The drilling was conducted by Envirotech Drilling LLC of Winnemucca, Nevada. All drill samples were collected at the rig using a wet splitter.

6.5.10 Summary Statement – Historical Drilling

The predominant down-hole length of the historical drill samples in the resource database is 1.52 m (5 feet), with 96% of the historical sample intervals with gold analyses in the resource database having this length. Other sample intervals are predominantly 3.05 m (10 feet) or 6.1 m (20 feet) (<2% each), with the small percentage of the remaining intervals having varied lengths.

Although surveys of the historical drill hole collars are limited, Liberty Gold has carefully checked the transformed database locations against historical aerial photos and drill hole plan maps. MDA (Gustin et al, 2021) completed similar checks. Two holes were found to be mislocated, and these were corrected. While the locations of some of the historical holes in the resource database undoubtedly have errors, the verification steps undertaken by Liberty Gold serve to significantly limit the magnitude of these potential inaccuracies.

There is no down-hole survey data in the resource database for the historical holes. This lack is not unusual for projects drilled prior to the late 1990s, especially considering the shallow depths of most of the drilling. Only 5% of the historical holes were drilled at angles of -75° or shallower, with a similar percentage having down-hole depths in excess of 150 m. Hole deviations are typically limited for near-vertical holes, as well as for holes drilled to shallow depths.

Historical references to “percussion drilling” are not clear as to whether the holes were drilled by RC or conventional rotary methods. While both drilling methods can experience down-hole contamination issues, RC can be superior to conventional rotary under certain drilling conditions. This topic is further discussed in Section 10.0.

The QP is unaware of any drilling, sampling, or recovery factors that could materially impact the use of the historical drill-hole data in the estimation of the current project Mineral Resources.

6.6 Historical Resource and Reserve Estimates

Several estimations of mineralized material at Black Pine were carried out by historical operators, only a few of which are summarized herein. Most of the mineralized material included in these historical estimates was subsequently mined. A qualified person has not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves. As such, Liberty Gold is not treating these historical estimates as current Mineral Resources or Mineral Reserves, and the estimates should not be relied upon.

The classification terminology is presented as described in the original references. It is not known if this terminology conforms to the meanings ascribed to the Measured, Indicated, and Inferred Mineral Resource classifications, or the Proven and Probable Reserve classifications of the Canadian Institute of Mining, Metallurgy and Petroleum’s (CIM) “CIM Definition Standards for Mineral Resources and Reserves,” adopted on May 19, 2014 (the “CIM Standards”). All of the estimates were originally reported in Imperial units of measure, and these units are retained for historical accuracy.

6.6.1 Noranda Historical Reserve Estimates

Prochnau (1985) carried out a “reserve estimate” for Noranda in the course of evaluating the Black Pine property for potential acquisition. The Tallman Pit area was divided into three zones (Tallman Pit, South Ore Body, and North Ore Body). Using a polygonal estimate with a cut-off grade of 0.03 oz Au/ton, a tonnage factor of 13 ft^3/ton, and no dilution, Prochnau (1985) estimated "reserves" of 393,700 tonnes (434,000 tons) at a grade of 0.068 oz Au/ton. Other key assumptions, parameters, and methods used to prepare this estimate are not known. The classification of these reserves differs from the CIM Standards, but the extent and nature of these differences is not known. This estimate provides some information as to the potential grade of small quantities of mineralized material at Black Pine. A qualified person has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves, and Liberty Gold is not treating this historical estimate as current Mineral Resources or Mineral Reserves. This historical estimate should not be relied upon. Noranda outlined "reserves" for various deposit areas at Black Pine as summarized in Table 6-3 (Noranda, 1989).

Table 6-3: Mid-**1989 Noranda "Reserves"**

Deposit	Classification	Tons	Grade (opt)	Gold (ounces)
Tallman, A, B	Proven	5,357,000	0.040	163,200*
Tallman, A, B	Probable	1,016,000	0.040	30,900*
C, D, E	Drill Indicated	3,597,000	0.057	155,900*
A-west	Proven**	2,753,000	0.025	68,800*
G, A-south, J	Drill Indicated	2,381,000	0.035	62,800*
Total		15,094,000	0.040	481,600*
Total in-ground				633,700

Notes:
* = recoverable gold ounces
** = "sub-economic"

6.6.2 Pegasus Historical Reserve Estimates

Pegasus produced a number of estimates of "reserves", "mineralized material", and "additional mineralized material" from 1991 through 1996 as summarized in Table 6-4. The key assumptions, parameters and methods used to prepare these estimates are not known. The classification of these estimates differs from the CIM Standards, but the extent and nature of these differences is not known. These estimates are not subject to upgrading to become current Mineral Resources or Mineral Reserves, because a significant quantity, if not all, of the estimated materials were subsequently mined out. These estimates are relevant because they represent Pegasus' estimates of Black Pine mineralization during the historical open-pit mining operation. A qualified person has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves. Liberty Gold is not treating these historical estimates as current Mineral Resources or Mineral Reserves, and the estimates should not be relied upon.

Table 6-4: 1990s Pegasus Historical Reserve Estimates

Area	1991		1992		1993		1994		1995		1996	
	oz	opt Au	oz	opt Au	oz	opt Au	oz	opt Au	oz	opt Au	oz	opt Au
Tallman Pit	68,487	0.023										
B Pit	88,731	0.036	47,826	0.048	18,876	0.036						
A Pit	181,345	0.019	181,345	0.02	249,840	0.018	49,667	0.19				
E Pit			67,655	0.07	58,770	0.057	50,981	0.072				
B Extension			39,471	0.023	25,106	0.026						
C/D			125,600	0.022	61,600	0.027	155,700	0.016	94,768	0.015	6,539	0.014
A Basin			50,500	0.03								
J Anomaly					20,300	0.025						
I Pit									21,410	0.014		
NE Tallman											26,320	0.017
Internal Documents	338,563	?	336,297	0.026	415,809	0.02	256,358	0.019	116,178	0.015	32,859	0.0165
Annual Report			336,297	0.026	346,000	0.018	256,000	0.019	116,000	0.015	29,959	0.017
Mineralized Material[1]	271,839	0.023	242,878	0.022	32,950	0.023	41,000	0.016			24,702	0.0135
Mineralized Material[1] + Addl. Mineralized Material[2]	271,839	0.023	242,878	0.022	32,950	0.023	421,000	?	420,417	0.013	443,802	0.013

Notes:

1. Mineralized Material defined as "within a floating cone or whittle pit that is not included in the current mine plan, or that needs better sampling to better define the zone."
2. Additional Mineralization defined as "all material within the computer block model at the measured/indicated level of geologic confidence but outside the current defined pits used for reserve definition. At Black Pine, some of this mineralization is surrounding mined-out pits and has a very low chance of becoming a future reserve." (Pegasus Gold Interoffice Memorandum, January 23, 1997)

In February 1997, late in the Black Pine mine life, "reserves" were estimated to be 1.8 Mt with a grade of 0.58 g Au/t, with "additional mineralized material" that totaled 1.7 Mt at a grade of 0.46 g Au/t (Metals Economics Group Report, 2012, quoting a 2/19/97 Pegasus press release). Key assumptions, parameters, and methods used in these historical estimates are not known. The classification of these reserves differs from the CIM Standards, but the extent and nature of these differences is not known. These estimates are considered relevant because they represent the mine operator's estimation of materials that remained very late in the mine life. SLR has not done sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves, and Liberty Gold is not treating these historical estimates as current Mineral Resources or Mineral Reserves. These estimates should not be relied upon.

6.6.3 Estimate of Remaining Gold in the Historic Heap Leach Pad

Powell (2012a) of Tetra Tech, prepared an internal report for Western Pacific Resources, with an estimate of the remaining gold in the historic heap leach pad, based on production numbers (tonnage placed on the pad, head grade, recovery, total ounces produced, etc.) provided by Pegasus in annual reports and other sources. Powell's (2012a) summary states *"the gold content remaining on the Black Pine reclaimed leach pad is estimated at 243,543 ounces. It is possible the above estimated metal content could be mis-calculated either high or low by 10 to 15%. This could have occurred through mining dilution, assay error, sample error, sample losses, or survey error. The writer has no clear understanding of the ore control procedures practiced at the mine during operations. Typical of the time would be some sort of blast hole assaying (fire assay) on 12 – 20-foot hole spacing and similar depth. This writer was unable to discover any geologic block model to reconcile back to the actual mine model (plan). Given the low-grade nature of the ore body, it certainly was critical for the operators to accurately forecast and mine the ore zones as efficiently as possible. Tetra Tech recommends that given the limited historic column heap leach testing, the future heap leach recovery be set at no higher than 25%. This is, in no way, anything more than an educated guess by the author at this time based on very limited information currently available."*

Powell (2012a) further estimates that 50,899 recoverable ounces may remain in the historic leach pad.

A qualified person has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves, and Liberty Gold is not treating these historical estimates as current Mineral Resources or Mineral Reserves. This estimate should not be relied upon.

Additional work to identify current Mineral Resources in the historic leach pad would include RC and Sonic drilling and column testing to identify what portion, if any, of the gold remaining in the leach pad is recoverable.

These Mineral Resource estimates are historical in nature, however, they are relevant as they indicate the mineralization on the Project. It is important to note that these historical Mineral Resources have been superseded by subsequent Mineral Resource estimates and that Liberty Gold is not treating any of these Mineral Resource estimates as a current Mineral Resource estimate.

6.7 Past Production

The Silver Hills, Ruth, Mineral Gulch, and Hazel Pine mines, all within the current property boundary, were located along the eastern edge of the Black Pine Mountains and operated between approximately 1915 and 1920, with the Silver Hills mine producing until 1932. Production was mostly on the order of a few tens to hundreds of tons from veins containing quartz, tetrahedrite, sphalerite, jamesonite, pyrite, and oxides of copper, zinc, antimony, and iron (Anderson, 1931; Brady, 1984).

According to Prochnau (1985), the Virmyra Mining Company operated the Tallman pit from 1949 through 1955. Gold production from this operation was estimated to be 98,884 tonnes (109,000 tons) with an average gold grade of 5.14 g/t Au (Hefner et al., 1991). The rock was treated by cyanide vat leaching. The tailings from this operation contained an estimated 0.026 oz Au/ton (0.89 g/t Au), indicating recoveries of approximately 80% (Prochnau, 1985).

After acquiring the Black Pine property from Noranda in mid-1990, Pegasus constructed a cyanide heap-leach pad and gold recovery plant and began extraction of mineralized material from the Tallman pit in October 1991 (Pegasus 1993 Annual Report). The first gold was poured on January 9, 1992. Pegasus subsequently mined five additional pits through 1997. Material was mined from the open pits at a rate of approximately 37,000 tons (33,600 tonnes) per day and ROM was placed on a multiple-lift, valley-fill leach pad. Gold was recovered using carbon adsorption and doré bars were produced after solvent electrowinning. Approximately 26.5 Mt of waste rock and 31 Mt of ore were mined between 1991 and 1997 (Sawyer, 1997).

Mining ceased at Black Pine in late 1997 and the heap-leach pad was subsequently rinsed and the surface was reclaimed (Sawyer, 1997; Powell, 2012a). Table 6-5 summarizes the production reported by Pegasus in annual reports and SEC filings, which differ slightly from similar information found in other reports (e.g., Pegasus internal reports, Intierra website, Sawyer, undated).

In January 1999 Pegasus Gold and its subsidiary company Black Pine Mining Inc filed for Chapter 7 bankruptcy and there was a voluntary reclamation bond forfeiture. Following the signing of an agreement with state and federal agencies regarding the reclamation status a trustee was appointed, and the remaining reclamation was completed. During the period from 1999 to 2002, under the direction of the bankruptcy trustee, additional leaching of the heap ramp was completed with proceeds from gold production providing additional funds for the site reclamation. Reliable records of gold production during this time are not available.

Table 6-5: 1990s Production Summary of the Black Pine Mine

	Units	1992	1993	1994	1995	1996	1997	1998	Totals
ROM ore mined[1]	kt	2,850	3,270	5,810	7,050	8,730	2,650	-	30,360
Stripping ratio[1]		-	1.3	1.16	1.16	0.98	2.43	-	1.13
Average gold grade[1]	g/t Au	0.55	0.82	0.69	0.72	0.52	0.55	-	
Gold recovery percentage[1]	%	-	80%	54%	59%	60%	61%	-	
Gold to heap leach[2]	oz	109,080	88,438	130,270	164,316	147,186	26,320		665,610
Gold recovered[1]	oz	48,700	66,100	65,700	108,500	87,900	44,100	13,800	434,800
Calculated gold recovery	%								65%
Silver recovered[1]	oz	14,900	28,600	39,100	59,300	31,000	16,200	-	189,100

Notes:
1. from Pegasus Gold Annual Reports, SEC Form 10-K filings, and BPMI closure report by Sawyer et al.
from Pegasus Gold internal yearly production statements

7 GEOLOGICAL SETTING AND MINERALIZATION

The information presented in this section of the Technical Report is derived from multiple sources, as cited.

7.1 Regional Geology

The Black Pine property is located in the northeastern portion of the Basin and Range physiographic province, near the late Proterozoic rifted continental margin of North America. Rifting was followed by late Proterozoic and early Paleozoic subsidence, and accumulation of a thick sequence of continental margin siliciclastic and carbonate rocks ranging from near-shore sandstone and shale to offshore carbonate reef and lagoonal deposits (e.g., Cook, 2015).

Beginning in the middle of the Paleozoic era, plate collisions from the west led to a series of intra-plate contractional orogenic events, starting with the emplacement of the Roberts Mountains allochthon ("RMA") in Late Devonian and Early Mississippian time. Although the RMA is located to the west of the Black Pine Mountains, it shed siliciclastic material into a foreland basin that stretched across much of what later became the eastern Great Basin, defined as the hydrographic region across the western United States that has no hydrologic connectivity to the ocean, including portions of Nevada, Oregon, Utah, California, Idaho, and Wyoming (e.g., Hintze, 1991).

In Pennsylvanian time, the Humboldt orogeny (Theodore et al., 1998) affected areas to the west of Black Pine. In the Middle to Late Jurassic, much of the area along the Nevada-Utah border was affected by an orogenic event known as the Elko orogeny, characterized by thrusting and attenuation faulting, with local areas of low-grade metamorphism (Thorman and Peterson, 2004). It is not clear whether some of the folding seen at Black Pine can be attributed to this orogenic event, although the presence of a phyllitic cleavage locally in sheared Mississippian strata indicates that some rocks were affected by low grade metamorphism.

Subsequently, the Late Cretaceous Sevier orogeny resulted in development of widespread, primarily thin-skinned, east-vergent folds and thrust faults throughout the eastern Great Basin (e.g., DeCelles, 2004). There is some evidence that the Laramide orogeny may also have affected this region in latest Cretaceous-Paleocene time.

In the early Eocene, contractional deformation gave way to extensional deformation and intermediate to felsic volcanism across the Great Basin. Throughout most of the Cenozoic, extension involved movement along low-angle normal faults, with up to 100 km of offset. Listric normal faults associated with these low-angle normal faults have tilted hanging wall strata as young as Miocene in age, generally in an eastward direction (e.g., Mueller et al, 1999).

The Black Pine Mountains lie in the hanging wall of the Raft River-Albion metamorphic core complex, located approximately 20 km to the southwest. In this area, high-grade metamorphic rocks are separated from weakly or unmetamorphosed strata along a series of low angle detachment faults with top to the east displacement and likely tens of km of movement. The Black Pine Mountains are interpreted to lie in the hanging wall of one of these faults (Konstantinou et al, 2012). The faults were active between approximately 14 and 8 million years ago, thus likely post-dating gold mineralization.

The latest manifestations of extension are "Basin and Range" style block faults that divide the Great Basin into its characteristic horsts and grabens. Some of these faults are still active today.

The Black Pine Mountains are predominantly underlain by Devonian to Permian sedimentary rocks, some of which are weakly metamorphosed. These occur in two major structural blocks, separated by a fault which transects the range from southwest to northeast (Figure 7-1). The southern block, which includes the Black Pine Project, consists largely of structurally interleaved members of the Permo-Pennsylvanian Oquirrh Group, including limestone, sandstone, dolomite, and siltstone. The Oquirrh Group is a regionally significant unit that hosts mineralization elsewhere in the northeastern Great Basin, for example in the Bingham Canyon District (Shaddrick et al., 1991; Hintze, 1991). It is described in more detail below.

The southern block can be divided into three structural plates, bounded by low angle faults (Figure 7-1 and Figure 7-2). The lowest plate comprises the Devonian Jefferson Formation and the Upper Mississippian-Lower Pennsylvanian Manning Canyon Shale, the latter of which was deposited in the Antler orogenic foreland basin. The middle plate consists of structurally interleaved members of the Oquirrh Group, including silty/sandy limestone and minor dolomite, variably calcareous sandstone, siltstone, and quartzite, and it is of primary interest as a host rock for gold mineralization. The upper plate consists primarily of sandstone and siltstone of the upper portion of the Oquirrh Group. The lowermost plate is believed to structurally overlie a basement of weakly metamorphosed rocks of suspected Cambro-Ordovician age (Smith 1982; Figure 7-2).

The northern block is comprised of two thrust plates. The lower thrust plate consists of four informally-named stratigraphic units, ranging from Late Pennsylvanian to Early Permian in age, probably corresponding to the upper portion of the Oquirrh Formation. The upper plate consists of limestone and silicified limestone of Early Permian age.

Igneous rocks are widespread but not abundant in the Black Pine Mountains. The Paleozoic rocks have been intruded by narrow, altered, intermediate to mafic dikes and sills. Tertiary ash-flow tuff and a rhyolitic flow-dome overlie the Paleozoic rocks outside the property boundary (Smith, 1982; Brady, 1984).

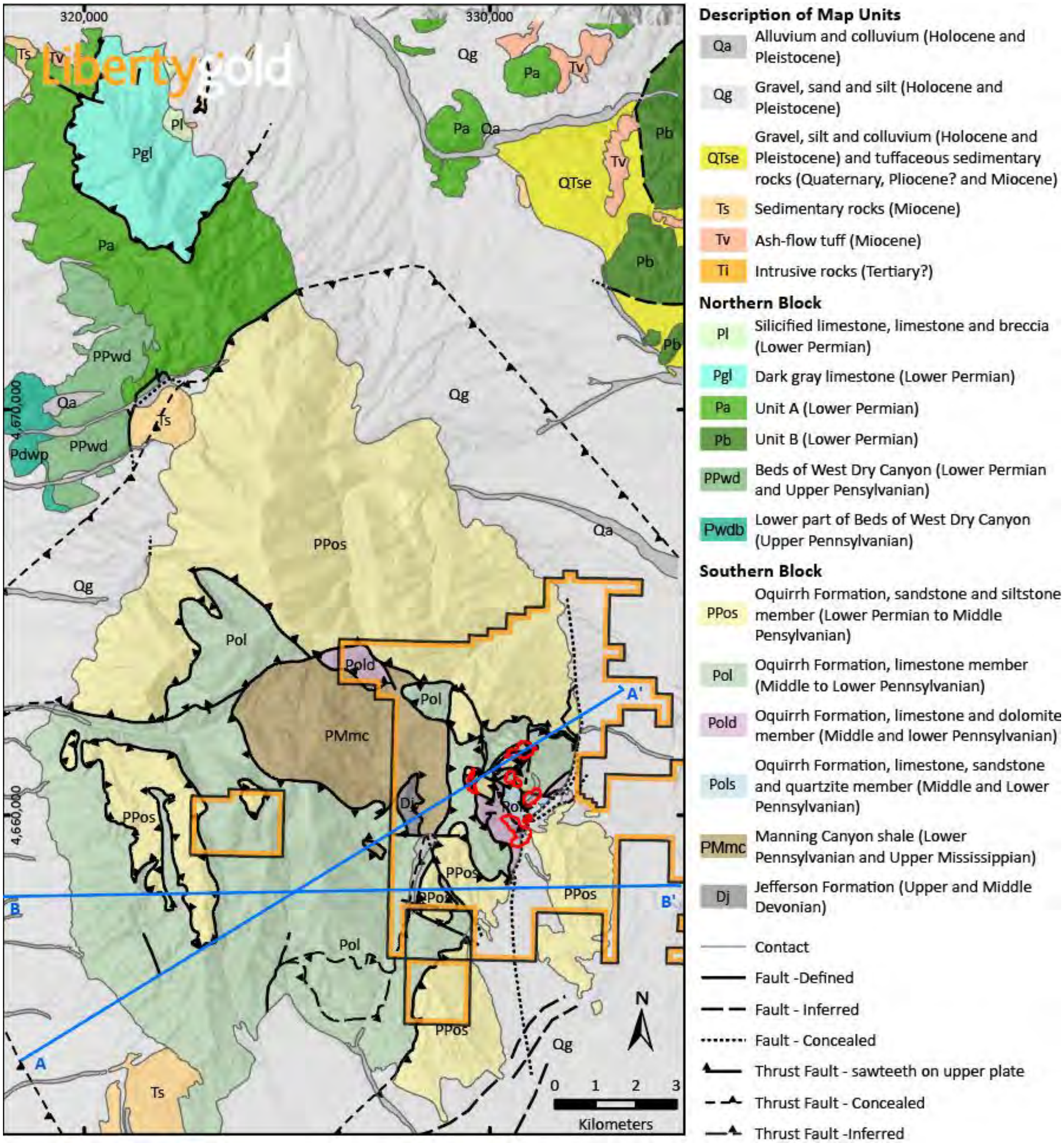


Figure 7-1: Regional Geology

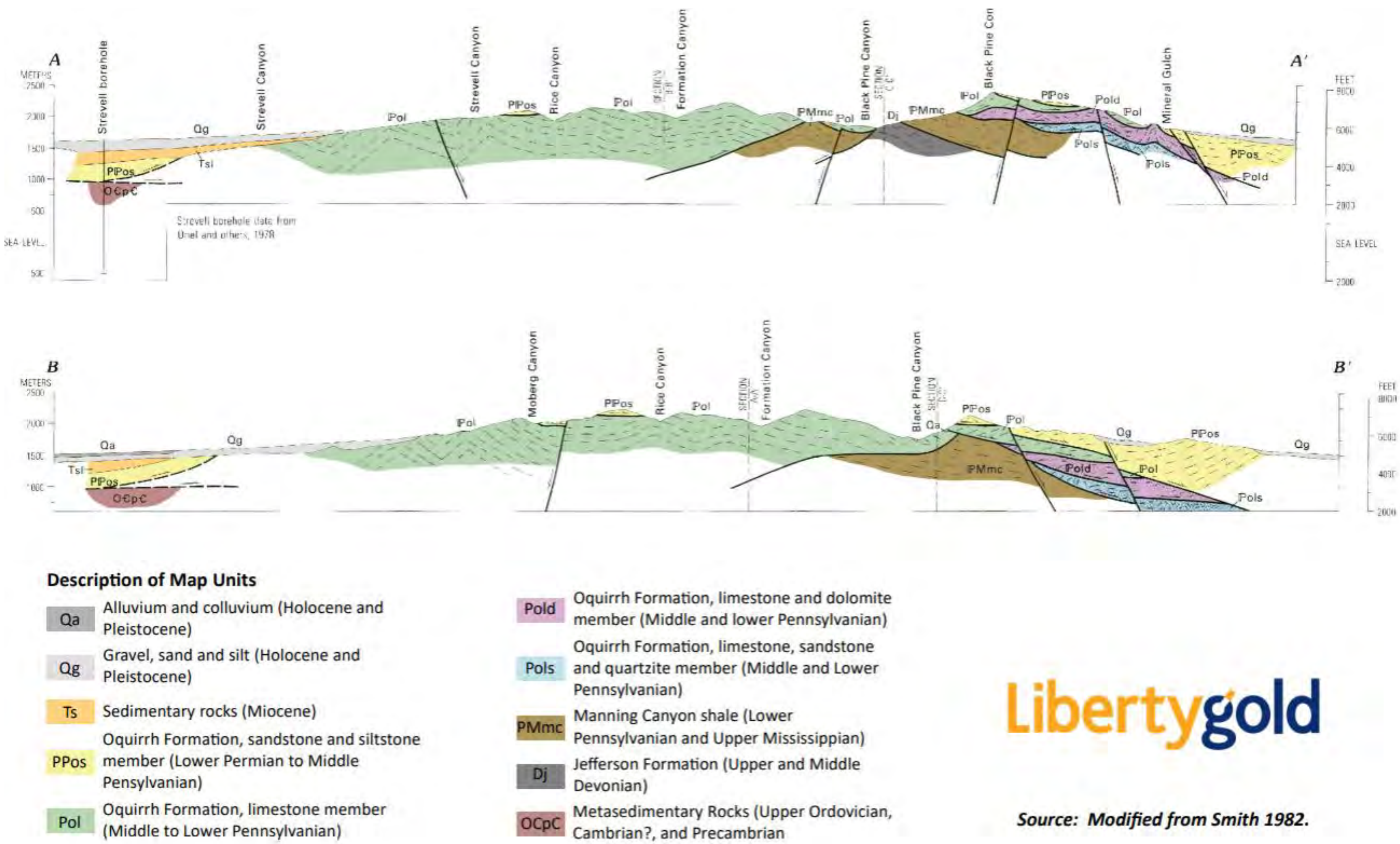


Figure 7-2: Schematic Cross Sections through the Black Pine Mountains

7.2 Property Geology

The Black Pine property is located within the southern structural block of the Black Pine Mountains where exposures consist of the lower plate units of the Jefferson Formation and Manning Canyon Shale, along with middle and upper plate units of the Oquirrh Formation, including weakly metamorphosed limestone and dolomite, silty and sandy limestone, calcareous sandstone and siltstone, quartzite, and shale Figure 7-3).

The pre-Cenozoic strata shown in Figure 7-3 are strongly folded and cut by faults. Virtually all contacts between formations and units are interpreted or observed to be fault contacts (Smith, 1982, 1983; Smith et al., 2020), making construction of a true stratigraphic sequence for the Project area problematic, although fossil data do loosely constrain ages of the units (Smith, 1982, 1983).

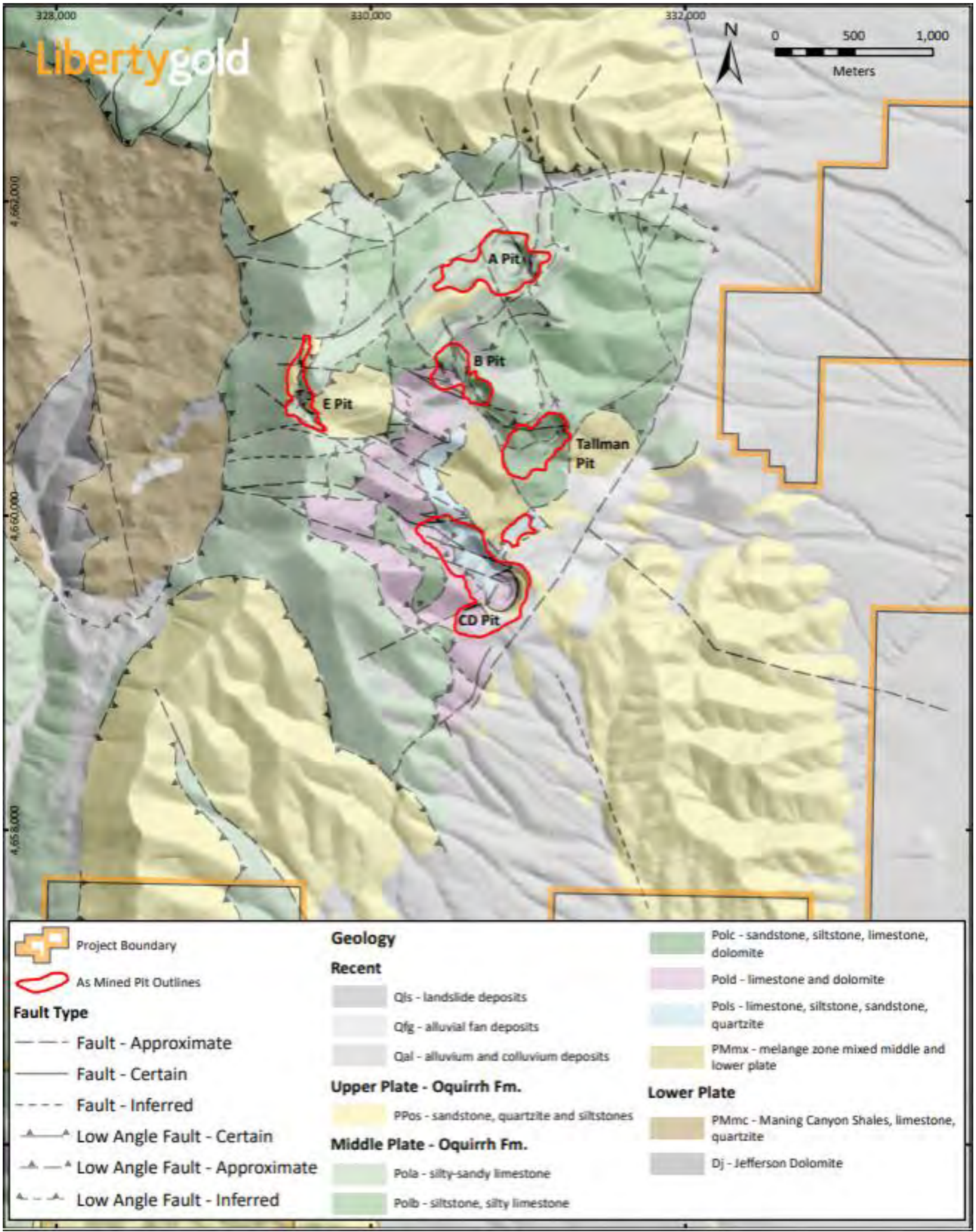


Figure 7-3: Geologic Map

7.2.1 Stratigraphy

The stratigraphy in the Project area records the transition from the top of the Devonian shelf and platform, through foreland-basin sedimentation associated with the mid-Paleozoic Antler orogeny, to basin and platform conditions that persisted throughout much of the late Paleozoic era. Figure 7-4 illustrates a simplified stratigraphic section for the property based on Liberty Gold mapping and drilling.

Jefferson Formation (Dj): Strata assigned to the Jefferson Formation comprise the oldest stratigraphic unit exposed in the Project area (Smith, 1983). It is Devonian in age, and consists of dolostone with minor sandstone and quartzite, representing very shallow water to intertidal conditions on the inner shelf, with some contribution of siliciclastic material from highlands to the east. It is found in the lower structural plate in the lowest-elevation areas in Black Pine Canyon in the western part of the property.

Manning Canyon Shale (PMmc): Strata assigned to the Manning Canyon Shale consist of up to 2,000 m of recessive-weathering, carbonaceous, dark grey to black argillite, shale, and siltstone with minor quartzite and limestone. It is Late Mississippian in age in the Black Pine area (Smith, 1982). The Manning Canyon Shale formed in response to emplacement of the RMA over areas to the west, reflecting foreland-basin sedimentation. It is present on surface in the lowest structural plate in the western part of the property, as well as in an area south of the historical Tallman and B pits. The upper contact of the Manning Canyon Shale is an important marker horizon in drilling, as it generally marks the bottom of the gold-bearing middle plate. In many locations, a weak phyllitic cleavage is present in shales along the upper (faulted) contact.

Oquirrh Group: The Oquirrh Group reflects complex sedimentation patterns established over a long period of time with terrigenous input into a shallow basin and carbonate platform setting. Rocks assigned to the Oquirrh Group are present over much of the northwestern part of Utah and locally into southern Utah. In more well-studied portions of the Oquirrh Group, thicknesses and rock types vary significantly between adjacent mountain ranges, as well as between thrust sheets. In general, however, it consists of a lower Pennsylvanian unit dominated by limestone, a middle Pennsylvanian unit that is a mixture of quartz sandstone, shale, and limestone, and an upper Pennsylvanian/lower Permian unit dominated by quartz sandstone. These have been divided into a number of formations and members, depending on location.

The Oquirrh Group may range up to 5,000 m thick in the Black Pine area, although interleaving and attenuation of the section by low-angle faults makes stratigraphic analysis extremely difficult. In the southern structural block in the Black Pine Mountains, Smith (1982), Loptien (1986), and Ohlin (1989) divided the Oquirrh Group into four informal members, three in the middle structural plate and one in the upper plate. Given that the rock descriptions and ages are overlapping, and the rocks are complexly interleaved along faults in and between the middle and upper structural plates (see Section 7.2.2), they may, in part, represent age-equivalent packages of rock that were subsequently brought into juxtaposition by faulting (Shaddrick, 2013).

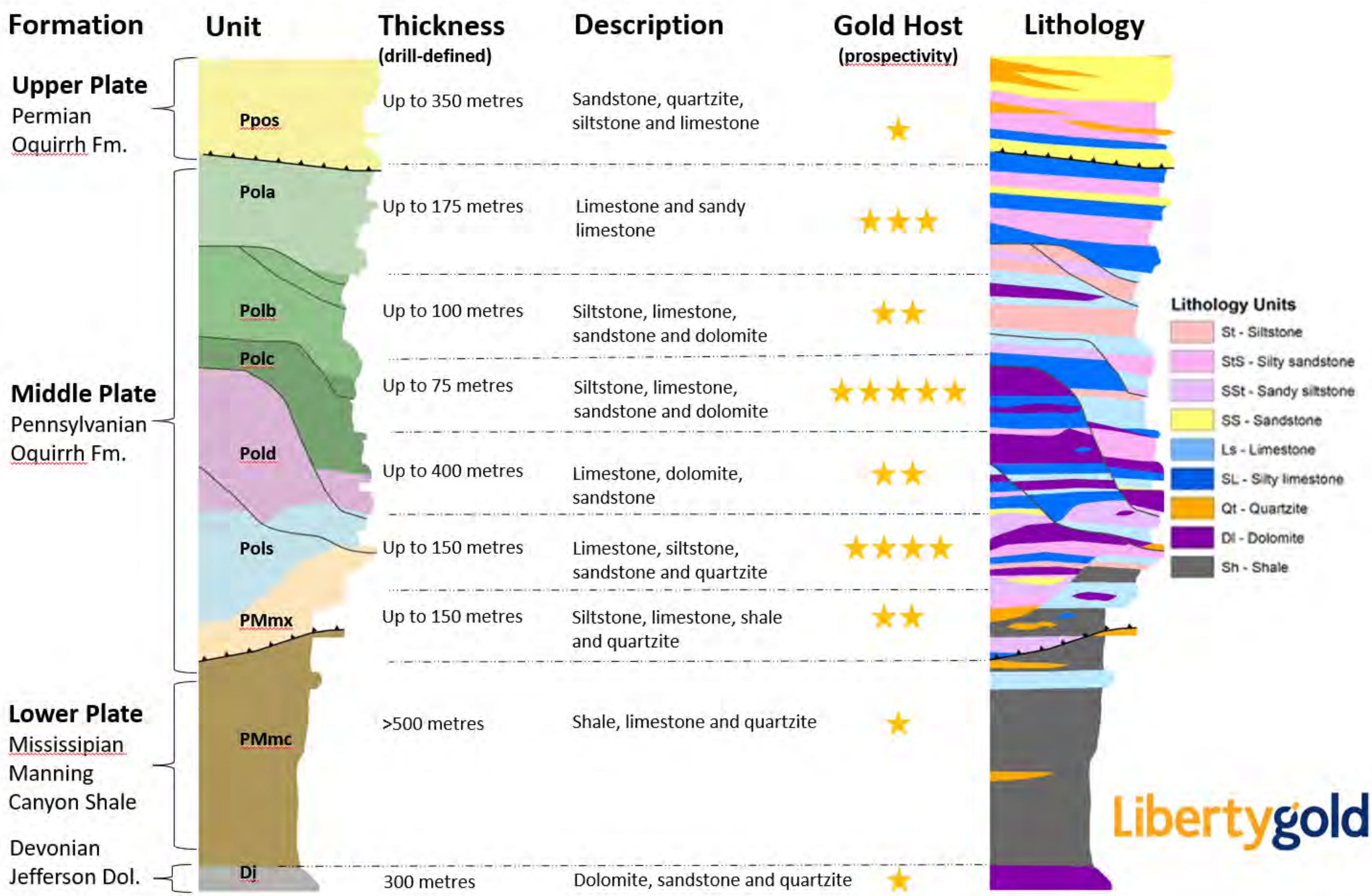


Figure 7-4: Stratigraphic Column

Pol - Limestone Member: The Limestone Member of the Oquirrh Group is the thickest and most widespread of the three members of the middle structural plate. It forms the structural upper member of the middle plate and consists of a diverse assemblage of carbonate rocks, shale, siltstone, and sandstone. This unit may be overturned (Smith, 1982) or may be stratigraphically continuous with the underlying Pold (Hefner et al., 1991). It is distinguished from the middle member of the middle plate by the first appearance of siltstone or sandy siltstone with interbedded limestone lenses. In the northeastern portion of the Black Pine Mine area, Liberty Gold recognizes the following submembers of the Pol unit, from stratigraphic/structural top to bottom of the sequence:

- Pola: Pola consists largely of medium gray, sandy limestone whose sandy texture is easily differentiated from Polb but harder to differentiate from overlying limy portions of the PPos. This unit has a “dirty” appearance due to the irregular mottled oxidation pattern of detrital sand grains in the limestone. Pola is thickly to massively bedded, with silty interbeds and rare dolomitic beds. In pit walls, the massive bedding and alternating thin interbeds are readily apparent. Black, recrystallized wavy calcite veins are common. This unit is typically highly fractured and brecciated. A 15-20 m-wide fault and breccia zone normally separates this unit from the underlying Polb. The contact with the overlying PPos unit is often faulted, but may be stratigraphic, putting the Pola in the upper structural plate, rather than the middle plate.

- Polb: Polb consists mainly of calcareous to non-calcareous siltstone with thick to massive beds and lenses of limestone and dolostone. The massive limestone and dolostone beds are poorly understood, possibly they are faulted blocks from Pola or Pold or discontinuous lenses deposited in Polb. The non-calcareous siltstone is often a pale pinkish tan color and strongly sheared. Overall, this unit contains more evidence of ductile deformation than Pola, Polc, or Pold (see structural geology section below). Where less affected by structural deformation, Polb can host homogenous intervals of siltstone up to 75 m thick.

- Polc: Polc is the basal unit of the Pol, a member of the Oquirrh Group. Polc can include nearly all lithological types in the middle plate, including calcareous siltstone, limestone, dolomite, and sandstone, often in alternating, 1-10 m-thick beds. However, the dominant rock type is a brownish, massive calcareous siltstone. Polc is a receptive host for decalcification and hosts some of the highest grades had Black Pine.
- Pold – Limestone and Dolomite Member: This middle member of the middle structural plate is characterized by thick-bedded to massive, cliff-forming silty to sandy limestone and dolostone, limestone breccia, and local beds of sandstone and siltstone. The contact with the overlying Pol appears stratigraphic where exposed in roadcuts south of the CD pit, though elsewhere it is commonly faulted (Hefner et al., 1991); others believe that the contact is not conformable (Smith, 1982). The contact with the lower Pols member of the middle plate is faulted. The Pold member is up to 300 m thick in the southwestern part of the Mine area but thins dramatically and is discontinuous to the north and east, reflecting attenuation along a low angle normal fault or faults.
- Pols – Limestone, Sandstone and Quartzite Member: This unit consists dominantly of thin-bedded to massive calcareous siltstone alternating with thick beds of silty and sandy limestone, with minor lenticular beds of calcareous sandstone and quartzite. Wavy bedding, crossbedding, and ripple marks characterize the limestone (Smith, 1982; Ohlin, 1989). The age is given as Early to Middle Pennsylvanian. Where exposed, the top and bottom contacts of the Pols Member are faulted. It is not present everywhere, suggesting that it has been faulted out along the lower plate contact.
- PPos – Sandstone and Siltstone Member: This unit, comprising the upper structural plate, consists dominantly of poorly-sorted, quartz-rich, calcareous to non-calcareous sandstones and siltstones with minor silty, bioclastic limestone lenses. Calcite-cemented breccia zones and extensive fracturing are extremely common. It is brownish-weathering and relatively distinctive due to its structural position and relative lack of limestone, and appears to correlate with the upper, sandstone-dominated formations in the Oquirrh Group in more well-studied areas to the south (Smith, 1983). On this basis, it is assigned an age of Middle Pennsylvanian to Early Permian. This unit is at least several hundred m thick in the southern Black Pine Range and north of Mineral

Gulch. It forms isolated klippe at the highest elevations in the central Mine area and flanks a dome of older rocks to the north, east and south of the Mine area.

In the Rangefront target area, the upper portion of the PPos unit contains a grey limestone unit with thin to thick bands of brown weathering sandy and silty limestone.

7.2.1.1 Rock Types Not Associated with Specific Stratigraphic Intervals

Several rock types are logged in RC chips and drill core that cannot necessarily be correlated with discrete stratigraphic units, either because, as defined, they occur more than once in the stratigraphic sequence or because they are partly structural and/or hydrothermal in nature. These include:

- Poc: A unit consisting of dark grey to black, carbonaceous, variably calcareous siltstone, shale, and limestone. This unit is commonly found at the base of the Polb unit, where it ranges in thickness from 0 to locally over 200 m in the northeastern part of the Black Pine Mine area. While carbonaceous material can be found throughout the stratigraphic section it often is logged near the base of the middle plate interleaved in the PMmx (see below).
- PMmx – Shale, Siltstone, Sandstone, and Quartzite Mixed Member: This unit, recognized through drilling and geochemical analysis, is interpreted as a fault mélange containing a mix of rock types, consisting of lower plate carbonaceous shales and middle plate siltstone, sandstone, quartzite, and limestone. A phyllitic cleavage is often noted. Zones of carbonaceous siltstone (Poc) are also present. This unit is discontinuous and is often similar in nature to the Manning Canyon Shale. However, the Manning Canyon Shale can be distinguished by the presence of elevated Cesium relative to the PMmx. PMmx often contains gold mineralization, whereas the underlying Manning Canyon Shale typically does not.
- CalFm – “Calcite Formation”: This designation is applied to massive zones of primarily white calcite up to tens of m thick. It can include massive coarse calcite, marble, and calcite breccia, often with two or more of these end members observed together. This unit designation is necessary given that most of the drilling is RC, where it is often difficult or impossible to discern the origin of the calcite.

7.2.1.2 Cenozoic Intrusive Rocks

Narrow dikes and sills 0.1 m to 2 m in width have intruded the middle plate rocks throughout the Black Pine Project area. They are typically less than 1 m in width with chilled margins and range from aphanitic to finely porphyritic, with small phenocrysts of feldspar, hornblende, and biotite. Alteration typically consists of chlorite, sericite, and pyrite with some clay. At surface and in drill holes, the dikes are usually strongly oxidized to a deep orange-brown color and strongly sericitized. In outcrop, they can be seen as sills in the Pold unit in the C/D pit highwall, and as clasts in and cross-cutting a large collapse breccia body in the highwall of the A pit. A single whole rock analysis of a sample from one dyke returned an SiO_2 content of 46.9%, suggesting that the rock is a lamprophyre or other ultramafic rock. However, the rock is relatively low in in magnesium, chromium, and other elements that are normally elevated in lamprophyres. Additional whole rock analysis is necessary to further characterize these intrusive rocks and determine whether they represent one or more intrusive episode.

A sample was sent to the University of Arizona Laserchron Laboratory for U-Pb zircon analysis. 30 small zircons were analyzed with a 20 micron laser spot using an Element single-collector ICPMS. Grains ranged in age from 226 to 2,580 Ma, more characteristic of entrained detrital grains than igneous grains.

Fine-grained, pale grey-green intrusive rock has been noted in drill chips primarily hosted in lower plate shales in some locations. These dykes are strongly clay or sericite altered and may contain up to a few percent disseminated brassy pyrite. A sample was sent to the University of Arizona Laserchron Laboratory for U-Pb zircon analysis. 30 small zircons were analyzed with a 20-micron laser spot using an Element single-collector ICPMS. Grains ranged in age from 419 to 1,645 Ma, more characteristic of entrained detrital grains than igneous grains.

7.2.2 Structural Geology

As discussed above, there are three stacked structural plates at the Black Pine property: a lower plate, comprising the Jefferson Formation and Manning Canyon Shale; a middle plate comprising the Pol, Pold, and Pols units of the Oquirrh Group, and an upper plate consisting of the PPos member of the Oquirrh Group. Shale and siltstone in the lower plate are sheared, and strata in the middle plate are very complexly structurally interleaved. The middle plate in the Project area is approximately 100 to 500 m in thickness, decreasing in thickness in all directions from a maximum thickness of approximately 500 m near the E pit. Rocks of the middle plate show evidence of at least two major deformation events, including thrust faults and folds, overprinted by low- to high-angle normal faults.

7.2.2.1 Mesozoic Contractional Event(s)

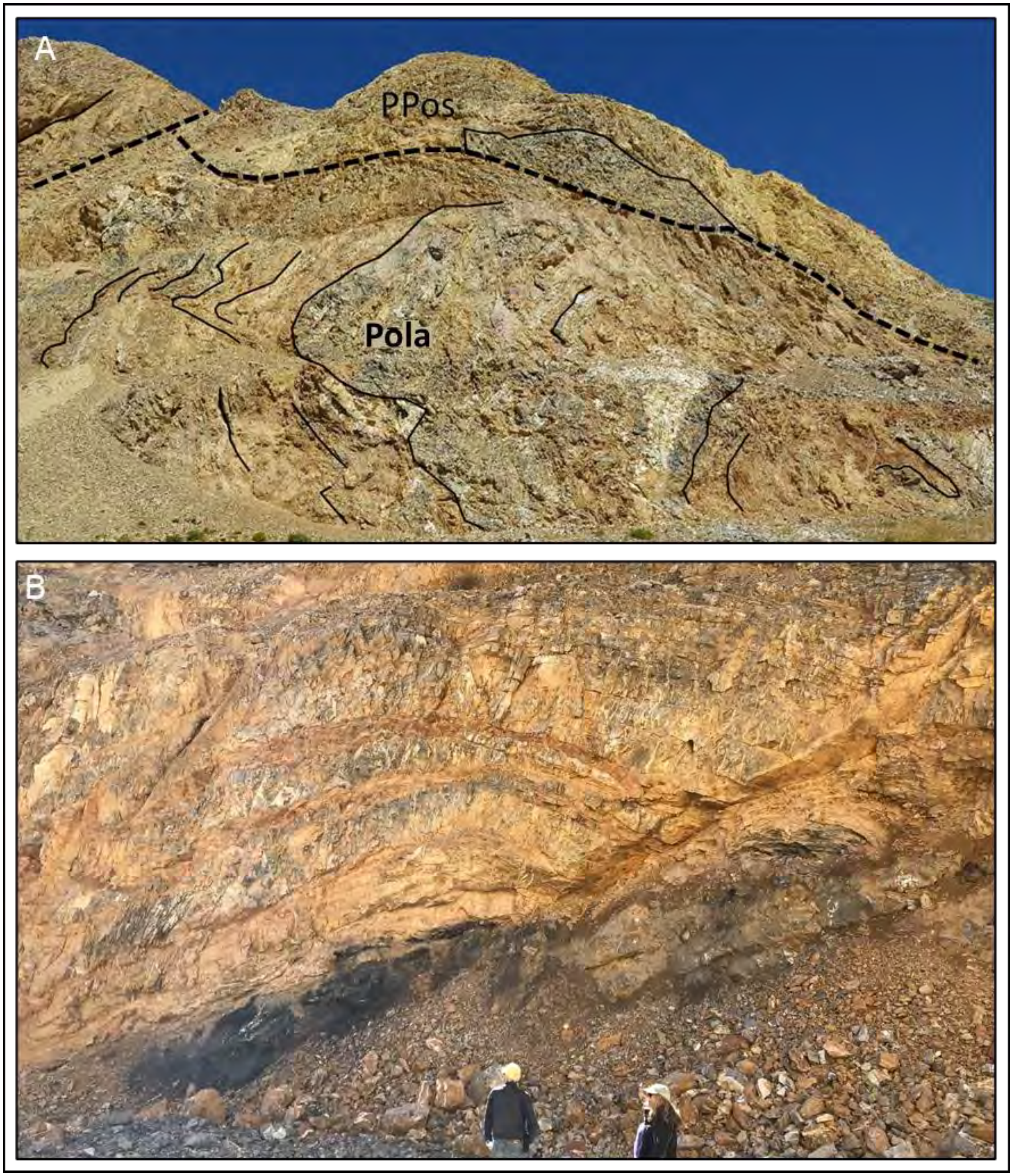
A polyphase, Mesozoic contractional event or events strongly affected rocks in the Black Pine Mountains.

Two contractional events of presumed Mesozoic age are interpreted at the Black Pine Property (Figure 7-5). The first is manifested by: 1 to 30 m scale, generally east- to northeast-vergent recumbent folds; weak axial-planar slaty to (rarely) phyllitic cleavage in silty to shaly rocks; and low angle, semi-ductile faults with reverse motion (as deduced from shear sense indicators, etc.). Higher strain zones are associated with areas of dominantly calcareous siltstone, such as in the Polb, Pols, and upper PMmc units, whereas significantly less first-phase contractional deformation is recorded in more massive units such as the Pola and Pold.

The second phase of contractional deformation is manifested by open to tight, upright folds with rounded (in massive limestone) to chevron geometry (silty strata). Some folds appear to be drag folds located in the hanging walls of relatively flat, semi-brittle faults. Calcite veins perpendicular to beds are common in limestone, on a m- to cm-scale. Sense of vergence is inconsistent across the property, suggesting that some of the faults could be back-thrusts associated with an otherwise northeast-vergent system. Property-wide, this deformation appears to be most prevalent in the Polc unit as observed in pit walls.

It is not known whether the two events represent a continuum or separate events. It is possible that both are associated with the Late Cretaceous Sevier Orogeny, which affected rocks throughout the eastern Great Basin and the Oquirrh Group in mountain ranges to the south and west of Black Pine. Another possibility is that the first event is associated with the mid-Jurassic Elko Orogeny (Thorman and Peterson, 2004) and the second with the Sevier orogeny.

On a property scale, middle plate strata dip southward in the southern part of the property and northward in the northern part of the property, forming an open dome that is apparent in outcrop due to the presence of an over-thickened package of resistant massive dolostone and limestone beds in the Pold unit, possibly a result of duplexing during the first phase of folding. Stratigraphic units in the middle plate are faulted out against the lower plate contact, which is relatively flat-lying except where cut and offset by Cenozoic normal faults.



Styles of Mesozoic folding at Black Pine. A) First-phase recumbent folds with axial planar cleavage in limestone and siltstone of the Pola unit in the E Pit highwall. View to the east. The upper plate Ppos sandstone is faulted over the top. B) Second phase open, upright fold with axial planar calcite veins in the hanging wall of a small thrust fault in the Tallman Extension Pit. View is to the southeast. The fault may be a back thrust, as sense of motion is to the southwest.

Figure 7-5: Styles of Mesozoic Folding

7.2.2.2 Cenozoic Extension

Episodic extension in the Great Basin commenced in the Eocene and persists to present day, accompanied by intermediate to felsic volcanism. Several major episodes of extension affect the Black Pine area, including:

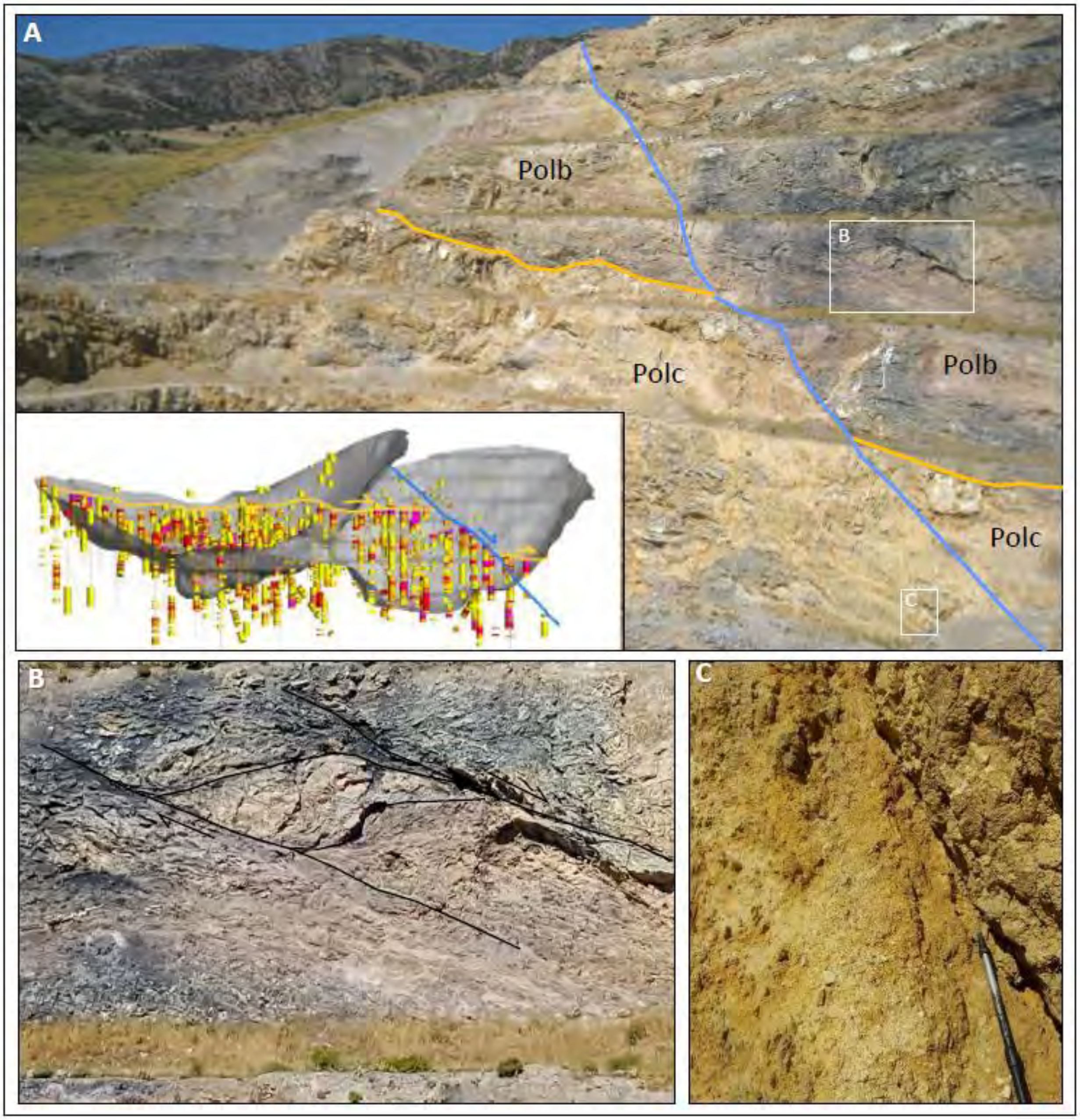
- Eocene listric normal faulting and volcanism (likely timing of gold mineralization)
- Oligocene extension, deep metamorphism and plutonism
- Miocene unroofing of the Albion-Raft River metamorphic core complex along low angle detachment faults
- Basin and Range block faulting

Elucidation of the structural framework of the mine area is hampered by a number of factors, including poor outcrop, almost exclusive use of RC drilling, relative lack of deep drill holes, lack of continuous marker beds, lack of fossils to constrain relative ages of rock units and a large number of overprinting, largely coaxial episodes of contraction and extension. The structural model continues to evolve as more mapping and drilling is carried out.

Eocene normal faults: The normal faults of possible Eocene age in the Project area are low to moderate angle, semi-ductile to brittle in nature and overprint the earlier contractional deformation (Figure 7-6). These faults are interpreted to be listric in nature and appear to exert significant control on the distribution of mineralization.

Two listric normal faults are interpreted to control mineralization over a wide area. One fault hosts mineralization in the D-1 zone along a relatively shallow, moderately NW-dipping segment of the fault, extending from north of A Basin to the Tallman Extension Pit. It is interpreted to flatten to the northeast, underlying the D-2 zone. A second, north-south striking listric normal fault is interpreted to underlie the D-3 zone and may extend as far south as the CD Pit. It is interpreted to join the D-1 listric fault to the north. The fault has a relatively steep orientation along the west side of the D-3 zone, then shallows to the east, eventually soling into the lower plate contact. This fault may be responsible for a dramatic thinning of the Pold unit to the east.

Oligocene – Miocene extension: The Black Pine area was affected by two or more episodes of post-mineral normal faulting, including movement on faults related to unroofing of the Raft River-Albion metamorphic core complex, located to the west. Up to several kilometres of top-to-the-east movement is postulated along one or more low angle detachment faults located between Black Pine and the Raft River Range, with related, down to the east listric normal faults tilting and extending strata in the hanging wall. It is possible that some of the normal faults in the Black Pine area may be related to this episode of extension, and that some or most of the movement along the middle plate – lower plate contact may be of this age. The latter would imply that the ultimate source of fluids for the gold mineralization may not be situated under the Black Pine Mine area at present but may be located up to several tens of kilometres to the west.



View to the northwest in the Tallman Extension Pit. A) the Polb unit low angle faulted contact with the Polc unit along a relatively low angle contact, which in turn is offset along a higher-angle, post-mineral fault. Inset map shows the distribution of gold, almost entirely within the Polc unit, which contains a high degree of internal brittle deformation. B) Detail of the pit wall showing semiductile shears with down to the east or northeast displacement in the Polb unit. C) Detail of post-mineral normal fault showing brittle gouge zone. Gold mineralization in this location is probably the result of entrainment in a post-mineral fault.

Figure 7-6: Examples of Normal Faults in the Tallman Extension Pit

Pliocene-Recent extension: Basin and Range (Pliocene-Recent) faults are in evidence along the eastern range front. A large, steep to moderate east-dipping fault extends along the range front, bending southwest to extend along the southeastern edge of the C-D pit. This fault offsets stratigraphy in a down-to-the-east sense of displacement, and brittle, calcite- and silica-cemented breccias and gouge are in evidence in some locations. The fault is well-exposed in the south highwall of the CD Pit, cutting a lower-angle normal fault separating the upper and lower plates. A number of parallel faults are in evidence to the west of the fault in the Mineral Gulch area, and a number are inferred to exist to the east under gravel cover, as evidenced in ground gravity data (see section 9).

7.2.2.3 Breccias

Breccias are very common at Black Pine and are present on all scales, from centimeter- to pit wall-scale. Breccia types include:

Collapse breccias. Collapse breccias are widespread, and generally consist of angular, polymictic clasts ranging from cm-sized to large blocks of limestone, dolostone, siltstone, sandstone, altered intrusive rock, calcite vein material and earlier breccias (Figure 7-7). Matrix, where present, consists of orange-brown iron oxides and silty material. Abundant calcite cement is typically white and coarse-grained. Breccias range from clast supported to clasts floating in calcite cement and/or sandy matrix. Collapse breccias can be seen in pit walls and in drill holes on a cm to pit-wall scale. Collapse breccias are thought to be intimately related to hydrothermal alteration and gold mineralization through a process of “hydrokarst”, with progressive erosion and hollowing out of fault zones or stratigraphic contacts by acidic fluids, followed by collapse and cementation by calcite.

Tectonic breccias: Breccias are associated with late, low- to high-angle normal faults. These breccias contain milled clasts, are typically poorly indurated and accompanied by clay alteration and gouge (Figure 7-6, inset C). A second type of tectonic breccia is observed in upper plate rocks and consists of angular fragments of sandstone in a limy matrix. These breccias form north- to northeast-trending arrays throughout the Rangefront Zone.

Crackle and Mosaic breccias: Crackle and mosaic brecciation are almost ubiquitous throughout the Black Pine property, particularly in the PPos, Pola and Polc units. Fractures are cemented by calcite and iron oxide.

Calcite breccias: Breccias consisting almost entirely of clasts of white calcite and cemented by white calcite and variable amounts of iron oxide, are common at Black Pine. Clasts range from angular to rounded, with a milled appearance. The genesis of these breccias is uncertain. In many cases, they appear to form relatively flat sheets that may correspond to calcite-filled fault zones. In other cases, the clasts appear to resemble coarse, white, recrystallized marble. Additional observation and study are needed to describe and understand the genesis of these breccias.

A large collapse breccia body occupies the southeast portion of the A pit. Limestone and dolomite clasts in a matrix of sand and calcite cement, giving way to nearly all calcite near the top of the body. Reddish-brown, elongate zones are lamprophyre dikes, which crosscut the breccia, but can also be found as clasts in it. Inset: detail of breccia with angular limestone clasts in a sandy, calcite cemented matrix.

Figure 7-7: Collapse Breccia in the A Pit

7.2.2.4 Strain Partitioning and Tectonostratigraphy

As the foregoing indicates, strain partitioning is observed within and between the different members and submembers of the Middle Plate, with each member or submember characterized by the presence or prevalence of different structural fabrics on a meso-scale:

Pola consists primarily of panels of massive thick-bedded limestone beds that are cut and offset on low-angle thrust and normal faults. The limestone is also strongly fractured, often overprinted by a stockwork of calcite veins.

Polb consists largely of calcareous to non-calcareous siltstone with thick beds of limestone and dolomite. Polb is thus a relatively high-strain zone due to preponderance of “weak” siltstone. Recumbent folds are common in this unit, along with low angle thrust and normal faults. The dominant structural fabric is relatively low angle and ductile to semi-ductile in nature.

Polc, consisting of alternating beds of non-calcareous siltstone, limestone, and dolomite is characterized by the presence of low to moderate angle thrust faults with second phase folds, and brittle structures including moderate to high angle brittle normal faults, jointing, and widespread brecciation.

Pold consists largely of massive limestone and dolostone. Where the Pold member forms thick, resistant outcrops in the core of the Black Pine gold system, it is characterized by low-strain zones showing relatively little evidence of internal folding or faulting, with the exception of some bedding-parallel, low angle normal or reverse faults such as observed in the C-D Pit highwall.

Pols, consisting of silty limestones and calcareous siltstone, is rarely observed in outcrop, but where exposed in the C-D pit, it can be seen to contain tight, recumbent folds.

Gold mineralization can be found in all of the middle plate units but is particularly well-developed in the Pola and Polc members, both characterized by the presence of areas of significant brittle deformation.

7.3 Alteration

Strata throughout the Black Pine Mine area is weakly to strongly hydrothermally altered and contain widespread gold mineralization over the entire thickness of the middle plate, and over an area measuring at least 14 km^2. In general, the rock types with higher porosity, permeability, and geochemical reactivity, such as calcareous siltstone and sandstone, and a higher degree of brecciation, are more strongly altered.

Alteration types closely associated with gold mineralization include:

- *Decalcification:* Defined as the removal by dissolution of calcite from the matrix of a carbonate rock, some degree of decalcification is common within gold-mineralized rocks. The highest gold grades are found in calcareous siltstone and sandstone, where decalcification forms spongy, porous zones with relatively low specific gravity. Selective decalcification is present along bedding planes, fractures and breccias. Large bodies of collapse breccias are present in the B Pit, A Pit and A Basin area, the end result of dissolution of more massive limestones and dolomites along faults, etc. and subsequent collapse of the resulting cavities (Figure 7-7). Sanding is observed locally in dolomitized rocks, a result of removal of calcite cement around dolomite rhombs.

- *Silicification:* Silicification is present throughout the mineralized zones, but it rarely manifests as discrete zones of jasperoid. Silicification is far more common as areas of very weak, non-texture-destructive silicification in calcareous siltstone and sandstone, as small (dm-scale) patches of more well-developed silicification or jasperoid locally, or as networks of veinlets in dolomitic rocks.

- *Marblization:* Zones of bleaching and recrystallization of grey limestone to a medium to coarse-grained marble are present throughout the Mine area, but are most common in the northern Discovery Zone and A Basin area, apparently in relation to large, variably brecciated, calcite veins. The marbleized zones can be distinguished from calcite veins by the preservation of relict bedding in the marble. This phenomenon is not directly spatially related to gold mineralization, but these zones are often located adjacent to zones of mineralization.

- *Clay alteration:* Lithologic types such as siltstone and shale, or rocks containing a significant component of fine silt, are clay altered to some degree. The clay species have not been investigated, but it is likely that the clay is largely illite, based on analogy to other Carlin-type gold systems.

- *Carbon:* Carbonaceous material is present to some degree in all calcareous rock types, particularly those with a high component of silt, and is probably derived from organic material incorporated when the rocks were deposited. Carbonaceous material is notably present in the Poc unit at the base of the Polb submember, in the PMmx unit and in the lower plate shales, forming bedding-parallel, tabular lenses. Carbonaceous material is also present as irregular small lenses throughout the middle plate, particularly in high-strain areas, suggesting that the carbonaceous material was mobilized and deposited in these locations. The stratigraphic carbonaceous zones rarely contain significant gold, but the irregular lenses may contain gold mineralization.

- *Iron Oxide:* Virtually the entire rock mass in the middle and upper plates, with the exception of some massive limestones, contains at least some iron oxide, primarily limonite and goethite with minor hematite and jarosite, an indication of the original presence of pyrite and the degree to which most of the rock mass is thoroughly oxidized. Iron oxide ranges from disseminated in porous siltstones to fracture controlled in more massive rocks. Visually, the amount of iron oxide in the rock can be used as an indication of the presence of gold in the rock, with increasing amounts of iron oxide generally a favorable indicator of gold.

- *Calcite:* Coarse, white calcite is ubiquitous throughout the Black Pine Mine area as veins, veinlets, breccia cement and breccia clasts. In some cases, calcite forms both the clasts and cement in some breccias. While a large amount of calcite is generally associated with lower gold grades, its presence in large quantities is a good indication of gold in adjacent rocks. Preliminary study of calcite fluorescence suggests that there are several distinct phases of calcite veining.

7.4 Gold Mineralization

7.4.1 Style of Mineralization

Gold mineralization, consisting of finely disseminated, micron- and submicron-size gold particles, is hosted in middle plate, calcareous shale and siltstone, as well as fault and dissolution/collapse and tectonic breccias and zones of heavy fracturing developed in more massive limestone and dolomite. Gold mineralization is enhanced where these favorable stratigraphic units intersect, or lie along, large, pre- to syn-mineral, primarily listric normal faults. Gold was likely hosted within the lattice of arsenical pyrite rims on pyrite grains, but the mineralized rocks are now thoroughly oxidized, such that gold is present as “free” gold, associated with goethite, hematite, limonite, barite, calcite, quartz, and rare scorodite. Gold-bearing rocks are typically strongly decalcified and weakly clay altered, with areas of weak to (rarely) moderate silicification. Areas of calcite veining or calcite-cemented breccias are common, probably as a result of decalcification. Lenses of carbonaceous material, either remobilized or concentrated by decalcification, are locally present. The alteration, gold mineralization, host rocks, and geochemical associations are consistent with a Carlin-style deposit model.

Reflected-light microscopy has shown that native gold occurs in quartz and calcite veins, in hematite pseudomorphs after pyrite, and along grain boundaries (Hefner et al., 1991). In (rare) unoxidized material, an electron microscope is required to detect the gold grains, which are commonly less than 0.5 microns in diameter (Brady 1984). Gold is disseminated in clayey or silty matrix of clastic rocks and micrite groundmass of limestone. Carbon is locally present as both graphite and organic matter; gold is associated with organic matter in both clastic and carbonate sedimentary rocks.

Geochemically, gold shows an association with the typical Carlin pathfinder trace elements of arsenic, antimony, mercury, thallium, and tellurium, which are all elevated in the presence of elevated gold. However, while some samples with high gold grades do have a correspondingly high arsenic or antimony values, these elements do not always correlate strongly and sometimes even correlate negatively, possibly a result of mobility of the elements in the supergene environment. A correlation matrix of Liberty Gold drill holes (Figure 7-8) shows positive linear correlations between gold, arsenic, mercury and tellurium, typically strong Carlin-style pathfinder elements. There are no recognized alteration or spatial patterns to these positive or negative correlations between gold and the trace elements. Silver, copper, lead, and zinc are correlated with each other and with arsenic, antimony, mercury, and selenium. Areas of elevated silver, lead, zinc, and copper generally do not spatially coincide with gold mineralization but there are areas where there appears to be overlapping base metal and later Carlin gold systems.

	Au_ppm	Ag_ppm	As_ppm	Ba_ppm	Bi_ppm	Cu_ppm	Hg_ppm	Pb_ppm	Sb_ppm	Se_ppm	Te_ppm	Tl_ppm	Zn_ppm
Au_ppm	1												
Ag_ppm	0.100621	1											
As_ppm	0.16458	0.40097	1										
Ba_ppm	0.0365	0.078785	0.185666	1									
Bi_ppm	0.189322	0.391089	0.265943	0.099065	1								
Cu_ppm	0.05938	0.78932	0.349851	0.138848	0.29591	1							
Hg_ppm	0.143009	0.669556	0.40706	0.061599	0.414723	0.427059	1						
Pb_ppm	0.06105	0.501436	0.362811	-0.01368	0.255994	0.169467	0.672718	1					
Sb_ppm	-0.01513	0.187985	0.079821	0.00762	0.296976	0.04909	0.532696	0.0593	1				
Se_ppm	0.127681	0.500551	0.35509	-0.03425	0.353383	0.153344	0.516521	0.595799	0.021744	1			
Te_ppm	0.527426	0.116209	0.12909	-0.06327	0.301961	0.031905	0.165374	0.209305	0.013236	0.248856	1		
Tl_ppm	0.029998	0.104959	0.213028	0.07629	0.292789	0.03026	0.375538	0.054201	0.679786	0.003174	0.060604	1	
Zn_ppm	0.090712	0.468074	0.532172	0.124811	0.205097	0.341395	0.583356	0.65837	0.039366	0.52894	0.107934	0.032849	1

Source: Liberty Gold (2023)

Note. 51,900 Liberty Gold drill samples containing >0.1 g Au/t)

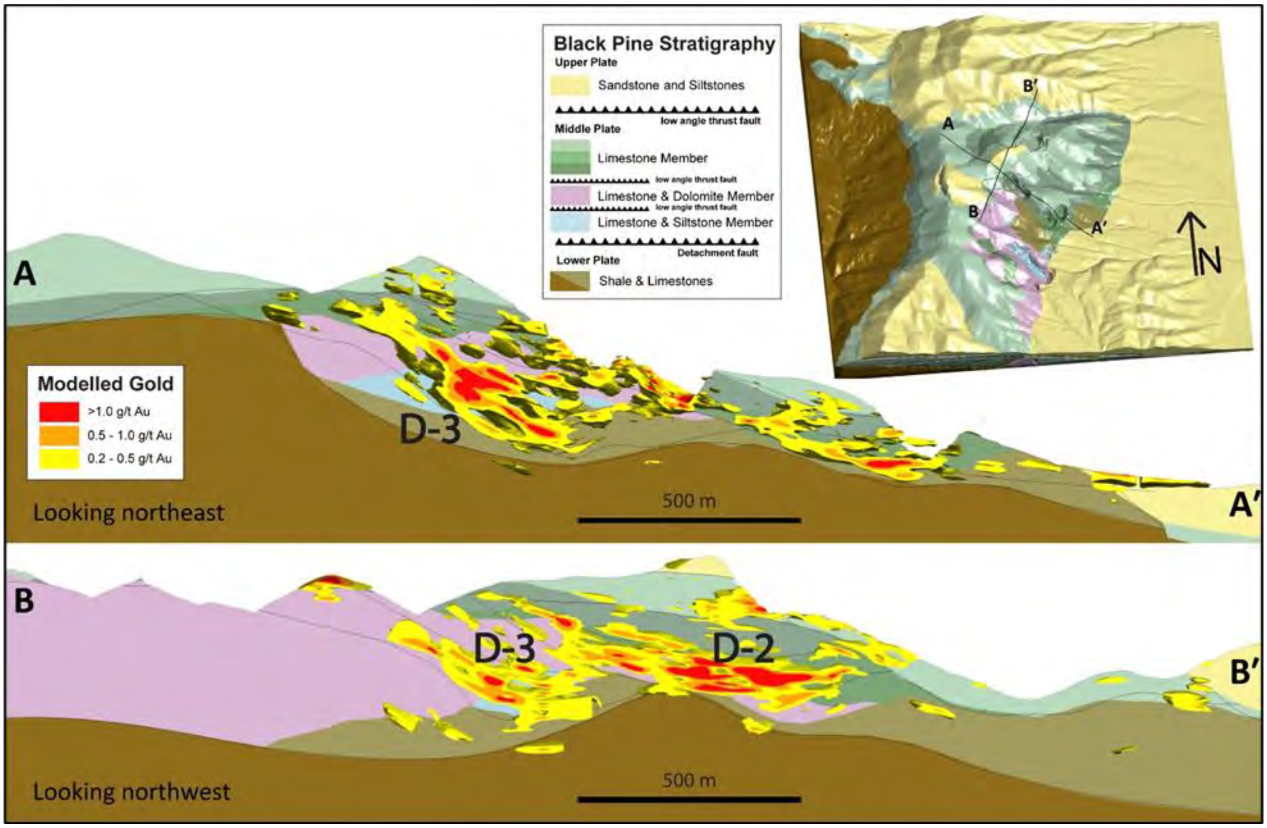
Figure 7-8: Geochemical Correlation Matrix

7.4.2 Location of Mineralization – Historical Pits and Vicinity

Silver and Base Metals: Silver and base-metal mineralization were historically mined on a small scale prior to the 1940s. These occurrences were located at Hazel Pine and along the Range Front, north of A Basin, northwest of D Pit, and in the Silver Hills (Back Range target; Figure 6-4) around the margins of the Black Pine gold system. This type of mineralization is associated with steep faults, brown iron-oxide-stained hematitic silicification and quartz veins (Ohlin, 1988). Host rocks are typically thick-bedded limestone with massive white calcite replacement beds. Liberty Gold has sampled stockpiles from historical mining containing >10% Zn in iron-oxide-cemented breccia. Short intervals of elevated silver-lead-zinc are relatively common in and around zones of gold mineralization, but there is generally a weak correlation between elevated gold and elevated silver, lead, and zinc. Therefore, it is likely that the two events are not closely related.

Gold: During the historical Pegasus mining operation, gold-mineralized material was extracted from six pits, including the Tallman pit, the B/B Expansion pit, A pit, E pit, I pit, and the C/D pit (Figure 6-4). Gold is distributed throughout the middle structural plate, but higher grades are focused on more favorable stratigraphic units such as calcareous siltstones, and in association with low to moderate- to high-angle faults. Favorable faults are brittle in nature and strike northwest in the Tallman, B, C, D, and E pits. Others strike northeast in the Tallman, C, D, A, and I pits and north in the E pit. Gold appears to be concentrated along and in the immediate footwall of some of these faults, where less favorable massive limestone, non-calcareous shale or sandstone are present in the hanging wall (Tallman NE and B Extension pits).

Gold is present in a large number of historical drill holes in unmined areas, particularly in areas adjacent to the historical open pits as shown in Figure 7-9. For example, historical "reserves" disclosed in Section 0 were defined to the north and west of the A pit, but these areas were never mined. Gold mineralization remains in-situ beneath and peripheral to the historical pits, as demonstrated in both historical and Liberty Gold drilling. The reader is referred to Section 10 of this report for descriptions and illustrations of the major zones of gold mineralization at Black Pine, defined through historical and Liberty Gold drilling.



Source: Liberty Gold (2021)

Figure 7-9: Schematic Cross Section of Middle Structural Plate (looking south)

7.4.3 Gold Mineralization and Soil Anomalies

Throughout the exploration history of Black Pine, the presence of gold in soils in areas underlain by middle plate rocks over a roughly 12 km^2 area has been a reliable indicator of gold in the subsurface. Most of the original (mined) deposits were located at least partially through drill testing of high-amplitude (>100 ppb) soil anomalies. Subsequent drilling of the wider area of anomalous gold in soil led to the identification of smaller, unmined satellite deposits including the J, F, M, and Bobcat zones. With at least a cursory drill test of the larger, high-amplitude soil anomalies on the property largely complete, testing of more subtle anomalies in upper plate rocks, such as over portions of the southern Rangefront Zone, Burnt Basin and Back Range Connector, remain a priority.

8 DEPOSIT TYPES

Black Pine mineralization is best described to be in the class of sedimentary rock-hosted, Carlin-type gold deposits ("CTGDs"). While CTGDs are not unique to the eastern Great Basin, they exist in far greater numbers and total resource size in northern Nevada than anywhere else in the world. They are characterized by concentrations of very finely disseminated gold principally in silty, carbonaceous, and calcareous marine sedimentary rocks. The gold is present as micron-size and smaller disseminated grains, often internal to iron-sulfide minerals (arsenical pyrite is most common), or with carbonaceous material in the host rock. Free particulate gold, and particularly visible free gold, is not a common characteristic of these deposits except where strongly oxidized.

CTGDs in the Great Basin have some general characteristics in common, although there is a wide spectrum of variants (Cline et al., 2005; Cline, 2018). Anomalous concentrations of silver, arsenic, antimony, and mercury are typically associated with the gold mineralization. Elevated concentrations of thallium, tungsten, tellurium, and molybdenum can also be present in trace amounts. Alteration of the gold-bearing host rocks is typically manifested by decalcification, often with the addition of silica, fine-grained disseminated pyrite and marcasite, remobilization and/or the addition of carbon, and the deposition of late-stage barite and/or calcite veins. Small amounts of white clay (illite or kaolinite) are generally present. Decalcification of the host produces volume loss, with incipient collapse brecciation that enhances the pathways of the mineralizing fluids. Due to the small size of the gold grains, CTGDs generally do not have coarse-gold assay issues common in many other types of gold deposits.

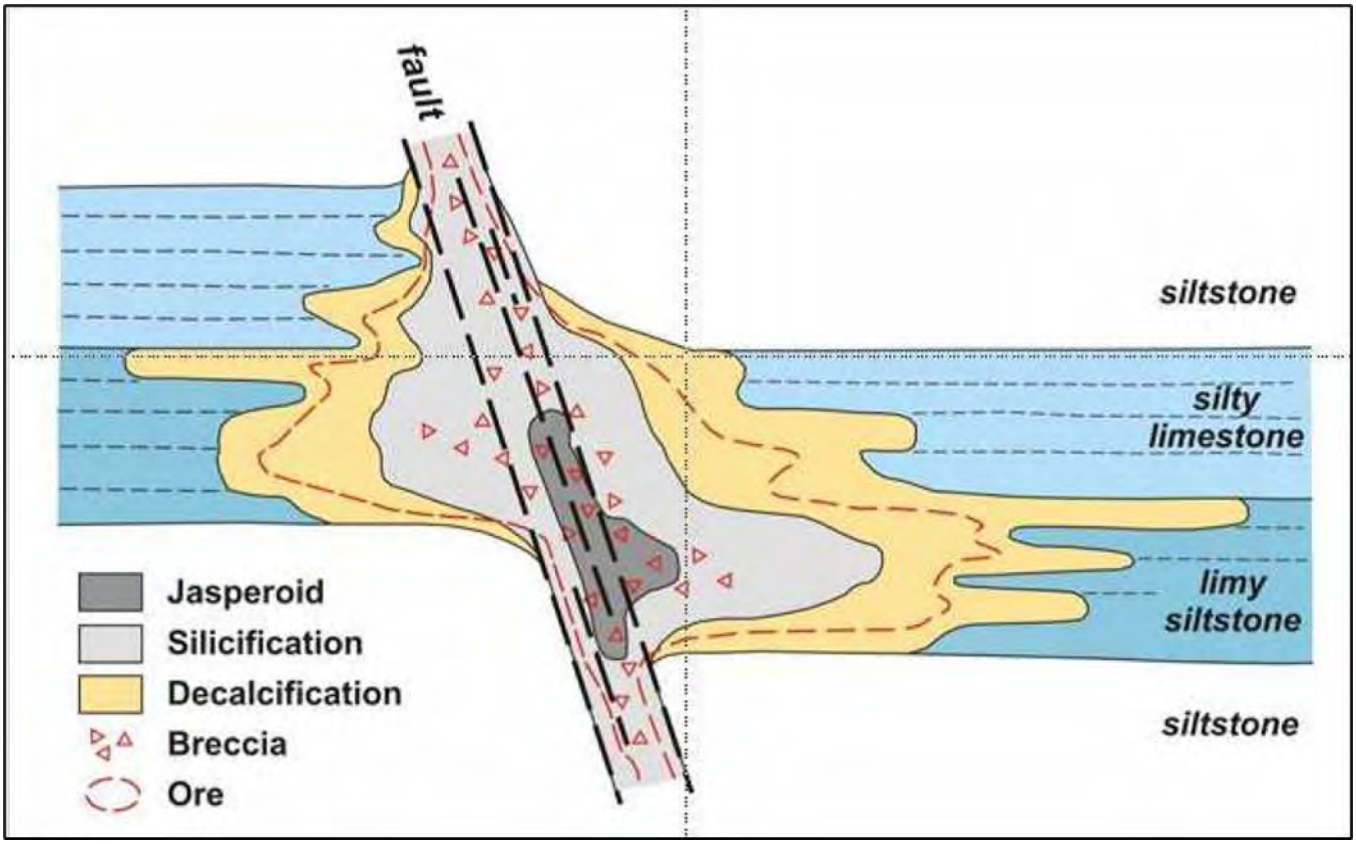
Deposit configurations and shapes are quite variable. CTGDs are typically somewhat stratiform in nature, with mineralization largely confined within specific favorable stratigraphic units. Fault and solution-collapse breccias can also be primary hosts to mineralization (Figure 8-1).

The gold mineralization identified at Black Pine shares many of the characteristics of CTGDs, including:

- Stratigraphic control of mineralization, primarily in calcareous siltstone units within the Pennsylvanian Oquirrh Group.
- Structural control in and adjacent to low-angle to high-angle normal faults, and in tectonic, collapse, and hydrothermal breccias.
- Geochemical association with elevated arsenic, mercury, antimony, and thallium, as well as silver and tellurium; base metals are elevated around the north and east sides of the system.
- Gold is very fine grained, disseminated, and associated with decalcification, silicification, calcite and clay, as well as pyrite, arsenical pyrite, and their oxidized variants (limonite, goethite, hematite, etc.).

The Black Pine gold deposits also have characteristics that differ from typical CTGDs. The general location of the Project is outside the major gold deposit trends in Nevada. There are multiple silver-lead-zinc occurrences within the Black Pine property, although the temporal association with the gold mineralization is not clear.

Using the Carlin Type model as a guide, exploration at Black Pine has been focused on targeting zones of favorable stratigraphy primarily silty, sandy carbonates in Pola, Polb, Polc and sections of Ppos units where they intersection high or lower angle fault zones. Exploration has also been driven by surface geochemical anomalies in soils and rock chip sampling where elevated Carlin-type pathfinder elements such as arsenic, antimony and mercury are present.



Source: Robert et al. (2007)

Figure 8-1: Cross-Section Model of a Carlin-Style Sedimentary Rock-Hosted Gold Deposit

9 EXPLORATION

This section summarizes exploration work carried out by Liberty Gold at the Black Pine Project. Section 9.1 is excerpted from Gustin et al (2021).

9.1 Historical Data Compilation and Project Database Construction

Liberty Gold inherited several historical data packages from Western Pacific Resources. The historical database upon which Western Pacific based their exploration program contained primarily exploration and development data up to the 1989 sale of the Project to Pegasus, including compiled digital and hardcopy records of surface rock and soil samples, geological mapping, exploration drill-hole locations, assays, surveys, geological logs, and copies of drill assay certificates. Also included were various internal and external memoranda and reports.

After the purchase of Black Pine from Western Pacific, a hard drive was conveyed to Liberty Gold containing .zip files created during the Pegasus mining operation, with file stamps dating principally from 1990 to 1997. The data comprises numerous Surpac, PC EXPLOR, PC MINE, and Gemcom project files, mine topography, and permitting design CAD files from throughout the mine life, as well as bench, road, and topographic survey files. Gemcom extraction files were recovered containing rock and soil sample databases and a compiled drill-hole database. This drilling database contains drill hole location and orientation data, gold and silver assays, lithological data, and carbon analyses for all historical drilling on the property, notably including 1,098 Pegasus drill holes. Blast-hole data for the E pit, A pit, and some of the C, D and I pits have been recovered, representing approximately 40% of the total. Very few hard copy files from the Pegasus operation have been recovered.

Liberty Gold's compilation and verification efforts as of the Effective Date of this report include:

- <u>Assembly and verification of raw data export files of drill-hole data into a coherent Access database.</u> Pegasus data files without column headers were re-organized and verified using assay certificates and drill logs from pre-1990 drill-hole data. Assay data reported in troy ounces per short ton were converted to grams per metric tonne using a conversion factor of 34.286. Laboratory assay certificates and drill logs were available for most Noranda holes and some earlier holes, and these were used to validate down-hole assays. Down-hole lithological and alteration data were obtained from the same raw files, which included a primary lithological unit abbreviation and a secondary lithology or alteration, sometimes including presence of carbon.

- <u>Conversion of historical mine-grid coordinates into the UTM NAD 83, Zone 12 coordinate system.</u> Historical drill hole collar coordinates, surface-sample locations, and topographic information were transformed using Western Pacific and 2010 Olympus aerial-survey data. The horizontal error ranged from less than one meter near the grid origin (near the C/D pit,) to 1.0 m about one kilometer away, to 3.0 m at the far edges of the Project. This error range was determined by using 11 historical mine-grid control points that were found in the field and subsequently surveyed in UTM coordinates by Olympus Aerial Surveys, Western Pacific, BLM, and Liberty Gold. These survey results were then compared to the UTM locations of the control points as determined by the same transformation applied to the historical drill-collar locations.

- <u>Verification of historical collar locations and surface samples after coordinate transformation.</u> Air-photo disturbance images from 1992 and 1998, georeferenced drill hole maps from Noranda, and CAD maps from Pegasus were used to validate drill-collar locations following the coordinate transformation. This led to the identification of only two drill holes that were mis-located, and the locations of these holes were corrected. Noranda road-cut rock samples from in the lower F zone and J anomalies were adjusted following coordinate transformation, with their correct locations apparent from sample distributions relative to present-day reclaimed road alignments and historical aerial photos, as well as geo-referenced sample maps.

- Creation of an as-mined bedrock surface topography through clipping and merging pre-mine topography beneath dumps. As-built pit topographic maps were merged, and as-mined pit topography maps were created by digitizing bench surveys in ArcGIS 3D. A pre-mining topographic surface was also created. For the as-mined topography compilation, CAD files in the local mine grid were imported into an ArcGIS Geodatabase using the coordinate transformation, and elevations in feet were converted to meters. Historically surveyed, as-mined topographic maps for the Tallman, B pit, I pit, and D-north pits, all currently partially back-filled, were used to create the as-mined topography. A 2010 Orthophoto digital elevation model ("DEM") was to create the as-mined topography for the Tallman NE, B Extension, A, and C/D pits, as these pits were for the most part not backfilled. Pit-wall failures or partial back filling occurred in the E, C/D, and A-West pits. Portions of historical topographic data, consisting of either pit designs corroborated with blast-hole data or digitized bench surveys, were used to reconstruct an accurate as-mined bedrock surface for these pits.

- Recovery and compilation of surface geochemical data (soil and rock samples) from Pegasus database exports. Verification of soil-sample locations included comparisons to georeferenced maps of original soil grids and rock-sample locations, where available. As of the Effective Date of this report, a total of 12,453 soil samples and 4,516 rock samples within the Liberty Gold property boundary have been attributed with coordinates and gold assay data. Of these, 8,029 soil samples and 1,664 rock samples have both assay certificates and location data.

- Geologic map compilation. Surface geological maps created by Noranda were not updated significantly during the Pegasus operations. The Noranda map by Ohlin (1989) is still the best available historical property-scale geological map. Registration, digitization, and spot checking of Ohlin's map have been performed. Pit maps by Willis (2011) for Western Pacific have been registered and transformed into UTM NAD83, but these have not been used or extensively field-checked, although the mapping correlates well with down-hole lithology. USGS mapping by Smith (1982) provides geological information on a regional scale. These maps are gradually being amalgamated into a single geological map for the entire property, as the pit maps provide geological information that was not available prior to mining.

- Recovery of blast hole data. As of the Effective Date of this report, a database of 61,704 blast hole data points have been recovered, verified, and assembled. The blast holes are from E pit (12,987 - complete), A pit (36,398 – partial), C/D pit (7,418 – partial), and I pit (4,901 – partial). Also recovered are 63,861 blast hole intervals from C/D pit with corrupted coordinates (currently unusable). Liberty Gold is of the opinion that there is more blast hole data contained within the data files, and recovery efforts remain ongoing. Comparison of the complete set of blast hole data and exploration drill-hole assays within the E pit demonstrates the importance of the data density provided by the blast holes in modelling the complex, strongly structurally controlled gold mineralization at Black Pine.

- SLR conducted an additional review of the 'below detection limit' ("BDL") of historical assay values. As laboratory technology improves, so do the detection limits that the equipment can measure. Common practice is to replace the BDL value with the value of the current detection limit or half of the detection limit. SLR identified multiple detection limits throughout the Project's drilling campaigns. Liberty Gold took a prudent approach in identifying any assays at the detection limits prior to 1998 and replaced them with a value of 0.017 ppm. The total number of assays which were reduced in value is approximately 43,000 entries and reduced the entire database grade by 4% from 0.238 g/t Au to 0.229 g/t Au. This process will allow greater confidence in historical assays not artificially increasing grade in the Mineral Resource estimation.

9.2 Liberty Gold Rock Sampling

Liberty Gold has carried out several surface rock-sampling programs to characterize mineralization and alteration on the Black Pine property on underexplored gold-in-soil anomalies beyond the limits of historical pits. Between 2017 and the Effective Date of this report, 702 rock samples were collected throughout the property, primarily as grab samples and chip/channel samples in prospective rocks along newly exposed road-cuts, including in the CD Zone SW Extension, F Zone, Back Range and Bobcat Zone (Figure 9-1). Gold assays range from below detection limit to a high of 3.01 g Au/t. Liberty Gold has interpreted that the rock-chip sampling indicates gold is most closely associated with iron oxide, decalcification, and argillization, primarily in deformed silty limestones and calcareous siltstones, and is spatially associated with faults.

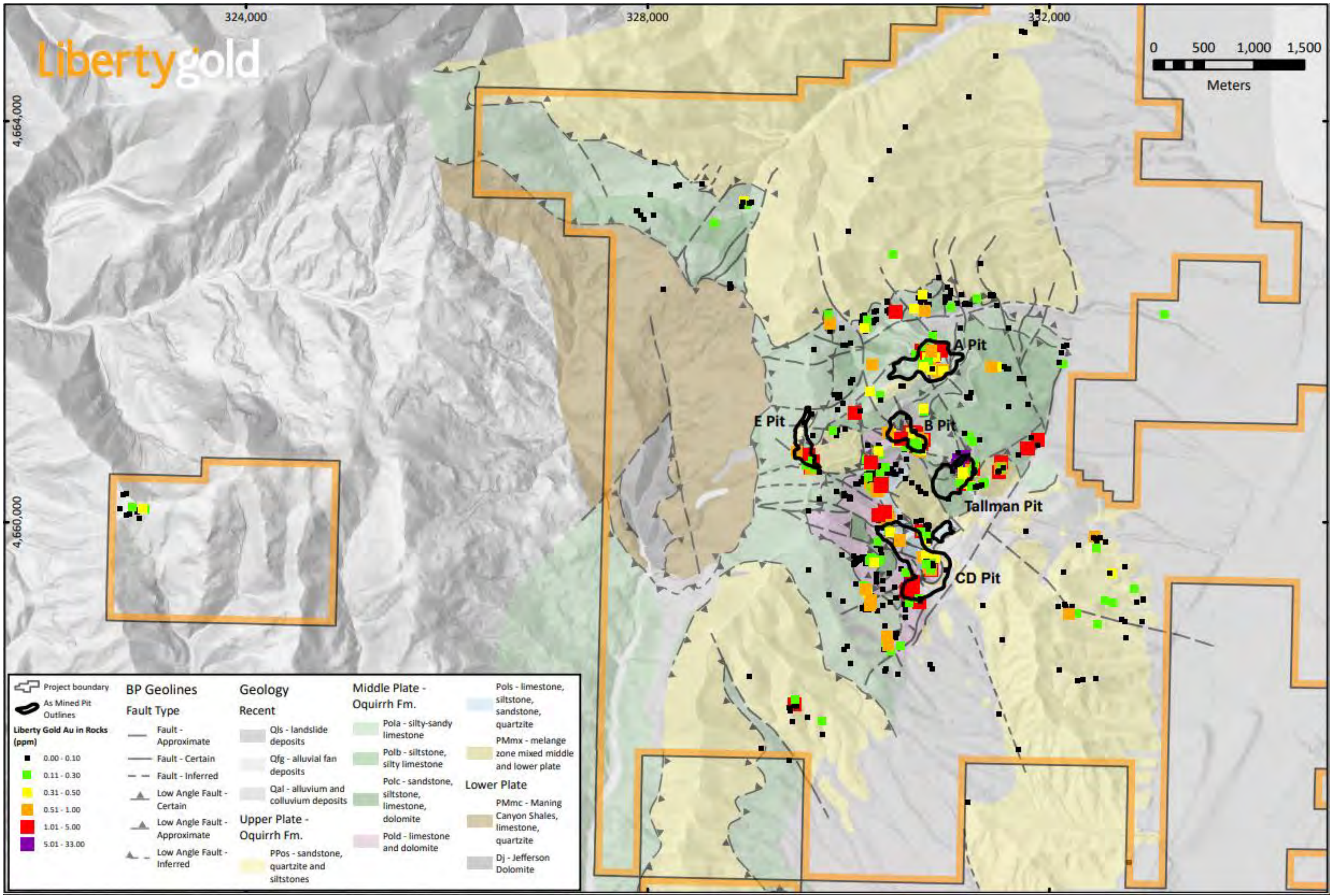


Figure 9-1: Liberty Gold Rock Sampling, 2017-2023

9.3 Liberty Gold Geologic Mapping

Liberty Gold has carried out geologic mapping at various times throughout the life of the Project, primarily by April Barrios, William Lepore, Randy Hannink, Moira Smith, and consulting geologist Tracy Dembrowski. Geologic mapping was facilitated using a combination of digital pads (ArcPad, Field Maps, FieldMove Clino etc.) and paper maps, and has been integrated by Liberty Gold geologist April Barrios into a master property map in ArcGIS that is used as a base for a number of figures in this report.

9.4 Ground Gravity Survey

A ground gravity survey was carried out in 2022 by MaGee Geophysical Services LLC, as summarized in Wright (2022). A total of 1,168 stations were acquired in two phases on 200 m and 400 m grids covering central and eastern parts of the property and adjacent areas, as well as widely spaced stations on public roads surrounding the grid. Relative gravity measurements were made with LaCoste & Romberg Model-G gravity meters and one Scintrex CG-5 meter. Topographic surveying was performed with Trimble Real-Time Kinematic and Fast-Static GPS units referenced to two base stations. Data processing was performed with the Xcelleration Gravity module of Oasis Montaj, V. 7.0. Additional processing methods are described in Wright (2022). All rock units in the area were represented by a density of 2.45 g/cc. Complete Bouguer anomaly data were produced and gridded using a kriging algorithm using a 50 m spacing. The data were further processed to produce residual gravity, first vertical derivative and horizontal gradient models.

The horizontal gradient model illustrates the rate of change in gravity response over horizontal distances, and clearly delineates the Rangefront Fault, as well as other likely faults to the east that are concealed by gravel cover (Figure 9-2). Of note is a north-south linear immediately to the east of the Rangefront Zone, as well as another approximately 3 km to the east. The residual gravity model reflects the relative depth of the (relatively less dense) gravel cover, with higher measurements reflecting areas with shallower gravel cover (Figure 9-3).

The gravity data will be reprocessed when sufficient drilling has been carried out in the subject area to generate a 3-D geology model.

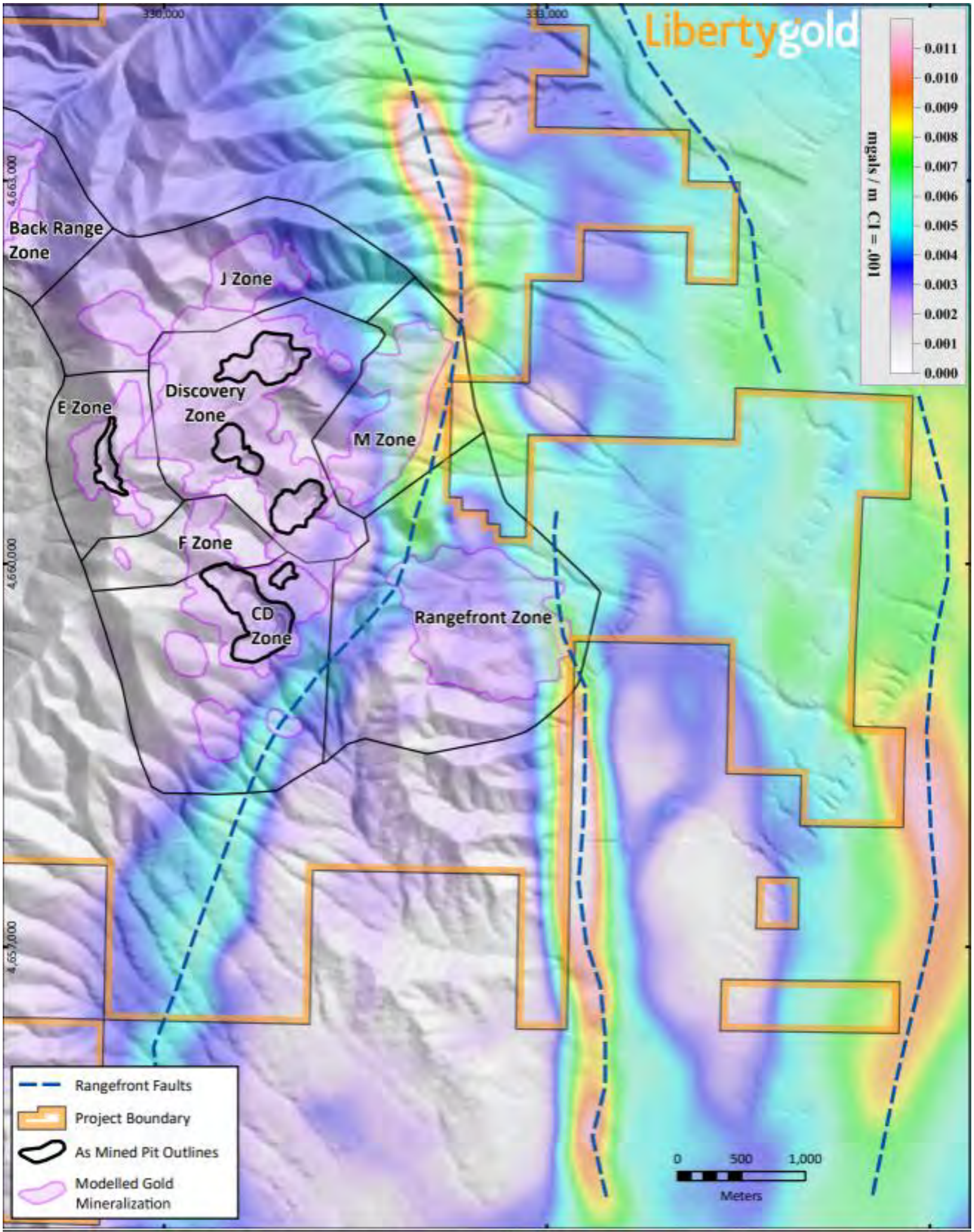


Figure 9-2: Horizontal Gradient Gravity

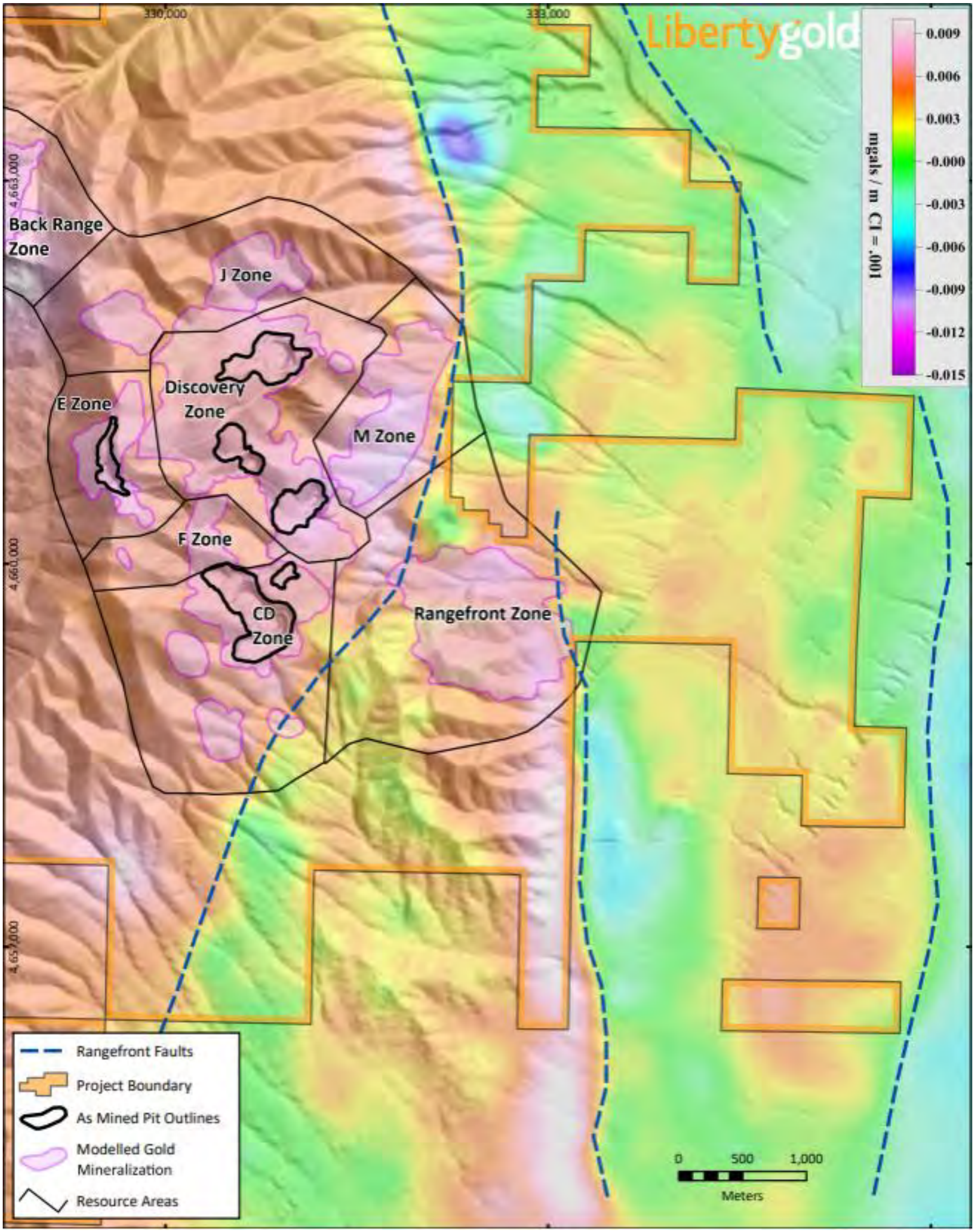


Figure 9-3: Vertical Derivative Gravity

9.5 Three-Dimensional Modeling

Liberty Gold has created a three-dimensional geological model for the Black Pine property in Leapfrog, in order to integrate surface mapping, drilling and structural data, and interpretations. The model is subject to revision as new data becomes available and is the primary platform for real-time analysis of drill data and drill hole planning. The geological model also forms the basis for the resource estimate. The 3-D model includes fault surfaces and solids representing the primary stratigraphic units described in Section 7.0, as well as surficial deposits including alluvium, pit backfill, and waste dumps.

9.6 Geotechnical Study

Golder Associates USA Inc. (Novak and Pegnam, 2022) was commissioned to provide geotechnical services related to three core holes (LBP456C, LBP429CA and LBP489C) drilled in 2021 near the western margins of the Discovery and CD resource pits. Data collection included comprehensive geotechnical logging, point load testing, and optical and acoustic televiewer surveying, as appropriate. Geotechnical logging included recording of core recovery, rock quality designation, fracture count, alteration and weathering, joint condition, geological strength, identification of weak zones, and descriptions of discontinuities. Televiewer reconciliation was completed using Advanced Logic Technology WellCAD software, with manual identification of structural orientations on 23 holes (13 core and 10 RC). An additional 46 samples were sent to Call and Nicolas Inc for point load testing and unconfined compression tests.

9.7 Summary Statement

The QP has not analyzed the sampling methods, sample quality, sample representativity, or possible presence of bias related to the Black Pine surface samples at the Black Pine Project because these data are superseded in relevance by the available drill data. Drill procedures and results are described in Section 10.0.

10 DRILLING

10.1 Summary

Liberty Gold carried out drilling programs in 2017 and 2019-2023 as tabulated in Table 10-1. Figure 10-1 shows the locations of all Liberty Gold drill hole collars within the Black Pine property by year, including holes that were abandoned at shallow depth due to adverse drilling conditions and water-related holes. Figure 10-2 shows all drill holes in the drill database relative to the outline of modelled gold mineralization.

Table 10-1: Summary of Liberty Gold Black Pine Project Drilling

Year	No. RC Holes[1]	RC (m)[2]	No. Core Holes[1]	Core (m)[2]	Total (Holes)[1]	Total (m)[2]
2017	14	2,274	0	0	14	2,274
2019	86	22,537	6	1,252	92	23,788
2020	160	43,874	10	2,352	170	46,227
2021	245	69,647	11	2,163	256	71,811
2022	318	66,381	6	1,118	324	67,499
2023[3]	149	27,461	11	1,401	160	28,862
Total	972	232,174	33	8,287	1,016	240,461

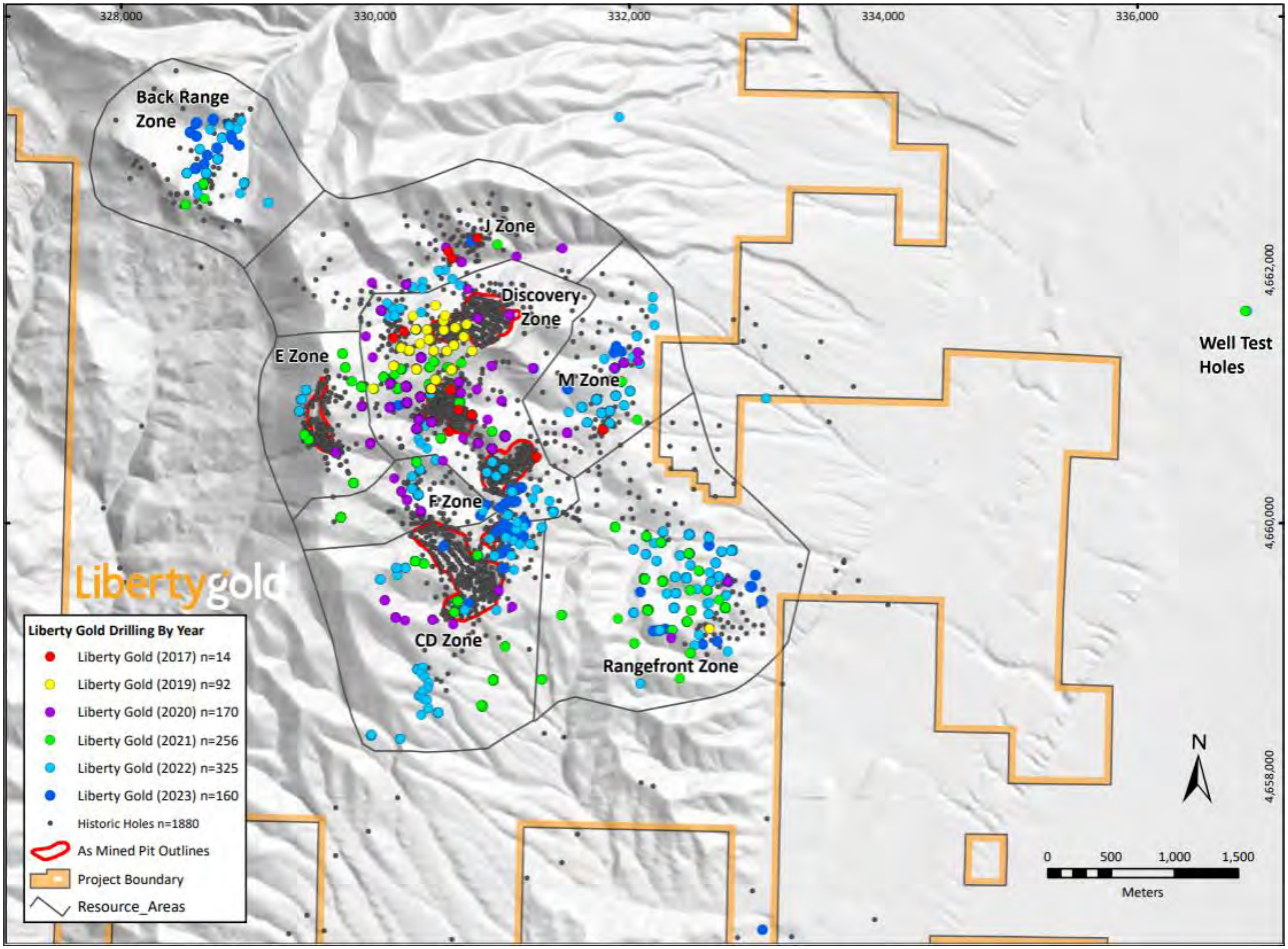
Notes:
1. The number of holes includes both holes that were abandoned prior to reaching target and were re-drilled, resulting in an "A" designation for redrilled holes, and holes that were drilled well outside of the resource area. SLR did not use abandoned holes in the resource database.
2. Meterage includes meters drilled in abandoned holes.
3. 2023 drilling statistics include 10 core holes that were drilled in 2023 but not included in the resource estimate.

10.2 Drilling Description

The 2017 drilling contractor was Boart Longyear Company of Elko, Nevada. A track-mounted Foremost MPD 1500 drill rig was utilized with 14.0 cm diameter center-return bits. On very deep holes (usually >1500') where significant groundwater was intersected, the center return hammer was switched out for a tri-cone rock bit. All drilling was done with water injection. Drill cuttings were split and sampled over 1.52 m (5 ft) intervals using a rotating wet "cyclone" vane-type splitter. The split samples fell directly into pre-labeled water-permeable cloth sample bags that were sealed. Excess water drained from the sample bags as they sat at the drill sites. Sample weights were generally in the range of 3 - 10 kg after drying.

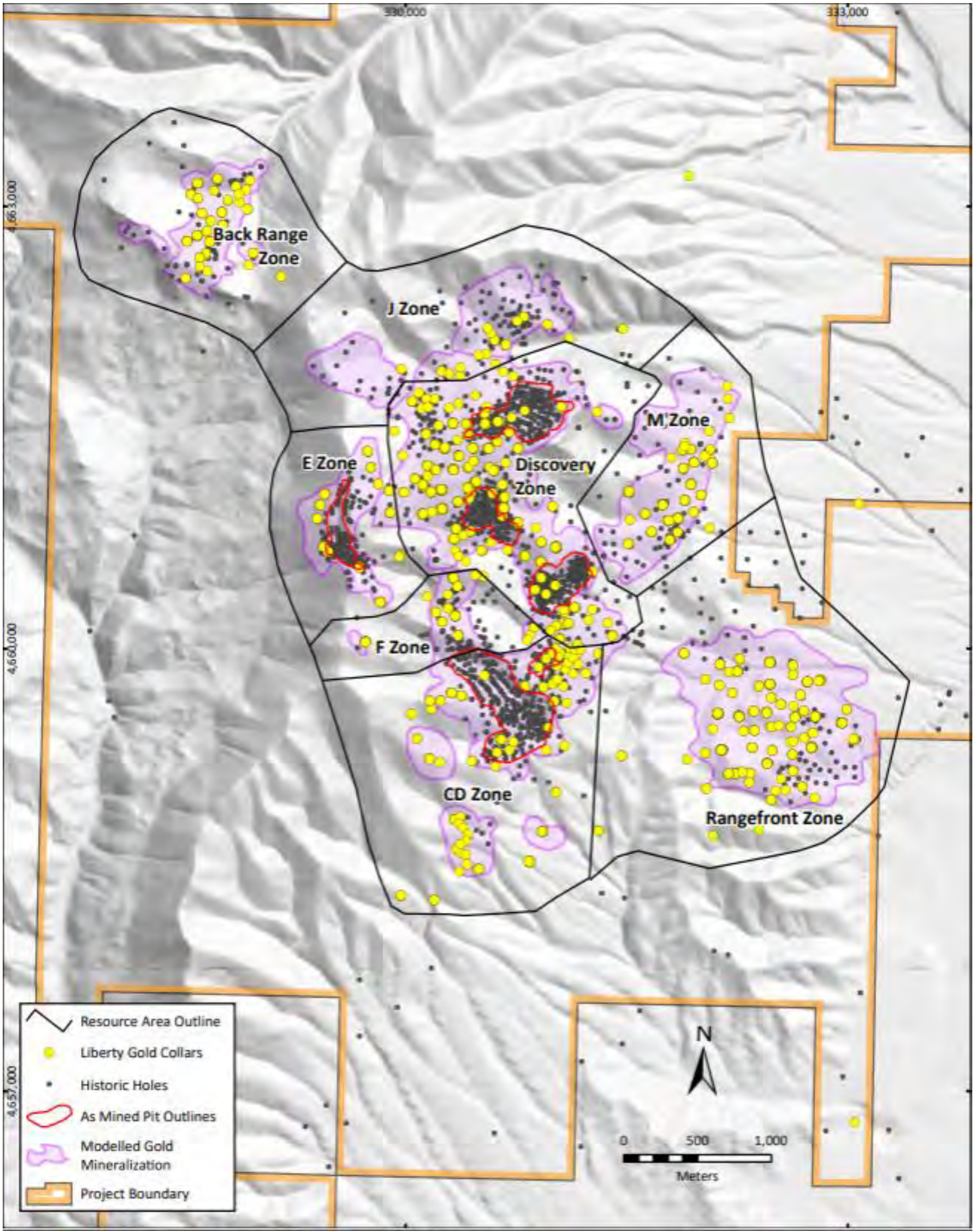
The 2019 RC drilling was undertaken by Boart Longyear with two track-mounted Foremost MPD 1500 drill rigs that used 14.0 cm diameter center-return bits. Sample handling techniques were identical to that in 2017. The 2019 core drilling contractor was Timberline Drilling Inc. of Elko Nevada. Drilling was carried out using a track-mounted Atlas Copco CS14 using primarily PQ (85 cm diameter core) tools.

The 2020 to 2023 RC drilling contractor was Boart Longyear. Two or more track-mounted Foremost MPD 1500 drill rigs were utilized with 14.0 cm center-return bits, and one truck-mounted Atlas Copco Super 10 also with the same bits. Sample handling techniques were identical to that in 2019. The 2020 – 2022 core drilling contractor was Major Drilling of Salt Lake City, Utah. Drilling was carried out using a track-mounted LF90 and a truck-mounted LF230 to recover primarily PQ and some HQ (63.5 cm diameter) core. The 2023 core drilling contractor was Drilcor of Coeur d'Alene, ID. Drilling was carried out using a track-mounted omni x-101 drill to recover PQ core.



Source: Liberty Gold (2024)

Figure 10-1: Liberty Gold Drilling by Year



Source: Liberty Gold (2024)

Figure 10-2: Drill Hole Collars with Outline of Modelled Gold Mineralization

10.3 Drill-Hole Collar Surveys

For the 2017 and 2019 drill collars, locations were initially marked in the field by Liberty Gold personnel using a Trimble GeoXH hand-held GPS receiver with differential correction accuracy of 0.5 m horizontally and 1.0 m vertically. Drill holes were abandoned in accordance with the State of Idaho Rules for Exploration Operations and Required Reclamation (IAR 20.03.02.06) as well as Sawtooth National Forest policies. After completion of the holes, the collars were marked with stamped brass tags fastened to a steel wire, and their locations were surveyed by Liberty Gold personnel using the Trimble GeoXH GPS receiver.

From 2020 through 2023, drill collars were marked and abandoned in the same manner as the 2019 holes. A Juniper Systems Geode Sub-meter GPS receiver was used by Liberty Gold staff to survey the collar locations.

10.4 Down-Hole Surveys

For Liberty Gold's 2017 drilling, down-hole deviation surveys were carried out by International Directional Services ("IDS") of Elko, Nevada. IDS utilized a truck-mounted surface-recording gyro (SRG) survey instrument for some holes, and a north-seeking gyro (NSG) instrument for other holes. Readings were taken at the bottom and top of the holes, as well as at intervals of 15.2 m along the lengths of each hole.

For Liberty Gold's 2019 drill program, downhole surveys were conducted using a north-seeking, solid state gyroscopic tool (Reflex EZ-Gyro) that was rented from Imdex Limited, along with a paired depth counter and a wireline winch mounted on a trailer. The tool was programmed to read at set depth intervals of 15.2 m as it traveled down the hole, with a second survey run at the same intervals on the way out of the hole. The downhole surveys were completed by Liberty Gold personnel, who then manually downloaded the data from the tool to a handheld device where the data was checked for accuracy before the hole was abandoned.

For Liberty Gold's 2020 through 2023 surveys, a north-seeking, solid-state gyro tool made by SPT was rented from IDS, along with a blue-tooth-paired depth counter and wireline winch. The tool reads continuous as it travels the hole, with the Gyromaster software programmed to export a reading at each 15.2 m interval. The downhole surveys were completed by Liberty Gold personnel, with specific protocol followed to verify precision of the survey before the hole was abandoned. The Gyromaster software compares the in-run and the out-run of each survey, and a threshold of <1.4% variance is met before any survey is considered complete. If needed, such as on deeper holes and holes steeper than -75 degrees, spring centralizers were installed on the tool to reduce rotation during the survey.

10.5 Sample Quality and Down-Hole Contamination

Down-hole contamination is always a concern with holes drilled by rotary (RC or conventional) methods. Contamination occurs when material originating from the walls of the drill hole above the bottom of the hole is incorporated with the sample being extracted at the bit face at the bottom of the hole. The potential for down-hole contamination increases substantially if significant water is present during drilling, whether the water is from in-the-ground sources or injected by the drillers. Conventional rotary holes, in which the sample is returned to the surface along the space between the drill rods and the walls of the drilled hole, are particularly susceptible to down-hole contamination.

While small areas of perched water have been intersected by Liberty Gold, no consistent groundwater table has been intersected Liberty Gold in the main mineralized zones, and there are no records that indicate that the historical holes intersected significant water either. A Draft Environmental Impact Statement prepared for Pegasus states "Pegasus exploration wells did not encounter any measurable groundwater at depths between 300 and 700 feet (91 to 213 m) below the surface and there are no perennial or intermittent streams in the project area" (USDA Forest Service, 1993). Only on deep holes, that tested well into the lower plate section of rocks, was sustained groundwater intersected.

During the detailed explicit modelling of the gold mineral domains discussed in Section 14.0, none of the signs of potential down-hole contamination were recognized in any of the holes drilled by historical operators or Liberty Gold.

The lack of groundwater in the mineralized zones, coupled with the relatively shallow depths of the historical holes (average and median down-hole depths of 102 m and 92 m, respectively) played significant roles in the mitigation of material contamination issues at Black Pine. In addition, Liberty Gold required center-return bits to be used for all of their RC holes, except in cases where excellent ground conditions were encountered or exploratory drilling in areas without known mineralization. This method minimizes the distance between the bit face as it breaks the rock and the collection of the sample into the inner tube of the RC drill pipe, which thereby further minimizes the potential for contamination. In cases where a conventional hammer bit and interchange were used, care was taken to make sure the return sample stream was coming back clean to ensure that no material was flushing downhole after adding rods, prior to sampling. In nearly every RC drill hole, drillers pump a light-to-medium-weight mixture of bentonite between the drill rods and the walls of the drill hole, which greatly reduces hole caving and downhole contamination.

10.6 Liberty Gold Drilling Summary

Drilling prior to mid-2021 is described in detail in Gustin et al (2021), including tables of results for various target areas. The reader is also referred to numerous Liberty Gold press releases between 2017 and 2023 available on the Company website, with links to graphics and tables of results with collar locations, downhole information, and assay results for all holes drilled from the inception of drilling in 2017 through 2023.

In late 2017, Liberty Gold drill-tested five target areas with 14 RC holes (B Pit Extension, Tallman Pit NE, and A Basin (Discovery Zone), J Anomaly (J Zone), and Hazel Pine Mine (M Zone); Figure 10-2). The primary purposes of this drilling were to validate drilling carried out by previous operators and to familiarize Liberty Gold with both mineralized and unmineralized rock. As such, drill sites were sited either immediately adjacent to historical pits or in established target areas. The 2017 holes were drilled from sites permitted under Western Pacific's 2012 PoO. These site locations were designed by Western Pacific without the benefit of knowledge of over 1,300 historical drill holes, the data from which were obtained later. Consequently, sites were not always optimally located relative to drill targets. Hole LBP012 was lost in underground mine workings at a depth of 13.2 m and redrilled. All drill holes were inclined at angles ranging from -45° to -80°.

In 2019, after the receipt of a PoO that allowed access to most of the area of surface mineralization at Black Pine, Liberty Gold completed a larger drill program encompassing 86 RC holes and six core holes. The core program was designed to obtain large diameter core for metallurgical testing, as described in Section 13.0. The RC program was primarily designed to explore an area between the historical B Pit, historical A Pit and historical A Basin target, where 3D geological modelling had identified a large area thought to contain extensions of surface gold mineralization in the pit highwalls and A Basin beneath the limit of historical drilling. The 2019 drilling identified two significant zones of mineralization: "Discovery 1", or "D-1", a northwest-striking, moderately northeast-dipping zone of mineralization extending from the A Basin area to the historical B extension Pit; and "Discovery 2", or "D-2", a relatively flat-lying zone of mineralization extending in a north-easterly direction from the Discovery 1 zone to the A Pit highwall. This nomenclature was later abandoned in favor of "Discovery Zone" (Figure 10-2) as the two zones eventually expanded and merged, and with the recognition that the D-2 zone was a down-dip extension of mineralization along a listric normal fault.

In 2020, RC drilling continued in the Discovery Zone, discovering a north-striking, moderately to steeply east-dipping zone of gold mineralization lying immediately west of, and eventually merging with, the Discovery Zone. This zone was named the "Discovery 3", or "D-3" zone. This terminology was also later abandoned, and the mineralization lumped with the rest of the Discovery Zone, although it is still recognized as controlled by a separate listric normal fault with a more northerly orientation that continues southward through the F Zone and merges with the north end of the CD Zone. Other targets of RC drilling in 2020 included the:

- Southeast extension of the Discovery Zone between the B Extension Pit and the Tallman Pit
- Northwest extension of the Discovery Zone
- J Zone
- F Zone between the CD Pit and B Extension Pit
- M Zone
- Southwest Extension target west of the historic I Pit

Core drilling was carried out primarily for the purpose of obtaining samples for additional metallurgical testing in the Discovery, E, and CD Zone areas, as described further in Section 13.0.

In 2021, RC drilling resumed on April 1, targeting primarily infill drilling in the Discovery Zone for resource classification upgrade, expansion of known mineralization along the CD pit western margin, and exploratory drilling in the Rangefront area. In July 2021, gold mineralization was encountered in five holes drilled from a single site west of the 2021 Rangefront resource pits, leading to discovery of a zone of mineralization, briefly designated “D-4” in press releases from that time, that now encompasses a roughly 1 km^2 area. Infill and step-out drilling continued in the (expanded) Rangefront Zone throughout the remainder of the year.

Core drilling targeted areas not well covered by previous metallurgical drilling, including the newly defined Rangefront Zone, as well as the M, E, CD, and F zones.

Three core holes were drilled, including two in the Discovery Zone and one in the CD Zone for the primary purpose of collecting geotechnical data through use of a televiewer, point load tester, and geotechnical logging (see Section 7.0).

Core orientation was carried out subsequent to drilling by Golder Associates USA Inc. (Novak and Pegnam, 2022), who reconciled features in the drill core with the oriented televiewer data furnished by IDS out of Elko, Nevada.

In 2022, RC drilling commenced in early January and continued through mid-December. Drilling encompassed:

- Infill and step-out drilling in the Rangefront Zone for resource upgrade and expansion purposes.
- Targeting of areas of near-surface, above average grade that might favorably impact the first few years of a mining operation. Target areas included the M, F, and Back Range zones.
- Amalgamation of zones consisting of two or more small resource pits defined by primarily shallow historical drilling. Target areas included the M, F, and Back Range zones.
- Expansion drilling along the margins of existing zones, including the western margins of the CD and E zones.
- Identifying and defining areas of mineralization associated with surficial deposits, including waste rock storage areas and historic pit backfill.
- Testing of new targets including Bobcat, South Rangefront, Section 36, and Next Canyon Up targets.

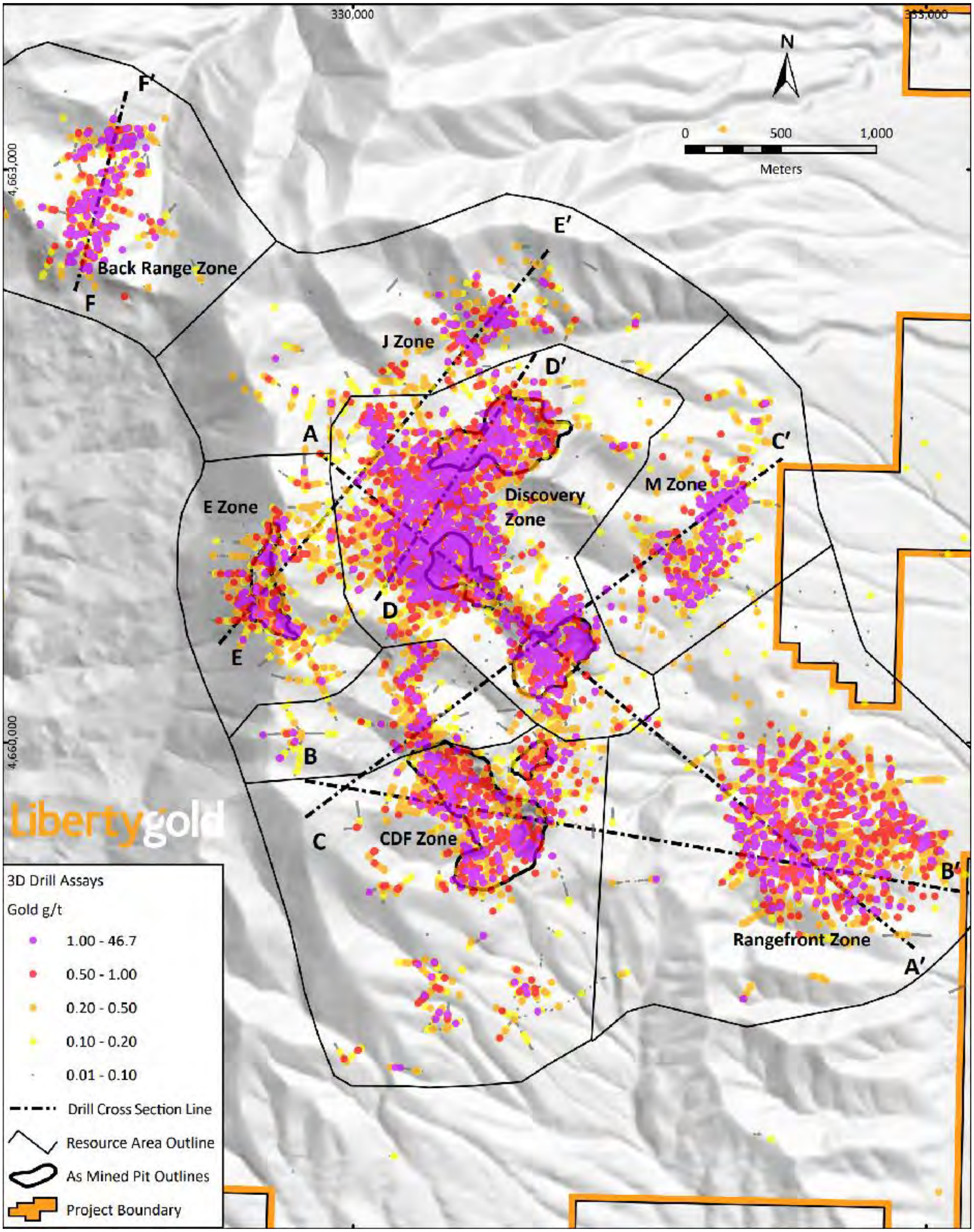
In 2023, RC drilling commenced in early January and continued through the end of August. Drilling focused on:

- Resource upgrade in Rangefront, M, Back Range and Discovery zones
- Step out and expansion drilling in Rangefront east, M zone and Back Range and between the historic CD Pits and Talman Pits, which included further testing through historic waste rock dumps.
- One hole also drilled in South Rangefront. (approximately 2 km south of the Rangefront Zone)
- Eleven PQ sized core holes testing Back Range, J, M, F, Rangefront and Discovery zones to fill metallurgical data gaps.

10.7 Mineralized Zone Descriptions

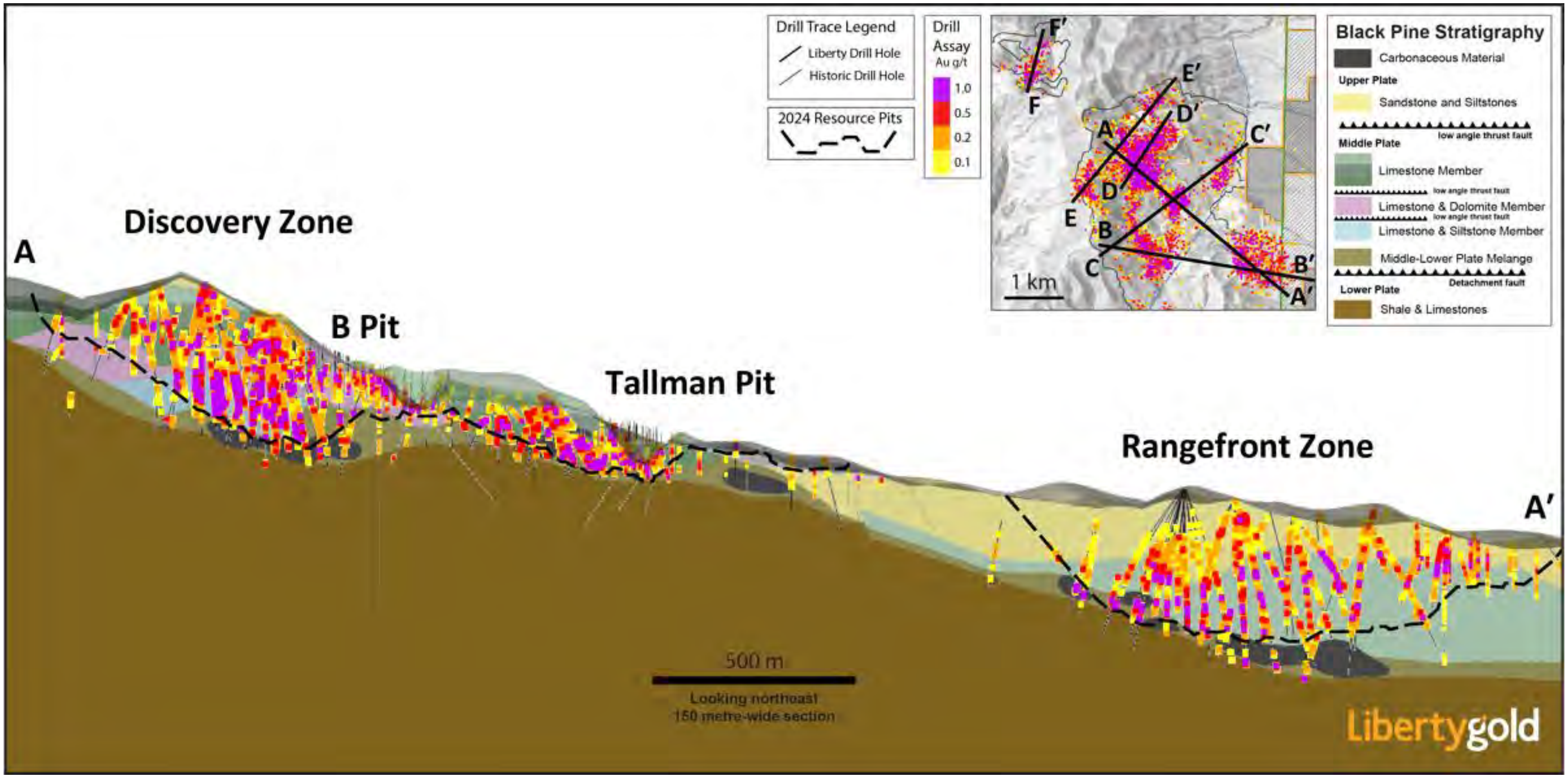
Zones subject to Liberty Gold drilling are described in more detail below and illustrated in Figure 10-3 through Figure 10-9. Zone descriptions reflect the most recent drilling and interpretations in these areas since the previous resource estimate (Gustin et al, 2021). Additional tables of drill results from 2017 – 2020, cross sections and zone descriptions can be found in Gustin et al (2021).

Over time, continued drilling by Liberty Gold demonstrated continuity between smaller zones of mineralization identified by previous operators, such that some zone and target names were expanded to include nearby named targets, with the older names abandoned. Also, the large size of the Black Pine gold system made modelling of the system as a single entity impractical. As such, zone and target names were grouped into seven primary zones as shown Figure 10-3. Figure 10-4 through Figure 10-9 illustrate the major mineralized zones in cross section.



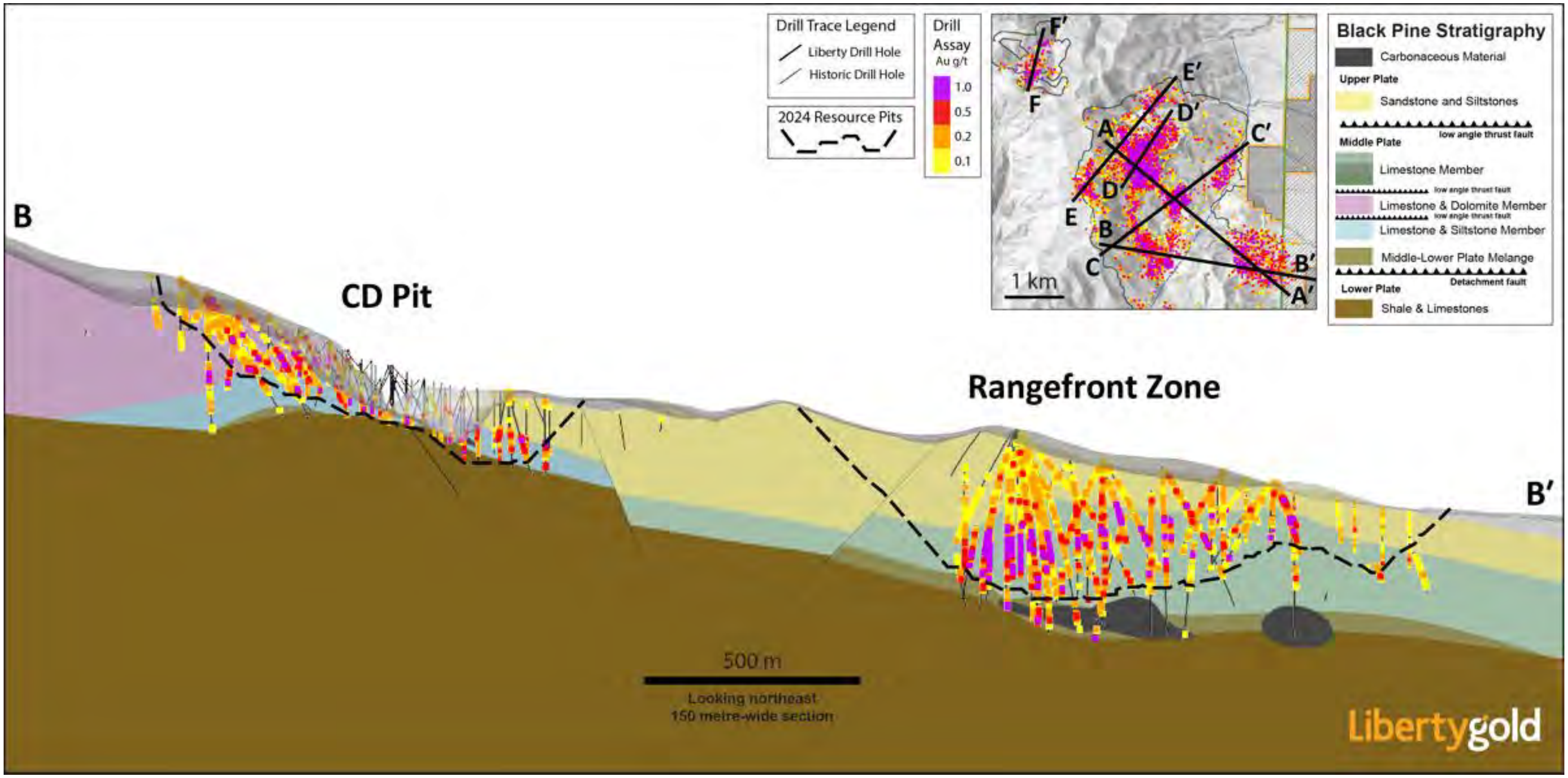
Source: Liberty Gold (2024)

Figure 10-3: Map of 3D Drill Assays and Cross Section Locations



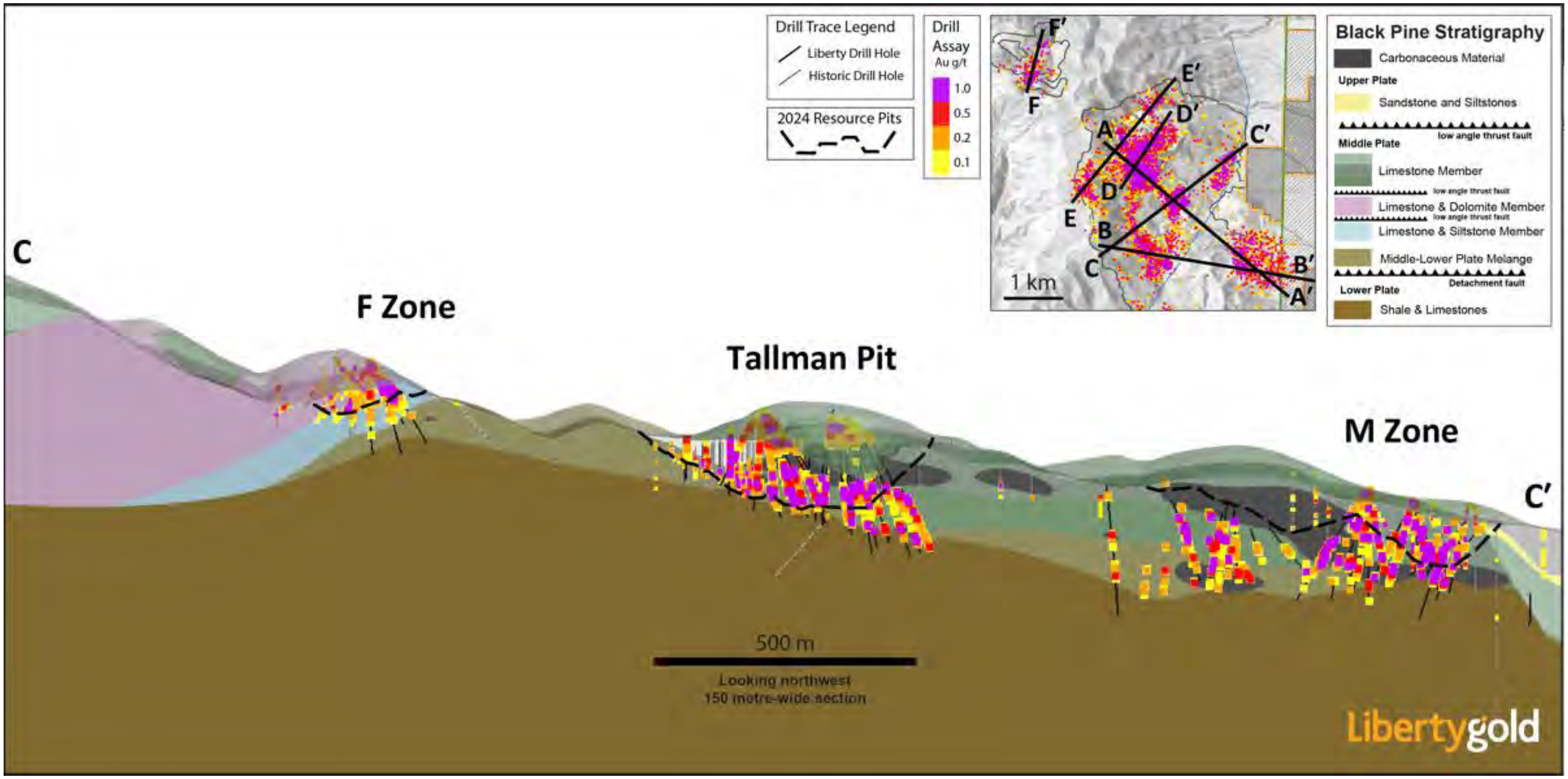
Source: Liberty Gold (2024)

Figure 10-4: Cross Section A-**A'**



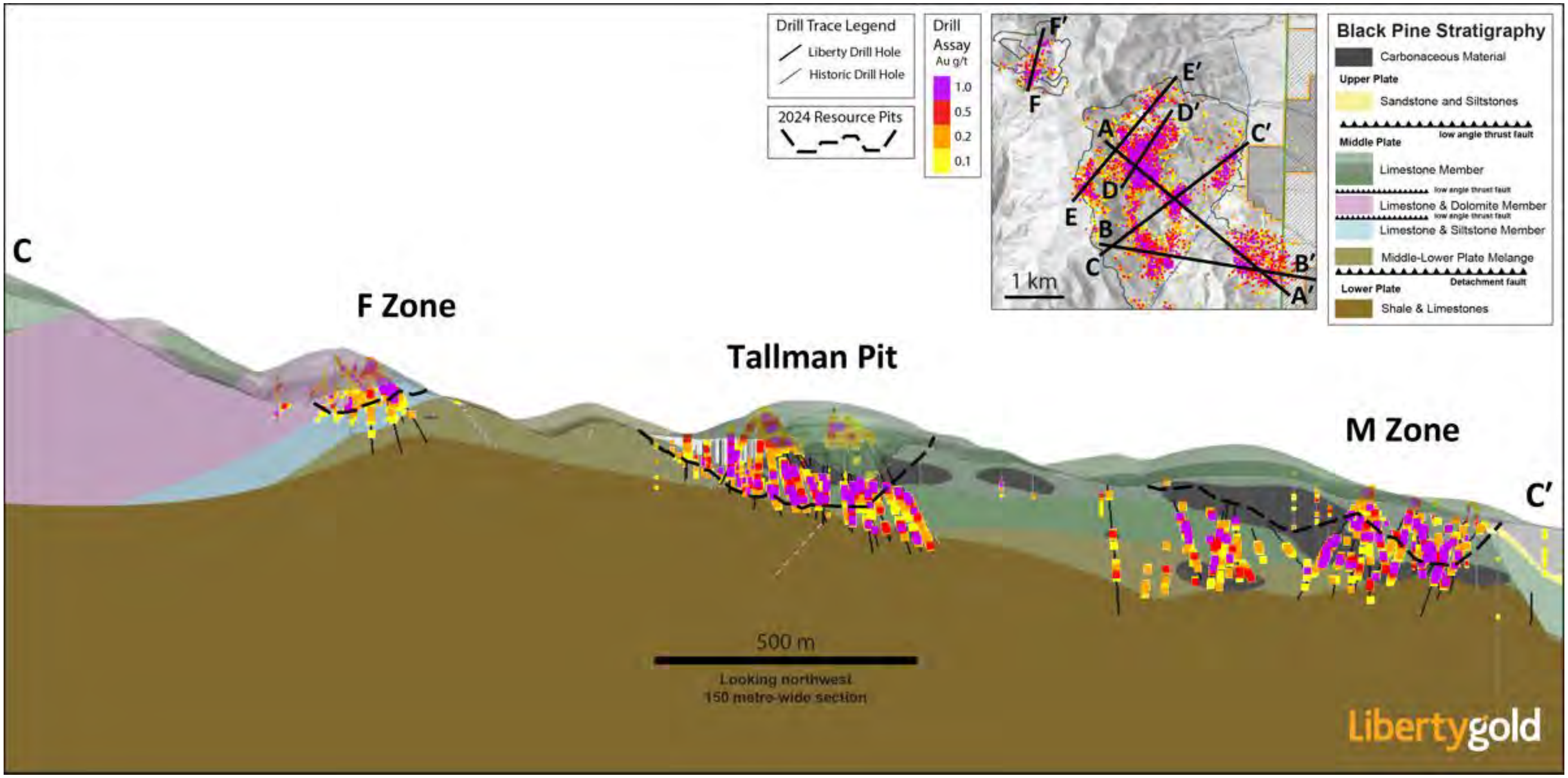
Source: Liberty Gold (2024)

Figure 10-5: Cross Section B-**B'**



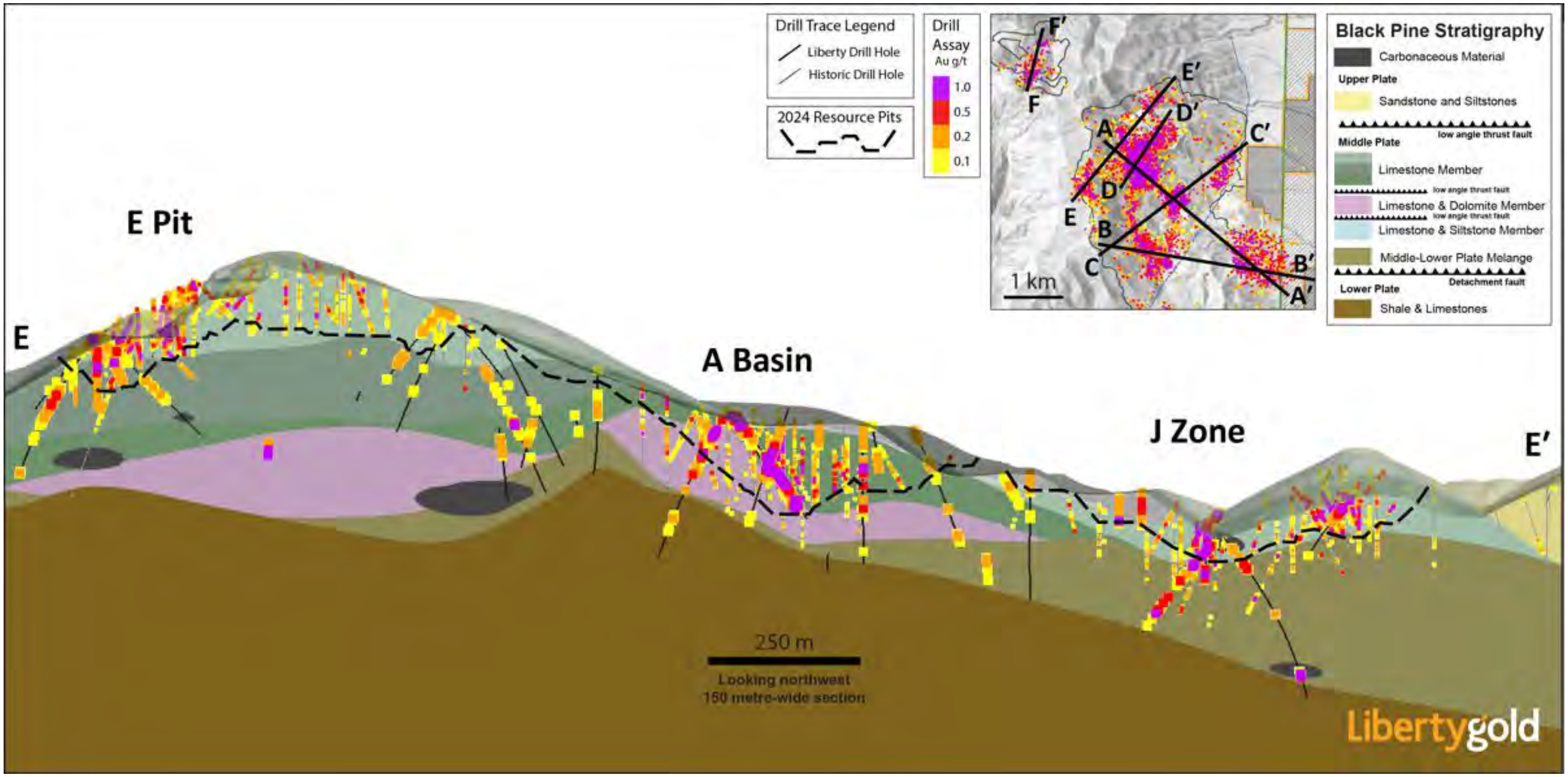
Source: Liberty Gold (2024)

Figure 10-6: Cross Section C-**C'**



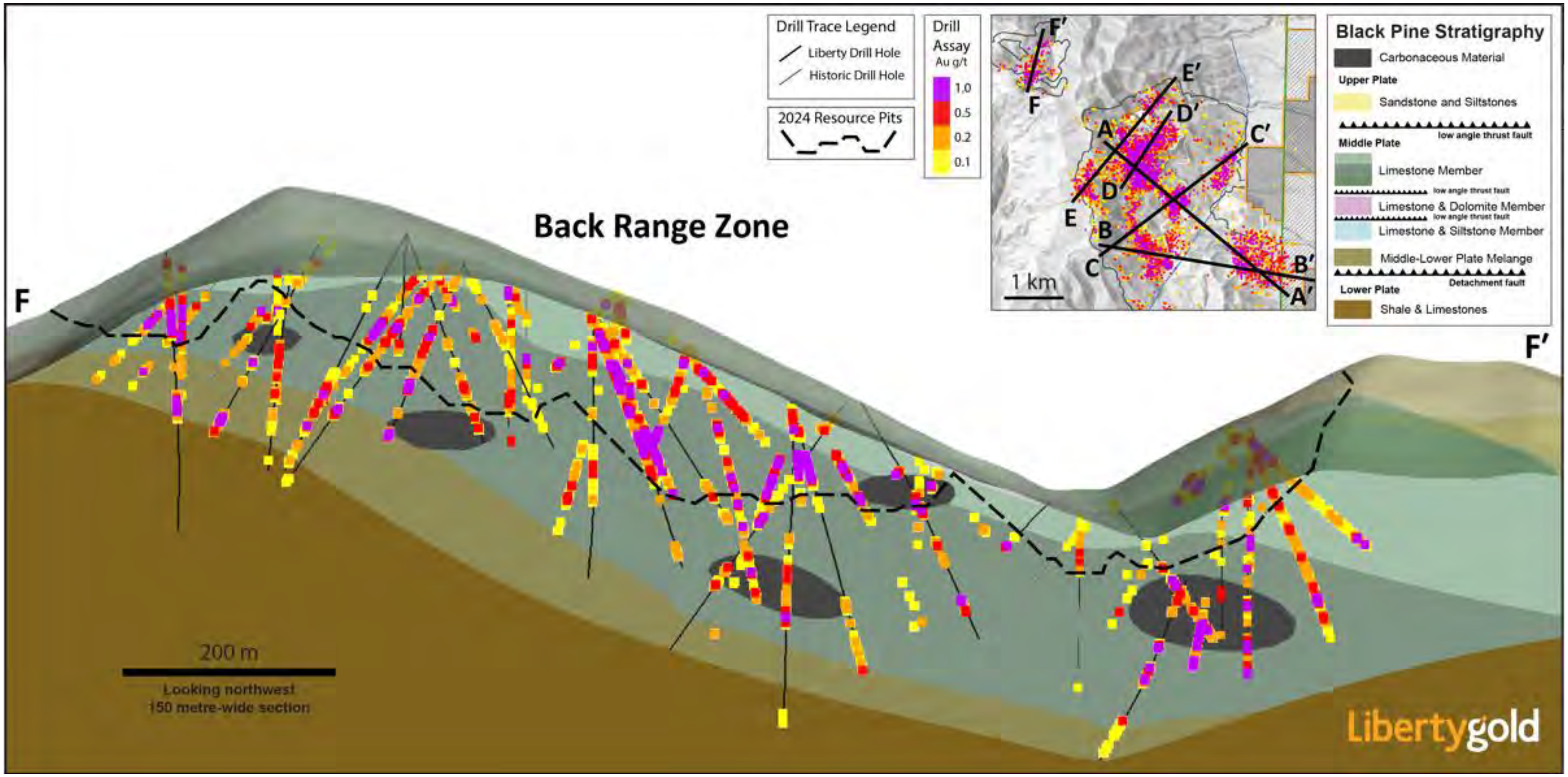
Source: Liberty Gold (2024)

Figure 10-7: Cross Section D-**D'**



Source: Liberty Gold (2024)

Figure 10-8: Cross Section E-**E'**



Source: Liberty Gold (2024)

Figure 10-9: Cross Section F-**F'**

10.7.1 Discovery Zone

The D-1, D-2, and D-3 zones are referred to collectively as the “Discovery Zone”, as they lie in close proximity, with the D-2 Zone interpreted as a down-dip extension of the D-1 zone in the immediate hanging wall of a listric normal fault, and the D-3 Zone interpreted as lying in the hanging wall of a second listric normal fault to the immediate west that may join into the D-1/D-2 structure at depth (Figure 10-7).

Shallow historic drill holes defined mineralization along a ridge between the historic B and A pits and shallow mineralization in the A Basin area to the northwest. Deeper mineralization in the Discovery Zone was discovered in 2019 with hole LBP021, drilled in the 700 m gap between LBP002 in the A Basin area and the highwall of the B Pit. This hole returned a 47.2 m interval at an average grade of 1.78 g/t Au. Gold mineralized rocks are highly oxidized, and higher grades are associated with reddish brown, variably brecciated, and strongly decalcified calcareous siltstone of the Polc member of the middle plate of the Oquirrh Group. Drilling continued throughout 2019 and 2020, defining a relatively flat zone of mineralization underlying most of the area between the historic B and A pits, and a steeper, northeast-dipping zone of mineralization extending to the northwest and southeast, merging with remanent mineralization in the vicinity of the historic Tallman Pit. With further drilling, the steeper and flatter zones are now interpreted as mineralization in the hanging wall of a listric normal fault.

In 2021, a second listric normal fault with mineralization in the hanging wall was discovered through drilling targeting the up-dip extension of the D-1 Zone to the west. LBP127 intercepted 33.5 m grading 1.98 g/t Au and 30.5 m grading 1.11 g/t Au in the footwall below the intended target area. This zone is relatively steeply dipping and is interpreted to lie in the hanging wall of a north-striking, east-dipping listric normal fault that merges with the original Discovery Zone listric normal fault to the north and down dip to the east along the low-angle normal fault that separates the middle plate from the lower plate. Drilling in 2022 demonstrated that the second listric normal fault continues to the south through the F zone and CD Zone. Gold mineralization is hosted in brownish, oxidized, variably brecciated, and decalcified calcareous siltstone and limestone of the Pols and PMmx members of the middle plate of the Oquirrh Group.

As of the Effective Date of this report, a total of 784 historic RC and Rotary holes, 13 historic core holes, 396 Liberty Gold RC holes, and 16 Liberty Gold core holes have been drilled in the Discovery Zone, including the Tallman pit area and A Basin.

10.7.1.1 Discovery Zone Southeast Extension

Drilling in 2019 through 2021 in the area southeast of the main Discovery Zone demonstrated continuity of mineralization from the Discovery Zone through the B Extension Pit to the Tallman Extension Pit along the previously identified listric normal fault at the base of the Discovery Zone. This 400-metre-long area was historically tested with a very few shallow holes. Gold mineralization extends from surface along the southeast side of the prominent ridge between the B and Tallman Extension pits northeastward, and it remains open down-dip to the northeast. The host rock for oxidized mineralization consists of strongly decalcified calcareous siltstone suspected to be the Polc or Pols member of the middle plate. Some of the mineralization, particularly down dip to the northeast, is hosted in black, carbonaceous siltstone with variable, but generally low, cyanide solubility. The D-1 SE Zone is still open for expansion to the north and east, although poor cyanide solubility in this area limits economic upside.

10.7.1.2 Discovery Zone Northwest Extension

The Discovery Zone Northwest Extension, starting with drill hole LBP002 and extending northwest of the historic A Basin target along the postulated extension of the Discovery Zone listric normal fault, has been tested with several drill holes. Most drill holes contain shallow, relatively low-grade gold mineralization associated with abundant calcite veins and calcite-cemented breccia (Calfm). Mineralization in this area is still open to the northwest.

10.7.2 J Zone

The J Zone lies on the immediate north side of Mineral Gulch and was tested by a number of shallow historical holes. Liberty Gold followed up with two drill holes in 2017, several holes in 2021 and one metallurgical core hole in 2023. J Zone mineralization appears to lie partially within massive, brecciated, variably decalcified limestone underlain by carbonaceous, pyritic siltstone (PMmx). Shallow J Zone mineralization above the level of the floor of Mineral Gulch is thoroughly oxidized, while much of the mineralization below the valley floor is carbonaceous with sporadic areas of disseminated pyrite. The J Zone is still open for expansion to the west, east, and north, although it is overlain by a thick sequence of unmineralized upper plate rocks to the north.

10.7.3 CD-F Zone

The CD-F Zone comprises the area around the historic CD and I Pits, with a narrow zone of mineralization (F zone) extending northward to join the Discovery Zone.

The CD and I Pit areas are flanked on all sides by a rind of unmined gold mineralization defined by shallow historic drilling. Drilling by Liberty Gold has extended mineralization down-dip to the southwest (CD Southwest Extension area), to the south around and under the I Pit, and down-dip to the east.

Historical drilling in the F Zone demonstrated that mineralization mined in the CD Pit continues to the north-northwest at shallow depth in the Pold and Pols units of the middle plate of the Oquirrh Group. Drilling beyond a few hundred meters north of the CD Pit is sparse due to steep terrain, but a series of historical holes drilled near the base of resistant outcrops of Pold returned shallow gold intercepts in what is interpreted as Pols siltstone and limestone overlying the PMmc. Drilling by Liberty Gold further to the west of these holes encountered scattered, relatively low-grade mineralization in the Pold unit, which appears to extend much deeper in this area than in the drill holes to the east. Drilling by Liberty Gold further to the south on a ridge separating the CD pit from the rest of the F zone to the north returned shallow mineralization up to 75 m thick. Drilling in 2022 demonstrated continuity of mineralization across the entire length of the F zone from the northwest edge of the CD pit to the southwest edge of the Discovery Zone along a strongly sheard silty and oxidized silty unit.

The Bobcat Zone, a separate area of mineralization located to the south of the I Pit, was tested with a number of shallow historic holes. Follow-up drilling of 27 RC holes by Liberty Gold in 2022 identified shallow oxide gold mineralization over a 400 m by 200 m area, with mineralization primarily focused along the gently dipping contact between the Pold and Pol members.

The Southwest Extension is a large area to the southwest of the CD and I historic pits. In this area, a gold-in-soil anomaly extends for over one square kilometer, where an over-thickened section of Pold is overlain by a sequence of platy limestone and siltstone equivalent to the Pola-b-c sequence to the north. Drill holes in this area returned shallow intercepts of relatively low-grade gold mineralization, extending gold mineralization up to several hundred meters to the southwest of historic drilling.

10.7.4 Rangefront Zone

The Rangefront Zone lies immediately south of the Black Pine Mine access road along the mountain front (Figure 10-3, Figure 10-4, and Figure 10-5). In this area, the Ppos unit of the upper plate of the Oquirrh Group is exposed on surface. The unit is variably limy and brecciated, with widespread but relatively weak gold-in-soil anomalies. The eastern portion of what is now identified as the Rangefront Zone was tested by 92 shallow historical holes, revealing widespread shallow gold mineralization. In 2019, Liberty Gold drilled a shallow core hole (LBP093C) in the middle of the eastern Rangefront Zone, which returned 55.3 m at a grade of 0.49 g Au/t from a depth of 46.2 m. Mineralization appeared to start at the base of the Ppos unit at the contact of a limestone unit assigned to the Pola member of the middle plate. A small resource in two separate pits was estimated in this area by Gustin et al (2021).

Liberty Gold returned to the Rangefront area in 2021, drilling five holes from a site approximately 700 m to the west of the 2021 resource pits, and along trend with the modelled extension of the NW-trending listric normal fault at the base of the Discovery Zone. LBP356 and LBP358 hit long intervals of relatively high-grade oxide gold mineralization (Table 10-2). Drilling continued in late 2021 and throughout 2023 and identified a zone measuring approximately 1 km by 1.3 km in 223 RCholes and 8 core holes drilled as of October 2023. Mineralization is still open to the north, east, and west.

The Rangefront Zone consists of gently east- to north-dipping anastomosing lenses of gold mineralization hosted in limy portions of the upper plate PPos member and middle plate Pola member over a vertical distance of up to 400 m. Mineralization at the base of the zone is transitional into the PMmx unit at the base of the middle plate and is locally carbonaceous with reduced cyanide solubility.

Table 10-2: Liberty Gold Discovery Holes in the Expanded Rangefront Zone

Hole ID (Az, Dip) (degrees)	From (m)	To (m)	Intercept (m)	Au (g/t)	Cut off Au (g/t)	Hole Length (m)
LBP356 (0, -55)	94.5	100.6	6.1	0.44	0.2	470.9
and	141.7	157	15.2	0.24	0.15	
and	253	339.9	86.9	0.91	0.15	
including	253	271.3	18.3	0.25	0.2	
including	285	339.9	54.9	1.32		
and including	286.5	309.4	22.9	2.15	1	
and including	315.5	318.5	3	2.83		
and including	289.6	291.1	1.5	5.75	5	
and	346	367.3	21.3	2.03	0.2	
including	349	365.8	16.8	2.52	1	
and including	352	353.6	1.5	5.15	5	
and	174.9	396.2	21.3	0.23	0.15	
LBP358 (45, -50)	222.5	251.5	29	0.22	0.15	434.3
including	225.5	230.1	7.6	0.33	0.2	
and	262.1	286.5	24.4	1.23	0.15	
including	263.7	281.9	18.3	1.58	0.2	
and including	266.7	281.9	15.2	1.85	1	
and	295.7	346	50.3	1.37	0.15	
including	295.7	339.9	44.2	1.54	0.2	
and including	295.7	297.2	1.5	2.65	1	
and including	307.9	330.2	22.9	2.37		
and including	324.6	326.1	1.5	6.75	5	

Notes:
1. Intercept length is assumed to approximate true thickness.

10.7.5 M Zone

The M Zone is an area of shallow gold mineralization lying along the range front north of the historical heap leach pad (Figure 10-3 and Figure 10-6). The M Zone originally consisted of two separate zones (M Zone and Hazelpine, located approximately 0.5 km south of the original M Zone) defined by 85 historic RC and rotary holes. At Hazelpine, historic shallow workings mined zinc and silver in addition to gold. Subsequent drilling of 88 RC and 4 core holes by Liberty Gold in 2017 through 2023, including the area between the two zones amalgamated them into a single zone, designated the M Zone. Gold mineralization is present in the immediate footwall of the moderately east dipping

Rangefront Fault in a zone that extends to the southwest. Shallower intervals, hosted in the Polc member, are strongly oxidized, while some deeper intervals, likely hosted in the PMmx member, are hosted in black, carbonaceous siltstone. Within the broader M zone, a number of en echelon, northeast-trending, higher-grade tabular bodies of mineralization have been identified. As the Effective Date of this report, the M Zone is still open to the north and west.

10.7.6 E Zone

A number of historical drill holes with shallow gold intercepts extend south, west, and north of the historical E Pit. Between 2020 and 2023, Liberty Gold drilled 41 RC and 2 core holes in this area, primarily along the west side of the zone. While mineralization in most areas of the Black Pine deposit is overall gently east-dipping, mineralization in this area, as well as the controlling stratigraphy and faults, rolls over on the eastern edge of the E Pit and assumes a westerly dip, nearly parallel to the west side of the ridge bounding the east side of the Black Pine Creek drainage. Mineralization in this area is still open down-dip to the west. (Figure 10-3 and Figure 10-8).

10.7.7 Back Range

A number of small historical workings in the Silver Hills area, now called Back Range, were mining for silver, gold and some base metals, similar to the Hazelpine mine (now M-Zone). Historical drilling of 66 RC and 3 core holes highlighted two mineralized zones with up to 2.30 g/t Au over 25.9 m. The geologic controls were poorly understood until Liberty Gold was able to add an additional 66 RC hole and 3 metallurgical core holes in 2022 and 2023. The modern drilling helped merge the two zones together and highlighted a strong N-NE trend to the mineralization. This NNE trend is controlled by highly deformed siltstone interpreted to be part of the Polb middle plate formation (Figure 10-9). These mineralized siltstones are bounded on the east and west by weakly mineralized more resistive limestone beds. Based on the amount of deformation observed in road cuts and in drill core there is a high possibility of a parallel siltstone unit further west of the limestone units, that remains untested.

10.8 Summary Statement

The overwhelming majority of sample intervals in the Black Pine resource database have a down-hole length of 1.52 m (5 feet) are appropriate for the style of the Black Pine mineralization.

The mineralization at Black Pine is predominated by gently dipping zones that mimic stratigraphic and low-angle, and structural controls, and the drill holes cut these zones at high to moderate angles. There are a few, relatively small areas where mineralized dips increase, and some holes cut this mineralization at acute angles that can yield exaggerated downhole widths. This effect is entirely mitigated by the explicit modelling techniques employed in the estimation of the current resources, which constrain all intercepts to lie within explicitly interpreted domains that appropriately respect the known and inferred geologic controls.

1) The QP is unaware of any sampling or sample-recovery factors that would materially impact the accuracy and reliability of the drill-hole data and is of the opinion that the drill samples are of sufficient quality for the purposes used in this report.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section summarizes all information known that relates to sample preparation, analysis, and security, as well as quality assurance/quality control ("QA/QC") procedures employed and results obtained, that pertain to the Black Pine Project. Historical information has been compiled from historical records in Liberty Gold's possession and is largely unchanged from Gustin et al (2021).

11.1 Sample and Preparation and Analysis

11.1.1 Historical Surface and Drilling Samples

With the exception of Pegasus drill samples from 1990 through 1997, which were assayed at the Black Pine mine laboratory, all historical samples were analyzed at laboratories independent of the historical operators, and it is not known what, if any, certifications these laboratories held at the times they were used.

Newmont 1963 and 1964: Newmont submitted rock-chip samples to Union Assay for gold and silver analyses. Newmont's 1964 drilling samples were also sent to Union Assay for gold and silver by fire assay. MDA has no information on sample preparation procedures used by Union Assay on the Newmont samples.

Newmont 1974: Newmont submitted soil samples to Rocky Mountain Geochemical Corp. ("Rocky Mountain") in Midvale, Utah. These were analyzed for gold, silver, lead, and zinc by atomic absorption ("AA"), and arsenic was determined calorimetrically. Cuttings from the initial 11 holes drilled in 1974 were sent to Rocky Mountain's laboratory in Salt Lake City, Utah for gold and silver assays by AA. Drill cuttings from the last nine holes drilled in 1974, as well as a few samples from the end of the last hole analyzed by Rocky Mountain, were sent to Skyline Labs ("Skyline") of Tucson, Arizona for gold and silver fire assays.

Check assays on 38 drill samples from the 1974 program were performed by fire assay by Union Assay for gold and silver. One hole was also assayed for lead and zinc by Union Assay, but the method of analysis is not known. MDA has no information on procedures and methods used for sample preparation by Rocky Mountain, Skyline, and Union Assay.

There is evidence that the Rocky Mountain 1974 AA gold analyses were cyanide-soluble analyses. If true, the gold values in the resource database would represent only the cyanide-soluble portion of the total gold contents of the samples.

Gold Resources 1974 - 1976: Gold Resources submitted rock and soil samples to Rocky Mountain, and these were analyzed for gold, silver, arsenic, mercury, and copper; the methods of sample preparation and analysis are not known. The cuttings from their 1974 through 1976 drill holes and selected intervals of core were sent to Union Assay for fire assay analyses of gold and silver.

Kerr Addison Mines Ltd. 1975: Kerr Addison used Vangeochem Lab Ltd. of North Vancouver, B.C. for copper, zinc, and gold analyses.

Pioneer Nuclear 1979 - 1981: Pioneer's 1979 drill samples were sent to Rocky Mountain and analyzed for gold by fire assay. The 1980 drill samples were sent to Union Assay in Salt Lake City, Utah where gold and silver were analyzed by fire-assay methods. Gold and silver from the single hole drilled in 1981 were analyzed by fire assay at Cone Geochemical Inc. in Denver, Colorado. No additional information on the methods of sample preparation and analysis that were used by these laboratories is available.

Pegasus 1983 - 1985: Pegasus collected several hundred rock-chip and soil samples across the Black Pine Mountains and the future mine area. Assay certificates and sample locations are not available, but summary sheets indicate they were assayed for gold, silver, and mercury, with occasional antimony and arsenic analyses as well. Drill samples in 1983 were analyzed for gold by fire assay methods by Union Assay and Rocky Mountain, both at their Salt Lake City, Utah, laboratories. No records are available with respect to the assaying of the samples from the 36 holes drilled in 1984, and no other information is available on the sample preparation and analytical methods used.

Permian Exploration and Pegasus 1984: Drill samples were sent to Union Assay in Salt Lake City, Utah for fire assay of gold and silver with 30-gram aliquots. Some samples were also sent to Rocky Mountain in Salt Lake City, Utah. No other information is available on the sample preparation and analytical methods used.

Noranda 1986 - 1989: Noranda carried out extensive soil sampling across the property. In 1986 through 1989, soil samples were analyzed at Chemex Labs Inc. ("Chemex") in Sparks, Nevada for gold by fire assay with an AA finish. In 1988 and 1989, Noranda's rock-chip samples were analyzed at Geochemical Services Inc. ("GSI") in Torrance, California for silver, arsenic, gold, mercury, and antimony. No other information is available on the methods and procedures used for sample preparation and analyses.

Samples from Noranda's 1986 drilling were analyzed at several laboratories. Rocky Mountain in West Jordan, Utah determined gold and silver by fire assay on 30-gram aliquots. Samples from previously analyzed holes were sent to Assay Lab Inc. ("Assay Lab") in West Jordan, Utah for 30-gram fire assay of gold and silver. Cuttings for at least 12 holes were sent to GSI for 30-gram fire assay with gravimetric finish. Samples from multiple holes were also sent to Chemex in North Vancouver, British Columbia, for 30-gram fire assays for gold. No other information is available on the methods and procedures used for sample preparation and analyses.

The 1987 and 1988 drilling samples were mainly sent to Analytical Services Inc. ("ASI") in Elko, Nevada for 30-gram fire assay of gold with a gravimetric finish. For some samples, gold was determined by fire assay at GSI and Chemex. Some check assays for gold were also done by ASI, and others were conducted by Legend Metallurgical Laboratory Inc. (Legend) in Reno, Nevada and GSI using 30-gram fire assay with a gravimetric finish. No other information is available on the methods and procedures used for sample preparation and analyses.

All of the 1989 drill samples were analyzed for gold by Legend using a 30-gram fire assay procedure. No other information is available on the methods and procedures used for sample preparation and analyses in 1989.

Pegasus 1990 - 1997: Pegasus collected several thousand rock-chip samples across the Black Pine property. These were routinely analyzed for gold, silver, arsenic, barium, bismuth, antimony, and mercury, and occasionally for copper, lead, zinc, and molybdenum. No sample certificates are available and there is no information regarding assay laboratories, sample preparation, or analytical methods.

The Pegasus drill samples during this time period were assayed on-site at the Black Pine mine laboratory. Every sample was analyzed for gold by a hot cyanide leach ("HCL") procedure. If the HCL analysis reported was greater than 0.005 oz Au/ton (0.17 g Au/t), the sample was also analyzed for gold by fire assay, and the fire assay value was entered into the historical drill-hole database. If runs of typically up to five to ten consecutive samples returned HCL values of 0.005 oz Au/ton or less, a fire assay was also completed irrespective of the HCL grade. The remaining four to nine samples returning HCL values of 0.001 to 0.005 oz Au/ton for which no fire assay was completed were factored to create an "estimated fire assay" value, based on HCL-to-fire-assay ratios obtained from nearby sample intervals. This factoring led to estimated values that either did not increase from the HCL values or were increased by 0.001 or 0.002 oz Au/ton (0.034 or 0.068 g Au/t). These factored values were then entered into the database. There is no record of whether laboratory personnel or exploration staff assigned these factored gold values. The factoring for low-grade HCL assays was referenced in 1992 through 1997 internal annual reports and evident in 1996 and 1997 assay worksheets from the Black Pine mine laboratory in the possession of Liberty Gold.

No other information is available on the methods and procedures used for sample preparation and gold assaying at the mine laboratory. The mine laboratory was not independent of Pegasus, and it is not known if the mine laboratory held any certifications.

Western Pacific 2011 - 2012: Drill samples were initially stored on site, then transported to the ALS Minerals sample preparation facility in Elko, Nevada by an ALS representative. No QA/QC samples were inserted.

Surface rock-chip and drilling samples were sent to the ALS Minerals (“ALS”) laboratory in Elko, Nevada for sample preparation. The pulps were analyzed at ALS’ facilities in Reno, Nevada. Gold was analyzed using a 30-gram fire-assay fusion with an AA finish (ALS method code Au-AA23). Separate 1-gram aliquots of some samples were analyzed for 51 major, minor, and trace elements at the ALS laboratory in North Vancouver, B.C. using a combination of inductively-coupled-plasma atomic emission (“ICP-AES”) and mass spectrometry (“MS”) following an aqua-regia digestion (ALS method code ME-MS41).

There is no evidence that QA/QC samples were inserted for analysis along with the Western Pacific rock samples.

11.1.2 Liberty Gold Surface Samples

Between 2017 and the Effective Date of this report, a total of 702 rock samples were collected by Liberty Gold personnel and transported to the ALS sample preparation facility in Elko, Nevada or Twin Falls Idaho. Sample weights were generally between 1 and 2 kilograms. The samples were crushed to 70% passing 2.0 mm, split to obtain a 250-gram subsample, and the subsample was pulverized to 85% passing 75 microns. The pulverized splits were shipped by ALS either to their assay laboratory in Reno, Nevada or North Vancouver, B.C., where in both cases gold was determined by 30-gram fire assay with an AA finish (method code Au-AA23). Separate 1.0-gram aliquots were analyzed for 51 major, minor, and trace elements by ICP-AES and MS following aqua-regia digestion (ALS method code ME-MS41).

ALS is independent of Liberty Gold. The ALS analytical facility in North Vancouver, British Columbia, is certified to ISO 9001:2008 standards and has received ISO/IEC 17025:2005 accreditation from the Standards Council of Canada. The ALS laboratory in Reno, Nevada, is certified to ISO 9001:2008 standards and has received ISO/IEC 17025:2005 accreditation.

11.1.3 Liberty Gold Drilling Samples

Prior to May 2021, RC drill samples were transported periodically by Liberty Gold personnel to the ALS laboratory in Elko, Nevada, or otherwise by ALS personnel or by a third-party contractor, Stott Trucking of Elko, Nevada. Subsequently, after ALS commissioned a new preparation laboratory in Twin Falls, Idaho, samples were transported to the new laboratory by Liberty Gold personnel, ALS personnel, or by either of George’s Transfer of Twin Falls, Idaho or Bill Evans Trucking of Twin Falls, Idaho.

After drying and weighing, the samples were crushed to 70% passing 2.0 mm. The crushed material was riffle split to obtain a 250-gram subsample that was ring-mill pulverized to 85% passing 75 microns. In 2019 through 2022, depending on sample load at the Elko and Twin Falls facilities, samples were shipped at various times to prep labs in: Tucson, Arizona; Thunder Bay, Ontario; Reno, Nevada; Vancouver, British Columbia; or Hermosillo, Chihuahua, or Guadalajara, Mexico.

Prior to 2023, after logging on site, core samples were transported by Liberty Gold personnel to Liberty Gold’s Elko office and core cutting facility for cutting by an independent contractor provided by Rangefront Mining Services of Elko, Nevada. After cutting the core into two halves lengthwise, one half was cut in half, and quarter core samples were placed in numbered sample bags and then picked up by ALS personnel for transport to the Elko preparation facility. Core samples were prepared for analysis by ALS with the same procedures as the RC samples. Core drilled in 2023 was logged and cut on site by Liberty Gold staff using the same procedures described above. Samples were shipped by an independent contractor, Bill Evans Trucking to the ALS Twin Falls prep Lab.

The sample pulps were shipped by ALS to their assay laboratory in Reno, Nevada, where 30-gram aliquots were analyzed for gold by fire assay fusion with an AA finish (ALS method code Au-AA23). If the resulting fire assay exceeded 0.1 g/t Au, separate aliquots were also analyzed for cyanide-soluble gold by AA after a 1-hour agitated leach in a 0.25% NaCN solution (ALS method code Au-AA13). In 2020 through 2023, ALS' fire assay laboratories in Lima, Peru and Vancouver, Canada were also utilized. Silver in addition to 50 major, minor, and trace elements were analyzed for all samples by a combination of ICP-AES and MS using a 1-gram aliquot following an aqua-regia digestion (ALS method code ME-MS41) at the ALS laboratory in North Vancouver, British Columbia.

Drill samples returning results greater than 5.0 g/t Au were re-assayed using a new 30-gram aliquot and fire assay fusion followed by a gravimetric finish (ALS method code Au-GRA23).

Liberty Gold employed a blind numbering system for RC and core samples, such that the hole number and down-hole footage are not known to the assay laboratory.

11.2 Sample Security

No information is available concerning security measures used by historical operators for surface and drilling samples.

Drill samples were stored at the Black Pine drill sites for a few days prior to transport. In 2022 and 2023, drill samples were also stored at the Company's exploration office until ready for shipment to the prep lab.

Liberty Gold's surface and RC samples were transported by Liberty Gold, ALS, Stott Trucking, George's Transfer or Bill Evans Trucking to the ALS sample preparation laboratories in either Elko, Nevada or Twin Falls, Idaho. Core samples were transported to the Elko office core cutting facility by Liberty Gold personnel, or the core remained on site. Cut core was transported from the Elko Office to the ALS Elko prep lab by ALS. In 2023, cut core was transported by Liberty Gold personnel or Bill Evans Trucking to the ALS Twin Falls Prep lab. Chain of custody forms from the lab are archived at the Liberty Gold office.

All pulps were returned to Liberty Gold and are stored in Liberty Gold's secure warehouse. A selection of coarse rejects are stored in the warehouse, or (since 2021) at the Liberty Gold exploration office adjacent to the Black Pine property.

11.3 Quality Assurance/Quality Control

11.3.1 Historical QA/QC Procedures

This section is derived from Gustin et al (2021). Historical records in the possession of Liberty Gold indicate that QA/QC procedures used by at least some of the historical operators involved check assays and, in certain cases, the submission of analytical standards, RC rig duplicates, and/or duplicates prepared from the coarse rejects of the original samples (preparation duplicates).

As a check on sampling procedures, Newmont collected coarse and fine materials that were not captured in the rotary or RC drill samples sent for assay (Hardie, 1964). These coarse and fine materials from 530 feet of drilling from seven of the 17 holes drilled in 1964 were sampled at the same five-foot intervals as the drill samples sent for assay. The fine materials consisted of "dust-sized particles caught in the cyclone dust collector", while the coarse materials were comprised of "particles caught between the dust collector and the sample collector." The fine and coarse samples were sent along with the standard drill samples to Union Assay Office, Inc. of Salt Lake City, Utah ("Union Assay") for gold assay. The results of Newmont's study are described in an untitled, anonymous memo and summarized in Table 11-1.

Newmont noted that both the fines and coarse materials that were collected to check the sampling methodology are representative of only small quantities of material that are not sampled relative to the original samples. Newmont further commented that the results of the study indicate that no serious downgrading or upgrading in gold grades are indicated, and therefore the original samples were sufficiently representative, in terms of gold grade, of the full volume of cuttings that were returned to the surface.

Table 11-1: Newmont Evaluation of Black Pine Project Drilling

Hole ID	From (ft)	To (ft)	Original (opt Au)	Fines (opt Au)	Coarse (opt Au)
BP-5	285	330	0.011	0.007	0.008
BP-6	80	100	0.007	0.007	0.006
BP-8	250	300	0.022	0.027	0.021
	30	95	0.016	0.015	0.017
BP-11	110	185	0.009	0.008	0.006
	210	240	0.013	0.010	0.011
BP-13	55	70	0.030	0.039	0.034
BP-14	50	120	0.027	0.029	0.028
BP-15	100	260	0.003	0.005	0.004

(Liberty Gold Corp. – Black Pine Project)

In 1974, Newmont sent 38 drill-sample pulps from five of the holes drilled in 1974 to Union Assay for gold and silver check fire assays. While the means of the original and check analyses differ by only 3%, the dataset is small. In addition, the original analyses of the 21 drill samples were done by Rocky Mountain and the remainder by Skyline, complicating an evaluation of the results. In 1985 and 1986, Permian had check assays done at Rocky Mountain on 48 pulps from 38 holes drilled in 1983 by Pegasus. No further information is available. The Newmont and Permian/Pegasus check assays represent approximately 3% and 2% of the drilling assays of these operators in 1974 and 1988, respectively.

Noranda analyzed duplicates each year using “selected secondary splits stored at the drill sites.” In 1986, an unknown number of samples from 1.52 m intervals were sent to ASI for “check assays” of gold and silver to allow comparisons with 6.1 m drill samples originally analyzed at Rocky Mountain. For the 1987 drilling, a total of 23 “check assays” of 1.52 m samples from one hole were completed. In 1988, a total of 113 pulps and coarse rejects were analyzed. No further information is available concerning possible QA/QC procedures implemented by Noranda.

Records are incomplete, but 1996 and 1997 assay worksheets from the Black Pine mine laboratory refer to inserted standards for samples analyzed by the HCL procedure. The rate of standard insertion and the expected gold values for the standards are not known.

11.3.2 Liberty Gold QA/QC

The QA/QC program instituted by Liberty Gold for drilling in 2017 through the Effective Date of this report included the insertion of coarse blanks, certified reference materials (“CRMs” or “standards”), and RC or Core field duplicates into the sample stream. A minimum of one CRM, one blank, and one field duplicate was inserted into the sample stream for every 36 drill samples, which is the number of samples in each ALS analytical batch. The results of these inserted control samples are summarized below.

11.3.3 QA/QC Results

11.3.3.1 Certified Reference Materials

CRMs were used to monitor and evaluate the analytical accuracy and precision of the Liberty Gold drill sample assays performed at ALS (Table 11-2). The insertion of CRMs can also be useful for detecting sample switches and numbering issues that can occur with primary drill samples.

A total of 16 CRMs has been used at Black Pine as of the Effective Date of this report. Twelve CRMs were prepared by Minerals Exploration and Environmental Geochemistry ("MEG") of Carson City and Lamoille, Nevada, using drill samples from Liberty Gold. Three of these were prepared from samples from the Kinsley sedimentary rock-hosted gold deposit in eastern Nevada ("PG" prefixes in Table 11-2). Five CRMs are derived from Black Pine drill samples (LG 19001, LG.21.01, LG.21.02, LG.22.01 and LG.22.02), and one is from drill samples from Fronteer Gold's Long Canyon sedimentary rock-hosted Gold Project in eastern Nevada (FGS2011A). An additional three commercial standards, Au-19.09, Au-19.10 and Au-21.01, were purchased from MEG. One CRM (CDN-GS-P6A) was purchased from CDN Resource Laboratories of Langley, B.C. Three CRMS (OxH66, SG40 and SJ53) were purchased from Rocklabs Ltd. of Auckland, New Zealand.

A total of 4,131 CRMs were inserted into the drill sample stream between 2017 and late 2023.

Table 11-2: Liberty Gold Certified Reference Materials

CRM	Source	Expected Value	SD	Num Samples	Num Outliers	Bias (%)	Comments
Au-19.09	MEG	0.71	0.032	111	3	-0.44	Commercial, volcanic
Au-19.10	MEG	0.81	0.036	69	3	-1.99	Commercial, siliceous
Au-21.01	MEG	0.41	0.023	90	17	8.20	Commercial, doped, siliceous matrix
CDN-GS-P6A	CDN	0.74	0.027	102	2	0.42	Commercial, volcanic
FGS2011A	MEG	7.13	0.373	47	0	-3.08	Long Cyn custom std
LG.21.01	MEG	0.41	0.014	756	27	1.14	BP custom standard
LG.21.02	MEG	1.10	0.042	555	21	-2.16	BP custom standard
LG.22.01	MEG	0.30	0.008	328	46	3.80	BP custom standard
LG.22.02	MEG	0.70	0.018	206	10	-0.34	BP custom standard
LG19001	MEG	0.70	0.03	556	12	-2.22	BP custom standard
OxH66	Rocklabs	1.29	0.032	35	1	1.65	Commercial, volcanic
PG13001X	MEG	1.87	0.075	382	5	-2.78	Kinsley custom std
PG13002X	MEG	2.19	0.087	346	7	-1.57	Kinsley custom std
PG14001X	MEG	0.33	0.017	484	2	-0.96	Kinsley custom std
SG40	Rocklabs	0.98	0.022	2	0	-1.28	Commercial, volcanic
SJ53	Rocklabs	2.64	0.048	62	1	0.25	Commercial, volcanic
Totals	-	-	-	4,131	157	-	-

(Liberty Gold Corp. – Black Pine Project)

In the case of normally distributed data, 95% of the CRM analyses would be expected to lie within two standard-deviations ("SD") of the certified value, while only 0.3% of the analyses are expected to lie outside of the three SD limits. Note, however, that most assay datasets from metal deposits are positively skewed.

CRM analyses outside of three SD limits defined by the CRM are typically considered to be failures. As it is statistically unlikely that two consecutive analyses of standards would lie between the two and three SD limits, such samples are also considered to be failures unless further investigations suggest otherwise. All potential failures should trigger investigation, possible laboratory notification of potential problems, and possible re-analysis of all samples included with the failed standard result.

ALS's performance with respect to assaying PG13001X is shown on Figure 11-1. The certified value of the CRM (gray line) along with one, two, and three SD limits (dark blue, light blue, and red lines, respectively) for the CRM are shown, as are the ALS analytical results of the CRM (red dots). The x-axis plots the ALS certificate numbers by increasing dates.

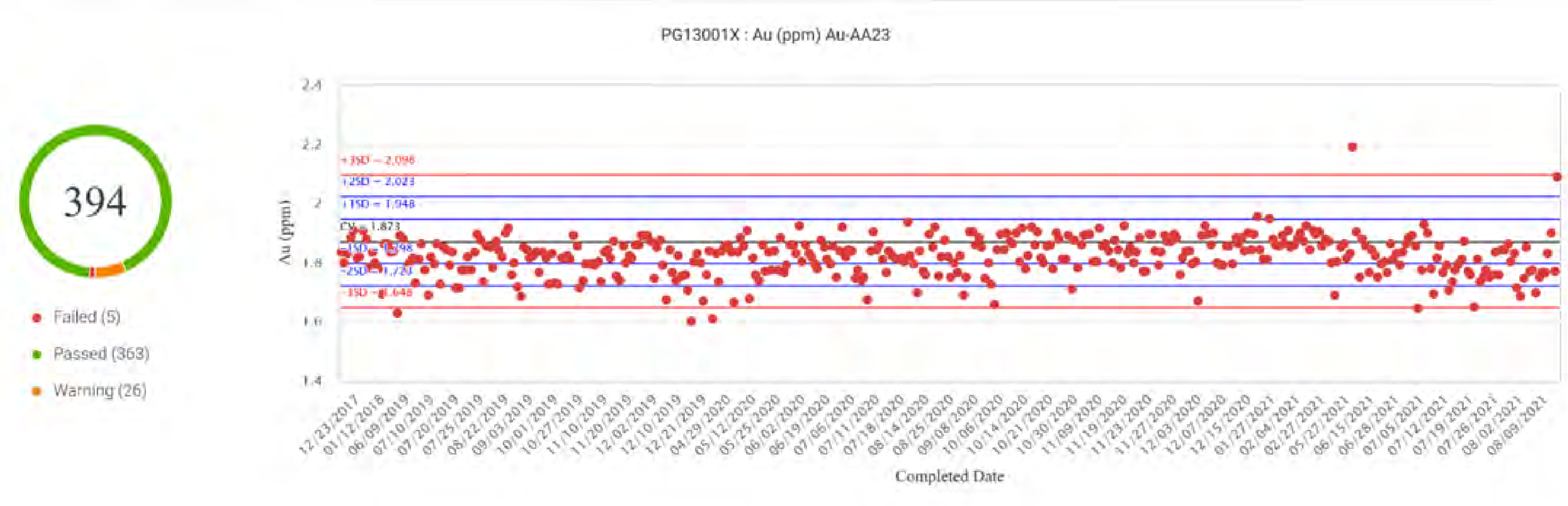


Figure 11-1: Graph of ALS Analyses of CRM PG13001Xs – 2017 through 2021 Drill Programs

The PG13001X chart documents a consistent low bias in the ALS gold assays relative to the certified value. Similar results are seen in the PG13002X, FGS2011A, and, to a less consistent extent, LG19001 CRMs. PG13001X, PG13002X, and FGS2011A are the three highest-grade CRMs utilized at the Project. Approximately 1% of the PG13001X standards failed low, with a significant number below two SDs. The same trend was noted with the LG19001 standard. The ALS performance improved in late 2020 in LG19001 and, to some extent, PG13001X, worsening again in mid-2021, suggesting that instrument drift at the lab may have been a factor.

The chart for LG.21.01 (Figure 11-2) shows six samples that are extremely out of range, both high and low, with most of the failures recorded in October 2021 and late May 2022. Investigation by Liberty Gold and ALS determined that the failures were the result of sample switches at the lab or loss of the sample prill rather than any deficiency in assaying or related to the CRMs themselves. The large number of sample switches that were revealed by insertion of the CRMs is a concern.

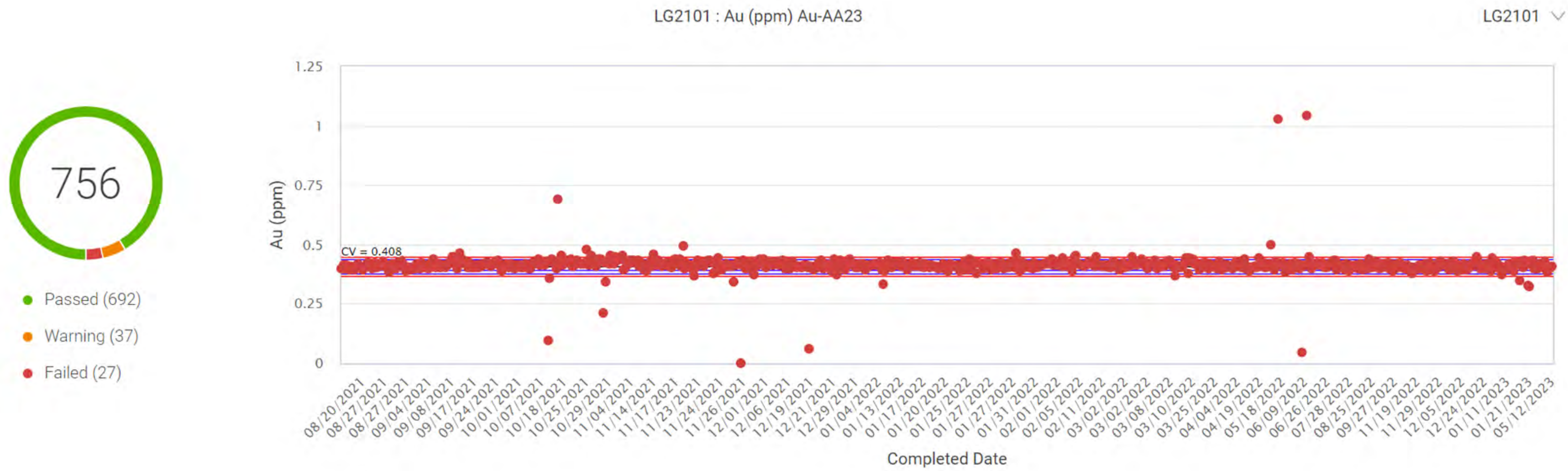


Figure 11-2: Graph of ALS Analyses of CRM LG.21-01 – 2021 Through 2022 Drill Programs

Near the end of November 2022, some CRMs labeled "LG21.02" began returning from ALS with gold values well outside of the expected mean value of 1.098 ppm gold. These samples returned values ranging from 0.38-0.42 ppm, within the error limits for CRM LG21.01. The multi-element geochemistry associated with these CRMs was a close match for LG21.01 CRMs. Liberty Gold hypothesized that some of the LG21.02 CRMS were mislabeled as LG21.02 during preparation at MEG (they were prepared during the same time period). Audits were made at all stages, aided by photo evidence and duplicate labels, to eliminate the possibility of mislabeling by Liberty Gold field staff. Subsequently, all standards that were labeled as LG21.02 but were determined to be LG21.01 (40 samples from 11/29/2022 – 12/31/2022), were changed to LG21.01.

A total of 157 failures were identified out of the 4,131 ALS analyses of the CRMs, or approximately 3.8%. However, a significant number of the failures are attributable to the low bias of the analysis. In other words, absent the low bias that characterizes the ALS analyses of some of the standards, many of these analyses, and perhaps more, would not be considered to have failed.

The PG13002X, PG14001X, LG19001, LG.21.01 and SJ53 standards each produced one or more very low value, and the CDN-GS-P6A and LG.21.01 standards one or more very high value, all of which were determined to be caused by sample switches at the lab. The number of issues determined to be such sample switches is a concern, and this issue needs to be investigated further.

All CRM failures were reported to the lab, triggering an investigation. If the failure was determined to be a sample switch or loss of the gold prill, the situation was remedied. If the cause of the failure was not determined to be a mechanical failure such as a sample switch, and if the failure occurred in an interval with a reportable interval of gold mineralization, the failure triggered a rerun of the standard and the ten drill samples analyzed immediately before and after the failed standard. Reruns typically fell within acceptable (SD) limits, but the associated drill samples typically returned results similar to the original analyses. To the extent that the 'failures' are actually caused by the low bias, the lack of change in the rerun analyses of drill samples is actually expected.

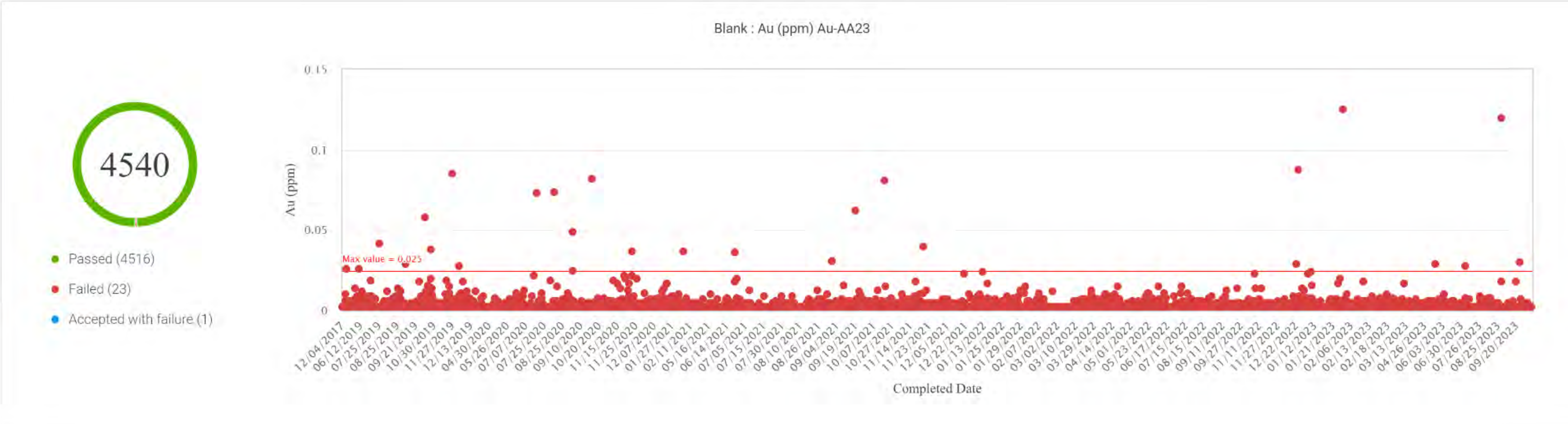
The QP has reviewed the performance and protocols of the CRMs and finds them to be to industry standards. The QP is of the opinion that the results of the CRMs allow for the associated assays to be acceptable for use in the Resource Estimation.

11.3.3.2 Coarse Blanks

Coarse blanks are samples of barren material that are used to monitor for possible contamination during sample preparation stages in the laboratory, and they are also useful for detecting sample switches and numbering issues. The detection limit of the ALS fire assay with AA finish is 0.005 g Au/t; blanks with assays in excess of 0.025 g Au/t (five times the lower detection limit) were therefore considered failures requiring investigation.

Liberty Gold's blanks consisted of Vigoro brand "pond pebbles", which are coarse enough to require primary and secondary crushing and thereby allows for the monitoring of the entire sample-preparation process applied to the drill samples. Blanks were inserted every approximately 36 samples, except in drill-core samples from intervals judged unlikely to be mineralized based on rock type or lack of macroscopically visible alteration. Where possible, the blanks were inserted within core intervals that were judged to have the potential to be mineralized.

A total of 4,540 blanks were inserted in the sample stream for the 2017 through 2023 programs. Of these, 27 ALS analyses of the blanks returned values in excess of 0.025 ppm Au, with 10 of these exceeding 0.050 g Au/t (Figure 11-3). Without special protocols (e.g., crushing unmineralized quartz between drill-sample crushing), ALS accepts any blank that contains <1% of the metal content of the preceding samples. Using this metric, all of the failed (above 0.025 ppm) samples are acceptable.



Source: Liberty Gold 2024
Note: dashed red line is upper acceptable limit

Figure 11-3: Coarse Blank Analyses – 2017-2023 Drilling Program

In the case of a failure, an internal review takes place, but for RC holes, no further action is taken because, in the case of RC chips, the entire drill sample is crushed, making it impossible to replicate all stages of sample preparation if a new pulp was prepared for re-analysis.

The QP has reviewed the performance and protocols of the blank samples and finds them to be to industry standards. The QP is of the opinion that the results of the blank samples allow for the associated assays to be acceptable for use in the Mineral Resource estimate.

11.3.3.3 RC Field Duplicates

RC field duplicates are second splits of the RC chips collected at the sample splitter at the same time as the original sample splits during active drilling. Field duplicates are mainly used to assess geologic variability and sub-sampling variance. The field duplicate samples were submitted to ALS at the same time as their associated drill samples.

The cyclone discharge of the RC drill rig used by Liberty Gold was set up with a “Y” splitter. The primary samples were consistently collected from the same outlet of the “Y” splitter throughout the drilling campaign, while the field duplicates were collected separately from the other outlet of the Y splitter, simultaneously with the primary sample. The field duplicates were collected approximately every 36 samples throughout the entirety of each drill hole, which resulted in a large number of duplicates of unmineralized intervals. A total of 4,249 of the RC field duplicates collected were paired with an original drill sample that assayed in excess of 0.01 ppm Au/t in the course of the 2017 through 2023 drill programs. The data set excludes one extreme outlier that is likely due to a sample switch at the lab.

Figure 11-4 shows two graphs that plot the gold assay values of the original RC drill sample versus those of the duplicate samples, using arithmetic and log scales; the log scale shows more detail in the lower-grade range. Trend lines are shown for both sets, and 30% variance lines are shown on the arithmetic chart.

Of the 4,249 samples, 216 pairs differ by more than 30%. While no bias is evident in the data, several data pairs in the RC subset (in addition to the outlier that was excluded) show substantial differences, most of which are suspected to be sample switches based on multielement signatures. As with the sample switches determined by the CRM and blank analysis, this is a concern and needs to be investigated further with ALS.

The data exhibit a strong correlation between original and duplicate samples, with a Correlation Coefficient (R^2) value of 0.951, and an average difference of 0.37% between the means.

The QP has reviewed the performance and protocols of the RC field duplicates and finds them to be to industry standards. The QP is of the opinion that the results of the RC field duplicates allow for the associated assays to be acceptable for use in the Mineral Resource estimate.

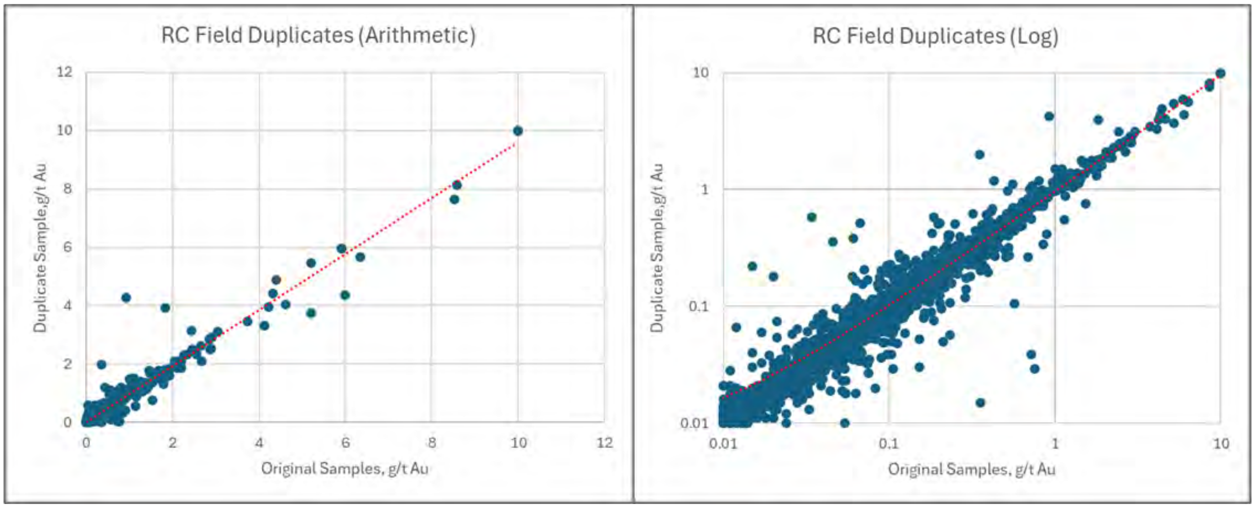


Figure 11-4: RC Field Duplicate Data – Liberty Gold 2017-2022 Drilling Programs

11.3.4 Core Field Duplicates

A total of 88 core duplicates were collected during the 2019 through 2022 programs, by quartering the half core and submitting one quarter as the primary sample and one quarter as the duplicate sample. Data are summarized Figure 11-5.

A minimal number of sample pairs showed a variance greater than 30%, primarily occurring in the lowest-grade samples. No extreme outliers were identified. The data demonstrate a high correlation, with an R^2 coefficient of 0.998 and an average difference of -0.391% between means.

The QP has reviewed the performance and protocols of the core field duplicates and finds them to be to industry standards. The QP is of the opinion that the results of the core field duplicates allow for the associated assays to be acceptable for use in the Resource Estimation.

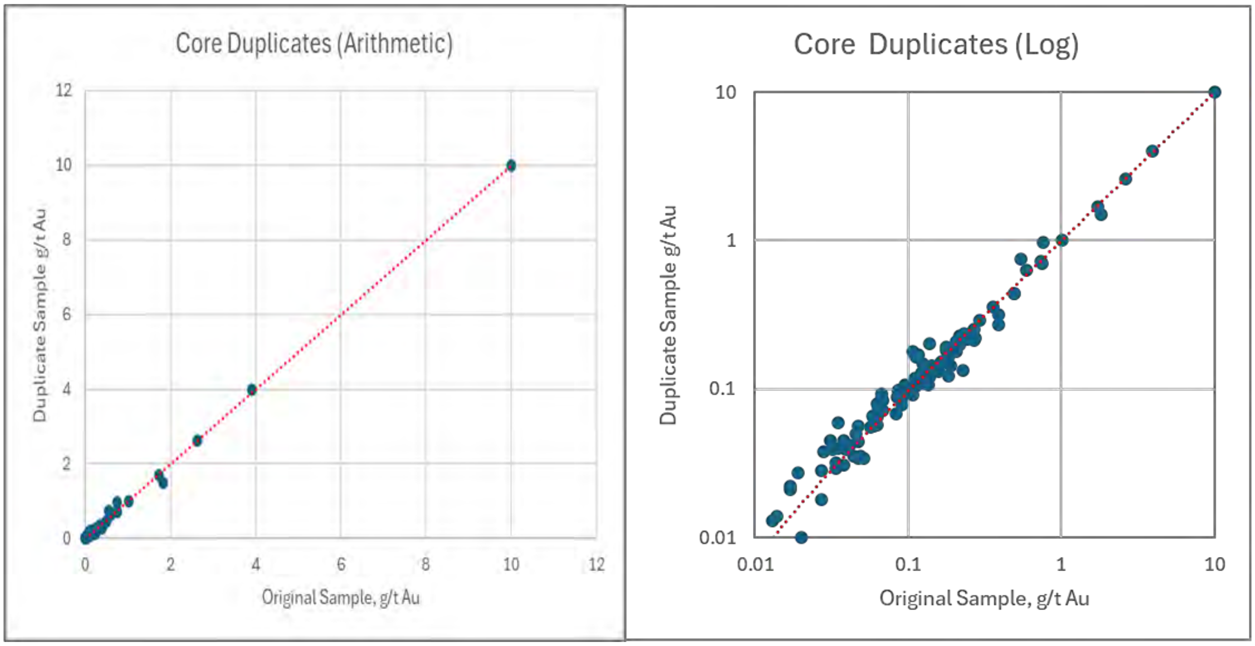


Figure 11-5: Core Field Duplicate Sample Comparison, Liberty Gold 2019-2022 Drilling

11.3.4.1 Check Assays

As a further check on analytical accuracy, Liberty Gold selected a portion of the original drill-sample pulps prepared and analyzed by ALS from the 2017 and 2019- 2023 drill programs, and these pulps were sent to Inspectorate/Bureau Veritas Laboratories ("BV") for re-assaying of gold content by fire assay with an AA finish. BV is an ISO 17025 accredited, independent laboratory located in Sparks, Nevada. The procedure for selection of check assays consisted of querying all drill samples that returned greater than 0.1 g/t Au and assigning these a random number The selection was then sorted on the random number and approximately 7% of these were selected for re-assay, for a total of 3,052 RC sample pulps and 190 core sample pulps. It should be noted that in 2022, Liberty Gold staff mistakenly submitted 764 (RC) and 78 (Core) assay samples with values <0.1 g/t Au. The low grade, near detection limit in some instances, result in skewed data and for this reason these 842 samples are not used in the following summary. CRMs were also submitted to BV along with the ALS pulps. BV analyzed the samples by a method similar to that used by ALS.

The BV pulp-check analyses are compared to the original ALS assays in Figure 11-6, with 31 outlier RC pairs removed representing sample pairs with >40% variance, for a total of 2,370 check assay pairs.

The mean of all of the BV analyses is slightly lower than the original ALS assays (0.433 ppm versus 0.440 ppm, respectively). However, Figure 11-6 shows the BV analyses have a consistent positive bias (BV assays greater than ALS) at lower gold concentrations. The percent relative difference of pairs between 0.1 and 0.4 g/t Au is 1.37%; between 0.4 and 1.0 g/t is 2.24%; and between 1.0 and 14.0 g/t Au is 0.24%. This low bias, particularly in the lower grade ranges, is consistent with the low bias seen in assay results from control samples.

The QP has reviewed the performance and protocols of the check assays and finds them to be to industry standards. The QP is of the opinion that the results of the check assays allow for the associated assays to be acceptable for use in the Mineral Resource estimate.

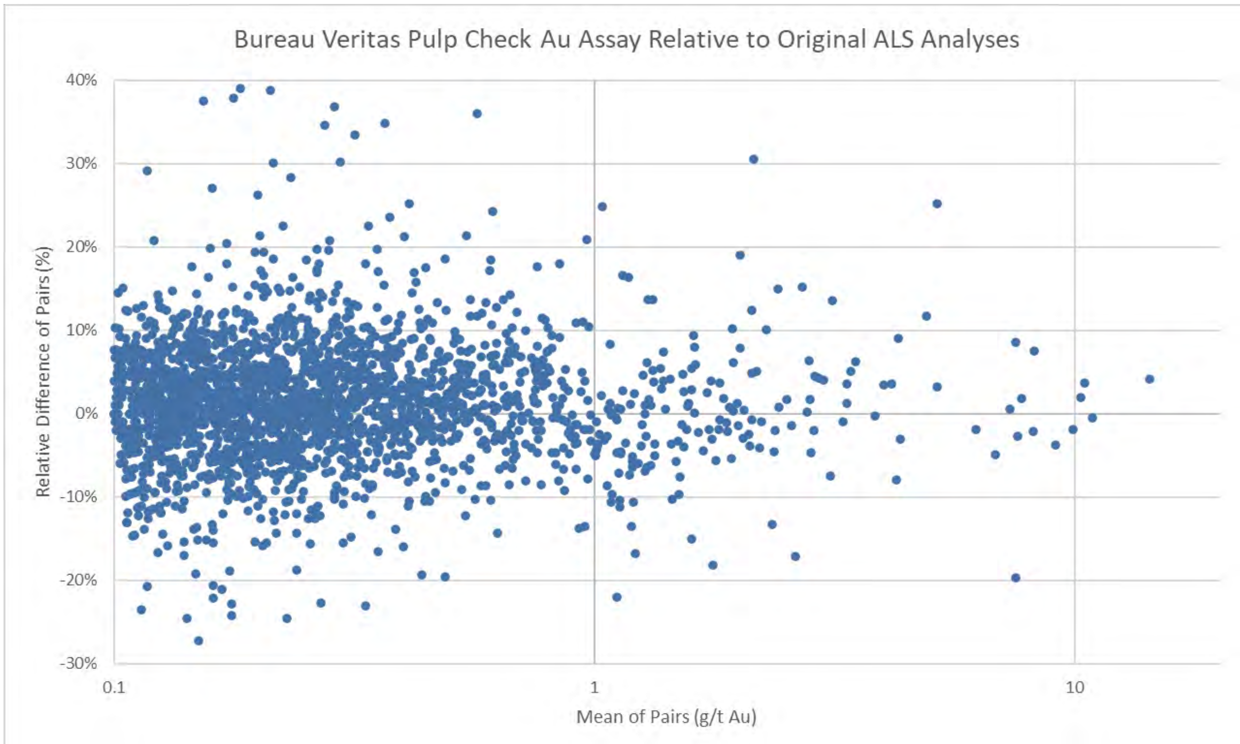


Figure 11-6: Check Assay Data Analysis

11.4 Cyanide Soluble Gold Assay Investigation

In August 2021, it was noticed that results for cyanide soluble (AuCN) assaying of some pulps were lower than expected based on previous assaying of similar material. This led to an internal investigation by Liberty Gold examining the possible causes of the anomalous results, including elevated preg-robbing carbon not detected in visual logging, elevated and possibly preg-borrowing clay, etc. After several months, no clear indication of the nature of the low AuCN results was identified, and Liberty Gold began to suspect a laboratory error. At this time, data was returned from a core twin of an RC hole in the Rangefront Zone. AuCN results were materially higher for the core hole (prepped in Elko) than for the RC hole (prepped in Twin Falls). The ALS QA/QC manager was engaged and began an investigation. They noted that some of the equipment and protocols were slightly different between the two prep labs, and between the Reno and Vancouver assay labs, but the slight differences were not sufficient to account for the large differences in AuCN results between the core and RC twin holes.

By mid-2022, the matter was elevated to the North American QA/QC manager, who quickly identified that the operators in the Twin Falls laboratory had been adding stearic acid well in excess of recommended amounts during the pulping step, which in turn had a strongly negative impact on the AuCN results.

As of the Effective Date of this report, coarse rejects from approximately 296 holes with 18,100 samples have been rerun for AuCN.

A follow-up investigation testing whether any amount of stearic acid might suppress AuCN results was carried out, with the conclusion that even small amounts of stearic acid, consistent with ALS protocols, has a deleterious effect on recovery of gold by cyanidation. With the coarse rejects from earlier holes that were prepped in this manner having been discarded, the AuCN results were removed from the database.

11.5 Summary Statement

The independent laboratories used to analyze the primary drill samples of the historical operators prior to the open-pit mining operation at the Black Pine Project include ASI, Chemex, GSI, Legend, Rocky Mountain, Skyline, and Union Assay. All of these laboratories were independent of the historical operators, widely known, and commonly used by the exploration and mining industry at the time. During the mining operation, the Pegasus drill samples were analyzed at the on-site mine laboratory.

While documentation is incomplete for the methods and procedures used for historical sample preparation, analyses, and sample security, as well as for the QA/QC procedures and results, it is important to note that the historical sample data were used to develop a successful commercial mining operation that produced more than 400,000 ounces of gold.

Liberty Gold's sample preparation and analyses were performed at a certified laboratory, and their sample security and QA/QC procedures were consistent with industry norms.

Despite ongoing issues with sample swaps and other issues at the lab identified through monitoring of CRMs and blanks, the QP is of the opinion that the Black Pine drill hole assay data is reliable and can be used to support the current resources, interpretations, and conclusions summarized in this report.

12 DATA VERIFICATION

Data verification is the process of confirming that data has been generated with proper procedures, transcribed accurately from its original source into the Project database, and is suitable for use as described in this technical report.

12.1 Verification of Historical Drill Data

Liberty Gold's construction and initial verification of the historical database is described in Section 9.1. The process Liberty Gold used to compile the Project data was actively reviewed during this process by MDA, who also provided guidance (Gustin et al, 2021).

SLR performed cross-validation procedures between the Black Pine assay database and the ALS assay certificates. A total of 1,153 certificates spanning the years 2017 to 2023 were compiled and compared against Au and Ag values within the "Drilling_Samples.csv" assay database. The entire dataset underwent cross-referencing with both client-provided certificates and certificates retrieved directly from the Laboratory system for gold, while 85% of the samples were compared for silver.

Of the 169,854 total samples recorded in the database, 1,005 samples do not contain assay results for Au (gold) or Ag (silver). In addition, 206 samples did not meet Au cross-check requirements due to various issues:

- Two of these samples were identified as sample swaps.
- 17 samples have values recorded in the database that do not match any certified values in the corresponding assay certificates.
- 187 samples lack supporting assay certificates.

For the remaining samples, most align with the assay database values. Minor discrepancies were observed, primarily in Au values, with recorded discrepancies ranging between 0.005 g/t and 0.1 g/t Au when compared to the certified assay values, but most of the Ag values align well. The minor discrepancies are likely due to the selection of a preferred value from multiple re-assays.

SLR concludes that the database is strongly validated by the provided assay certificates and the database verification procedures for Black Pine comply with industry standards and are adequate for the purposes of Mineral Resource estimation.

12.2 Verification of Liberty Gold Data

12.2.1 Software Validation and Audit of Drill Hole Database

SLR conducted a number of digital and visual queries on the resource database. SLR inspected the drill hole traces, reviewed the drill hole traces in 3D, level plan, and vertical sections and found no unreasonable geometries. SLR also confirmed that there are no duplicate sample numbers and that sample numbers are available for every assayed interval.

SLR performed cross-validation procedures between the Black Pine assay database and the ALS assay certificates. A total of 1,153 certificates spanning the years 2017 to 2023 were compiled and compared against Au and Ag values within the "Drilling_Samples.csv" assay database. The entire dataset underwent cross-referencing with both client-provided certificates and certificates retrieved directly from the Laboratory system for gold, while 85% of the samples were compared for silver.

Of the 169,854 total samples recorded in the database, 1,005 samples do not contain assay results for Au (gold) or Ag (silver). In addition, 206 samples did not meet Au cross-check requirements due to various issues:

- Two of these samples were identified as sample swaps.
- 17 samples have values recorded in the database that do not match any certified values in the corresponding assay certificates.
- 187 samples lack supporting assay certificates.

For the remaining samples, most align with the assay database values. Minor discrepancies were observed, primarily in Au values, with recorded discrepancies ranging between 0.005 g/t and 0.1 g/t Au when compared to the certified assay values, but most of the Ag values align well. The minor discrepancies are likely due to the selection of a preferred value from multiple re-assays.

SLR concludes that the database is strongly validated by the provided assay certificates and the database verification procedures for Black Pine comply with industry standards and are adequate for the purposes of Mineral Resource estimation.

12.3 Site Inspection

The QP visited the Black Pine Project site on April 11, 2024. The site visit included inspections of the historical open pits, traverses outside of the pits, and detailed discussions with Liberty Gold technical staff. Mineralization from open-pit exposures was examined, as were numerous unaltered and altered (and possibly mineralized) outcrops beyond the limits of the open pits. Various active core and RC drill sites were visited during the visit. RC drill chips and drill core from representative areas of the deposit were reviewed with the Liberty Gold team.

The QP experienced no limitations with respect to data verification activities related to the Black Pine Project. In consideration of the information summarized in this and other sections of this report, the QP has verified that the Project data is acceptable as used in this report, most significantly to support the estimation and classification of the Mineral Resource estimation.

12.3.1 Drill Hole Collars

The QP visited a number of the Liberty Gold drill pads. The locations of the drill pads were confirmed using a detailed topographic map showing drill roads. While many of the drill collars have been buried or destroyed by subsequent traffic, tags with hole numbers were found for at least one of the holes sited on the pads. GPS coordinates were recorded for holes which tags were inspected and confirmed against the collar coordinates in the database.

In the QP's opinion, the Black Pine database is adequate for Mineral Resource estimation.

12.4 Summary Statement

The modelling of the Black Pine resources is based on a database that includes 1,848 historical RC holes, 26 historical core holes, and 970 RC and 31 core holes drilled by Liberty Gold.

In the QP's opinion, the Black Pine database is adequate for Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section has been prepared under the supervision of Mr. Gary Simmons based on historical and Liberty Gold test results as cited. The term “ore” is used in this section to refer to mineralized material used for test process feed and has no economic significance. Historical test parameters and results originally reported in imperial units have not been converted to metric units and in some cases the author prefers to use a mixture of imperial and metric units throughout the text.

13.1 Metallurgical Work Completed Prior to Mining Operations

A significant number of historical reports are available that document metallurgical testing completed prior to the Pegasus mining operations that began in 1991. The reports reviewed by the authors as of the Effective Date of this report are summarized in chronological order below.

- Potter (1974): The U.S. Bureau of Mines Salt Lake City Metallurgy Center carried out column-percolation cyanidation tests on two samples (BP7 and BP9) with calculated head assays of 2.71 g Au/t and 6.75 g Au/t, respectively. A total of 5 kg of minus 2-inch material from sample BP7 and 8 kg of minus 2-inch material from BP9 were leached in glass columns. BP7 was leached for 191 hours, recovering 87.4% of the gold to activated carbon. BP9 was leached for 701 hours, with 80.2% extracted to activated carbon.
- Ennis (undated – estimated 1975): Gold Resources commissioned Newport Minerals, Inc. of Cripple Creek, Colorado to carry out crush-leach testing on a 136 kg composite sample with a head grade of approximately 15 g Au/t. Five tests were done at various particle sizes, including “as received”, 1-inch, ¾-inch, ½-inch, and 3/8-inch. Samples were leached “in a barrel” for seven days. The “as received” sample showed “approximately 70%” extraction, with 73% for the 3/8-inch sample.
- Dawson (1980): Pioneer commissioned Dawson Metallurgical Laboratories, Inc. of Murray, Utah to carry out a 48-hour leach of a “composite of samples” ground to 90% passing 200 mesh. The conclusion was that “an appreciable portion of the gold does not leach”, possibly “due to carbonaceous matter” in the tested sample.
- Dix (1984): Kappes, Cassiday & Associates (“KCA”) of Reno, Nevada carried out cyanide leach tests on three samples from the Tallman mine. Sample BP1 had a grade of 7 g Au/t; BP2 assayed 1.37 g Au/t, and BP3 had a gold content of 0.21 g Au/t. Two 58-day leach tests were carried out on minus 4-inch and minus ½-inch material from BP1, with gold extractions of 75% and 81%, respectively. Agitated cyanide tests were run for 24 hours on portions of pulverized head samples. The average extraction for BP1 and BP2 was 93%. BP3 was found to contain strongly “preg-robbing” carbonaceous material.
- Defilippi (1988): The KCA report (KCA-1988a) is of particular interest as KCA tested a series of samples from four large diameter core holes and three bulk samples taken from historical pit locations and some core extending below the historical Black Pine pits mined by Pegasus. The materials sampled represent material types that will be mined in future operations. A typical log-normal plot of the various test feed sizes (P_{80}) vs. gold and silver extraction is shown in Figure 13-1 and Figure 13-2, for Noranda Core Hole BP87-93.

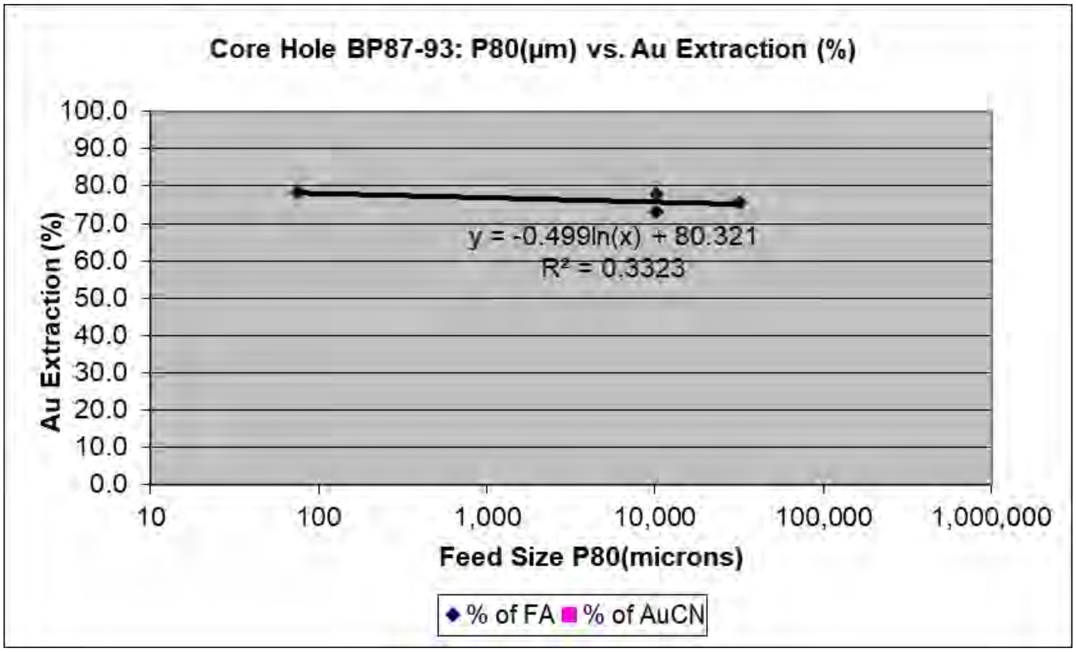


Figure 13-1: Plot of 1988 Column P_{80} (microns) vs. Gold Extraction (%)

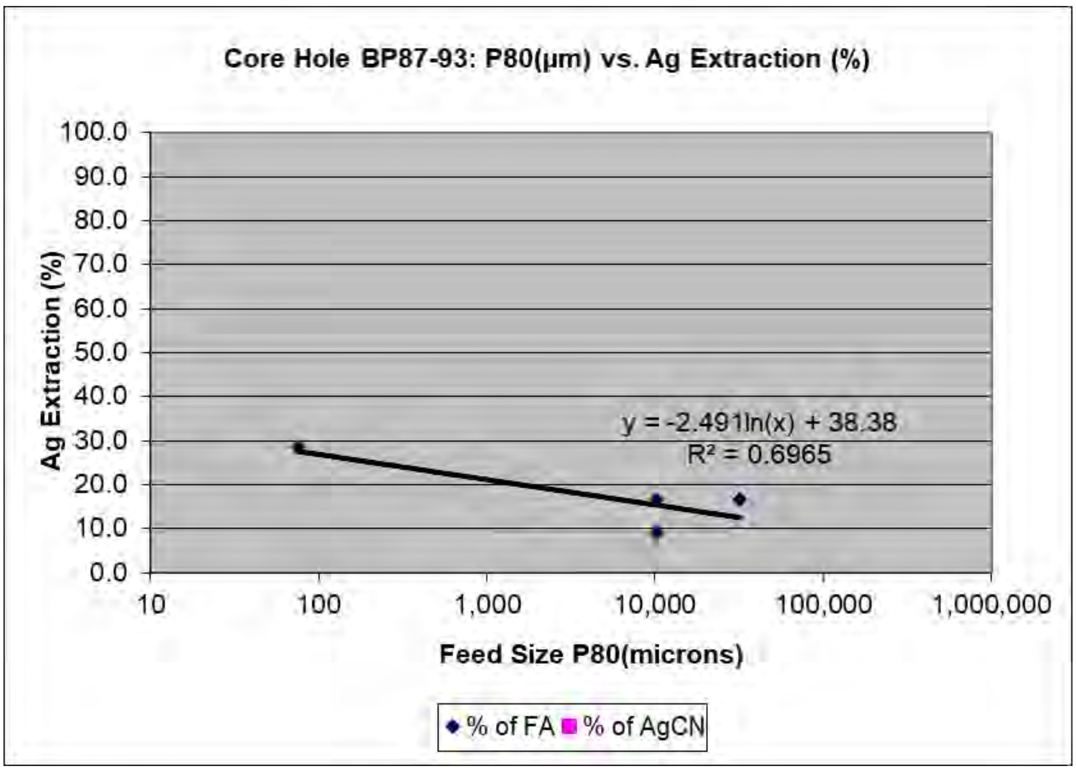


Figure 13-2: Plot of 1988 Column P_{80} (microns) vs. Gold Extraction (%)

The test results derived from the KCA-1988 report were included with the more recent Liberty Gold testing by KCA in 2020 to develop gold and silver recovery models for Black Pine.

KCA also carried out tests on a composite sample of Black Pine carbonaceous mineralization, made up of 34.14 m of drill core and a total weight of 372.2 kg. The sample was subjected to double oxidation, chlorination with hypochlorite, thiourea leaching, carbon-in-leach (“CIL”), and roast/cyanide leach tests. Most techniques did not significantly increase extractions over those obtained from direct cyanidation. However, “straight oxidation with hypochlorite gave gold recoveries of 88% with the addition of 320 pounds (145 kg) of calcium hypochlorite per ton of ore” and “roasting the ore at 540 degrees C for two hours followed by straight cyanidation gave gold recoveries of 80%.”

- Yernberg (1988): According to a copy of a report by Senior Metallurgist W.R. Yernberg of KCA that is missing the first 18 pages and some details and results, eight bottle-roll tests were carried out on 500 g of pulverized material that was agitated for 24 or 48 hours in different sets of tests. With one exception, gold extractions ranged from 78.3 to 89.7%. A single sample had an extraction of 50% and was found to be moderately preg-robbing.

  Continuously drained drip-leach column tests were carried out with backhoe samples and drill core. Backhoe samples included splits of three samples processed at minus 3-inch and minus 1-inch particle sizes, and these were leached for 60 days. Five core samples were crushed to 1.5 inch (37.5 mm) and 0.5 inch (12.5 mm) and were leached for 40 or 60 days in separate tests. Two of the 0.5-inch (12.5 mm) columns required agglomeration. Tailings screen analyses were employed to look at the effectiveness of leaching in different size fractions within the samples. Leaching was significantly more effective for the smaller size fractions than the larger ones.

- Clemson (1988): This study provided an in-depth look at the distribution of gold in oxidized and unoxidized mineralized materials in the Black Pine deposits. Extremely fine-grained native gold was noted in oxidized samples, averaging two microns in diameter, associated with hematite, quartz, and calcite. Some silica encapsulation was noted.

  The report describes bottle-roll testing undertaken at Lakefield Research of Peterborough, Ontario, Canada. Samples of drill chips were ground to minus 20 mesh and screened at minus 35, 100, 200, and 500 mesh, and the various screen fractions were assayed for gold. No enrichment of gold in any of the size fractions was noted. Ten samples were used for the study, with results for the minus 200 mesh fraction reported for all samples. Gold extractions for seven of the ten samples ranged from 81.9% to 92.4%. Three of the samples yielded very low recoveries; these samples contained preg-robbing carbonaceous material. A number of techniques were applied to these samples in an attempt to improve extraction: grinding to 86% passing minus 400 mesh, roasting at 600 degrees C, and then leaching was found to be the most effective method.

- Dix (1990): KCA performed 4-hour agitated cyanide-leach tests on ten 1 kg “as received” chip samples (nominally ¼-inch [6.25 mm] particle size), and the data were compared to conventional fire assays. Gold extractions ranged from 78.1% to 97.5% and averaged 87.5%.

13.2 Metallurgical Work Completed by Pegasus

Liberty Gold has no historical records documenting metallurgical testing that Pegasus may have carried out. However, Western Pacific Resources acquired the Black Pine Project in October 2012 and produced a summary report documenting Pegasus gold production records on December 13, 2012, titled: “*Report on Heap Leach Production and Recovery - Black Pine Mine, Idaho”*. Production records from the Pegasus operation indicate that from 1991 through 1998, the average gold recovery by ROM heap leaching was 64.1% (Table 13-1). These numbers do not include additional ounces recovered from wash/rinse closure and reclamation activities, as these were carried out after Pegasus ceased to be operator.

Table 13-1: Pegasus Heap Leach Production Summary

Year	Material to Leach Pad			Reported Prod		HL Feed	HL Pad Remaining	Calc. HL Pad Grade	Annual Rec.	Cumulative Recovery
	Tonnage (000 t)	Head Grade (g/t)	Contained Metal (kg Au)	(oz Au)	Rec (%)	(oz Au)	(oz Au)	(opt Au)	(%)	(%)
1991	--	--	--	--	--	--	--	--	--	--
1992	2,850	1.200	3,420.0	48,700	65.0	109,947	61,247	0.0195	44.3	44.3
1993	3,270	0.820	2,681.4	66,100	80.0	86,202	81,349	0.0121	76.7	58.5
1994	5,810	0.690	4,008.9	65,700	54.0	128,879	144,527	0.0110	51.0	55.5
1995	7,050	0.720	5,076.0	108,500	59.0	163,184	199,211	0.0095	66.5	59.2
1996	8,730	0.520	4,539.6	87,900	60.0	145,940	257,251	0.0084	60.2	59.4
1997	2,572	0.534	1,373.7	44,080	--	44,172	257,343	0.0077	99.8	62.1
1998	--	--	--	13,800	--	--	243,543	0.0073	--	64.1
Totals	30,282		21,099.6	434,780		678,324	243,543	0.0073	64.1	64.1

Source: Compiled by Western Pacific Resources, Dec 13, 2012, and published in Gustin et al, 2021.

Western Pacific Resources speculated in their report that the total leach cycle may also have been compromised and an additional 50,899 recoverable ounces remained in the pad. If correct, this would have resulted in an increase of the overall expected gold recovery to 71.6%.

13.3 Liberty Gold 2019-2020 Bulk Sample Bottle-Roll and Column Leach Testing

In 2019 Liberty Gold initiated bottle-roll and column-leach testing at KCA on six backhoe extracted bulk samples taken from existing pit walls and benches from five of the historical Black Pine open pits and one road cut through the F Zone resource area (KCA 2020a).

Bulk Sample pit locations where metallurgical samples were excavated are as follows:

1. A Pit
2. Tallman Pit
3. Upper BX Pit
4. Lower F Zone
5. C/D Pit
6. I Pit

A map showing more precise locations of the Liberty Gold 2019 bulk samples (including the location of the Noranda bulk samples (Defilippi, 1988) and large diameter core) is shown in Figure 13-3.

The Liberty Gold bulk samples were collected in new 55-gallon steel drums and the average weight was 1,000 kg each. All composites were subjected to bottle-roll leach testing at target P_{80} sizes of 75 µm and 1,700 µm, and to column-leach testing at 75.0 mm crush size. The main objective of the tests was to evaluate the laboratory-scale leachability character of the Black Pine resources in terms of gold extraction, extraction rate, reagent consumption, and sensitivity to feed size.

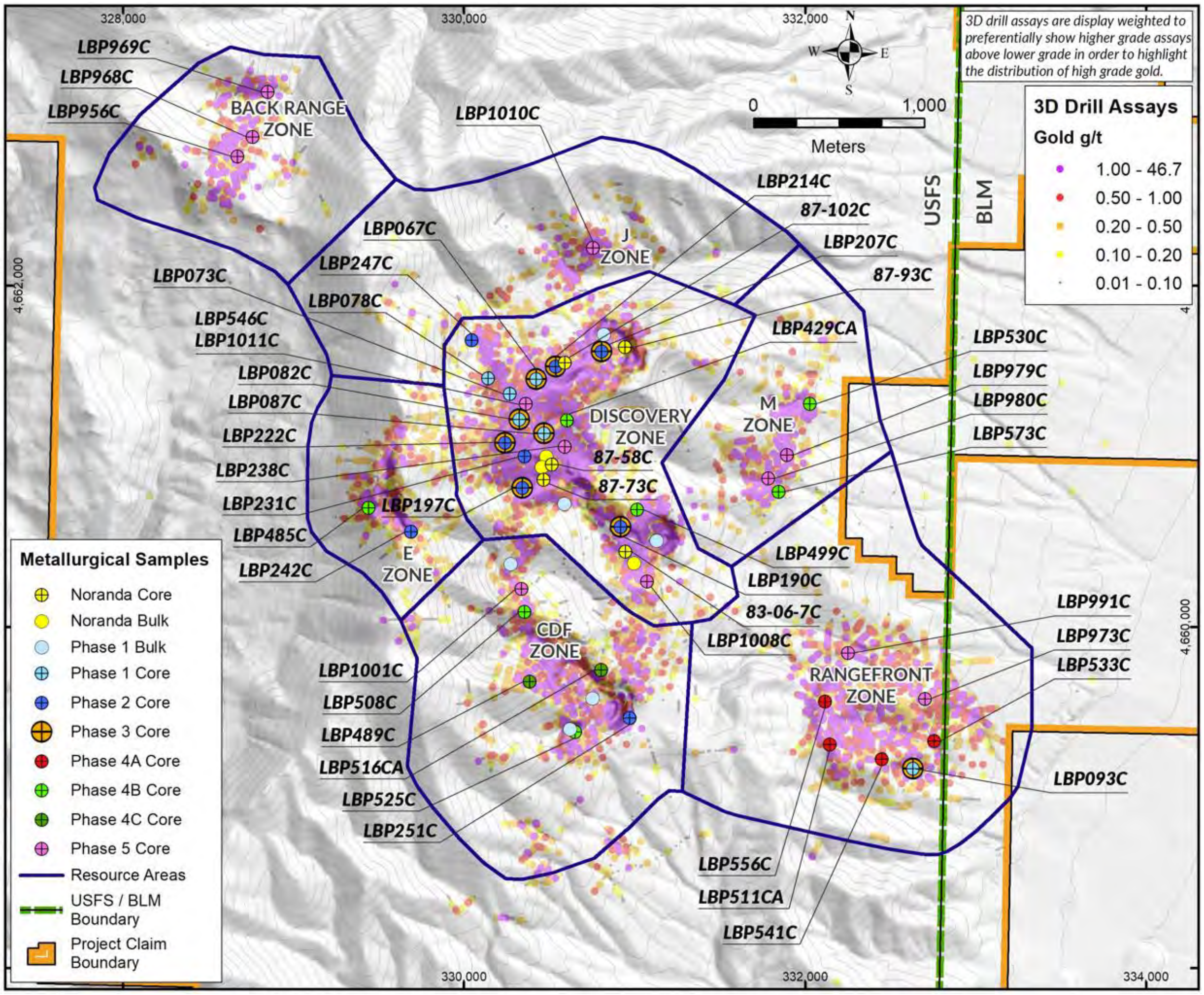


Figure 13-3: Liberty Gold and Noranda Metallurgical Samples

13.3.1 2019–2020 Black Pine Bulk Sample Head Assays

Head assays and geo-metallurgical characterization were obtained for the six bulk samples using a combination of four separate laboratories: KCA, ALS, University of British Columbia ("UBC") and FL Schmidt ("FLS"). The head assays and geo-metallurgical characterization data are tabulated in Table 13-2. A summary of findings is provided below:

- Gold grade ranged from 0.23 ppm to 4.04 ppm and averaged 1.82 ppm.
- Silver grade ranged from 0.63 ppm to 9.92 ppm and averaged 3.62 ppm.
- Organic carbon ranged from 0.06% to 0.12% and averaged 0.08%.
- Sulfide sulfur ranged from <0.01% to 0.03% and averaged 0.02%.
- Preg robbing analysis ranged from 2.6% to 9.9% and averaged 4.5%. There is indication of minor clay and carbon preg-borrowing in some samples, but it does not appear to materially affect gold or silver extraction.
- Copper values by ICP were very low, ranging from 9 ppm to 465 ppm and averaged 95 ppm.
- Cyanide solubility of gold ranged from 51.9% to 86.1% and averaged 75.8%.
- Concentrations of the deleterious elements by ICP were as follows: 9 ppm selenium; mercury ranged from 4.0 ppm to 14.9 ppm; and arsenic was low at 31 ppm to 1,105 ppm and averaged 271 ppm.

- Concentrations of the primary cyanide consumers were low and suggest minimum potential for affecting cyanide-consumption rates. Copper averaged 95 ppm, nickel averaged 98 ppm, and zinc averaged 250 ppm.
- Silica (SiO_2) content ranged from 9.8% to 45.0% (by whole-rock analysis) and averaged 28.0%.

13.3.2 2019-2020 Bulk Sample Bottle-Roll Test Results

Bottle-roll leach cyanidation testing was conducted on six bulk composites to evaluate the leachability characteristic of the Black Pine historical pit resources at fine to coarse particle size. Bottle-roll testing was conducted at two targeted feed sizes: 80% passing (P_{80}) 75 µm (200 mesh) and 80% passing 1,700 µm (10 mesh). Retention times were 72 hours for the 75 µm bottle-roll tests and 144-hrs for the 1,700 µm tests. A second P_{80} = 75 µm CIL bottle-roll test was conducted to evaluate preg-robbing potential. The 75 µm bottle-roll and CIL testing followed a standard procedure outlined detail in the final KCA report (KCA 2020a). The 75 µm direct leach and CIL bottle-roll test procedure was the same as for the 1,700 µm bottle-rolls, except the retention time was reduced from 144 to 72 hours. The 75 µm and 1,700 µm bottle-roll test results (along with the column leach test results – discussed later in this section) are shown in Table 13-2.

Three of the six bulk samples; A Pit, Upper BX Pit, and Lower F Zone, contained significant minus 75 µm fines (200 mesh), 12.5%, 17.7%, and 10.9%, respectively, and were agglomerated with 2.0 kg/t of cement. The three remaining bulk composites, with lower fines content, were leached without agglomeration.

Table 13-2: 2019-2020 Bulk Sample Head Assays

		KCA Analysis											ALS Analysis											
KCA Sample No.	Mine Area	Au & Ag Assays					Sulfur and Carbon Species									Preg-robb Analysis			Sulfur and Carbon Species					
		AuFA ppm	AuCN ppm	AuCN %	AgFA ppm	Cu ppm	C(tot) %	C(org) %	C(inorg) %	$S_{(total)}$ %	$S_{(sulfide)}$ %	$S_{(sulfate)}$ %	AuFA ppm	AuCN ppm	AuCN %	Au-AA31 w/spike	Au-AA31a w/o spike	Au PR %	C(tot) %	C(org) %	C(inorg) %	$S_{(total)}$ %	$S_{(sulfide)}$ %	$S_{(sulfate)}$ %
2019 Black Pine Bulk Samples																								
87201B	A Pit	1.182	0.82	69.4	5.23	30	5.91	0.26	5.65	0.07	<0.01	0.07	1.200	0.82	68.3	4.09	0.79	3.8	5.97	0.09	5.88	0.06	0.01	0.05
87202B	Tallman Pit	2.160	1.80	83.3	3.77	36	6.52	0.19	6.33	0.03	<0.01	0.03	2.270	1.80	79.3	5.17	1.90	4.7	6.60	0.09	6.51	0.02	0.01	0.01
87203B	BX Pit	4.040	3.48	86.1	9.92	465	3.30	0.17	3.13	0.03	<0.01	0.03	4.010	3.48	86.8	6.53	3.44	9.9	3.37	0.12	3.25	0.03	<0.01	0.04
87204B	F Zone	0.775	0.62	80.0	1.25	20	7.38	0.22	7.16	0.01	0.01	0.01	1.220	0.62	50.8	4.04	0.70	2.6	7.46	0.08	7.38	0.01	<0.01	0.02
87205B	CD Pit	0.231	0.12	51.9	0.90	9	6.59	0.30	6.30	0.01	<0.01	0.01	0.238	0.12	50.4	3.54	0.21	2.9	6.71	0.06	6.65	0.02	0.02	<0.01
87206B	I Pit	2.552	2.14	83.9	0.63	12	10.40	0.22	10.18	0.03	<0.01	0.03	2.770	2.14	77.3	5.65	2.32	2.9	10.75	0.06	10.70	0.04	0.03	0.01

The following is a summary of the findings from the bottle-roll test results.

13.3.2.1 75 µm (200-Mesh) Bottle-Roll Results

- Gold head grades for the bulk samples ranged from 0.22 ppm to 3.77 ppm Au (average = 1.77 ppm Au).
- Gold extraction results ranged between 64.6% and 91.3% (weight average = 81.8%).
- Silver head grades ranged from 0.50 ppm to 8.92 ppm Ag (average = 3.15 ppm Ag).
- Silver extraction results ranged from 25.3% to 69.8% (weight average = 56.6%).
- Cyanide consumption averaged 0.13 kg/t and lime consumption averaged 0.67 kg/t.

13.3.2.2 75 µm (200-Mesh) CIL Bottle-Roll Results

- Gold head grades for the bulk samples ranged from 0.26 ppm to 4.03 ppm Au (average = 1.91 ppm Au).
- Gold extraction results ranged between 76.6% and 93.3% (weight average = 85.8%).
- Silver head grades for the bulk samples ranged from 0.44 ppm to 9.78 ppm Ag (average = 3.56 ppm Ag).
- Silver extraction results ranged from 29.9% to 72.3% (weight average = 61.3%).
- Cyanide consumption averaged 0.67 kg/t and lime consumption averaged 0.75 kg/t.

The weighted average gold extraction percent for the 75 µm CIL bottle-roll tests was 85.8% vs. 81.8% for the direct leach 75 µm bottle-roll test, indicating mild preg-borrowing or preg-robbing potential.

13.3.2.3 1,700 µm (10-Mesh) Bottle-Roll Results

- Gold head grades for the bulk samples ranged from 0.25 ppm to 3.74 ppm Au (average = 1.81 ppm Au).
- Gold extraction results ranged between 64.5% and 90.1% (weight average = 79.1%).
- Silver head grades for the bulk samples ranged from 0.98 ppm to 9.50 ppm Ag (average = 3.88 ppm Ag).
- Silver extraction results ranged from 10.2% to 59.4% (weight average = 39.8%).
- Cyanide consumption averaged 0.18 kg/t and lime consumption averaged 0.71 kg/t.

13.3.3 2019-2020 Bulk Sample Column-Leach Program

Column-leach cyanidation testing was conducted on six Black Pine bulk composites to evaluate laboratory-scale leachability characteristics of historical pit resources, at coarse particle size, in terms of gold/silver extraction, extraction rate and reagent consumption (KCA 2020a). Column testing was conducted at a target P_{80} (feed size) of 75 mm. Laboratory column charges were leached for 100 days with dilute sodium cyanide solution.

The 75 mm column-leach testing followed a standard procedure outlined in detail in the final KCA-2020a report. The column-leach test results are shown in Table 13-3. The column leach test parameters are presented in Table 13-4.

Table 13-3: Summary Bottle-Roll, CIL, and Column Leach Test Results on 2019 Liberty Gold Bulk Samples

KCA Sample No.	Description	Test No	Geology		Feed Size		Leach Time (days/Hrs)	Au Balance		Ag Balance		Reagents		
			Mine Area	F-Form	Target P80 (µm)	Screen P80 (µm)		Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	NaCN kg/t	Lime kg/t	Cement kg/t
2019 Liberty Bulk Samples														
87201B	Bulk Sample	87212	A Pit	Polc	75,000	58,600	100d	**79.8**	1.161	**30.7**	5.27	0.46	0.00	2.1
	Bulk Sample	87207 A	A Pit	Polc	1,700	1,960	144	**74.6**	1.170	**27.3**	6.49	0.16	0.75	
	Bulk Sample	87230 A	A Pit	Polc	75	97	72	**76.0**	1.083	**47.6**	4.58	0.03	0.75	
	Bulk Sample	87232 A	A Pit	Polc	75CIL	97	72	**79.6**	1.162	**49.7**	4.61	0.58	0.75	
87202B	Bulk Sample	87215	Tallman Pit	Polc	75,000	74,400	100d	**79.7**	1.995	**29.1**	3.71	0.60	0.75	0.0
	Bulk Sample	87207 B	Tallman Pit	Polc	1,700	1,640	144	**79.5**	2.247	**36.0**	3.97	0.15	0.75	
	Bulk Sample	87230 B	Tallman Pit	Polc	75	88	72	**78.9**	2.151	**48.6**	3.33	0.15	0.75	
	Bulk Sample	87232 B	Tallman Pit	Polc	75CIL	88	72	**86.0**	2.294	**58.3**	3.86	0.68	0.75	
87203B	Bulk Sample	87218	Upper BX Pit	Polc	75,000	55,900	100d	**92.8**	3.377	**56.2**	9.25	0.76	0.00	2.0
	Bulk Sample	87207 C	Upper BX Pit	Polc	1,700	1,520	144	**90.1**	3.736	**59.4**	9.50	0.50	0.75	
	Bulk Sample	87230 C	Upper BX Pit	Polc	75	104	72	**91.3**	3.733	**69.8**	8.92	0.38	0.75	
	Bulk Sample	87232 C	Upper BX Pit	Polc	75CIL	104	72	**93.3**	4.033	**72.3**	9.78	1.10	1.00	
87204B	Bulk Sample	87221	Lower F Zone	Pols	75,000	51,300	100d	**78.2**	0.803	**8.5**	1.18	0.53	0.00	1.9
	Bulk Sample	87208 A	Lower F Zone	Pols	1,700	1,480	144	**79.3**	0.777	**12.6**	1.27	0.12	0.75	
	Bulk Sample	87230 D	Lower F Zone	Pols	75	102	72	**80.7**	0.787	**25.3**	0.95	0.08	0.75	
	Bulk Sample	87232 D	Lower F Zone	Pols	75CIL	102	72	**82.5**	0.819	**29.9**	0.87	0.58	0.75	
87205B	Bulk Sample	87224	C/D Pit	Pols	75,000	71,900	100d	**76.8**	0.233	**28.6**	0.49	0.60	0.76	0.0
	Bulk Sample	87208 B	C/D Pit	Pols	1,700	1,580	144	**64.5**	0.245	**14.8**	1.08	0.03	0.75	
	Bulk Sample	87231 A	C/D Pit	Pols	75	114	72	**64.6**	0.223	**40.7**	0.59	0.07	0.50	
	Bulk Sample	87233 A	C/D Pit	Pols	75CIL	114	72	**76.6**	0.261	**44.1**	0.59	0.54	0.75	
87206B	Bulk Sample	87227	I Pit	Pold	75,000	56,100	100d	**60.9**	2.677	**17.6**	0.51	0.60	0.75	0.0
	Bulk Sample	87208 C	I Pit	Pold	1,700	1,740	144	**66.7**	2.663	**10.2**	0.98	0.14	0.50	
	Bulk Sample	87231 B	I Pit	Pold	75	107	72	**74.8**	2.670	**34.0**	0.50	0.09	0.50	
	Bulk Sample	87233 B	I Pit	Pold	75CIL	107	72	**79.3**	2.888	**50.0**	0.44	0.54	0.50	

Table 13-4: 2019-2020 Bulk Sample Column-Leach Test Parameters

KCA Sample No.	KCA Test No.	Description	Crush Size (mm)	Column Diameter (m)	Initial Charge Height (m)	Charge Weight (kg)
87201 B	87212	A Pit	87.5	0.305	2.743	277.04
87202 B	87215	Tallman Pit	87.5	0.305	2.921	308.83
87203 B	87218	Upper BX Pit	87.5	0.305	2.553	283.44
87204 B	87221	Lower F Zone	87.5	0.305	2.896	285.96
87205 B	87224	C/D Pit	87.5	0.305	2.565	288.75
87206 B	87227	I Pit	87.5	0.305	2.597	282.84

13.3.3.1 Column-Leach Test Extractions

Gold extractions ranged from 60.9% to 92.8% based on calculated head grades, which ranged from 0.23 ppm to 3.38 ppm Au (Figure 13-4). The sodium cyanide consumptions ranged from 0.46 kg/t to 0.76 kg/t. The material utilized in leaching was blended with 0.75 kg/t or 0.76 kg/t hydrated lime, with three of the composites agglomerated with 1.86 kg/t to 2.08 kg/t cement. Column test extraction results are based upon carbon assays vs. the calculated head (carbon assays + tail screen assays). The solution balance gold extraction profiles are presented graphically in Figure 13-4.

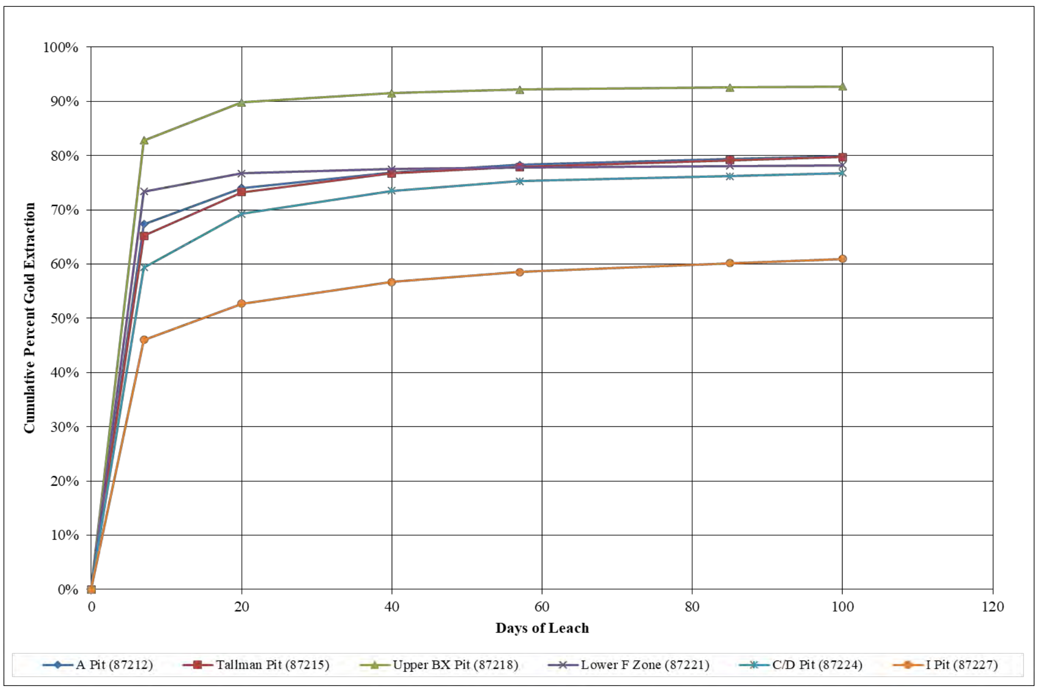


Figure 13-4: 2019-2020 Column-Leach Test Work, Gold Extraction vs. Days of Leach

13.3.3.2 Head vs. Tails Screen Analysis

Tails screen assays demonstrate that gold extraction for all six of the Black Pine bulk samples are minimally sensitive to feed particle size. Figure 13-5 is a typical example of results for the A Pit bulk sample.

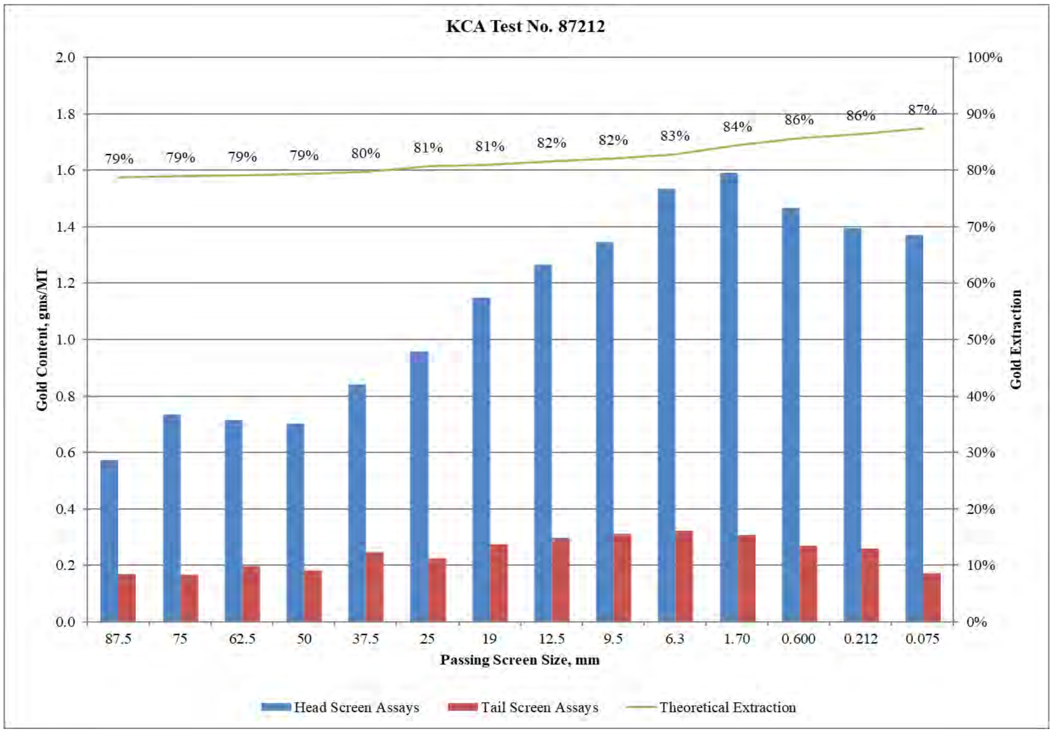


Figure 13-5: A Pit Head vs. Tail Screen Analyses and Gold Extraction by Size Fraction

13.4 2020 Phase 1 Variability Composite Testing

In 2019 Liberty Gold initiated bottle-roll and column-leach testing at KCA on 29 variability composites selected from six large diameter PQ metallurgical core holes, selected from Discovery Zone 1, Discovery Zone 2, and the Rangefront resource locations (KCA 2020b). The large diameter metallurgical core hole collar locations are shown on Figure 13-3.

Splits from eleven of the 29 composites were selected and shipped to Hazen Research, Inc. (Hazen) in Golden, Colorado, for SAG mill comminution ("SMC") testing (SMC Test®) and Bond Abrasion index (Ai) testing. Comminution and abrasion final test results are documented in a letter report from Stepperud (Hazen) to KCA (Stepperud, 2020).

One composite from the 2019 bulk sample program (the I Pit composite) and four variability composites from the 2019 PQ core drilling program (BP73-7, BP73-10, BP78-12, and BP87-25) were selected for gold deportment mineralogy study and were shipped to AMTEL Ltd in Canada ("AMTEL") and are reported in AMTEL (2020).

Splits of all 29 composite heads were delivered to three separate laboratories for additional geo-metallurgical and environmental characterization analysis:

1. ALS for ICP and gold cyanide solubility analysis
2. FLS for "XRD" and "Whole-Rock" analysis
3. Western Environmental Testing Laboratories ("WETLAB") for environmental characterization of solids and aqueous solutions

With reference to Section 13.10.2 of Section 13, the bulk sample and PQ core metallurgical composites reasonably sampled materials from the Pola, Polb, Polc, with minimal samples coming from PPos, Pold, and Pols, all members of the Oquirrh Group. A 2020 metallurgical PQ core drilling program was designed to fill major resource material gaps that were minimally sampled in 2019 (Polc, Pold, and Pols).

13.4.1 2020 Black Pine Variability Composite Head Assays

Head assay details and geo-metallurgical characterization results are in the KCA 2020b report. A high-level summary of the geo-metallurgical characterization is presented below for gold, silver, copper, cyanide gold solubility, carbon and sulfur species, preg-robbing analysis, as well as ICP multi-element analyses, whole-rock analyses, and QXRD analyses. Select composite summary results for gold, silver, copper, carbon and sulfur speciation, and preg-robbing analysis, are detailed in Table 13-5:

- Gold grades ranged from 0.20 ppm to 5.67 ppm and averaged 0.86 ppm.
- Silver grades ranged from 0.85 ppm to 4.0 ppm and averaged 1.9 ppm.
- Organic carbon ranged from 0.07% to 0.20% and averaged 0.11%.
- Sulfide sulfur ranged from <0.01% to 0.01% and averaged <0.01%.
- Preg-robbing analyses ranged from 0.7% to 19.0% and averaged 4.9% (using a 1 ppm spike). Preg-robbing values <10% are considered within the error band of the test procedure and are classified as non-preg-robbing by KCA. Only two composites (BP78-13 and BP78-15) were >10% at 19.0% and 13.3% respectively.
- Copper values were very low, ranging from 10 ppm to 78 ppm and averaged 34 ppm.
- Gold cyanide solubility ranged from 24.9% to 96.8% and averaged 83.4%.
- Concentrations of the deleterious elements were as follows: selenium averaged 19 ppm; mercury ranged from 2.0 to 26.5 ppm with an average of 5.6 ppm; and arsenic levels were low, ranging from 49 to 404 ppm with an average of 155 ppm.
- Concentrations of the primary cyanide consumers were low and suggest minimum potential for affecting cyanide consumption rates. Copper averaged 34 ppm, nickel averaged 73 ppm, and zinc averaged 229 ppm; and
- Whole-rock silica content ranged from 15.5% to 80.2% and averaged 47.8 %.

Table 13-5: 2019 Black Pine Variability Composite Head Assays by KCA and ALS

KCA Sample No.	Composite ID	Head Assays																	
		ALS			KCA												ALS (Preg-robbing)		
		AuFA ppm	AuCN ppm	AuCN %	AgFA ppm	AgCN ppm	AgCN %	Cu ppm	CuCN ppm	CuCN %	C(tot) %	C(org) %	C(inorg) %	S(total) %	S(sulfide) %	S(sulfate) %	AA31 w/spike	AA31a w/o	Au PR %
2019 Phase 1 Variability Core Composites																			
87234A	BP67-1	0.384	0.270	70.3	1.64	0.600	36.6	32	6.84	21.4	4.26	0.09	4.17	0.03	<0.01	0.03	1.143	0.153	1.0
87235A	BP67-2	0.346	0.150	43.4	2.29	1.353	59.1	30	5.24	17.5	0.90	0.20	0.70	0.09	<0.01	0.09	1.067	0.103	3.7
87236A	BP67-3	0.761	0.460	60.4	2.14	1.227	57.4	20	6.04	30.2	5.97	0.11	5.86	0.02	<0.01	0.02	1.113	0.207	9.3
87237A	BP67-4	5.860	5.670	96.8	4.01	1.387	34.6	24	5.73	23.9	2.43	0.14	2.29	0.08	<0.01	0.08	3.737	2.807	7.0
87238A	BP67-5	1.825	1.560	85.5	2.23	0.920	41.3	35	8.89	25.4	3.33	0.13	3.20	0.03	<0.01	0.03	1.650	0.710	6.0
87239A	BP67-6	1.285	1.030	80.2	2.33	1.073	46.2	22	7.60	34.5	6.15	0.14	6.15	0.05	<0.01	0.05	1.443	0.487	4.3
87240A	BP73-7	0.241	0.060	24.9	1.94	0.633	32.6	18	7.37	41.0	5.50	0.12	5.50	0.03	<0.01	0.03	0.943	0.030	8.7
87241A	BP73-8	0.381	0.170	44.6	2.53	1.067	42.2	38	14.21	37.4	5.20	0.12	5.20	0.04	0.01	0.03	0.993	0.070	7.7
87242A	BP73-9	0.273	0.140	51.3	2.23	1.007	45.2	36	7.69	21.4	3.09	0.15	3.08	0.02	<0.01	0.02	1.083	0.100	1.7
87243A	BP73-10	2.440	2.110	86.5	2.05	0.940	45.9	23	15.25	66.3	7.19	0.10	7.19	0.03	<0.01	0.03	1.970	1.003	3.3
87244A	BP73-11	0.533	0.460	86.3	0.85	0.393	46.2	10	10.43	104.3	10.10	0.11	9.99	0.09	<0.01	0.09	1.220	0.237	1.7
87245A	BP78-12	0.810	0.590	72.8	1.83	0.367	20.0	62	20.57	33.2	2.80	0.11	2.69	0.04	<0.01	0.04	1.270	0.300	3.0
87246A	BP78-13	0.436	0.290	66.5	1.37	0.400	29.2	23	8.19	35.6	9.20	0.09	9.11	0.03	<0.01	0.03	0.870	0.060	19.0
87247A	BP78-14	0.354	0.300	84.7	1.59	0.440	27.7	41	13.69	33.4	9.13	0.07	9.06	0.07	0.01	0.06	1.093	0.137	4.3
87248A	BP78-15	2.610	1.960	75.1	2.41	0.567	23.5	26	13.68	52.6	7.59	0.10	7.49	0.07	<0.01	0.07	1.890	1.023	13.3
87249A	BP82-16	0.405	0.310	76.5	1.60	0.307	19.2	41	14.93	36.4	4.80	0.07	4.73	0.03	<0.01	0.03	1.050	0.123	7.3
87250A	BP82-17	0.367	0.260	70.8	1.16	0.440	37.9	34	13.71	40.3	6.95	0.09	6.86	0.01	<0.01	0.01	1.123	0.133	1.0
87251A	BP82-18	0.317	0.130	41.0	2.09	0.980	46.9	78	31.50	40.4	4.45	0.11	4.34	0.01	<0.01	0.01	0.990	0.067	7.7
87252A	BP82-19	0.203	0.120	59.1	1.19	0.527	44.3	69	37.00	53.6	5.80	0.06	5.74	0.01	<0.01	0.01	1.043	0.060	1.7
87253A	BP82-20	0.835	0.730	87.4	1.41	0.473	33.7	49	29.74	60.7	5.23	0.09	5.14	0.03	<0.01	0.03	1.303	0.327	2.3
87254A	BP82-21	0.495	0.370	74.7	1.11	0.593	53.5	38	22.58	59.4	6.74	0.10	6.64	0.02	<0.01	0.02	1.110	0.153	4.3
87255A	BP87-22	0.251	0.110	43.8	1.73	0.613	35.5	53	12.89	24.3	4.99	0.09	4.90	0.01	<0.01	0.01	1.027	0.073	4.7
87256A	BP87-23	0.301	0.210	69.8	1.53	0.373	24.5	27	10.49	38.8	6.06	0.12	5.95	0.03	<0.01	0.03	1.063	0.100	3.7
87257A	BP87-24	0.204	0.060	29.4	3.34	0.713	21.4	57	17.43	30.6	2.60	0.12	2.48	0.02	<0.01	0.02	0.983	0.037	5.3
87258A	BP87-25	1.420	1.180	83.1	4.05	1.800	44.5	23	5.56	24.2	5.97	0.11	5.86	0.04	<0.01	0.04	1.560	0.567	0.7
87259A	BP93-26	0.332	0.280	84.3	1.36	0.587	43.1	24	7.45	31.1	3.53	0.09	3.44	0.04	0.01	0.03	1.120	0.137	1.7
87260A	BP93-27	0.361	0.330	91.4	0.82	0.280	34.0	10	4.31	43.1	4.71	0.14	4.58	0.03	<0.01	0.03	1.130	0.143	1.3
87261A	BP93-28	0.722	0.620	85.9	1.18	0.293	24.9	12	6.47	53.9	4.31	0.11	4.21	0.04	<0.01	0.04	1.263	0.283	2.0
87262A	BP93-29	0.279	0.220	78.9	1.62	0.493	30.4	19	10.29	54.2	5.20	0.08	5.12	0.03	<0.01	0.03	1.073	0.103	3.0

13.4.2 Acid-Base Accounting

A portion of the pulverized head material for each individual sample was submitted to WETLAB for Acid-Base Accounting ("ABA") testing. ABA is a static test to determine the acid producing or acid neutralizing potential of a material. It is a general analysis for the elements of acid generation and does not indicate the potential rate at which generation or neutralization may occur.

It is generally accepted that a net neutralization potential ("NNP") value greater than 20 is indicative of a non-acid producing material (acid neutralizing material), and that an NNP value less than -20 is an acid generating material. All 29 Black Pine composites tested had NNP values greater than 20 and are therefore considered to be non-acid producing.

13.4.3 Bottle-Roll and Column Leach Testing

Coarse and fine milled bottle-roll leach tests were completed on each of the 29 samples. A portion of the head material for each individual sample was subjected to bottle-roll leach testing at target P_{80} sizes of 75 µm and 1,700 µm, and to column-leach testing at either 12.5 mm or 25.0 mm crush sizes (Table 13-6). A second CIL bottle-roll test was conducted at the 75 µm feed size to evaluate the potential for preg-borrowing clays and/or preg-robbing organic carbon. The main objective of these tests was to evaluate the laboratory-scale leachability character of the Black Pine resources in terms of gold extraction, extraction rate, reagent consumption, and sensitivity to feed size.

Table 13-6: 2020 Nominal P_{80} for Bottle-Roll and Column Leach Tests

Bottle-Rolls			Columns	
75 µm	75 µm (CIL)	1,700 µm	12.5 mm	25 mm
n = 29	n = 29	n = 29	n = 9	n = 20

The bottle-roll testing used standard procedures that are described in the final laboratory report (KCA 2020b), using 144 hours of retention time for 1,700 µm tests, and 72 hours for 75 µm direct leach and CIL tests.

Column-leach tests were conducted utilizing material crushed to their target P_{80} sizes and placed in columns of 10 cm and 15 cm diameters. During testing the material was leached between 77 to 80 days with a dilute NaCN solution. After leaching, each column was washed/rinsed for four days with water. A portion of the leached and washed material ("tailings") from each column was assayed for "tail screen" analyses by size fraction.

Tailings material from all 29 columns was utilized for compacted permeability test work. Additionally, tailings material from 19 columns was submitted to WETLAB in Sparks, Nevada, for ABA and meteoric-water mobility tests ("MWMT").

The following is a summary of the findings from the KCA 2020b report on bottle-roll and column test results.

13.4.4 Direct Leach and CIL Bottle-Roll Tests on 75 µm Composite Samples

Fine milled bottle-roll leach tests were completed on each of the 29 composites. The milled slurry was utilized for direct bottle-roll leach testing as well as CIL bottle-roll testing. The bottle-roll test procedures and results are described in detail in KCA 2020b.

- The direct leach gold head grades for the composites ranged from 0.21 ppm to 5.78 ppm Au, with an average of 0.83 ppm Au. Gold extraction from this material ranged from 37.7% to 92.7%, with a weight average of 81.4%.

- The CIL leach gold head grades for the composites ranged from 0.18 ppm to 6.15 ppm Au, with an average of 0.87 ppm Au. Gold extraction from this material ranged from 42.1% to 93.2%, with a weight average of 84.1%.
- The direct leach silver head grades for the composites ranged from 0.47 ppm to 3.9 ppm Ag, with an average of 1.8 ppm Ag. Silver extraction from this material ranged from 11.3% to 59.0%, with a weight average of 26.0%.
- The CIL silver head grades for the composites ranged from 0.83 ppm to 4.1 ppm Ag, with an average of 1.9 ppm Ag. Silver extraction from this material ranged from 3.9% to 57.3%, with a weight average of 26.8%.
- Cyanide consumption for the direct leach bottle-roll tests averaged 0.15 kg/t and lime consumption averaged 0.71 kg/t.
- Cyanide consumption for the CIL bottle-roll tests averaged 0.73 kg/t and lime consumption averaged 0.60 kg/t.

13.4.5 Direct Leach Coarse Bottle-Roll Tests on 1,700 µm Composite Samples

Coarse bottle-roll leach tests were completed on each of the 29 composites. The coarse bottle-roll test procedure and results are described in detail in KCA 2020b.

- Gold head grades for the composites ranged from 0.20 ppm to 6.25 ppm Au and averaged 0.86 ppm Au. Gold extraction ranged from 35.8% to 87.2%, with a weight average of 78.8%.
- Silver head grades for the composites ranged from 0.8 ppm to 4.1 ppm Ag and averaged 2.0 ppm Ag. Silver extraction ranged from 6.3% to 41.5%, with a weight average of 16.4%.

13.4.6 Column-Leach Tests on Composite Samples

All 29 composites were subjected to laboratory column-leach testing at KCA. Nine columns were tested at a target P_{80} of 12.5 mm and twenty composites at a target P_{80} of 25 mm (KCA 2020b). Column test procedures are described in detail in KCA 2020b. Column test extraction results are based upon carbon assays versus the calculated head (carbon assays + tail assays) and test result details are in Table 13-7.

- Calculated gold head grades ranged from 0.214 ppm to 5.44 ppm and averaged 0.84 ppm. Gold extractions ranged from 42.0% to 94.5%, with a weight average of 89.2%.
- Calculated silver head grades ranged from 0.80 ppm to 3.83 ppm and averaged 2.1 ppm. Silver extractions ranged from 4.8% to 41.1%, with a weight average of 15.0%.
- Cyanide consumptions ranged from 0.29 to 0.90 kg/t and averaged 0.56 kg/t. Based upon KCA's experience with clean non-reactive ores, cyanide consumption in commercial production heaps would range between 25% to 33% of the laboratory column test consumptions.
- Lime consumption ranged from 0.99 kg/t to 1.52 kg/t. One column charge (BP67-2) was agglomerated with 4.0 kg/t of cement and did not require any lime.

Gold extraction plotted versus days under leach is shown graphically in Figure 13-6 and are based upon column solution balances.

Table 13-7: 2020 Variability Column Test Results

KCA Sample No.	Comp ID	Test No	Pilot Gold		Feed Size					NaCN (g/l)	Leach Time (days)	Au Balance		Ag Balance		Reagents		
			Structure	F-Form	Target P80 (µm)	Screen P80 (µm)	% - 200M	Load Perm Tests	Cement kg/t			Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	NaCN kg/t	Lime kg/t	Cement kg/t
2019 Variability Core Composites																		
87234A	BP67-1	87278	Fz>Bx	Pola	37,500	23,140	16.5	Fail 75m	0.0	0.5	79d	**81.8**	0.550	**19.5**	2.26	0.64	1.00	0.0
87235A	BP67-2	87281	Fz>Bx	Polb	37,500	13,780	32.2	Fail 50m	4.0	0.5	79d	**71.3**	0.418	**41.1**	2.65	0.56	0.00	4.0
87236A	BP67-3	87284	Bx>Fz	Polc	37,500	24,500	5.6	Pass	0.0	0.5	79d	**69.1**	0.836	**27.0**	2.59	0.53	1.00	0.0
87237A	BP67-4	87287	Fz>Bx	Polc	37,500	23,160	16.6	Pass	0.0	0.5	78d	**94.5**	5.438	**11.7**	3.83	0.55	0.99	0.0
87238A	BP67-5	87290	Fz	Polc	37,500	24,780	20.1	Fail 100m	0.0	0.5	78d	**86.9**	1.693	**16.5**	2.31	0.56	1.51	0.0
87239A	BP67-6	88101	Fz,Bx	Polc	37,500	25,190	6.5	Pass	0.0	0.5	78d	**79.3**	1.330	**12.4**	2.42	0.58	1.52	0.0
87240A	BP73-7	88104	FZ/Bx	Polb,FZ	37,500	22,760	24.5	Pass	0.0	0.5	78d	**42.0**	0.264	**11.7**	2.13	0.45	1.50	0.0
87241A	BP73-8	88107	Bx	Polb	19,000	12,400	14.0	Pass	0.0	0.5	78d	**54.9**	0.384	**20.0**	2.45	0.53	1.44	0.0
87242A	BP73-9	88110	Fz	Polb	19,000	11,110	21.9	Pass	0.0	0.5	78d	**83.5**	0.278	**18.5**	2.27	0.69	1.50	0.0
87243A	BP73-10	88113	Bx,Fz	Polc	37,500	23,280	11.0	Pass	0.0	0.5	78d	**86.7**	2.423	**28.2**	1.74	0.48	1.49	0.0
87244A	BP73-11	88116	0	Polc	19,000	12,360	5.2	Pass	0.0	0.5	77d	**87.1**	0.464	**18.8**	0.85	0.41	1.48	0.0
87245A	BP78-12	88119	Fz	Polb	37,500	20,490	15.7	Fail 100m	0.0	0.5	77d	**72.8**	0.817	**7.0**	2.30	0.57	1.50	0.0
87246A	BP78-13	88112	0	Polc	37,500	26,010	1.5	Pass	0.0	0.5	80d	**82.6**	0.455	**4.9**	1.83	0.29	1.50	0.0
87247A	BP78-14	88125	Bx	Pold	37,500	23,900	4.2	Pass	0.0	0.5	80d	**79.2**	0.380	**8.8**	1.59	0.31	1.34	0.0
87248A	BP78-15	88128	Bx	Pold	19,000	11,830	8.2	Fail 100m	0.0	0.5	80d	**89.1**	2.245	**9.0**	3.01	0.49	1.50	0.0
87249A	BP82-16	88131	Fz,Bx	Pola	19,000	12,190	17.2	Pass	0.0	0.5	80d	**79.2**	0.380	**14.8**	1.89	0.59	1.00	0.0
87250A	BP82-17	88134	Fz	Pola	19,000	12,240	19.6	Fail 50m	0.0	0.5	80d	**77.5**	0.351	**25.0**	1.44	0.70	1.01	0.0
87251A	BP82-18	88137	Fz	Fz/Pola	37,500	24,390	24.9	Fail 50m	0.0	0.5	79d	**56.8**	0.331	**24.0**	2.33	0.69	1.01	0.0
87252A	BP82-19	88140	Fz	Polb	37,500	23,660	15.6	Pass	0.0	0.5	79d	**63.1**	0.214	**7.6**	1.57	0.55	1.02	0.0
87253A	BP82-20	88143	0	Polc	37,500	23,720	8.9	Pass	0.0	0.5	79d	**81.7**	0.798	**5.1**	1.58	0.60	1.13	0.0
87254A	BP82-21	88146	Fz=Bx	Polc	37,500	25,190	19.0	Fail 100m	0.0	0.5	79d	**72.6**	0.453	**11.4**	1.32	0.57	1.01	0.0
87255A	BP87-22	88149	Fz,Bx	Pola	19,000	14,310	19.3	Fail 50M	0.0	0.5	79d	**69.4**	0.258	**23.3**	1.89	0.90	1.00	0.0
87256A	BP87-23	88152	Fz	Polb	37,500	23,940	13.4	Fail 75m	0.0	0.5	79d	**69.2**	0.338	**8.9**	1.58	0.62	1.00	0.0
87257A	BP87-24	88155	Fz	Polb	19,000	10,570	27.2	Fail 75m	0.0	0.5	79d	**60.3**	0.237	**10.0**	3.80	0.75	1.02	0.0
87258A	BP87-25	88158	Fz>Bx	Polc	37,500	25,650	6.2	Psss	0.0	0.5	79d	**85.5**	1.331	**12.6**	3.81	0.52	1.00	0.0
87259A	BP93-26	88161	Fz	PPos,FZ	37,500	24,340	14.6	Pass	0.0	0.5	79d	**77.9**	0.330	**13.9**	1.15	0.80	1.02	0.0
87260A	BP93-27	88164	Bx=Fz	PPos	37,500	22,970	13.2	Pass	0.0	0.5	79d	**81.1**	0.338	**4.8**	1.26	0.47	1.00	0.0
87261A	BP93-28	88167	Bx=Fz	Pola	37,500	22,560	20.5	Fail 75	0.0	0.5	79d	**81.5**	0.601	**9.2**	1.31	0.44	1.01	0.0
87262A	BP93-29	88170	Bx	Pola	37,500	25,370	8.7	Pass	0.0	0.5	79d	**58.8**	0.272	**9.8**	1.63	0.51	1.01	0.0

Source: GL Simmons Consulting LLC, 2020, published in Gustin et al 2021

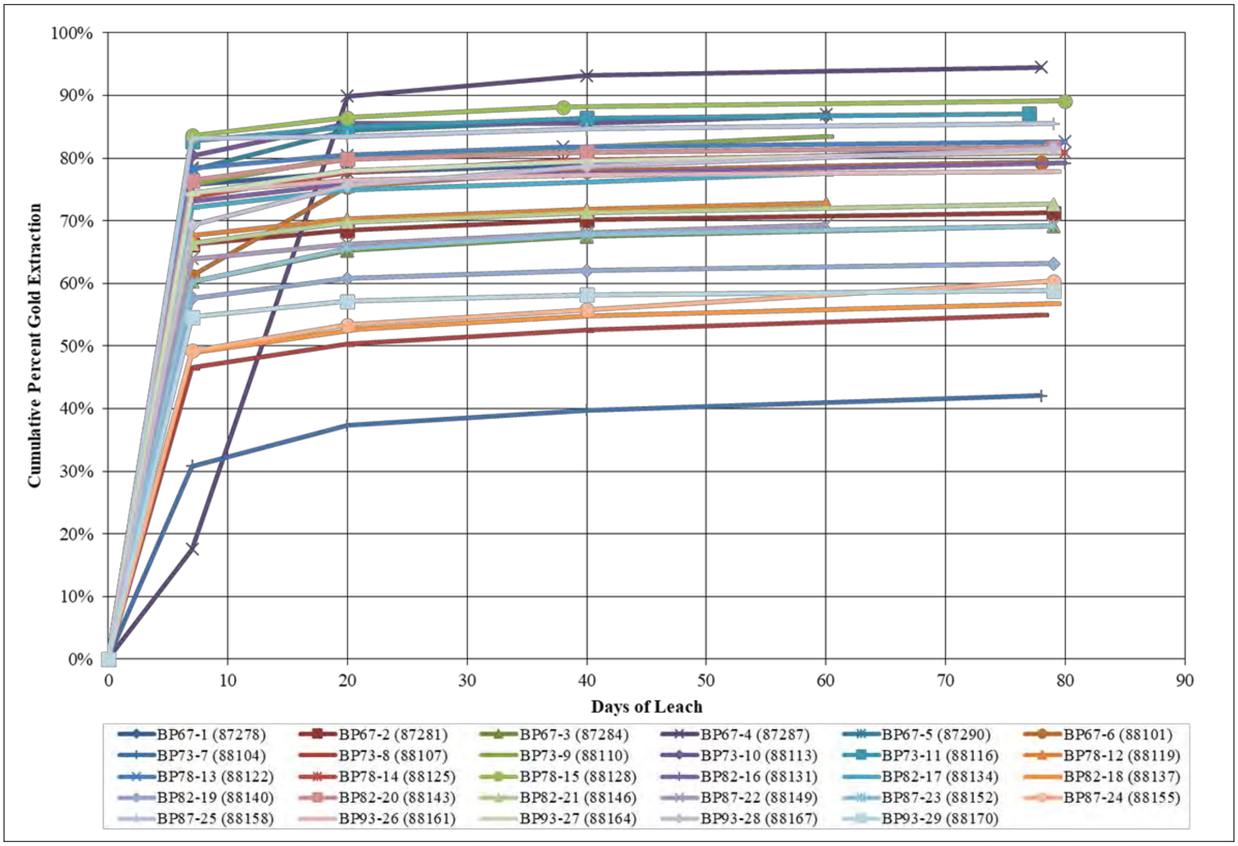


Figure 13-6: 2020 Gold Extraction vs. Days Under Leach for Column-Leach Tests

13.4.7 Comminution Characterization at Hazen

Portions of the head material for 11 composite samples were stage crushed to 100% passing 37.5 mm and submitted to Hazen Research, Inc. in Golden, Colorado for semi autogenous grinding (SAG) mill comminution (SMC), and Ai testing. Details of the comminution testing procedures and test results are reported in Stepperud (2020).

A summary of the Ai test work is presented in Table 13-8 and the SMC comminution characterization in Table 13-9.

Black Pine Ai test results can be characterized as having low to very mild abrasion characteristics, indicating low wear rates on mine ground engaging equipment and process related crushing, screening, and conveying equipment.

Table 13-8: Bond Abrasion Testing Results

HRI No.	Client ID	Ai (g)
55324-1	87234 A	0.0564
55324-2	87238 A	0.0269
55324-3	87240 A	0.0216
55324-4	87243 A	0.0078
55324-5	87248 A	0.0510
55324-6	87252 A	0.0386
55324-7	87254 A	0.0178
55324-8	87255 A	0.0114
55324-9	87258 A	0.0956
55324-10	87260 A	0.0396
55324-11	87261 A	0.0206

Table 13-9: SMC Comminution Characterization Test Results

HRI No.	Client ID	Black Pine Project – SMC Comminution Characterization										
		sg	A	b	A x b	Dwi kWh/m3	Dwi %	Mia kWh/t	Mih kWh/t	Mic kWh/t	ta	SCSE kWh/t
55324-1	87234 A	2.50	57.8	1.00	57.80	4.33	21	14.8	10.1	5.2	0.60	8.29
55324-2	87238 A	2.35	55.8	1.34	74.77	3.14	11	12.2	7.8	4.0	0.82	7.65
55324-3	87240 A	2.56	60.1	1.28	76.93	3.32	13	11.7	7.5	3.9	0.78	7.46
55324-4	87243 A	2.52	60.6	1.15	69.69	3.60	15	12.7	8.3	4.3	0.72	7.73
55324-5	87248 A	2.67	59.0	1.08	63.72	4.17	20	13.5	9.1	4.7	0.62	8.05
55324-6	87252 A	2.51	59.6	1.00	59.60	4.20	20	14.4	9.7	5.0	0.61	8.19
55324-7	87254 A	2.59	59.9	1.24	74.28	3.48	14	12.0	7.8	4.0	0.74	7.55
55324-8	87255 A	2.42	59.0	1.25	73.75	3.28	12	12.2	7.8	4.1	0.79	7.62
55324-9	87258 A	2.65	63.3	0.81	51.27	5.19	31	16.2	11.4	5.9	0.50	8.78
55324-10	87260 A	2.57	57.5	1.09	62.68	4.12	19	13.9	9.3	4.8	0.63	8.04
55324-11	87261 A	2.60	61.8	0.89	55.00	4.74	26	15.4	10.9	5.5	0.55	8.48

SMC Parameters:
A = maximum breakage
b = relation between energy and impact breakage
A x b = overall AG-SAG hardness
Dwi = drop weight index
Mia – coarse particle component
Mic = crusher component
Mih = high-pressure grinding roll component
SCSE = SAG circuit specific energy
sg = specific gravity of sample
ta = low energy abrasion component of breakage

The eleven composites were subjected to the modified SMC Test at Hazen to generate data for SAG mill comminution parameters and crushing index (Mic) by JKTech (Stepperud, 2020). The 2020 SMC Test® results for the 11 samples are given in Table 13-9. This table includes the average rock specific gravity, A x b (a measure of resistance to impact breakage) and drop-weight index (Dwi) values that are the direct result of the SMC Test® procedure. The values determined for the Mia, Mih, and Mic parameters, and the definitions provided by SMCT, are also presented in Table 13-9.

The Dwi ranged from 3.14 kWh/m^3 to 5.19 kWh/m^3, indicating soft materials. This is based on a comparison with the Dwi% column, which ranks the samples in terms of energy required in the SMC worldwide database, 0% being the lowest and 100% being the highest.

Mic (kWh/t) is the SMC crusher component energy required and is used to assist in design and selection of conventional crushing circuits. The Black Pine samples tested can be considered amenable to conventional, multi-stage crushing and screening circuit design. Mic, the SMC crusher component value, with an average of 4.7 kWh/t, would be ranked in the lower mid-range of the SMC worldwide database.

13.4.8 Load Permeability Test Work on Column Tailings

A portion of tailings material from each column-leach test was utilized for load permeability test work. The purpose of the load permeability testing was to examine the permeability of the crushed material under compaction loading equivalent to heap heights of 25 m, 50 m, 75 m, and 100 m.

The test cell utilized for modelling the permeability of stacked material, at various heap heights, was a steel column or cell. Staged axial (vertical) loading of the test material was utilized to simulate the incrementally increased pressure obtained when loading the heap. Drainage layers were installed at the top and at the base of the column. External load was applied to the charge of material in the column utilizing a perforated steel plate that moved freely within the walls of the column. A detailed description of the load permeability equipment, test procedure, and evaluation criteria is given in KCA (2020b).

All 29 columns were tested by KCA. Twelve of the columns failed at loading heights between 50 m and 100 m. Only one of the 12 columns was agglomerated with cement (BP67-2). Review of the column residue screen analysis show that 15 of the columns contained >15% of 75 µm (200 mesh) fines in the column feed. Of these 15 columns, 10 failed load permeability testing (Table 13-10).

It is recommended that future column-leach test programs include additional agglomeration testing and evaluation to devise methods to identify material types that will require ROM blending and/or crushing/agglomeration, before being placed on a heap leach pad. One approach may be to consider that materials containing <15% of 200 mesh fines are suitable for ROM blending on the leach pad, whereas materials containing >15% of 200 mesh fines may require primary blending at the mine bench level into haul trucks, secondary blending on the leach pad and ripping with a deep shank dozer or crushing and agglomeration prior to being placed on the pad for leaching.

Note: Subsequent phases of large diameter metallurgical core testing have shown that this first phase of variability composite test work, focused in the Discovery Zone, is an area of high clay content in Pola and Polb ore types and does not represent clay content of the mineralized resources on a property wide basis.

Table 13-10: Black Pine: % -200 Mesh vs. Pass/Fail Load Permeability Testing

KCA Sample No.	Comp ID	Test No	Pilot Gold Geology		Feed Size				
			Structure	F-Form	Target P80 (µm)	Screen P80 (µm)	% -200M	Load Perm Tests	Cement kg/t
2019 Variability Core Composites									
87234A	BP67-1	87278	Fz>Bx	Pola	37,500	23,140	16.5	Fail 75m	0.0
87235A	BP67-2	87281	Fz>Bx	Polb	37,500	13,780	32.2	Fail 50m	4.0
87236A	BP67-3	87284	Bx>Fz	Polc	37,500	24,500	5.6	Pass	0.0
87237A	BP67-4	87287	Fz>Bx	Polc	37,500	23,160	16.6	Pass	0.0
87238A	BP67-5	87290	Fz	Polc	37,500	24,780	20.1	Fail 100m	0.0
87239A	BP67-6	88101	Fz,Bx	Polc	37,500	25,190	6.5	Pass	0.0
87240A	BP73-7	88104	FZ/Bx	Polb,FZ	37,500	22,760	24.5	Pass	0.0
87241A	BP73-8	88107	Bx	Polb	19,000	12,400	14.0	Pass	0.0
87242A	BP73-9	88110	Fz	Polb	19,000	11,110	21.9	Pass	0.0
87243A	BP73-10	88113	Bx,Fz	Polc	37,500	23,280	11.0	Pass	0.0
87244A	BP73-11	88116	0	Polc	19,000	12,360	5.2	Pass	0.0
87245A	BP78-12	88119	Fz	Polb	37,500	20,490	15.7	Fail 100m	0.0
87246A	BP78-13	88112	0	Polc	37,500	26,010	1.5	Pass	0.0
87247A	BP78-14	88125	Bx	Pold	37,500	23,900	4.2	Pass	0.0
87248A	BP78-15	88128	Bx	Pold	19,000	11,830	8.2	Fail 100m	0.0
87249A	BP82-16	88131	Fz,Bx	Pola	19,000	12,190	17.2	Pass	0.0
87250A	BP82-17	88134	Fz	Pola	19,000	12,240	19.6	Fail 50m	0.0
87251A	BP82-18	88137	Fz	Fz/Pola	37,500	24,390	24.9	Fail 50m	0.0
87252A	BP82-19	88140	Fz	Polb	37,500	23,660	15.6	Pass	0.0
87253A	BP82-20	88143	0	Polc	37,500	23,720	8.9	Pass	0.0
87254A	BP82-21	88146	Fz=Bx	Polc	37,500	25,190	19.0	Fail 100m	0.0
87255A	BP87-22	88149	Fz,Bx	Pola	19,000	14,310	19.3	Fail 50M	0.0
87256A	BP87-23	88152	Fz	Polb	37,500	23,940	13.4	Fail 75m	0.0
87257A	BP87-24	88155	Fz	Polb	19,000	10,570	27.2	Fail 75m	0.0
87258A	BP87-25	88158	Fz>Bx	Polc	37,500	25,650	6.2	Psss	0.0
87259A	BP93-26	88161	Fz	PPos,FZ	37,500	24,340	14.6	Pass	0.0
87260A	BP93-27	88164	Bx=Fz	PPos	37,500	22,970	13.2	Pass	0.0
87261A	BP93-28	88167	Bx=Fz	Pola	37,500	22,560	20.5	Fail 75	0.0
87262A	BP93-29	88170	Bx	Pola	37,500	25,370	8.7	Pass	0.0

Source: GL Simmons Consulting LLC 2020, published in Gustin et al 2021

Note. Magenta color – correlation between % -200 Mesh and Load Permeability Pass/Fail Test Results

13.5 Mineralogy

Five column feed samples were selected for gold deportment mineralogy study and shipped to AMTEL in London, Ontario, Canada, including one sample from the Liberty Gold bulk sample program (I Pit) and four from the Phase 1 Variability testing program (BP73-7, BP73-10, BP78-12, and BP87-25) (AMTEL 2020). Select head assays for the five Black Pine mineralogy samples are provided in Table 13-11.

Table 13-11: Select Head Assays for Black Pine Mineralogy Samples

Sample ID	g Au/t		Fe (%)		TOC (%)		Stot (%)	S_{SO4} (%)	Ctot (%)
	Independent	Site	Independent	Site	Independent	Site			
BP73-7	0.264 ±0.004	0.23	1.4	1.2	0.08	0.05-0.08	0.02	<0.01	4.8
BP73-10	2.012 ±0.028	2.47	1.5	1.6	0.05	0.04-0.08	0.02	<0.01	7.4
BP78-12	0.789 ±0.009	0.83	2.6	2.7	0.03	0.04-0.04	0.035	<0.01	2.8
BP87-25	1.290 ±0.013	1.33	1.1	0.9	0.02	0.00-0.04	0.04	0.02	5.8
I Pit	2.070 ±0.020	2.55	0.2	0.6	0.01[1]	0.06	0.015	<0.01	10.8

Notes:

1. Independent assays by ALS Laboratories, Vancouver, Canada
2. Gold represents the average of triplicate 30g fire assays –AAS finish
3. Fe is the average of ICP & XRF assays
4. TOC independent method C-IR06a; comparable to site's upper range.

The I Pit sample was selected due to its high gold grade and lower gold extraction than all other higher-grade samples. The other four composite samples were selected based upon their variability in gold grade, clay, and organic carbon content (potential for preg-robbing).

13.5.1 Mineralogy Summary

Five samples were received for gold deportment analysis. The samples are oxide composites showing variable gold extractions. The scope of the examination was to identify and quantify all gold forms/carriers to understand the factor(s) limiting gold recovery.

The samples were received coarsely crushed and were milled to P_{80} of approximately 100 µm for analysis. Selected assays of the samples are given in Table 13-11. They compare well generally with site assays, except the gold grades for BP73-10 and I Pit samples that are approximately 20% lower than site. TOC assays are somewhat variable; grade discrepancies are attributed to low TOC content close to the method's detection limit.

General Mineralogy

- In all five samples the principal rock minerals are: quartz, carbonates, and clays/mica. The abundance of these minerals is highly variable in the five samples.
- Carbonates are calcite and dolomite with no iron content.
- Clay/mica minerals are essentially illite/muscovite with lesser kaolinite and very minor biotite. No other clay minerals are present.
- Iron is mostly oxide/oxyhydroxide minerals such as hematite and goethite. Combined they are termed FeO_x. The FeO_x content of the samples ranges from 0.2-3.4 wt%.
- FeO_x is present as massive particles, generally free, and fine-grained particles disseminated in rock. Disseminated FeO_x is preferentially found with quartz/clay particles and lesser carbonates.
- Sulphide minerals are very insignificant, consisting of trace pyrite and very rare chalcopyrite and covellite. The latter is too rare to impact cyanide consumption.

13.5.2 Gold Occurrence:

Gold grains:

- Are free milling gold particles >0.5 µm; and
- Are readily cyanidable; recovery is controlled by exposure versus locking in rock.

Colloidal-gold particulates:

- Are particulates <0.5 µm; they formed during weathering by coalescing gold atoms originally in solid solution form in pyrite.
- Colloidal particulates are associated with the FeO_x or in their near vicinity in rock.
- Colloidal gold is cyanidable. Recovery is ultimately controlled by exposure from rock.

13.5.3 Gold Deportment:

13.5.3.1 Exposed Gold:

At P_{80} of 90-100 µm, 46% to 87.5% of the gold is exposed and cyanidable. The least exposed gold is in BP73-7 and the most in BP73-10 and BP87-25. Exposed gold is in the form of

- Free gold grains characterized by:
    - Contribute 37-51% of the head grade of the five ores.
    - Composition: gold grains are gold-silver alloys with generally low silver content. The average silver content of gold particles is 7 wt% in BP73-7 and 2-3 wt% in the other four samples.
    - Grain size: gold grains are predominantly very small, generally <10 µm. Gold grains >10 µm are in quantifiable quantity in BP87-25 and the I Pit sample.
- Attached colloidal particulates and gold grains:
    - Attached gold contributes 9% to 44% of the head grades of the five samples; with the lowest contribution in BP73-7.
    - Except for I Pit, in the other four samples most of the attached gold is contributed by colloidal particulates with the FeO_x. In the I Pit sample; gold is also associated with rock.

13.5.3.2 Enclosed Gold:

- Enclosed gold is mostly in the form of tiny gold grains not yet liberated.
- Except for the I Pit sample, enclosed gold is minor, carrying 0.02-0.08 g Au/t. In the I Pit sample, enclosed gold is substantial, representing approximately 0.3 g Au/t or 13% of the head grade.
- In the I Pit ore, the quantity of enclosed gold is a function of the grind fineness. With finer grinding, enclosed gold is reduced, gradually liberating from carbonates and largely liberated from quartz at grinds finer than 45 µm.

13.5.3.3 Refractory Gold:

- Ranges from 9% to 49% of the gold in the ores examined.
- Refractory gold comprises colloidal gold particulates that are not cyanidable even after ultra-fine grinding. These are the tiniest colloidal particulates and are associated with the smallest and most disseminated FeOx.

13.5.3.4 Gold Preg-robbing by C-matter:

- C-matter occurs as very tiny ‘specks’ disseminated in quartz which liberate in the slimes. The tiny C-matter has a high surface area, hence increased capacity to preg-robbing gold.
- To determine C-matter ability to preg-rob gold an independent CN test with and without gold spike was performed and, when possible, C-matter was picked for direct analysis.
- It was determined that:
    - AMTEL’s spike test compares closely with KCA’s under similar gold spiking concentration.

- o C-matter from all five ores is preg-robbing. Even a very small TOC content causes measurable preg-robbing.
- o CIL reduced preg-robbing but it did not prevent it entirely.

13.5.3.5 Gold Preg-borrowing by Clays:

- To determine the ability of clay to preg-borrow gold, a clay-rich fraction was separated from the slimes (<10 µm). The clay fraction was measured by TOF-RIMS after contact with gold in solution, and in the absence and presence of free cyanide and activated carbon.
- It was determined that clays are minor preg-borrowers releasing the gold when free cyanide is present. In CIL, clays do not preg-borrow gold.

The relative contributions of the different modes of occurrence of gold to each sample is illustrated graphically in Figure 13-7.

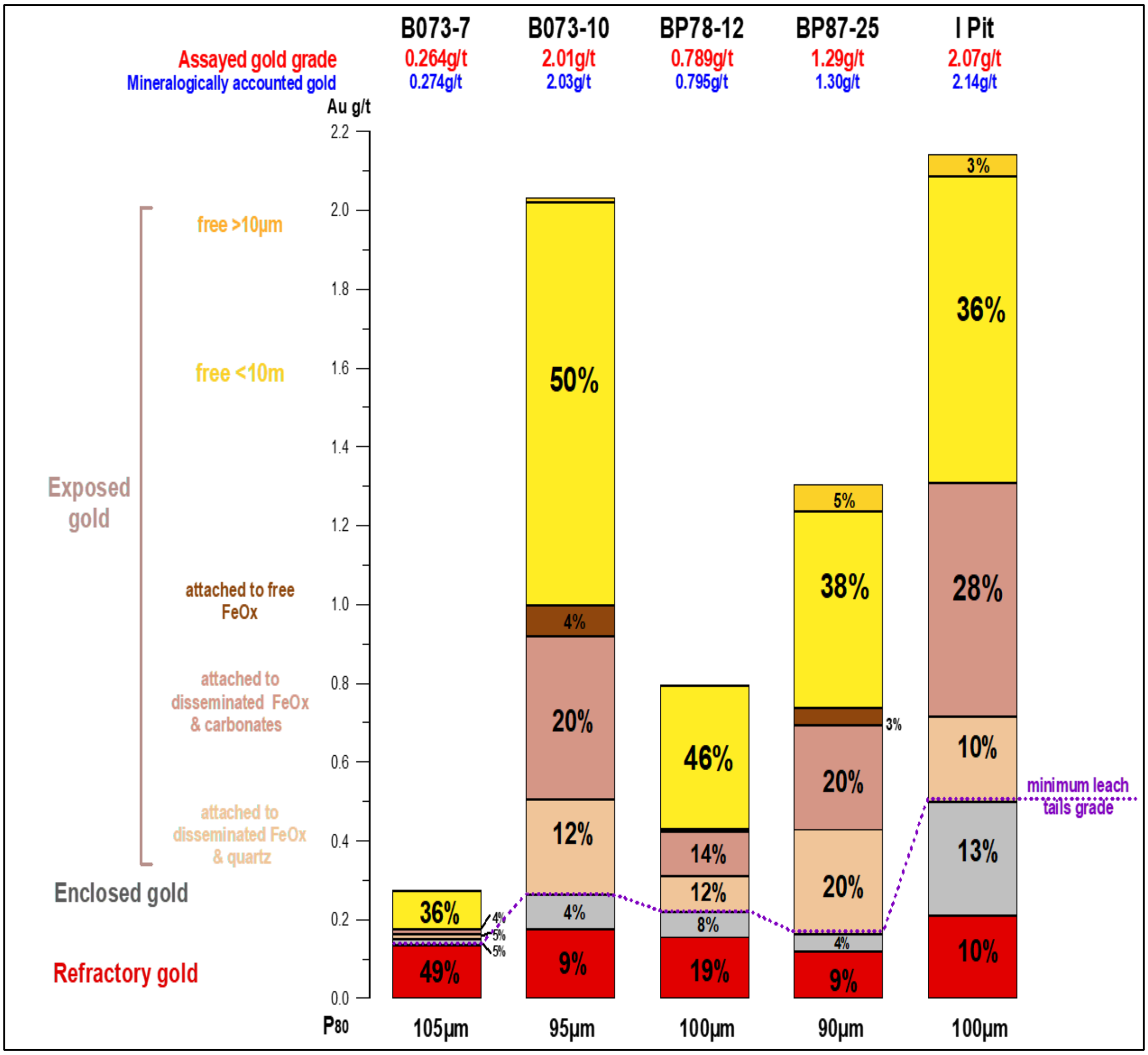


Figure 13-7: Gold Deportment from a Leach Perspective

13.5.4 AMTEL Mineralogy Conclusions

- Gold is present as tiny gold grains and even smaller colloidal particulates. The small size of gold grains and colloidal particulates contributes to fast leach kinetics.
- Gold grains are generally free. Colloidal gold particulates are largely associated with FeO_x, especially the fine-grained disseminated type.
- Gold liberation as a function of grind fineness is not an issue except for with respect to the I Pit sample. KCA test work for the four other samples determined only minor improvement in recovery by crushing from 10 mesh down to 200 mesh.
- Material refractoriness is the main factor limiting gold extraction. Refractoriness is attributed to very small colloidal gold particles carried by the highly disseminated FeO_x in quartz (formerly disseminated pyrite deposited during silicification events).
- An attempt to determine if increased FeO_x dissemination/material refractoriness correlates with the silica content or silica/alumina ratio in the material was unsuccessful.
- No other mineralogical indicators were identified to use as predictive tools for the lower- extraction samples. Samples' location within the geological domains may be more telling.
- C-matter affinity to preg-rob gold is an added problem. C-matter from all five samples showed preg-robbing capabilities. Under CIL conditions preg-robbing is reduced but not entirely eliminated. The TOC content of other future materials should be monitored.
- Gold preg-borrowing by clays is not an issue under CIL conditions.
- Predicted vs. achieved gold extraction in bottle-roll tests is tabulated below in Table 13-12. CIL test work performed at KCA is very well optimized, with the limitation of some preg-robbing taking place in cyanide leach bottle-roll tests.

Table 13-12: Predicted vs. Achieved Gold Extraction

	Predicted Extraction @ P_{80} of 100µm		KCA Attained Extraction @ P_{80} of 75µm	
	CIL Tails	Recovery	CN	CIL
BP73-7	0.148	45.9	29.5	42.1
BP73-10	0.261	87.2	82.3	87.4
BP78-12	0.218	72.6	71.7	74.2
BP87-25	0.163	87.5	83.8	85.8
I Pit	0.496	76.8	74.8	79.3

13.6 2021 Phase 2 Variability Testing

In February 2021 Liberty Gold delivered 45 individual Black Pine variability composites from ten large diameter PQ core holes to KCA for geo-metallurgical characterization, bottle-roll and column leach testing. All preparation, assaying, and metallurgical studies were performed utilizing accepted industry standard procedures, the same as used for Phase 1, and are documented in KCA-2021 Final Report. The variability composites represent resource materials from Discovery Zone 1, Discovery Zone 2, Discovery Zone 3, C/D Pit, E-Pit and F-Zone. The large diameter metallurgical core hole collar locations are shown on Figure 13-3.

Splits of all 45 composite heads were delivered to three separate laboratories for additional geo-metallurgical and environmental characterization analysis:

1. ALS for ICP and gold cyanide solubility analysis,
2. FLS for "XRD" and "Whole-Rock" analysis and
3. WETLAB for environmental characterization of solids and aqueous solutions.

The Phase 2 - 2021 metallurgical PQ core drilling program was designed to fill major resource gaps that were minimally sampled in Phase 1.

13.6.1 2022 Black Pine Variability Composite Head Assays

Head assay details and geo-metallurgical characterization results are in the KCA-2021 report. A high-level summary of the geo-metallurgical characterization is presented below for gold, silver, copper, gold cyanide solubility, carbon and sulfur species, preg-robbing analysis, as well as ICP multi-element analyses, whole-rock analyses, and QXRD analyses. Select composite summary results for gold, silver, copper, carbon and sulfur speciation, and preg-robbing analysis, are detailed in Table 13-5:

- Gold grades ranged from 0.21 ppm to 5.54 ppm and averaged 1.25 ppm.
- Silver grades ranged from 0.34 ppm to 7.3 ppm and averaged 2.1 ppm.
- Organic carbon ranged from <0.01% to 0.32% and averaged 0.10%.
- Sulfide sulfur ranged from <0.01% to 0.04% and averaged <0.01%.
- Preg-robbing analyses ranged from 0.0% to 56.4% and averaged 3.9% (using a 1 ppm spike). Preg-robbing values <10% are considered within the error band of the test procedure and are classified as non-preg-robbing by KCA. Only three composites (BP207-47, BP222-58, and BP231-65) were >10%.
- Copper values were very low, ranging from 2 ppm to 60 ppm and averaged 25 ppm.
- Gold cyanide solubility ranged from 24.3% to 95.8% and averaged 71.6%. Note: A number of composites are considered to be transitional and mildly sulfidic in nature and were tested to help determine the limit of cyanide solubility that can economically be placed on a heap leach.
- Concentrations of the deleterious elements were as follows: selenium averaged 13 ppm; mercury ranged from 1.9 ppm to 24.2 ppm with an average 6.9 ppm; and arsenic levels were low, ranging from 9 ppm to 222 ppm with an average of 95 ppm.
- Concentrations of the primary cyanide consumers were low and suggest minimum potential for affecting cyanide consumption rates. Copper averaged 25 ppm, nickel averaged 50 ppm, and zinc averaged 161 ppm.
- Whole-rock quartz content ranged from 14.7% to 80.6% and averaged 39.8%.

Table 13-13: 2021 Black Pine Variability Composite Head Assays by KCA and ALS

KCA Sample No.	Composite ID	Head Assays																	
		ALS			KCA														
		AuFA (ppm)	AuCN (ppm)	AuCN (%)	AgFA (ppm)	AgCN (ppm)	AgCN (%)	Cu (ppm)	CuCN (ppm)	CuCN %	C(tot) (%)	C(org) %	C(inorg) (%)	$S_{(total)}$%	$S_{(sulfide)}$ %	$S_{(sulfate)}$ %	Au-w/spike	Au-w/o spike	Au PR (%)
88826 A	BP190-30	0.213	0.120	56.5	2.40	1.080	45.0	30	6.28	20.9	3.46	0.23	3.23	0.01	<0.01	0.01	0.060	0.99	7.9
88827 A	BP190-31	0.463	0.240	51.9	2.40	0.520	21.7	28	4.76	17.0	5.99	0.11	5.88	0.02	<0.01	0.02	0.120	1.09	4.0
88828 A	BP190-32	0.614	0.540	88.0	1.65	0.480	29.2	38	6.04	15.9	6.54	0.06	6.48	0.04	<0.01	0.04	0.270	1.31	0.0
88829 A	BP190-33	0.487	0.360	73.9	2.30	0.540	23.5	38	6.00	15.8	5.69	0.12	5.57	0.02	<0.01	0.02	0.180	1.21	0.0
88830 A	BP190-34	0.727	0.440	60.5	1.51	0.480	31.8	7	2.24	32.0	7.57	0.11	7.46	0.04	<0.01	0.04	0.220	1.21	2.0
88831 A	BP190-35	0.360	0.320	88.9	2.78	0.640	23.0	13	4.84	37.2	8.15	0.02	8.13	0.05	<0.01	0.05	0.160	1.19	0.0
88832 A	BP190-36	3.233	3.020	93.4	2.98	1.340	44.9	15	3.48	23.2	5.87	0.10	5.77	0.05	<0.01	0.05	1.510	2.71	0.0
88833 A	BP197-37	1.406	0.760	54.1	0.96	0.360	37.5	2	1.44	72.0	9.36	0.13	9.23	0.03	<0.01	0.02	0.380	1.29	9.9
88834 A	BP197-38	0.396	0.320	80.8	1.03	0.240	23.3	3	1.86	62.0	8.85	0.07	8.78	0.02	<0.01	0.02	0.160	1.15	2.0
88835 A	BP197-39	0.662	0.620	93.7	1.89	0.420	22.3	14	2.56	18.3	6.96	0.07	6.89	0.04	<0.01	0.04	0.310	1.38	0.0
88836 A	BP197-40	2.019	1.700	84.2	1.47	0.280	19.0	12	3.90	32.5	6.69	0.09	6.60	0.01	<0.01	0.01	0.850	1.98	0.0
88837 A	BP197-41	0.530	0.440	83.1	1.20	0.240	20.0	7	3.36	48.0	8.07	0.07	8.00	0.01	<0.01	0.01	0.220	1.23	0.0
88838 A	BP197-42	1.203	1.040	86.4	2.13	0.560	26.3	32	8.52	26.6	3.70	0.08	3.62	0.01	<0.01	0.01	0.520	1.59	0.0
88839 A	BP197-43	0.432	0.240	55.6	0.79	0.160	20.3	15	1.86	12.4	5.73	0.06	5.67	0.03	<0.01	0.03	0.120	1.08	5.0
88840 A	BP207-44	0.247	0.180	72.9	2.13	0.900	42.3	9	2.18	24.2	9.08	0.04	9.04	0.04	<0.01	0.04	0.090	1.10	0.0
88841 A	BP207-45	1.392	0.940	67.5	6.17	3.320	53.8	26	3.58	13.8	4.18	0.06	4.12	0.03	<0.01	0.03	0.470	1.48	0.0
88842 A	BP207-46	0.885	0.440	49.7	1.95	0.860	44.0	23	3.28	14.3	5.35	0.11	5.24	0.02	<0.01	0.02	0.220	1.16	6.9
88843 A	BP207-47	2.657	0.660	24.8	2.78	1.060	38.2	15	5.04	33.6	7.04	0.26	6.78	0.04	<0.01	0.04	0.330	0.77	56.4
88844 A	BP207-48	4.173	3.280	78.6	2.57	0.760	29.6	13	2.00	15.4	6.66	0.09	6.57	0.02	<0.01	0.02	1.640	2.71	0.0
88845 A	BP207-49	1.358	1.000	73.7	1.92	0.520	27.1	26	1.70	6.5	6.43	0.23	6.20	0.03	<0.01	0.03	0.500	1.54	0.0
88846 A	BP214-50	0.295	0.200	67.8	1.61	0.760	47.2	30	4.30	14.3	7.11	0.04	7.07	0.06	<0.01	0.06	0.100	1.11	0.0
88847 A	BP214-51	0.298	0.260	87.2	3.50	1.180	33.7	60	5.90	9.8	0.04	0.04	<0.01	0.04	<0.01	0.04	0.130	1.17	0.0
88848 A	BP214-52	0.970	0.500	51.5	1.71	0.840	49.0	41	3.44	8.4	4.74	0.17	4.57	0.03	<0.01	0.03	0.250	1.20	5.0
88849 A	BP214-53	1.155	0.460	39.8	2.19	0.380	17.3	25	5.04	20.2	8.46	0.16	8.30	0.02	<0.01	0.02	0.230	1.26	0.0
88850 A	BP214-54	5.541	5.060	91.3	3.05	1.300	42.6	42	3.92	9.3	3.57	0.32	3.25	0.06	<0.01	0.06	2.530	3.73	0.0
88851 A	BP214-55	3.854	3.540	91.9	7.27	5.560	76.5	74	8.58	11.6	2.83	0.12	2.71	0.27	0.02	0.25	1.770	2.94	0.0
88852 A	BP214-56	2.229	1.580	70.9	3.05	1.940	63.6	36	8.54	23.7	4.15	0.09	4.06	0.26	0.04	0.22	0.790	1.84	0.0
88853 A	BP222-57	0.470	0.380	80.9	0.51	0.300	58.3	33	3.30	10.0	6.50	0.01	6.49	0.03	<0.01	0.03	0.190	1.21	0.0
88854 A	BP222-58	0.261	0.100	38.4	0.69	0.200	29.2	19	2.52	13.3	8.70	0.17	8.53	0.02	<0.01	0.02	0.050	0.88	17.0
88855 A	BP222-59	0.298	0.140	46.9	1.34	0.500	37.4	29	5.56	19.2	7.49	0.09	7.40	0.04	<0.01	0.04	0.070	0.99	8.0
88856 A	BP222-60	0.645	0.340	52.7	1.01	0.240	23.7	22	2.26	10.3	6.88	0.06	6.82	0.02	<0.01	0.02	0.170	1.11	6.0
88857 A	BP222-61	3.727	3.200	85.9	4.01	1.500	37.4	41	6.24	15.2	7.19	0.18	7.01	0.07	<0.01	0.07	1.600	2.89	0.0
88858 A	BP222-62	1.872	1.440	76.9	1.82	0.560	30.8	58	5.34	9.2	7.63	0.11	7.52	0.02	<0.01	0.02	0.720	1.82	0.0
88859 A	BP222-63	0.432	0.320	74.1	1.61	0.680	42.2	29	2.34	8.1	6.71	0.09	6.62	0.05	<0.01	0.05	0.160	1.18	0.0
88860 A	BP231-64	0.254	0.120	47.3	0.99	0.320	32.2	13	1.60	12.3	11.00	0.10	10.90	0.05	0.01	0.04	0.060	0.98	8.0
88861 A	BP231-65	0.329	0.080	24.3	0.34	0.160	46.7	14	3.56	25.4	9.73	0.12	9.61	0.01	<0.01	0.01	0.040	0.84	20.0
88862 A	BP231-66	0.710	0.520	73.3	2.19	0.480	21.9	41	5.64	13.8	3.99	0.10	3.89	0.03	<0.01	0.03	0.260	1.26	0.0
88863 A	BP231-67	0.669	0.540	80.8	1.27	0.460	36.3	24	1.52	6.3	7.34	0.04	7.30	0.04	<0.01	0.04	0.270	1.31	0.0
88864 A	BP231-68	2.623	2.440	93.0	2.95	0.680	23.1	38	4.52	11.9	1.90	0.06	1.84	0.01	<0.01	0.01	1.220	2.35	0.0
90915 A	BP242-69	1.186	1.080	91.1	2.63	0.760	28.9	36	4.10	11.4	3.94	<0.01	3.94	0.04	<0.01	0.04	0.540	1.56	0.0
90916 A	BP242-70	3.200	3.020	94.4	2.94	0.780	26.6	14	3.68	26.3	5.16	<0.01	5.16	0.03	<0.01	0.03	1.510	2.67	0.0
90917 A	BP247-71	0.299	0.260	87.0	2.01	0.900	44.7	10	2.60	26.0	3.17	0.05	3.12	0.03	<0.01	0.03	0.130	1.09	4.0
90918 A	BP247-72	0.441	0.340	77.1	2.01	0.480	23.9	23	8.06	35.0	7.23	0.01	7.22	0.03	<0.01	0.03	0.170	1.09	8.0
90919 A	BP251-73	0.616	0.520	84.4	1.29	0.120	9.3	7	5.34	76.3	8.12	<0.01	8.12	0.12	<0.01	0.12	0.260	1.25	1.0
90920 A	BP251-74	0.383	0.367	95.8	1.06	0.160	15.1	2	5.96	N/A	9.39	0.07	9.32	0.03	<0.01	0.03	0.183	1.16	2.7

13.6.2 Acid-Base Accounting

A portion of the pulverized head material for each individual sample was submitted to WETLAB for ABA testing. ABA is a static test to determine the acid producing or acid neutralizing potential of a material. It is a general analysis for the elements of acid generation and does not indicate the potential rate at which generation or neutralization may occur.

It is generally accepted that an NNP value greater than 20 is indicative of a non-acid producing material (acid neutralizing material), and that an NNP value less than -20 is an acid generating material. All of the 29 Black Pine metallurgical composites tested had NNP values >20 and are therefore considered to be non-acid producing.

13.6.3 Bottle-Roll and Column Leach Testing

Coarse and fine milled bottle-roll leach tests were completed on each of the 45 composites. A split of the head material, for each composite, was subjected to bottle-roll testing at target P_{80} sizes of 75 µm and 1,700 µm, and to column-leach testing at target P_{80}'s of 12.5 mm and 25.0 mm crush sizes (Table 13-14). A second series of CIL bottle-roll tests were conducted at the 75µm feed size to evaluate the potential for preg-borrowing clays and/or preg-robbing organic carbon. The main objective of these tests was to evaluate the laboratory-scale leachability character of the Black Pine resources in terms of gold extraction, extraction rate, reagent consumption, and sensitivity to feed size.

Table 13-14: Summary Black Pine Phase 2 Bottle-Roll and Column Leach Tests

Bottle-Rolls			Columns	
75 µm	75 µm (CIL)	1,700 µm	12.5 mm	25 mm
n = 45	n = 45	n = 45	n = 24	n = 21

The bottle-roll testing used standard procedures that are described in the final laboratory report (KCA-2021), using 144 hours of retention time for 1,700 µm tests, and 72 hours for 75 µm direct leach and CIL tests.

Column-leach tests were conducted utilizing material crushed to their target P_{80}'s and placed in columns of 10 cm and 15 cm diameters. During testing the material was leached between 82-99 days with a dilute NaCN solution. After leaching, each column was washed/rinsed for four days with water. A portion of the leached and washed material ("tailings") from each column was assayed for "tail screen" analyses by size fraction.

Tailings material from 39 columns was utilized for compacted permeability test work. Additionally, tailings material from 16 columns was submitted to WETLAB in Sparks, Nevada, for ABA and MWMT.

The following is a summary of the findings from the KCA (2021) report on bottle-roll and column test results.

13.6.4 Direct Leach and CIL Bottle-Roll Tests on 75 µm Composite Samples

Fine milled bottle-roll leach tests were completed on each of the 45 variability composites. The milled slurry was utilized for direct bottle-roll leach testing and CIL bottle-roll testing. The bottle-roll test procedures and results are described in detail in KCA 2021.

- The direct leach gold head grades for the composites ranged from 0.22 ppm to 4.95 ppm gold, with an average of 1.25 ppm. Gold extraction from this material ranged from 32.6% to 93.9%, with a weight average of 80.1%.
- The CIL leach gold head grades for the composites ranged from 0.24 ppm to 5.77 ppm gold, with an average of 1.32 ppm. Gold extraction from this material ranged from 59.3% to 94.0%, with a weight average of 85.4%.
- Cyanide consumption for the direct leach bottle-roll tests averaged 0.12 kg/t and lime consumption averaged 0.52 kg/t; and
- Cyanide consumption for the CIL bottle-roll tests averaged 0.74 kg/t and lime consumption averaged 0.51 kg/t.

13.6.5 Direct Leach Coarse Bottle-Roll Tests on 1,700 µm Composite Samples

Coarse bottle-roll leach tests were completed on each of the 45 composites. The coarse bottle-roll test procedure and results are described in detail in KCA 2021.

- Gold head grades for the composites ranged from 0.22 ppm to 5.51 ppm gold and averaged 1.24 ppm. Gold extraction ranged from 28.2% to 94.3%, with a weight average of 78.8%.

13.6.6 Column-Leach Tests on Composite Samples

All 45 composites were subjected to laboratory column-leach testing at KCA. Twenty-four columns were tested at a target P_{80} of 12.5 mm and 21 composites at a target P_{80} of 25 mm. Column test procedures are described in detail in KCA 2021. Column test gold and silver extractions are based upon loaded carbon assays and tails screen assays and are in Table 13-15.

- Calculated gold head grades ranged from 0.183 ppm to 5.86 ppm and averaged 1.24 ppm. Gold extractions ranged from 44.5% to 94.8%, with a weight average of 80.8%.
- Calculated silver head grades ranged from 0.32 ppm to 6.84 ppm and averaged 1.9 ppm. Silver extractions ranged from 3.5% to 71.1%, with a weight average of 26.6%.
- Cyanide consumptions ranged from 0.55 kg/t to 1.13 kg/t and averaged 0.78 kg/t. Based upon KCA's experience with clean non-reactive ores, cyanide consumption in commercial production heaps would range between 25% to 33% of the laboratory column test consumptions.
- Lime consumption ranged from 0.00 kg/t to 1.04 kg/t and averaged 0.91 kg/t. Four columns (BP190-36, BP214-51, BP-231-66, and BP247-71) were agglomerated with 4.0, 4.5, 5.9 and 2.0 kg/t cement respectively and did not require any lime.

Gold extraction plotted versus days under leach is shown graphically in Figure 13-8 and are based upon column solution balances.

Table 13-15: 2021 Variability Column Test Results

KCA Sample No.	Comp ID	Test No	Pilot Gold Geology		Feed Size					NaCN (g/l)	Leach Time (days)	Au Balance		Ag Balance		Reagents		
			Structure	F-Form	Target P80 (µm)	Screen P80 (µm)	% - 200M	Load Perm Tests	Cement kg/t			Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	NaCN kg/t	Lime kg/t	Cement kg/t
88826 A	BP190-30	88876	Msv	Pola	12,500	11,300	13.6	Pass		0.5	99d	**47.4**	0.211	**29.0**	2.86	1.11	1.01	
88827 A	BP190-31	88879	Mix	Pola	25,000	24,000	3.2	Pass		0.5	99d	**70.5**	0.400	**11.5**	1.88	0.66	0.84	
88828 A	BP190-32	88882	Bx,Msv	Pola	12,500	12,600	5.0	Pass		0.5	99d	**82.7**	0.660	**14.5**	1.79	0.89	1.00	
88829 A	BP190-33	88885	Mix	Polb	12,500	12,500	14.3	Pass		0.5	99d	**89.0**	0.463	**24.1**	2.12	1.09	1.01	
88830 A	BP190-34	88888	Msv,Bx	Polc	25,000	23,800	3.0	Pass		0.5	99d	**78.5**	0.763	**16.3**	1.47	0.71	1.01	
88831 A	BP190-35	90701	CaBx,Bx,Fz	Polc	25,000	23,900	5.4	Pass		0.5	99d	**80.9**	0.383	**32.4**	0.82	0.74	1.00	
88832 A	BP190-36	90704	Mix	Polc	12,500	10,900	16.2	Pass/close	4.0	0.5	98d	**87.0**	3.987	**38.3**	1.75	0.78	0.00	4.0
88833 A	BP197-37	90707	Mc,Msv	Pold	25,000	27,900	1.0	Pass		0.5	98d	**78.4**	1.308	**13.0**	1.15	0.59	1.00	
88834 A	BP197-38	90710	Mc,>Bx	Pold	25,000	24,500	1.8	Pass		0.5	98d	**76.7**	0.236	**8.6**	0.99	0.56	1.01	
88835 A	BP197-39	90713	Bx,Fz	Pold	25,000	22,500	8.1	Pass		0.5	98d	**85.3**	0.688	**14.0**	1.43	0.69	1.01	
88836 A	BP197-40	90716	Bx=Fz,Mc	Pols	12,500	13,700	8.2	Pass		0.5	98d	**88.9**	2.012	**10.2**	1.18	0.90	1.00	
88837 A	BP197-41	90719	Bx=Mc	Pols	25,000	24,200	2.3	Pass		0.5	98d	**72.7**	0.509	**5.3**	0.86	0.64	0.95	
88838 A	BP197-42	90722	Msv	Pols	12,500	11,800	5.8	Pass		0.5	98d	**86.5**	1.156	**11.4**	1.40	0.74	1.00	
88839 A	BP197-43	90725	Msv,Bx,Fol	Pols	25,000	22,600	6.6	Pass		0.5	98d	**76.1**	0.335	**31.3**	0.32	0.77	1.00	
88840 A	BP207-44	90728	Bx,CaBx	Pola	12,500	12,600	5.9	Pass		0.5	98d	**62.6**	0.246	**28.3**	1.59	0.80	1.04	
88841 A	BP207-45	90731	Fz,Bx	Polb	25,000	22,100	9.5	Pass		0.5	96d	**80.2**	1.350	**53.5**	5.68	1.15	1.01	
88842 A	BP207-46	90734	Fz,Msv	Polb	9,500	8,900	23.6	N/A		0.5	96d	**82.5**	0.733	**36.6**	2.02	1.01	1.00	
88843 A	BP207-47	90737	Msv	Polc	12,500	12,500	6.3	Pass		0.5	96d	**44.5**	2.768	**37.4**	2.14	0.82	1.00	
88844 A	BP207-48	90740	Msv	Polc	25,000	24,700	1.5	N/A		0.5	96d	**80.4**	4.750	**7.3**	2.75	0.69	1.00	
88845 A	BP207-49	90743	Msv	Polc	25,000	24,300	4.9	Pass		0.5	96d	**80.2**	1.469	**10.0**	1.40	0.70	1.00	
88846 A	BP214-50	90746	Msv,Bx	Pold	12,500	11,600	7.1	Pass		0.5	96d	**76.0**	0.283	**18.6**	1.72	0.86	1.00	
88847 A	BP214-51	90749	Fz	Polb	9,500	240	72.2	N/A	4.5	0.5	95d	**70.9**	0.323	**26.2**	4.35	1.13	0.00	4.5
88848 A	BP214-52	90752	Fz,Msv	Polb	25,000	23,500	15.6	Pass		0.5	95d	**67.1**	0.920	**29.2**	1.71	1.00	1.01	
88849 A	BP214-53	90755	Msv	Polb	12,500	12,000	10.3	Pass		0.5	95d	**66.8**	1.251	**61.4**	0.78	0.86	1.00	
88850 A	BP214-54	90758	Bx,Fz,Msv	Polc	12,500	11,100	23.2	Pass		0.5	95d	**92.2**	5.856	**33.7**	3.01	0.74	1.01	
88851 A	BP214-55	90761	Fz,Bx	Polc	12,500	11,300	23.0	Fail 100		0.5	95d	**93.2**	3.046	**71.1**	6.84	0.97	1.02	
88852 A	BP214-56	90764	Msv,Bx	Polc	12,500	12,500	8.7	Pass		0.5	95d	**76.1**	2.222	**49.5**	2.89	0.74	0.99	
88853 A	BP222-57	90767	Bx,Fz	Pola	12,500	12,100	11.0	Pass		0.5	95d	**78.1**	0.424	**24.1**	0.83	0.63	1.00	
88854 A	BP222-58	90770	Msv,Mc	Polb	25,000	25,300	1.7	Pass		0.5	95d	**51.3**	0.277	**25.9**	0.54	0.65	1.00	
88855 A	BP222-59	90773	Msv,Bx	Polc	25,000	25,100	2.4	Pass		0.5	95d	**72.8**	0.268	**13.1**	1.37	0.67	1.01	
88856 A	BP222-60	90776	Mc,Msv,FZ	Pold	12,500	11,200	12.2	Pass		0.5	93d	**91.7**	0.663	**14.5**	1.66	0.63	1.00	
88857 A	BP222-61	90779	Fz=Bx=Mc	Pols	12,500	12,000	10.5	Pass		0.5	93d	**83.1**	3.484	**17.3**	3.98	0.65	1.00	
88858 A	BP222-62	90782	Mix	Pols	25,000	23,400	3.2	Pass		0.5	93d	**78.2**	1.820	**11.3**	1.77	0.65	1.00	

Source: GL Simmons Consulting LLC, 2023

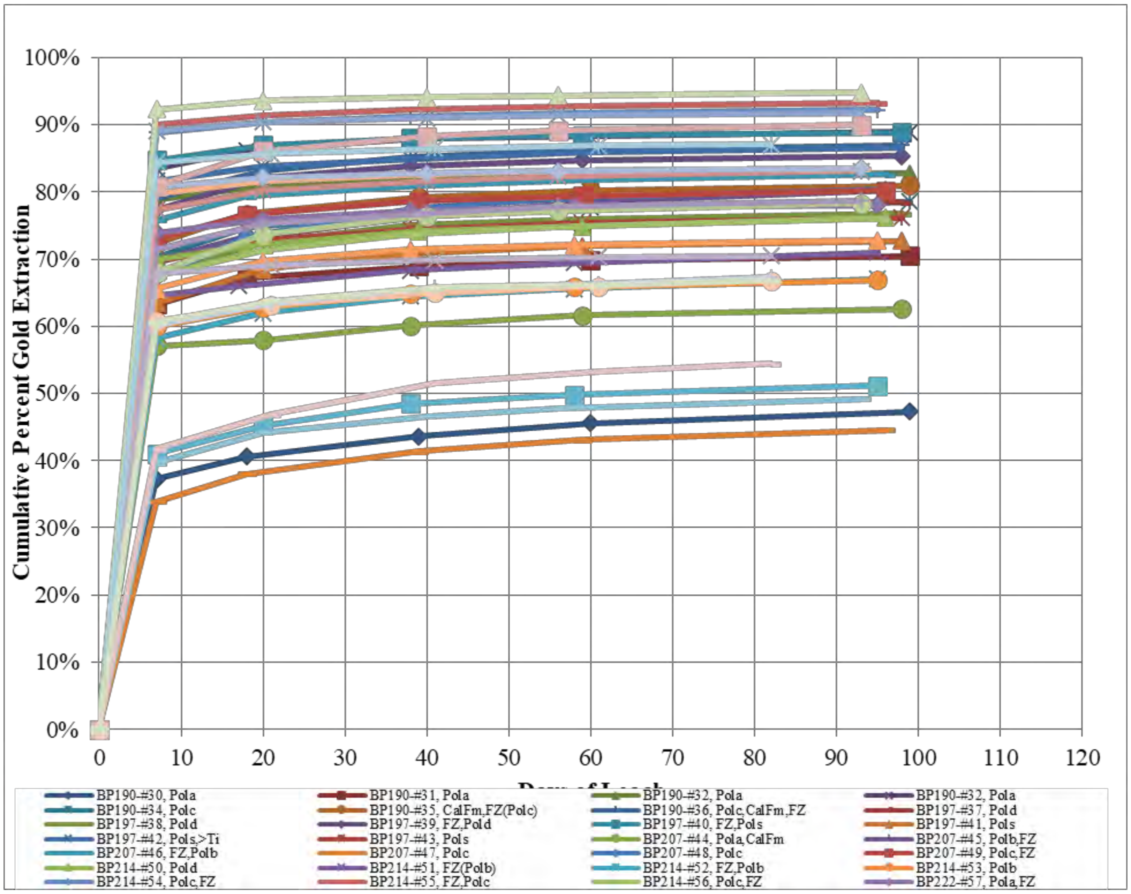


Figure 13-8: 2021 Gold Extraction vs. Days Under Leach for Column-Leach Tests

13.6.7 Comminution Characterization at Hazen

Portions of the head material for nine composite samples were stage crushed to 100% passing 37.5 mm and submitted to Hazen Research, Inc. in Golden, Colorado for SMC and Ai testing. Details of the comminution testing procedures and test results are reported in Stepperud (2021).

A summary of the Ai test work is presented in Table 13-16 and the SMC comminution characterization in Table 13-17.

Table 13-16: 2021 Bond Abrasion Test Results

HRI No.	Client ID	Composite ID	Ai (grams)
55234-12	88827 A	BP190-31	0.0167
55234-13	88831 A	BP190-35	0.0325
55234-14	88833 A	BP197-37	0.0617
55234-15	88837 A	BP197-41	0.0778
55234-16	88841 A	BP207-45	0.0507
55234-17	88845 A	BP207-49	0.0055
55234-18	88848 A	BP214-52	0.0045
55234-19	88859 A	BP222-63	0.0401
55234-20	88861 A	BP231-65	0.0220

Table 13-17: SMC Comminution Characterization Test Results

HRI No.	Client ID	Composite ID	Black Pine Project - Comminution Characterization										
			sg	A	B	A x b	Dwi kWh/m³	DWi %	M_{ia} kWh/t	M_{ih} kWh/t	M_{ic} kWh/t	t_a	SCSE kWh/t
55234-12	88827 A	BP190-31	2.55	57.8	0.86	49.7	5.13	30	16.6	11.7	6	0.5	8.81
55234-13	88831 A	BP190-35	2.63	60.3	0.73	44	5.96	41	18.2	13.2	6.8	0.43	9.35
55234-14	88833 A	BP197-37	2.71	72.7	0.43	31.3	8.61	75	23.7	18.4	9.5	0.3	11.12
55234-15	88837 A	BP197-41	2.63	62	0.68	42.2	6.22	44	18.8	13.7	7.1	0.42	9.53
55234-16	88841 A	BP207-45	2.57	57	1.35	77	3.32	13	11.7	7.5	3.9	0.78	7.46
55234-17	88845 A	BP207-49	2.57	58.8	0.67	39.4	6.54	49	20.1	14.8	7.6	0.4	9.75
55234-18	88848 A	BP214-52	2.47	55	1.58	86.9	2.84	9	10.7	6.7	3.5	0.91	7.18
55234-19	88859 A	BP222-63	2.63	54.6	1.29	70.4	3.72	16	12.5	8.2	4.2	0.69	7.72
55234-20	88861 A	BP231-65	2.64	59.6	1.31	78.1	3.37	13	11.5	7.4	3.8	0.77	7.44

A = maximum breakage
b = relation between energy and impact breakage
A x b = overall AG SAG hardness
DWi = drp (-weight index)
Mia = coarse particle component
Mic = crusher component
Mih = high-pressure grinding roll component
SCSE = SAG circuit specific energy
sg = specific gravity of sample
ta = low energy abrasion component of breakage

13.6.8 Load Permeability Test Work on Column Tailings

A portion of tailings material from forty (40) column leach residues were utilized for load permeability test work. The purpose of the load permeability test work was to examine the permeability of the crushed material under compaction loading equivalent to heap heights of 25 m, 50 m, 75 m, and 100 m.

All of the 40 column leach residues passed load permeability testing up to 100 meters height, except for one (1) composite (BP214-#55, FZ,Polc), which failed at 100 meters. This was a marked improvement over Phase 1 as Phase 2 met core moved out of the Discovery Zone 1 and 2 area where Pola and Polc high-clay zones are located.

Refer to Section 13.4.8 in this report for a description of the Load Permeability test apparatus and testing procedures and (KCA 2021a) report for complete details.

13.7 2022 Phase 3 Low-Grade Composite Variability Testing

In October 2021, Liberty Gold delivered fifteen (15) Black Pine low-grade variability composites to the laboratory facility of KCA in Reno, Nevada. The received material represented fifteen (15) individual composites. The received samples were utilized for metallurgical test work.

All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. The same as what is described in the original Phase 1 (KCA 2020a) and Phase 2 (KCA 2020b) final reports.

The low-grade variability composites were retrieved from existing Black Pine Phase1 and Phase 2 core that was not tested in the original programs.

Splits of all 15 composite heads were delivered to three separate laboratories for additional geo-metallurgical and environmental characterization analysis:

1. ALS for ICP and gold cyanide solubility analysis,
2. FLS for “XRD” and “Whole-Rock” analysis and
3. WETLAB for environmental characterization of solids and aqueous solutions.

The Phase 3 program was designed to test the lower-end (mining cut-off gold grade range) resources to better define the lower end of the head-grade vs. gold recovery models.

13.7.1 2022 Black Pine Low-Grade Variability Composite Head Assays

Head assay details and geo-metallurgical characterization results are in the KCA 2022 report. A high-level summary of the geo-metallurgical characterization is presented below for gold, silver, copper, gold cyanide solubility, carbon and sulfur species, preg-robbing analysis, as well as ICP multi-element analyses, whole-rock analyses and QXRD analyses. Select composite summary results for gold, silver, copper, carbon and sulfur speciation and preg-robbing analysis, are detailed in Table 13-18:

- Gold grades ranged from 0.122 ppm to 0.169 ppm and averaged 0.146 ppm.
- Silver grades ranged from 0.13 ppm to 1.69 ppm and averaged 1.07 ppm.
- Organic carbon ranged from 0.12% to 0.84 % and averaged 0.28%.
- Sulfide sulfur showed no variation and averaged <0.01%.
- Preg-robbing analyses ranged from 0.0% to 23.7% and averaged 3.9% (using a 1 ppm spike). Preg-robbing values <10% are considered within the error band of the test procedure and are classified non-preg-robbing by KCA. Only one composite (BP067-5) was >10%.
- Copper values were very low, ranging from 14 ppm to 68 ppm and averaged 35 ppm.
- Gold cyanide solubility ranged from 47.3% to 100.0% and averaged 77.4%.
- Concentrations of the deleterious elements were as follows: selenium averaged 19 ppm; mercury ranged from 4.9 ppm to 11.5 ppm with an average 6.7 ppm; and arsenic levels were low, ranging from 27 ppm to 112 ppm with an average of 59 ppm.
- Concentrations of the primary cyanide consumers were low and suggest minimum potential for affecting cyanide consumption rates. Copper averaged 35 ppm, nickel averaged 72 ppm, and zinc averaged 177 ppm.
- Whole-rock quartz content ranged from 22.9% to 71.8% and averaged 38.5%.

Table 13-18: 2022 Black Pine Low Grade Variability Composite Head Assays by KCA and ALS

		Head Assays																	
		ALS			KCA														
KCA Sample No.	Composite ID	AuFA ppm	AuCN ppm	AuCN %	AgFA ppm	AgCN ppm	AgCN %	Cu ppm	CuCN ppm	CuCN %	C(tot) %	C(org) %	C(inorg) %	$S_{(total)}$ %	$S_{(sulfide)}$ %	$S_{(sulfate)}$ %	Au-w/spike	Au-w/o spike	Au PR %
93001 A	BP067-75	0.169	0.080	47.3	1.79	1.26	70.2	68	15.06	22.1	4.62	0.27	4.35	0.03	<0.01	0.03	0.76	0.02	23.7
93002 A	BP067-76	0.159	0.100	62.9	0.69	0.50	72.3	43	6.04	14.0	7.70	0.17	7.53	0.02	<0.01	0.02	0.92	0.02	7.2
93003 A	BP067-77	0.136	0.100	73.5	1.96	1.32	67.3	44	15.28	34.7	9.14	0.23	8.91	0.05	<0.01	0.05	0.90	0.03	10.3
93004 A	BP067--78	0.143	0.110	76.9	0.40	0.44	110.0	36	5.16	14.3	6.74	0.14	6.60	0.03	<0.01	0.03	1.02	0.04	0.0
93005 A	BP082-79	0.160	0.090	56.3	2.69	0.76	28.3	57	6.66	11.7	0.87	0.84	0.03	0.02	<0.01	0.02	1.01	0.04	0.0
93006 A	BP087-80	0.122	0.100	82.0	1.22	0.44	36.0	52	8.22	15.8	5.32	0.81	4.51	0.03	<0.01	0.03	0.95	0.03	5.2
93007 A	BP093-81	0.153	0.150	98.0	0.26	0.28	106.5	26	4.92	18.9	2.59	0.12	2.47	0.07	<0.01	0.07	1.06	0.06	0.0
93008 A	BP190-82	0.131	0.090	68.7	0.29	0.42	147.0	24	3.78	15.8	6.70	0.19	6.51	0.31	<0.01	0.31	0.98	0.03	2.1
93009 A	BP197-83	0.139	0.150	100.0	0.13	0.14	111.4	14	2.28	16.3	8.37	0.17	8.20	0.03	<0.01	0.03	1.05	0.06	0.0
93010 A	BP197-84	0.142	0.110	77.5	0.41	0.32	78.9	29	4.24	14.6	7.00	0.13	6.87	0.04	<0.01	0.04	1.08	0.05	0.0
93011 A	BP197-85	0.127	0.120	94.5	0.49	0.24	48.8	17	3.12	18.4	7.02	0.29	6.73	0.01	<0.01	0.01	0.98	0.03	2.1
93012 A	BP207-86	0.162	0.120	74.1	1.89	0.86	45.6	25	5.38	21.5	7.16	0.10	7.06	0.08	<0.01	0.08	0.99	0.04	2.1
93013 A	BP214-87	0.137	0.100	73.0	1.91	1.86	97.5	26	6.08	23.4	7.72	0.25	7.47	0.78	<0.01	0.78	1.08	0.04	0.0
93014 A	BP207-88	0.163	0.130	79.8	0.93	0.56	60.5	22	5.38	24.5	7.89	0.21	7.68	0.08	<0.01	0.08	1.05	0.05	0.0
93015 A	BP222-89	0.145	0.140	96.6	1.00	0.32	32.0	35	3.88	11.1	5.51	0.25	5.26	0.02	<0.01	0.02	0.94	0.03	6.2

Source: Gary Simmons Consulting LLC, 2023

13.7.2 Bottle-Roll and Column Leach Testing

Coarse and fine milled bottle-roll leach tests were completed on each of the 15 composites. A split of the head material was subjected to direct leach ("DL") and CIL bottle-roll testing at target P_{80} sizes of 75 µm and 1,700 µm, and to column-leach testing at target P_{80}'s of 12.5 mm or 25.0 mm crush sizes. Because of the very low-grade gold in these composites and the inherent low-level preg-borrowing clays present in some of the Black Pine materials, CIL bottle-roll tests were conducted at the 75 µm and 1,700 µm feed size. The main objective of these tests was to evaluate the laboratory-scale leachability character of the Black Pine low-grade resources in terms of gold extraction, extraction rate, reagent consumption, and sensitivity to feed size.

The bottle-roll testing used standard procedures that are described in the final laboratory report (KCA 2022), using 144 hours of retention time for 1,700 µm tests and 72 hours for 75 µm tests.

Column-leach tests were conducted utilizing material crushed to their respective target P_{80}'s and placed in columns of 10 cm and 15 cm diameters. During testing the material was leached for 97 days with a dilute NaCN solution. After leaching, each column was washed/rinsed for four days with water. A portion of the leached and washed material ("tailings") from each column was assayed for "tail screen" analyses by size fraction.

Tailings material from ten of the fifteen composites was utilized for compacted permeability test work.

The following is a summary of the findings from the KCA 2022 report on bottle-roll and column test results.

13.7.3 Low-Grade Direct Leach and CIL Bottle-Roll Tests on 75 µm Composite Samples

Fine milled bottle-roll leach tests were completed on each of the 15 variability composites. The milled slurry was utilized for direct bottle-roll leach testing and CIL bottle-roll testing. The bottle-roll test procedures and results are described in detail in KCA 2022.

- The direct leach gold head grades for the composites ranged from 0.12 ppm to 0.21 ppm gold, with an average of 0.16 ppm. Gold extraction from this material ranged from 36.5% to 77.8%, with a weight average of 55.4%.
- The CIL leach gold head grades for the composites ranged from 0.13 ppm to 0.20 ppm gold, with an average of 0.16 ppm. Gold extraction from this material ranged from 50.1% to 78.2%, with a weight average of 66.9%.
- Cyanide consumption for the direct leach bottle-roll tests averaged 0.20 kg/t and lime consumption averaged 0.88 kg/t.
- Cyanide consumption for the CIL bottle-roll tests averaged 1.05 kg/t and lime consumption averaged 0.60 kg/t.

13.7.4 Low-Grade Direct Leach and CIL Coarse Bottle-Roll Tests on 1,700 µm Composite Samples

Coarse bottle-roll leach tests were completed on each of the 15 composites. The coarse bottle-roll test procedure and results are described in detail in KCA 2022.

- The direct leach gold head grades for the composites ranged from 0.10 ppm to 0.20 ppm gold and averaged 0.15 ppm. Gold extraction ranged from 41.9% to 73.6%, with a weight average of 57.9%.
- The CIL leach gold head grades for the composites ranged from 0.10 ppm to 0.18 ppm gold, with an average of 0.15 ppm. Gold extraction ranged from 51.9% to 76.8%, with a weight average of 65.2%.
- Cyanide consumption for the direct leach bottle-roll tests averaged 0.13 kg/t and lime consumption averaged 1.10 kg/t.
- Cyanide consumption for the CIL bottle-roll tests averaged 1.63 kg/t and lime consumption averaged 0.53 kg/t.

13.7.5 Low-Grade Column-Leach Tests on Composite Samples

All 15 composites were subjected to laboratory column-leach testing at KCA. Six columns were tested at a target P_{80} of 12.5 mm and nine composites at a target P_{80} of 25 mm. Column test procedures are described in detail in KCA 2022. Column test gold and silver extractions are based upon loaded carbon assays and tails screen assays and in Table 13-19.

- Calculated gold head grades ranged from 0.119 ppm to 0.183 ppm and averaged 0.149 ppm. Gold extractions ranged from 50.8% to 80.3%, with a weight average of 65.2%.
- Calculated silver head grades ranged from 0.42 ppm to 2.58 ppm and averaged 1.35 ppm. Silver extractions ranged from 6.4% to 55.0%, with a weight average of 25.9%.
- Cyanide consumptions ranged from 0.36 kg/t to 0.67 kg/t and averaged 0.44 kg/t. Based upon KCA's experience with clean non-reactive ores, cyanide consumption in commercial production heaps would range between 25% to 33% of the laboratory column test consumptions.
- Lime consumption ranged from 0.00 kg/t to 1.02 kg/t and averaged 0.87 kg/t. Two columns (BP067-75 and BP093-81) were agglomerated with 2.0 kg/t and 4.0 kg/t cement respectively and did not require any lime.

Gold extraction plotted versus days under leach is shown graphically in Figure 13-9 and are based upon column solution balances.

Table 13-19: 2022 Low-Grade Variability Column Test Results

KCA Sample No.	Comp ID	Test No	Pilot Gold Geology		Feed Size					NaCN (g/l)	Leach Time (days)	Au Balance		Ag Balance		Reagents		
			Structure	F-Form	Target P80 (µm)	Screen P80 (µm)	% - 200M	Load Perm Tests	Cement kg/t			Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	NaCN kg/t	Lime kg/t	Cement kg/t
2022 Phase 3 - LG Variability Core Composites																		
93001 A	BP067-75	93026	Bx,FZ	Pola,FZ	12,500	12,100	21.3	N/A	2.0	0.5	97d	**50.8**	0.183	**45.9**	1.830	0.67	0.00	2.0
93002 A	BP067-76	93029		Polc	12,500	14,200	3.1	N/A		0.5	97d	**51.7**	0.172	**25.7**	1.010	0.39	1.02	
93003 A	BP067-77	93032		Pold	25,000	24,800	1.8	Pass		0.5	97d	**77.3**	0.132	**39.1**	1.690	0.40	1.00	
93004 A	BP067--78	93035	Bx,	Pola	12,500	14,000	9.5	N/A		0.5	97d	**53.2**	0.139	**30.6**	0.850	0.48	1.00	
93005 A	BP082-79	93038	Fz	FZ,Polb	25,000	3,300	42.3	Fail 25m		0.5	97d	**65.5**	0.177	**23.4**	2.560	0.60	1.02	
93006 A	BP087-80	93041	Bx,Fz	Polb,FZ	25,000	24,600	10.2	Pass		0.5	97d	**69.8**	0.139	**12.3**	2.030	0.57	1.00	
93007 A	BP093-81	93044	Fz,	Ppos,FZ	25,000	22,100	21.0	Pass	4.0	0.5	97d	**79.0**	0.167	**21.9**	0.730	0.36	0.00	4.0
93008 A	BP190-82	93047	Msv,Fz	Polc,FZ	25,000	28,500	7.5	Pass		0.5	97d	**57.5**	0.127	**20.8**	1.010	0.36	1.00	
93009 A	BP197-83	3050	Mc,Fol	Pold	25,000	29,300	1.6	Pass		0.5	97d	**72.0**	0.164	**11.9**	0.420	0.36	1.01	
93010 A	BP197-84	93053	Bx,Mc,Msv	Pols.FZ	25,000	24,700	4.5	Pass		0.5	97d	**52.3**	0.149	**9.4**	0.640	0.39	1.01	
93011 A	BP197-85	93056	Fz>Bx	Pola	12,500	14,000	4.4	N/A		0.5	97d	**68.9**	0.119	**16.3**	0.800	0.39	1.00	
93012 A	BP207-86	93059	Bx	Pola	25,000	24,600	2.8	Pass		0.5	97d	**65.6**	0.151	**14.7**	1.900	0.42	1.01	
93013 A	BP214-87	93062	FZ/Bx	Polb,FZ	12,500	12,800	3.0	N/A		0.5	97d	**80.3**	0.132	**55.0**	2.090	0.40	1.02	
93014 A	BP207-88	93065	Bx,CaBx	Pold,CalFn	25,000	24,700	2.7	Pass		0.5	97d	**73.6**	0.148	**15.7**	1.530	0.37	1.01	
93015 A	BP222-89	93068	Msv	Polb	25,000	28,200	3.2	Pass		0.5	97d	**65.4**	0.133	**6.4**	1.100	0.44	1.01	

A somewhat unique characteristic of the Black Pine lower grade heap leach resources is that they commonly demonstrate clay and/or smoky carbon (very slight residual oxidized organic carbon or residual organic carbon in silica or carbonate rocks, sometimes detectable via a change in hue/color, but not detectable via organic carbon assay) preg-borrowing characteristics. The CIL bottle-roll tests commonly have higher gold extractions than their corresponding direct leach bottle roll tests, a normal characteristic of preg-robbing. The column leach test gold extractions can be higher than the direct leach bottle-roll tests and the column leach profiles demonstrate a longer duration tail, demonstrating that the preg-borrowed gold is given back over time as the leach solution gold grade drops.

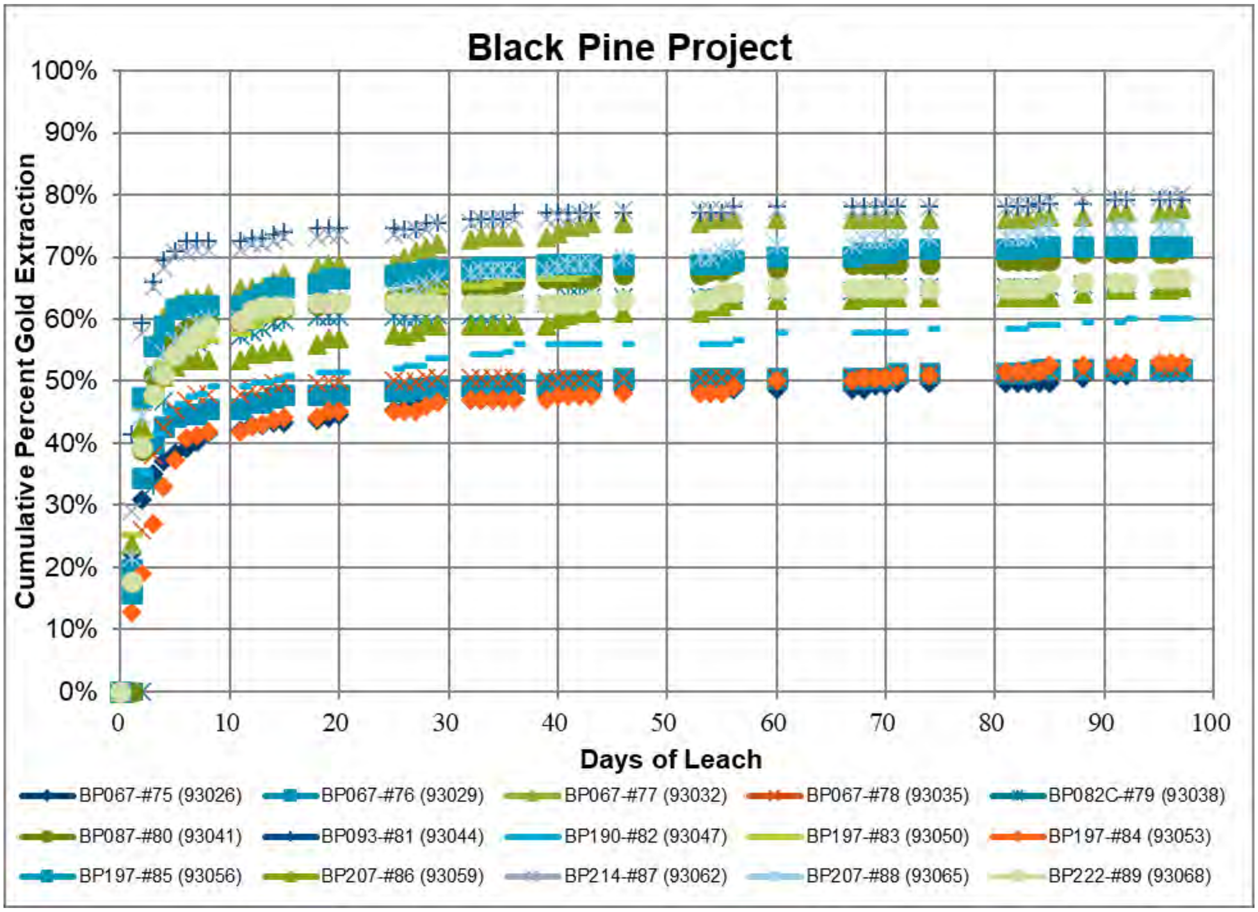


Figure 13-9: 2022 Low-Grade Variability Composites Gold Extraction vs. Days Under Leach for Column-Leach Tests

Ten (10) of the 15 column leach residues were utilized for load permeability test work. The purpose of the load permeability test work was to examine the permeability of the crushed material under compaction loading equivalent to heap heights of 25 m, 50 m, 75 m, and 100 m.

All ten column leach residues passed load permeability testing up to 100 m height, except for one composite (BP082-79), which failed at 25 m.

Refer to KCA (2022) report for detail on the Load Permeability test apparatus and testing procedures.

13.8 2023 Phase 4A, 4B, and 4C Variability Composite Testing

In October 2022-23, Liberty Gold delivered sixty (60) Black Pine variability composite samples, in three batches, to the laboratory facility of KCA in Reno, Nevada.

All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. The same as what is described in the original Phase 1 (KCA 2020a) and Phase 2 (KCA 2020b) final reports.

The variability composites were retrieved from Black Pine large diameter PQ metallurgical core drilling conducted in 2022.

Splits of all 60 composite heads were delivered to three separate laboratories for additional geo-metallurgical and environmental characterization analysis:

1. ALS for ICP and gold cyanide solubility analysis,
2. FLS for “XRD” and “Whole Rock” analysis and
3. WETLAB for environmental characterization of solids and aqueous solutions.

The Phase 4A, 4B and 4C three programs were designed to test the new Rangefront resource area and to fill gaps in the current resources. A brief description of the three sets of samples is provided here:

- Phase 4A (24 comps) – Test the new Rangefront deposit resources.
- Phase 4B (25 comps)– In-fill gaps in Discovery Zone (“DZ”), M-Zone, I-Pit, Tallman-Pit, E-Pit, and F Zone resource areas.
- Phase 4C – In-fill gaps in the C/D Pit and F-Zone resources.

13.8.1 2023 Black Pine Phase 4A, 4B, and 4C Variability Composite Head Assays

Head assay details, geo-metallurgical characterization and test results are in the following KCA reports:

- KCA 2023a report – Black Pine Project, Rangefront Deposit, Report of Metallurgical Test Work, May 2023.
- KCA 2023b report – Black Pine Project, Phase 4B, Report of Metallurgical Test Work, October 2023.
- KCA 2024a report – Black Pine Project, 4C Metallurgical Composites, Report of Metallurgical Test Work, January 2024.

A high-level summary of the geo-metallurgical characterization is presented below for gold, silver, copper, gold cyanide solubility, carbon and sulfur species, preg-robbing analysis, as well as ICP multi-element analyses, whole-rock analyses and QXRD analyses. Select composite summary results for gold, silver, copper, carbon and sulfur speciation and preg-rob analysis, are detailed in Table 13-20:

- Gold grades ranged from 0.124 ppm to 7.36 ppm and averaged 0.606 ppm.
- Silver grades ranged from 0.72 ppm to 40.8 ppm and averaged 2.45 ppm.
- Organic carbon ranged from 0.07% to 0.42 % and averaged 0.14%.
- Sulfide sulfur was very low, showing little variation with 50 of the 60 composites reporting <0.01% S=.
- Organic carbon preg-robbing analyses ranged from 0.0% to 5.9% and averaged 0.6% (using a 1 ppm spike). Preg-robbing values <5-10% are considered non-preg-robbing.
- Copper values were very low, ranging from 5 ppm to 179 ppm and averaged 25 ppm.
- Gold cyanide solubility ranged from 44.4% to 100.0% and averaged 86.0%.

- Concentrations of the deleterious elements were low: selenium averaged 15 ppm; mercury ranged from 2.0 ppm to 109.0 ppm with an average of 10.6 ppm and arsenic levels ranging from <2 ppm to 2078 ppm with an average of 90 ppm.
- Concentrations of the primary cyanide consumers were low and suggest minimum potential for affecting cyanide consumption rates. Copper averaged 25 ppm, nickel averaged 32 ppm, and zinc averaged 201 ppm.
- Whole-rock quartz content ranged from 13.9% to 76.7.8% and averaged 38.6%.

Table 13-20: 2023 Black Pine Phase 4A, 4B and 4C Variability Composite Head Assays by KCA and ALS

KCA Sample No.	Composite ID	ALS			KCA														
		AuFA ppm	AuCN ppm	AuCN %	AgFA ppm	AgCN ppm	AgCN %	Cu Ppm	DuCN ppm	CuCN %	C(tot) %	C(org) %	C(inorg) %	S(tot) %	S(Sulfide) %	S(SO$_4$) %	Au w/spike	Au-w/o spike	Au PR %
Phase 4A - Rangefront																			
95101 A	BP533-90	0.237	0.250	100.0	0.98	0.06	6.1	6	1.02	17.0	9.14	0.11	9.03	0.08	<0.01	0.08	1.15	0.11	0.0
95102 A	BP533-91	0.682	0.770	100.0	1.37	0.30	21.9	17	3.68	21.6	5.99	0.14	5.85	0.04	<0.01	0.04	1.38	0.35	0.0
95103 A	BP511CA-92	0.363	0.330	90.9	2.47	0.58	23.5	58	3.76	6.5	2.14	0.07	2.07	0.02	<0.01	0.02	1.15	0.13	-2.0
95104 A	BP511CA-93	0.159	0.140	88.1	0.87	0.16	18.4	19	1.46	7.7	4.29	0.16	4.13	0.03	<0.01	0.03	1.05	0.01	0.0
95105 A	BP511CA-94	0.124	0.120	96.8	0.73	0.20	27.3	12	1.60	13.3	3.20	0.07	3.13	0.01	<0.01	0.01	1.04	0.01	0.0
95106 A	BP511CA-95	0.197	0.200	100.0	0.89	0.20	22.4	21	1.34	6.4	7.08	0.08	7.00	0.04	0.02	0.02	1.03	0.04	1.0
95107 A	BP511CA-96	1.080	1.260	100.0	1.13	0.30	26.5	15	2.48	16.5	6.47	0.08	6.39	0.12	0.02	0.10	1.55	0.55	0.0
95108 A	BP511CA-97	1.635	1.570	96.0	3.67	1.02	27.8	38	4.06	10.7	4.29	0.10	4.19	0.11	<0.01	0.11	1.71	0.70	0.0
95109 A	BP511CA-98	0.900	0.990	100.0	1.34	0.30	22.4	17	1.80	10.6	5.82	0.29	5.53	0.17	0.02	0.15	1.51	0.44	0.0
95110 A	BP511CA-99	0.235	0.250	100.0	1.06	0.30	28.2	9	3.62	40.2	8.11	0.09	8.02	0.02	<0.01	0.02	1.12	0.07	0.0
95111 A	BP541-100	0.251	0.230	91.6	0.93	0.34	36.7	16	4.06	25.4	4.58	0.19	4.39	0.03	<0.01	0.02	1.09	0.06	2.8
95112 A	BP541-101	0.469	0.480	100.0	1.20	0.42	35.0	14	3.54	25.3	1.26	0.11	1.15	0.02	<0.01	0.02	1.19	0.17	3.8
95113 A	BP541-102	0.148	0.160	100.0	1.06	0.34	32.0	20	6.94	34.7	1.95	0.16	1.79	0.03	0.01	0.02	1.04	0.02	3.8
95114 A	BP541-103	0.226	0.230	100.0	0.73	0.18	24.6	8	1.72	21.5	10.10	0.42	9.68	0.11	0.05	0.06	1.09	0.07	3.8
95115 A	BP541-104	0.234	0.210	89.7	0.74	0.74	99.6	6	2.66	44.3	6.98	0.13	6.85	0.02	<0.01	0.02	1.11	0.08	2.8
95116 A	BP541-105	0.242	0.220	90.9	1.44	0.24	16.7	12	2.14	17.8	6.87	0.21	6.67	0.01	<0.01	0.01	1.10	0.05	0.0
95117 A	BP541-106	0.562	0.530	94.3	1.14	0.24	21.1	8	6.82	85.3	5.62	0.09	5.53	0.05	<0.01	0.05	1.20	0.20	2.0
95118 A	BP541-107	0.206	0.220	100.0	1.06	0.12	11.3	8	1.72	21.5	7.29	0.10	7.19	0.01	<0.01	0.01	1.06	0.03	0.0
95119 A	BP556-108	0.147	0.150	100.0	1.27	0.68	53.6	6	3.24	54.0	5.66	0.14	5.52	0.02	<0.01	0.02	1.01	0.01	2.0
95120 A	BP556-109	0.307	0.270	87.9	1.60	0.48	30.0	12	2.58	21.5	6.97	0.14	6.83	0.02	<0.01	0.01	1.03	0.07	5.9
95121 A	BP556-#110	0.137	0.130	94.9	0.73	0.25	34.2	14	1.81	13.1	7.45	0.19	7.26	0.02	<0.01	0.02	1.03	0.01	0.0
95122 A	BP556-111	1.035	0.980	94.7	3.60	1.07	29.7	16	2.81	17.3	6.19	0.14	6.05	0.15	<0.01	0.15	1.47	0.42	0.0
95123 A	BP556-112	0.749	0.730	97.5	5.50	2.94	53.4	30	6.88	22.9	5.14	0.10	5.04	0.25	<0.01	0.25	1.37	0.34	0.0
95124 A	BP556-113	0.258	0.230	89.1	2.07	1.08	52.1	19	5.83	31.4	9.37	0.08	9.29	0.10	0.06	0.04	1.16	0.08	0.0
Phase 4B – DZ, M-Zone, I-Pit, Tallman-Pit, E-Pit & F-Zone																			
96001 A	BP429CA-114	0.191	0.160	83.8	2.37	0.68	28.7	67	5.40	8.1	3.89	0.24	3.65	0.03	<0.01	0.03	1.08	0.09	1.0
96002 A	BP429CA-115	0.297	0.250	84.2	1.84	0.38	20.7	24	1.76	7.3	4.23	0.14	4.09	0.02	<0.01	0.02	1.12	0.12	0.0
96003 A	BP429CA-116	0.445	0.380	85.4	3.07	1.06	34.5	26	2.58	9.9	3.73	0.24	3.49	0.01	<0.01	0.01	1.19	0.19	0.0
96004 A	BP429CA-117	0.256	0.240	93.8	2.43	0.76	31.2	67	6.64	9.9	5.23	0.22	5.01	0.01	<0.01	0.01	1.09	0.09	0.0
96005 A	BP429CA-118	0.293	0.130	44.4	2.47	0.56	22.7	21	2.26	10.8	5.06	0.27	4.79	0.03	<0.01	0.03	1.09	0.09	0.0
96006 A	BP429CA-119	0.268	0.170	63.4	3.20	1.78	55.6	23	8.16	35.5	7.38	0.20	7.18	0.04	<0.01	0.04	1.07	0.08	1.0
96007 A	BP429CA-120	1.850	1.580	85.4	3.01	1.16	38.5	25	5.90	23.6	6.25	0.15	6.10	0.35	<0.01	0.35	1.73	0.75	2.0
96008 A	BP429CA-121	0.186	0.140	75.3	2.33	1.16	49.8	29	4.54	15.7	3.11	0.18	2.93	0.07	0.01	0.06	1.05	0.06	1.0
96009 A	BP485-122	0.165	0.140	84.8	1.94	0.64	32.9	15	2.82	18.8	6.08	0.19	5.89	0.02	<0.01	0.02	1.05	0.05	0.0
96010 A	BP485-123	0.347	0.280	80.7	1.81	0.46	25.5	23	3.70	16.1	3.48	0.19	3.29	0.04	<0.01	0.04	1.13	0.13	0.0
96011 A	BP485-124	0.268	0.190	70.9	2.10	0.45	21.2	24	3.38	14.0	4.32	0.16	4.16	0.02	<0.01	0.02	1.03	0.03	0.0
96012 A	BP508-125	0.348	0.290	83.3	1.67	0.27	16.3	11	1.64	15.6	10.10	0.11	9.99	0.06	<0.01	0.06	1.04	0.04	0.0
96013 A	BP508-126	0.168	0.150	89.3	1.47	0.31	21.1	8	2.82	34.1	8.81	0.14	8.67	0.01	<0.01	0.01	1.02	0.02	0.0
96014 A	BP508-127	0.841	0.700	83.2	2.67	1.05	39.5	19	2.66	14.1	3.65	0.15	3.50	0.03	<0.01	0.03	1.28	0.28	0.0
96015 A	BP508-128	7.360	7.180	97.6	2.93	0.89	30.3	18	1.46	8.3	5.48	0.22	5.26	0.02	<0.01	0.02	4.53	3.53	0.0
96016 A	BP499-129	0.525	0.380	72.4	7.40	4.83	65.2	179	120.6	67.5	8.57	0.20	8.37	0.08	<0.01	0.08	1.13	0.14	1.0
96017 A	BP499-130	0.265	0.160	60.4	1.73	0.48	27.5	25	6.54	26.2	6.25	0.13	6.12	0.03	<0.01	0.03	1.03	0.03	0.0
96018 A	BP499-131	4.170	3.870	92.8	6.87	4.95	72.1	47	12.9	27.3	2.92	0.16	2.76	1.34	<0.01	1.34	2.86	1.86	0.0
96019 A	BP499-132	0.139	0.080	57.6	40.8	6.69	16.4	172	87.2	50.7	8.26	0.15	8.11	0.04	<0.01	0.04	1.02	0.06	4.0
96020 A	BP573-133	0.339	0.300	88.5	2.53	1.27	50.1	24	14.8	60.8	6.53	0.12	6.41	0.02	<0.01	0.02	1.16	0.16	0.0
96021 A	BP525-134	0.528	0.420	79.5	1.17	0.10	8.7	22	15.72	72.2	9.09	0.12	8.97	0.01	<0.01	0.01	1.19	0.19	0.0

KCA Sample No.	Composite ID	ALS			KCA														
		AuFA ppm	AuCN ppm	AuCN %	AgFA ppm	AgCN ppm	AgCN %	Cu Ppm	DuCN ppm	CuCN %	C(tot) %	C(org) %	C(inorg) %	S(tot) %	S(Sulfide) %	S(SO_4) %	Au w/spike	Au-w/o spike	Au PR %
96022 A	BP525-135	0.578	0.480	83.0	1.13	0.07	6.5	18	14.74	81.9	10.20	0.10	10.10	0.03	<0.01	0.03	1.23	0.23	0.0
96023 A	BP525-136	0.177	0.120	67.8	1.13	0.09	8.1	22	17.26	78.5	8.29	0.13	8.16	0.01	<0.01	0.01	1.06	0.07	1.0
96024 A	BP530-137	0.630	0.510	81.0	1.07	0.14	12.9	29	25.60	88.7	6.03	0.09	5.94	0.17	<0.01	0.17	1.26	0.27	1.0
96025 A	BP530-138	1.215	1.080	88.9	1.60	0.23	14.1	10	9.08	86.8	5.62	0.08	5.54	0.03	<0.01	0.03	1.52	0.52	0.0
Phase 4C – C/D Pit & F-Zone																			
95155 A	BP251C-139	0.273	0.210	76.9	0.47	0.14	30.0	14	5.82	41.6	9.34	0.10	9.24	0.02	<0.01	0.02	1.09	0.09	0.0
95156 A	BP489C-140	0.196	0.140	71.4	0.87	0.22	25.8	7	3.92	53.0	8.97	0.11	8.86	0.04	0.01	0.03	1.05	0.06	1.0
95157 A	BP489C-141	0.125	0.090	72.0	0.80	0.16	20.2	5	3.56	72.2	9.78	0.12	9.66	0.03	<0.01	0.03	1.05	0.05	0.0
95158 A	BP489C-142	0.373	0.310	83.1	1.20	0.44	37.0	7	3.94	52.6	10.00	0.11	9.89	0.06	0.0	0.05	1.14	0.14	0.0
95159 A	BP489C-143	0.619	0.520	84.0	1.13	0.37	32.3	13	7.54	57.1	8.11	0.08	8.03	0.02	<0.01	0.02	1.23	0.23	0.0
95160 A	BP489C-144	0.239	0.170	71.1	0.80	0.24	29.5	8	7.18	92.9	8.35	0.09	8.26	0.02	<0.01	0.02	1.07	0.08	1.0
95161 A	BP516C-#145	0.197	0.140	71.1	0.67	0.22	33.3	13	12.22	91.7	8.52	0.10	8.42	0.01	<0.01	0.01	1.07	0.06	-1.0
95162 A	BP516C-146	0.348	0.290	83.3	0.67	0.21	32.1	23	22.40	97.9	6.90	0.10	6.80	0.03	0.01	0.02	1.11	0.11	0.0
95163 A	BP516C-147	0.261	0.200	76.6	0.47	0.12	24.9	9	11.54	133.2	8.85	0.11	8.75	0.02	<0.01	0.02	1.08	0.08	0.0
95164 A	BP516C-148	1.100	1.050	95.5	1.20	0.42	34.7	23	11.24	48.5	6.15	0.10	6.05	0.02	<0.01	0.02	1.41	0.41	0.0
95165 A	BP516C-149	0.199	0.190	95.5	0.47	0.13	27.4	15	17.82	120.2	9.12	0.11	9.01	0.01	<0.01	0.01	1.06	0.06	0.0

13.8.2 Bottle-Roll and Column Leach Testing

Coarse and fine milled bottle-roll leach tests were completed on each of the 60 composites. Splits of the head material was subjected to direct leach (DL) and CIL bottle-roll testing at target P_{80} sizes of 75 µm and 1,700 µm, and to column-leach testing at target P_{80}'s of 12.5 mm or 25.0 mm crush sizes. The main objective of these tests was to evaluate the laboratory-scale leachability character of the Rangefront and other Black Pine resources in terms of gold extraction, extraction rate, reagent consumption, and sensitivity to feed size.

The bottle-roll testing used standard procedures that are described in the final laboratory report (KCA 2023a, KCA2023b & KCA2024a), using 144 hours of retention time for 1,700 µm tests and 72 hours for 75 µm tests.

Column-leach tests were conducted utilizing material crushed to their respective target P_{80}'s and placed in columns of 10 cm and 15 cm diameters. Column tests were leached between 78 and 107 days (depending upon individual column charge leach rates) with a dilute NaCN solution. After leaching, each column was washed/rinsed for four days with water. A portion of the leached and washed material ("tailings") from each column was assayed for "tail screen" analyses by size fraction.

Tailings material from 40 of the 60 column composites was utilized for compacted permeability test work.

The following is a summary of the findings from KCA 2023a, KCA 2023b, and KCA 2024a final reports on bottle-roll and column test results.

13.8.3 Direct Leach and CIL Bottle-Roll Tests on 75 µm Composite Samples

Fine milled bottle-roll leach tests were completed on each of the 60 variability composites. The milled slurry was utilized for direct bottle-roll testing and CIL bottle-roll cyanide leach testing. The bottle-roll test procedures and results are described in detail in KCA 2023a, KCA 2023b, and KCA 2024a reports.

A summary of the 75 µm DL and CIL test results, containing average gold head grade, weighted average gold extraction %, cyanide and lime consumption are presented below. A total of 120 bottle roll tests are included in the summary table calculations:

Table 13-21: Summary 75 µm Bottle Roll Tests (60 comps, 120 tests)

Leach Test ID	Au Head	Au Ext	Reagents	
	ppm	%	Cyanide	Lime
75µm DL	0.644	83.6	0.230	0.563
75µm CIL	0.624	87.3	0.636	0.527

13.8.4 Direct Leach and CIL Coarse Bottle-Roll Tests on 1,700 µm composite Samples

Fine crush bottle-roll leach tests were completed on each of the 60 variability composites. The milled slurry was utilized for direct bottle-roll testing and CIL bottle-roll cyanide leach testing. The bottle-roll test procedures and results are described in detail in KCA 2023a, KCA 2023b, and KCA 2024a reports.

A summary of the 1,700 µm DL and CIL test results, containing average gold head grade, weighted average gold extraction %, cyanide and lime consumption are presented below. A total of 120 bottle roll tests are included in the summary table calculations:

Table 13-22: Summary 1,700 µm Bottle Roll Tests (60 comps, 120 tests)

Leach Test ID	Au Head	Au Ext	Reagents	
	ppm	%	Cyanide	Lime
1,700µm DL	0.648	83.5	0.144	0.794
1,700µm CIL	0.620	84.0	1.122	0.532

Note: Cyanide consumptions are always higher in CIL tests vs. DL tests due to catalytic destruction of cyanide when contacted with fresh activated carbon.

13.8.5 Column-Lech Tests on Composite Samples

All 60 composites were subjected to laboratory column-leach testing at KCA. Sixteen (16) columns were tested at a target feed P_{80} of 12.5 mm and forty-four (44) composites at a target feed P_{80} of 25 mm. Column test procedures are described in detail in KCA 2023a, KCA 2023b, and KCA 2024a reports. Column test gold and silver metallurgical balances (extractions) are based upon loaded carbon assays and tails screen assays and are in Table 13-19.

- Calculated gold head grades ranged from 0.111 ppm to 7.95 ppm and averaged 0.615 ppm. Gold extractions ranged from 43.6% to 95.8%, with a weight average of 84.3.
- Calculated silver head grades ranged from 0.27 ppm to 9.59 ppm and averaged 1.79 ppm. Silver extractions ranged from 1.5% to 48.9%, with a weight average of 8.9%.
- Cyanide consumptions ranged from 0.10 kg/t to 1.60 kg/t and averaged 0.74 kg/t. Based upon KCA's experience with clean non-reactive ores, cyanide consumption in commercial production heaps would range between 25% to 33% of the laboratory column test consumptions.
- Lime consumption ranged from 0.00 kg/ to 1.97 kg/t and averaged 0.78 kg/t. Five columns were agglomerated with 4.0 kg/t of cement and did not require any lime addition.
- Gold extraction % vs. days under leach are based upon column solution balances.

Table 13-23: Phase 4A, 4B & 4C Variability Column Test Results

KCA Sample No.	Comp ID	Test No	Mine Area	Geology Formation	Column Data						Au Balance		Ag Balance		Reagents		
					Target P_{80} (µm)	Screen P_{80} (µm)	% - 200M	Load Perm Tests [*1]	NaCN (g/l)	Time (days)	Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	Na CN kg/t	Lime kg/t	Cement kg/t
Phase 4A- Rangefront Variability Core Composites																	
95101 A	BP533-90	95501	Rangefront	Pola, Fz	25,000	24,200	2.6	Pass	0.5	90	90.8	0.239	2.1	0.96	0.68	0.76	
95102 A	BP533-91	95504	Rangefront	Pola	25,000	23,300	7.7	Pass	0.5	90	86.7	0.782	3.7	1.34	0.73	0.75	
95103 A	BP511CA-92	95507	Rangefront	Ppos	12,500	11,300	13.0	N/A	0.5	90	80.4	0.368	5.4	2.24	0.91	1.01	
95104 A	BP511CA-93	95510	Rangefront	Ppos, T1	25,000	25,200	6.3	Pass	0.5	90	82.7	0.15	3.2	1.25	0.73	0.74	
95105 A	BP511CA-94	95513	Rangefront	Ppos	25,000	24,000	15.4	Pass	0.5	90	79.6	0.137	13.0	0.77	0.69	0.76	
95106 A	BP511CA-95	95516	Rangefront	Pola, Ti/FZ	25,000	24,300	3.8	Pass	0.5	89	88.5	0.209	2.6	1.17	0.69	0.75	
95107 A	BP511CA-96	95519	Rangefront	Pola/FZ	25,000	24,700	4.0	Pass	0.5	89	92.4	1.322	4.4	1.14	0.66	0.76	
95108 A	BP511CA-97	95522	Rangefront	Pola/FZ	25,000	22,400	10.6	Pass	0.5	89	95.8	1.966	6.1	3.09	0.74	0.76	
95109 A	BP511CA-98	95525	Rangefront	Pola	25,000	25,000	4.5	Pass	0.5	89	88.7	0.938	2.3	1.71	0.63	0.50	
95110 A	BP511CA-99	95528	Rangefront	Pola, FZ	25,000	28,800	4.6	N/A	0.5	89	84.1	0.352	3.7	1.36	0.68	0.76	
95111 A	BP541-100	95531	Rangefront	Pola/FZ	25,000	27,800	6.4	N/A	0.5	89	78.5	0.261	9.2	0.87	0.73	0.76	
95112 A	BP541-101	95534	Rangefront	Pola/FZ	25,000	10,900	11.9	N/A	0.5	89	81.3	0.459	4.7	1.28	0.92	0.75	
95113 A	BP541-102	95537	Rangefront	Pola/FZ	25,000	22,200	8.4	Pass	0.5	78	91.2	0.148	4.2	0.95	0.47	0.00	4.0
95114 A	BP541-103	95540	Rangefront	Pola/FZ	25,000	25,300	4.1	Pass	0.5	78	89.1	0.239	3.5	0.86	0.60	0.77	
95115 A	BP541-104	95543	Rangefront	Pola	25,000	24,300	4.5	Pass	0.5	78	70.6	0.214	10.2	1.47	0.58	0.75	
95116 A	BP541-105	95543	Rangefront	Pola, FZ	12,500	12,400	8.8	N/A	0.5	78	87.5	0.224	2.0	1.47	0.43	0.00	4.0
95117 A	BP541-106	95549	Rangefront	Pola	12,500	13,700	7.2	N/A	0.5	78	88.6	0.597	2.1	0.94	0.64	0.50	
95118 A	BP541-107	95552	Rangefront	Pola	12,500	12,500	5.1	N/A	0.5	78	76.5	0.200	8.8	0.34	0.67	0.50	
95119 A	BP541-108	95555	Rangefront	Ppos	12,500	13,500	8.1	N/A	0.5	78	59.0	0.144	26.7	0.45	0.55	0.00	4.0
95120 A	BP541-109	95558	Rangefront	Pola	25,000	24,600	3.1	Pass	0.5	78	54.1	0.281	33.6	1.13	0.57	0.75	
95121 A	BP541-110	95561	Rangefront	Pola	25,000	24,500	2.3	Pass	0.5	78	66.7	0.123	5.4	1.11	0.52	0.75	
95122 A	BP541-111	95564	Rangefront	Pola	25,000	24,600	5.5	Pass	0.5	78	86.6	0.948	3.3	2.99	0.64	0.75	
95123 A	BP541-112	95567	Rangefront	Pola	12,500	12,300	5.9	N/A	0.5	78	90.0	0.738	27.0	4.56	0.42	0.50	
95124 A	BP541-113	96670	Rangefront	Pola	25,000	24,100	2.2	Pass	0.5	78	81.6	0.234	30.0	1.90	0.48	0.75	
Phase 4B – DZ, M-Zone, I-Pit & Tallman Variability Core Composites																	
96001 A	BP429CA-114	96051	DZ2	Pola, FZ	25,000	24,200	5.8	N/A	0.5	93	69.9	0.198	3.4	2.23	0.66	1.00	
96002 A	BP429CA-115	96054	DZ2	FZ, Pola	25,000	25,900	6.9	Pass	0.5	93	56.9	0.268	2.4	2.11	0.73	1.00	
96003 A	BP429CA-116	96057	DZ2	Polb	25,000	22,900	12.5	Pass	0.5	93	83.1	0.425	5.8	3.39	0.53	1.50	
96004 A	BP429CA-117	96060	DZ2	Polb, FZ	25,000	25,800	10.1	Pass	0.5	93	89.1	0.216	4.5	2.10	0.55	1.48	
96005 A	BP429CA-118	96063	DZ2	Polc	25,000	25,400	4.6	Pass	0.5	93	43.6	0.299	13.7	1.46	0.65	1.00	
96006 A	BP429CA-119	96066	DZ2	Pold	25,000	25,000	3.5	Pass	0.5	93	75.3	0.268	37.6	3.64	0.56	1.00	
96007 A	BP429CA-120	96069	DZ2	FZ, Pold	12,500	13,800	15.1	N/A	0.5	96	82.8	1.895	14.8	4.37	1.60	0.00	4.0
96008 A	BP429CA-121	96072	DZ2	PmMx	12,500	13,900	7.2	N/A	0.5	96	84.5	0.180	24.0	3.11	0.98	1.02	
96009 A	BP485-122	96075	E Pit	Pola, FZ	25,000	28,200	3.0	Pass	0.5	96	78.6	0.162	8.4	2.56	0.56	1.97	
96010 A	BP485-123	96078	E Pit	Pola, FZ	25,000	22,800	15.6	Fail 75m	0.5	96	86.1	0.342	8.9	1.77	0.83	0.99	
96011 A	BP485-124	96081	E Pit	Pola, FZ	25,000	24,900	5.3	Pass	0.5	96	81.8	0.258	6.2	2.05	0.73	1.00	
96012 A	BP508-125	96084	F Zone	Pold, CalFm	25,000	25,600	1.9	Pass	0.5	96	75.3	0.359	6.6	1.04	0.49	1.00	
96013 A	BP508-126	96087	F Zone	Pols	25,000	24,500	1.4	Pass	0.5	96	67.7	0.174	4.5	1.21	0.47	1.00	
96014 A	BP508-127	96090	F Zone	Pols, Ti	12,500	13,200	11.8	N/A	0.5	96	85.8	0.747	22.9	2.56	0.95	1.01	
96015 A	BP508-128	96093	F Zone	Pols, FZ	25,000	24,500	20.4	Fail 50m	0.5	96	94.2	7.995	14.6	4.11	0.52	0.00	4.0
96016 A	BP499-129	96401	Tallman	Pola, CalFm	25,000	28,700	2.8	Pass	0.5	96	75.5	0.500	42.1	9.58	0.75	1.00	
96017 A	BP499-130	96404	Tallman	Polb	12,500	12,600	6.3	N/A	0.5	94	57.0	0.243	12.2	2.50	0.68	0.75	
96018 A	BP499-131	96407	Tallman	Polc	25,000	25,600	4.0	Pass	0.5	94	87.6	3.893	34.0	4.82	0.65	1.00	

KCA Sample No.	Comp ID	Test No	Mine Area	Geology Formation	Column Data						Au Balance		Ag Balance		Reagents		
					Target P_{80} (µm)	Screen P_{80} (µm)	% - 200M	Load Perm Tests *1	NaCN (g/l)	Time (days)	Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	Na CN kg/t	Lime kg/t	Cement kg/t
96019 A	BP499-132	96410	Tallman	Polc, FZ	12,500	13,100	9.3	N/A	0.5	94	61.4	0.140	N/A	N/A	0.75	0.75	
96020 A	BP573-133	96413	M Zone	Polc, FZ	25,000	28,600	4.1	Pass	0.5	94	71.9	0.316	21.5	3.16	0.51	0.50	
96021 A	BP525-134	96416	I Pit Saddle	Pold	25,000	25,200	2.9	Pass	0.5	94	63.3	0.487	3.0	0.77	0.53	0.99	
96022 A	BP525-135	96419	I Pit Saddle	Pols	25,000	25,700	3.2	Pass	0.5	94	73.9	0.531	1.5	1.75	0.95	0.50	
96023 A	BP525-136	96422	I Pit Saddle	Pols	12,500	13,400	3.8	N/A	0.5	94	64.0	0.190	4.4	1.97	0.10	0.76	
96024 A	BP530-137	96425	M Zone	Polb, FZ	12,500	13,800	7.4	N/A	0.5	94	77.0	0.641	3.6	1.61	0.95	0.50	
96025 A	BP530-138	96428	M Zone	Polc	25,000	26,000	4.4	Pass	0.5	94	81.0	1.130	2.8	2.00	0.56	0.75	
Phase 4C – C/D Pit Variability Core Composites																	
95155 A	BP251C-139	96431	C/D Pit So	Pold	12,500	12,400	3.5	N/A	0.5	107	64.7	0.255	21.2	0.32	0.98	0.99	
95156 A	BP489C-140	96434	C/D Pit West	Pold, FZ	25,000	24,900	2.4	Pass	0.5	107	86.5	0.182	23.1	0.34	0.91	1.25	
95157 A	BP489C-141	96437	C/D Pit West	Pold	25,000	27,400	1.7	Pass	0.5	107	30.3	0.111	14.5	0.29	0.95	0.75	
95158 A	BP489C-142	96440	C/D Pit West	Pold, FZ	25,000	24,700	2.3	Pass	0.5	107	69.5	0.353	25.2	0.78	1.08	1.00	
95159 A	BP489C-143	96443	C/D Pit West	Pols, Ti	25,000	24,300	1.9	Pass	0.5	107	72.9	0.550	19.1	0.61	1.17	0.75	
95160 A	BP489C-144	96446	C/D Pit West	Pols, Ti	25,000	24,800	2.5	Pass	0.5	107	52.7	0.224	24.0	0.33	1.07	0.50	
95161 A	BP516C-#145	96449	C/D Pit East	Pols, Ti	12,500	14,100	3.3	N/A	0.5	107	54.8	0.179	24.0	0.33	1.19	0.74	
95162 A	BP516C-146	96452	C/D Pit East	Pols, FZ	25,000	27,600	1.1	Pass	0.5	107	73.9	0.336	21.0	0.32	1.21	0.75	
95163 A	BP516C-147	96455	C/D Pit East	Pols	25,000	29,000	8.4	Pass	0.5	107	57.1	0.240	5.6	0.26	1.01	0.75	
95164 A	BP516-148	96458	C/D Pit East	Pols, FZ	12,500	13,200	2.8	N/A	0.5	107	89.8	1.120	48.9	0.69	1.17	0.99	
95165 A	BP516-149	96461	C/D Pit East	Pols	25,000	25,400	25.4	Pass	0.5	107	60.6	0.204	14.3	0.29	0.97	0.74	

*1 – Load Permeability Tests were conducted at 25, 50, 75, and 100 meters simulated heap height. Pass indicated no problem up to 100 meters. N/A means column residue was not tested.

Thirty-nine (39) of the 60 column leach residues were subjected to load permeability test work. The purpose of the load permeability testing was to examine the permeability of the crushed material under compaction loading equivalent to heap heights of 25 m, 50 m, 75 m, and 100 m.

Two (2) of the 39 column leach residues failed load permeability testing up to 100 m height. Load permeability Pass/Fail criteria is included in Table 13-10.

Refer to KCA 2023a, KCA 2023b, and KCA 2024a reports for details on the Load Permeability test results, apparatus, and testing procedures.

13.9 2023 Phase 5A Variability composite Testing

In January 2024, Liberty Gold delivered twenty-five (25) Black Pine variability composite samples to the laboratory facility of KCA in Reno, Nevada.

All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures. The same as what is described in the original Phase 1 (KCA 2020a) and Phase 2 (KCA 2020b) final reports.

The variability composites were retrieved from Black Pine large diameter PQ metallurgical core drilling conducted in 2022-23.

Splits of all 25 composite heads were delivered to two separate laboratories for additional geo-metallurgical and environmental characterization analysis:

1. ALS for ICP and gold cyanide solubility analysis, and
2. FLS for “XRD” and “Whole Rock” analysis.

The Phase 5 program was designed to test the new Back Range and J Zone resource area and to fill gaps in current resources. A brief description of the Phase 5 variability composite samples delivered to KCA is provided here:

- Back Range (9 comps) – First testing of the Back Range deposit resources.
- J Zone (4 comps) – First testing of the J Zone deposit resources.
- Discovery Zone - Tallman Pit area (3 comps) – In-fill gaps in the South Tallman Pit resources.
- M Zone (6 comps) – In-fill gaps in the M-Zone resources.
- F Zone (3 comps) – In-fill gaps in the F-Zone resources.

13.9.1 2024 Black Pine Phase 5A Variability Composite Head Assays

Head assays, geo-metallurgical characterization and metallurgical test result details are in the Kappes, Cassiday and Associates (KCA 2024b) report:

- KCA 2024b report – Black Pine Project, Phase 5A Core Composites, Report of Metallurgical Test Work, June 2024.

A high-level summary of the geo-metallurgical characterization is presented below for gold, silver, copper, gold cyanide solubility, carbon and sulfur species, preg-rob analysis, as well as ICP multi-element analyses, whole-rock analyses and QXRD analyses. Select composite summary results for gold, silver, copper, carbon and sulfur speciation and preg-rob analysis, are detailed in Table 13-24:

- Gold grades ranged from 0.141 ppm to 5.09 ppm and averaged 0.994 ppm.
- Silver grades ranged from 0.33 ppm to 188.3 ppm and averaged 17.38 ppm.
- Organic carbon ranged from 0.07% to 0.25 % and averaged 0.16%.
- Sulfide sulfur ranged from <0.01% to 2.57% and averaged 0.10%, 21 of the 25 composites reporting <0.01% sulfide sulfur.
- Preg-robbing analyses ranged from 0.0% to 27.5% and averaged 2.4% (using a 1 ppm spike). 23 of the 25 composites had Preg-robbing values <5% and are considered non-preg-robbing.
- Copper values ranged from <2 ppm to 366 ppm and averaged 47 ppm.
- Gold cyanide solubility ranged from 28.4% to 95.1% and weight averaged 75.4%.
- Concentrations of the deleterious elements were low for selenium which averaged 20 ppm and mercury which ranged from 0.52 ppm to 107.5 ppm with an average of 14.9 ppm. Arsenic levels ranged from 32 ppm to

2959 ppm with an average of 315 ppm. The two highest arsenic samples (868 and 2,959 ppm As) came from M-Zone

- Concentrations of the primary cyanide consumers were low for copper (averaged 46 ppm) and nickel (averaged 44 ppm). Zinc ranged from 39 ppm to 19,597 ppm and averaged 1,589 ppm. The two highest Zn samples (12,063 ppm and 19,597 ppm Zn) correspond with the same two high arsenic samples in M-Zone. Removing these two samples reduces the average Zn grade to 351 ppm.
- Whole-rock SiO_2 content ranged from 33.6% to 78.6% and averaged 43.9%.

Table 13-24: 2024 Black Pine Phase 5A Variability Composites

KCA Sample No.	Composite ID	ALS			KCA														
		AuFA ppm	AuCN ppm	AuCN %	AgFA ppm	AgCN ppm	AgCN %	Cu ppm	CuCN ppm	CuCN %	$C_{(tot)}$ %	$C_{(org)}$ %	$C_{(inorg)}$ %	$S_{(tot)}$ %	$S_{(sulfide)}$ %	$S_{(SO4)}$ %	Au-w/spike	Au-w/o spike	Au PR %
Phase 5A – Back Range, J Zone and Gap In-fill																			
98104 A	BP956C-150	0.480	0.205	42.7	1.77	1.34	75.8	20	3.54	17.7	4.29	0.16	4.13	0.03	<0.01	0.03	1.11	0.11	0.0
98105 A	BP956C-151	2.295	1.495	65.1	5.17	3.72	72.0	26	3.32	12.8	2.06	0.19	1.87	0.01	<0.01	0.01	1.92	0.92	0.0
98106 A	BP956C-152	5.090	4.020	79.0	6.00	4.24	70.7	12	4.62	38.5	4.04	0.18	3.86	0.01	<0.01	0.01	3.07	2.07	0.0
98107 A	BP956C-153	1.133	0.900	79.5	2.57	0.12	4.7	11	1.72	15.6	4.68	0.16	4.52	0.02	<0.01	0.02	1.50	0.50	0.0
98108 A	BP956C-154	0.313	0.180	57.5	1.17	0.88	75.4	15	1.94	12.9	3.66	0.20	3.46	0.14	0.02	0.12	1.12	0.12	0.0
98109 A	BP956C-155	0.355	0.160	45.1	2.00	1.58	79.0	25	3.08	12.3	2.66	0.23	2.43	0.01	<0.01	0.01	1.12	0.13	1.0
98110 A	BP956C-156	4.485	3.565	79.5	3.33	2.34	70.2	35	2.10	6.0	0.64	0.25	0.39	0.01	<0.01	0.01	3.02	2.02	0.0
98111 A	BP956C-157	1.290	0.930	72.1	1.53	1.10	71.7	10	2.34	23.4	5.42	0.16	5.26	0.04	0.01	0.03	1.47	0.47	0.0
98112 A	BP956C-158	0.141	0.040	28.4	1.40	0.88	62.9	12	2.90	24.2	6.55	0.15	6.40	0.08	<0.01	0.08	0.73	0.01	27.5
98113 A	BP1010C-159	0.543	0.435	80.1	2.13	0.94	44.1	25	2.34	9.4	4.45	0.16	4.29	0.04	<0.01	0.04	1.15	0.15	0.0
98114 A	BP1010C-160	0.366	0.235	64.3	0.93	0.52	55.7	8	4.66	58.3	7.35	0.13	7.22	0.03	<0.01	0.03	1.24	0.24	0.0
98115 A	BP1010C-161	0.905	0.775	85.6	1.27	0.38	30.0	11	1.16	10.5	5.83	0.24	5.59	0.04	<0.01	0.04	0.73	0.01	27.5
98116 A	BP1010C-162	1.058	0.800	75.7	1.93	0.74	38.3	16	2.14	13.4	4.94	0.14	4.80	0.01	<0.01	0.01	1.31	0.31	0.0
98117 A	BP1008C-163	0.377	0.330	87.6	0.33	0.22	66.0	<2	1.12	56.0	3.45	0.14	3.31	0.02	<0.01	0.02	1.13	0.15	2.0
98118 A	BP1008C-164	0.359	0.265	73.9	0.68	0.18	26.3	<2	0.96	48.0	7.68	0.21	7.47	0.02	<0.01	0.02	1.07	0.07	0.0
98119 A	BP1008C-165	1.133	0.965	85.2	0.62	0.22	35.7	<2	0.86	43.0	8.86	0.16	8.70	0.02	<0.01	0.02	1.33	0.33	0.0
98120 A	BP980C-166	0.815	0.470	57.7	180.3	93.4	51.8	366	38.40	10.5	3.31	0.11	3.20	0.31	<0.01	0.31	1.33	0.34	1.0
98121 A	BP980C-167	0.270	0.155	57.5	14.9	9.6	64.7	86	5.92	6.9	4.23	0.12	4.11	0.07	<0.01	0.07	1.10	0.10	0.0
98122 A	BP980C-168	0.437	0.380	87.1	3.37	2.78	82.6	13	6.00	46.2	1.60	0.15	1.45	3.18	2.57	0.61	1.15	0.15	0.0
98123 A	BP979C-169	0.327	0.265	81.2	2.03	0.74	36.4	13	1.86	14.3	4.17	0.14	4.03	0.04	<0.01	0.04	1.04	0.04	0.0
98124 A	BP979C-170	0.709	0.530	74.8	188.3	156.4	83.0	281	64.60	23.0	7.58	0.08	7.50	0.15	<0.01	0.15	1.31	0.31	0.0
98125 A	BP979C-171	0.796	0.580	72.9	10.10	9.68	95.8	44	9.14	20.8	4.20	0.07	4.13	0.32	0.01	0.31	1.39	0.39	0.0
98126 A	BP1001C-172	0.752	0.715	95.1	1.17	0.42	36.0	2	1.88	94.0	7.81	0.09	7.72	0.01	<0.01	0.01	1.23	0.23	0.0
98127 A	BP1001C-173	0.238	0.195	81.9	1.03	0.44	42.6	4	2.00	50.0	7.70	0.12	7.58	0.02	<0.01	0.02	1.05	0.05	0.0
98128 A	BP1001C-174	0.177	0.145	81.9	0.33	0.34	102.0	9	2.44	27.1	5.44	0.15	5.29	0.06	<0.01	0.06	1.06	0.06	0.0

Source: Gary Simmons Consulting LLC, 2024

13.9.2 Bottle-Roll Testing

Coarse and fine milled bottle-roll leach tests were completed on each of the 25 composites. Splits of the head material was subjected to direct leach (DL) and CIL bottle-roll testing at target P_{80} sizes of 75 µm and 1,700 µm, and coarse crush bottle roll testing at target P_{80}'s of 12.5 mm crush sizes. The main objective of these tests was to evaluate the laboratory-scale leachability character of the Phase 5A variability composites in terms of gold extraction, extraction rate, reagent consumption, and sensitivity to feed size.

Due to the long duration time frame to complete laboratory column leach testing, the standard 12.5 mm and 25 mm Black Pine laboratory column leach tests are being delayed until later in 2025. Standard column leach testing was replaced with coarse-crush 12.5 mm bottle roll tests, to meet the Pre-Feasibility time schedule. It is planned to complete the standard column leach testing once the pre-feasibility study is completed and remodel the Phase 5A results using the column leach data.

The 1,700 µm and 75 µm direct leach and CIL bottle roll testing procedures are the same as all previous phases of testing for Black Pine and are described in the final laboratory report (KCA 2024b).

The coarse-crush (12.5 mm) bottle roll test procedure is briefly described here, with full details in KCA 2024b:

1. A 15 kg split of stage crushed head material (target P_{80} 12.5 mm) was placed into a 50-liter carboy and slurried with 22,500 milliliters of tap water. The pH of the slurry was checked and adjusted, as required, to 10.5 to 11.0 with hydrated lime. Sodium cyanide was added to the slurry to a target amount of 1.0 grams per liter sodium cyanide and maintained at 0.50 grams per liter. The carboy was subjected to intermittent rolling (rolling for two (2) minutes every hour) throughout the duration of the test
2. At 24 hours the test was allowed to settle. The preg solution was then siphoned off and filtered through a small column containing attritioned granulated activated carbon (GAC). The resulting barren solution was checked for pH, dissolved oxygen (DO), NaCN, Au, Ag and Cu, then returned to the carboy.
3. At 72 hours the test was allowed to settle. The preg solution was then siphoned off and filtered through the same small carbon column (C-1) as the 24-hour solution. The resulting barren solution was checked for pH, dissolved oxygen (DO), NaCN, Au, Ag, and Cu and returned to the carboy.
    a. The loaded carbon (C-1) from the 24- and 72-hour solution strip was then dried. The dried carbon was weighed and assayed for gold and silver content.
4. Items 2-3 were conducted to simulate conventional heap leach CIC extraction of gold and silver from pregnant solution, to simulate barren solution being returned to the heap.
5. The leach test continued with intermittent agitation for a total of 240 hours (10 days). The slurry was checked at 96, 120, 144, 168, 192, 216, and 240 hours for pH, dissolved oxygen (DO), NaCN, Au, Ag, and Cu. Additional hydrated lime and sodium cyanide were added after each sample period, if required, to adjust the slurry to the target levels.
6. During the 72 - 240-hour period a closed-end PVC pipe containing 150 grams of GAC carbon was placed into the carboy. Upon completion of the leach period, the incapsulated GAC (C-2) utilized during the test (72 to 240 hours) was removed and dried. The dried carbon was then weighed and assayed for gold and silver content.
7. The dried tailings were thoroughly blended then split into two (2) portions. One portion was crushed to a target size of 80% passing 1.70 millimeters and three (3) 200-gram portions were split out for triplicate gold and silver assays.
8. The remaining tailings material was wet screened at 0.075 millimeters. The undersized material was dried and set aside. The oversized material was dried and dry screened at 19, 12.5, 9.5, 6.3, 1.70, 0.600, 0.212, and 0.075 millimeters. The dry screened minus 0.075-millimeter material was then combined with the wet screened minus 0.075 millimeter material.

9. Each separate size fraction was then weighed, and the weights reported. The material was then recombined and stored.

13.9.3 Direct Leach and CIL Bottle-Roll Tests on 75 µm Composite Samples

Fine milled bottle-roll leach tests were completed on each of the 25 variability composites. The milled slurry was utilized for direct leach (DL) and CIL bottle-roll cyanide leach testing. The bottle-roll test procedures and results are described in detail in KCA 2024b.

A summary of the 75 µm DL and CIL test results, containing average gold head grade, weighted average gold extraction %, cyanide and lime consumption are presented below. A total of 50 bottle roll tests are included in the summary table calculations:

Table 13-25: Summary 75 µm Bottle Roll Tests (25 comps, 50 tests)

Leach Test ID	Au Head	Au Ext	Reagents	
	Ppm	%	Cyanide	Lime
75µm DL	0.978	77.1	0.428	0.680
75µm CIL	0.968	81.6	0.690	0.690

13.9.4 Direct Leach and CIL Coarse Bottle-Roll Tests on 1,700 µm Composite Samples

Fine crush bottle-roll leach tests were completed on each of the 25 variability composites. The milled slurry was utilized for direct leach (DL) and CIL bottle-roll cyanide leach testing. The bottle-roll test procedures and results are described in detail in KCA 2024b.

A summary of the 1,700 µm DL and CIL test results, containing average gold head grade, weighted average gold extraction %, cyanide and lime consumption are presented below. A total of 50 bottle roll tests are included in the summary table calculations:

Table 13-26: Summary 1,700 µm Bottle Roll Tests (25 comps, 50 tests)

Leach Test ID	Au Head	Au Ext	Reagents	
	Ppm	%	Cyanide	Lime
1,700µm DL	0.928	77.8	0.170	0.790
1,700µm CIL	0.947	81.5	0.915	0.970

Note: Cyanide consumptions are always higher in CIL tests vs. DL tests due to catalytic destruction of cyanide when contacted with fresh carbon.

13.9.5 Coarse-Crush (12.5 mm) Bottle Roll Tests on Composite Samples

All 25 composites were subjected to laboratory coarse-crush (12.5 mm) bottle roll tests as described in Section 13.9.2 of this report and in KCA 2024b. Gold and silver metallurgical balances (extractions) are based upon loaded carbon assays and tails screen assays and are in Table 13-27.

- Calculated gold head grades ranged from 0.165 ppm to 5.04 ppm and averaged 0.986 ppm. Gold extractions ranged from 48.4% to 86.8%, with a weight average of 78.9%.
- Calculated silver head grades ranged from 0.47 ppm to 194.1 ppm and averaged 15.61 ppm. Silver extractions ranged from 5.9% to 61.5%, with a weight average of 38.6%.

- Cyanide consumption ranged from 0.07 kg/t to 1.00 kg/t and averaged 0.25 kg/t.
- Lime consumption ranged from 0.50 to 2.25 kg/t and averaged 0.65 kg/t.

Table 13-27: Phase 5A Variability Coarse-Crush (12.5 mm) Bottle Roll Test Results

KCA Sample No.	Composite ID	Test No.	Geology		Coarse-Crush (12.5 mm) BR Data						Au Balance		Ag Balance		Reagents	
			Mine Area	F-Form	Target P_{80} (µm)	Screen P_{80} (µm)	% -200M	Load Perm *1	NaCN (g/l)	Time (days)	Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	NaCN kg/t	Lime kg/t
98104 B	BP956C-150	98150 A	Bankrange	Polb, Fz	12,500	11,500	20.1	BR-N/A	0.5	10	66.6	0.433	44.5	1.84	0.07	0.50
98105 A	BP956C-151	98150 B	Bankrange	Fz, Polb	12,500	9,200	38.1	BR-N/A	0.5	10	83.9	2.143	56.3	4.84	0.13	0.75
98106 A	BP956C-152	98150 C	Bankrange	Fz, Polb	12,500	10,200	36.3	BR-N/A	0.5	10	80.9	5.039	41.0	4.43	0.16	0.50
98107 B	BP956C-153	98151 A	Bankrange	Polb, Fz	12,500	11,600	12.4	BR-N/A	0.5	10	86.8	1.167	48.3	2.17	0.20	0.50
98107 A	BP956C-154	98151 B	Bankrange	PMmx, Fz	12,500	12,200	14.7	BR-N/A	0.5	10	75.3	0.347	50.0	1.07	0.19	0.50
98108 B	BP956C-155	98151 C	Bankrange	Polb, FZ	12,500	12,000	14.3	BR-N/A	0.5	10	64.1	0.381	56.2	2.01	0.20	0.50
98110 B	BP956C-156	98152 A	Bankrange	FZ, Polb	12,500	7,300	35.6	BR-N/A	0.5	10	83.6	3.983	59.0	2.62	0.11	0.75
98111 B	BP956C-157	98152 B	Bankrange	Polb	12,500	12,200	14.9	BR-N/A	0.5	10	75.4	1.372	49.3	1.42	0.14	0.50
98112 A	BP969C-158	98152 C	Bankrange	Polb	12,500	12,300	10.3	BR-N/A	0.5	10	48.4	0.173	52.6	1.18	0.24	0.50
98113 A	BP1010C-159	98153 A	J Zone	Pola, T1	12,500	11,700	17.8	BR-N/A	0.5	10	78.6	0.562	11.6	2.28	0.21	0.75
98114 B	BP1010C-160	98153 B	J Zone	Pola	12,500	13,000	9.5	BR-N/A	0.5	10	62.7	0.417	7.0	1.29	0.19	0.50
98115 A	BP1010C-161	98153 C	J Zone	Pola, FZ	12,500	11,400	14.6	BR-N/A	0.5	10	79.9	1.057	6.2	1.45	0.13	0.75
98116 A	BP1010C-162	98154 A	J Zone	FZ, PM mx	12,500	12,000	17.3	BR-N/A	0.5	10	86.0	0.836	5.9	2.36	0.12	0.50
98117 B	BP1008C-163	98154 B	Tallman	Polc	12,500	12,700	5.6	BR-N/A	0.5	10	70.4	0.492	8.6	0.78	0.16	0.50
98118 B	BP1008C-164	98154 C	Tallman	Polc, FZ	12,500	12,500	8.6	BR-N/A	0.5	10	65.6	0.342	6.1	0.85	0.14	0.50
98119 A	BP1008C-165	98155 A	Tallman	Polc	12,500	11,500	7.9	BR-N/A	0.5	10	81.0	1.191	10.7	0.79	0.21	0.50
98120 A	BP980C-166	98155 B	M Zone	FZ, Polc	12,500	11,700	18.3	BR-N/A	0.5	10	64.2	0.735	30.2	131.98	1.00	2.25
98121 B	BP980C-167	98155 C	M Zone	Polc, FZ	12,500	12,300	7.6	BR-N/A	0.5	10	73.9	0.263	38.0	14.86	0.91	1.00
98122 A	BP980C-168	98156 A	M Zone	PMmx, FZ	12,500	9,700	17.8	BR-N/A	0.5	10	83.4	0.509	47.3	4.18	0.51	0.75
98123 B	BP979C-169	98156 B	M Zone	Polc	12,500	12,200	10.6	BR-N/A	0.5	10	68.7	0.321	14.1	1.72	0.14	0.50
98124 B	BP979C-170	98156 C	M Zone	Polc	12,500	13,000	7.1	BR-N/A	0.5	10	82.2	0.859	43.9	194.11	0.56	0.50
98125 B	BP979C-171	98157 A	M Zone	Polc, FZ	12,500	12,000	13.4	BR-N/A	0.5	10	84.9	0.779	61.5	9.17	0.16	0.50
98126 B	BP1001C-172	98157 B	F Zone	Pols	12,500	12,100	9.0	BR-N/A	0.5	10	69.3	0.849	10.4	1.03	0.17	0.50
98127 B	BP1001C-173	98157 C	F Zone	Pols, FZ	12,500	11,900	15.9	BR-N/A	0.5	10	56.2	0.245	11.7	1.42	0.16	0.50
98128 A	BP1001C-174	98158 A	F Zone	PmMx, FZ	12,500	7,300	40.8	BR-N/A	0.5	10	71.6	0.165	45.7	0.47	0.16	0.75

13.10 Gold Recovery Methodology and Commercial Scale Recovery Models

13.10.1 Gold and Silver Recovery Methodology

The following is a brief description of the methodology used to derive the Black Pine gold-recovery models. Five steps are used in developing final commercial-scale gold recovery models:

- Step 1: Determining the gold extraction for each variability or Bulk composite, using a combination of fine grind/crush bottle-rolls and medium/coarse crush column tests.
- Step 2: Develop head grade versus tails grade models to use in final development of the gold recovery equations.
- Step 3: Estimate solution losses at the end of economic gold recovery operations/closure.
- Step 4: Determine operational scale-up inefficiencies for the heap leach flowsheet selected.
- Step 5: Incorporating steps 1-4 into final recovery models that reflect modelled test data met balances, commercial scale inefficiencies and deductions for solution losses.

13.10.2 Black Pine Ore Type Descriptions

Gold and silver mineralization are hosted in the Upper and Middle plate sequence of the Permian Oquirrh Formation. See Figure 13-10. Economic gold values are present in all geologic units in the Upper and Middle Plates: Ppos, Pola, Polb, Polc, Pold, Pols and Pmmx. Significant mineralization has not been found in the Lower Plate Manning Canyon Shale at Black Pine.

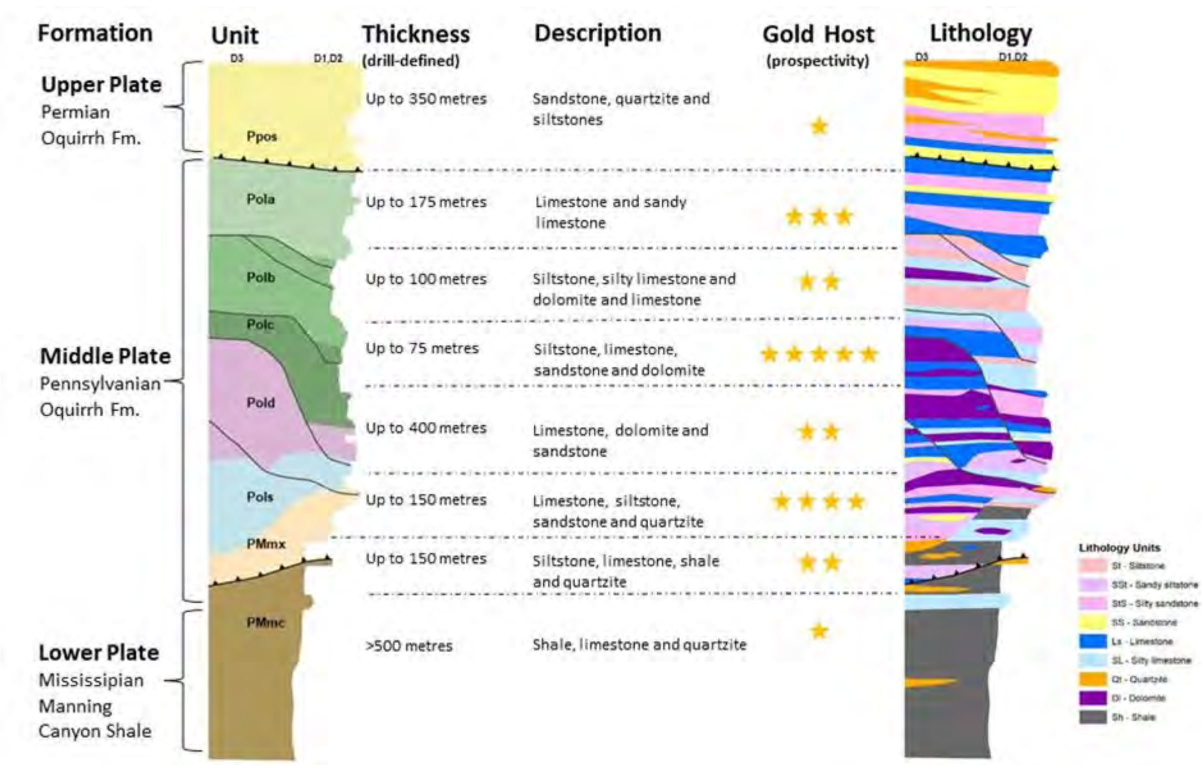


Figure 13-10: Black Pine Lithology

To facilitate gold and silver recovery modelling of the various lithologic units the following 3D cyanide solubility frames have been established:

- Met-1: Mineralized oxide ore with gold cyanide solubility >65%.
- Met-2: Mineralized oxide ore with gold cyanide solubility >50%, <65%.
- Met-3: Mineralized predominantly oxide ore with gold cyanide solubility >25%, <50%.
- Met-4: Mineralized mix of oxide, sulfide, and preg-robbing organic carbon material with gold cyanide solubility <25% ("WASTE"). This mineralized material is always WASTE and will be discarded with other mine waste.

Note: The Black Pine deposit is a carbonate hosted deposit that is essentially 98% oxide with very low sulfide sulfur content. Met-1, Met-2, and Met-3 ore type designations are basically oxide ores with varying degree gold cyanide solubility related to the following characteristics.

1. Medium to high-grade mineralization with consistently high gold cyanide solubility in decalcified low Mg-rocks.
2. Low grade, near economic cut-off, has naturally lower cyanide solubility, especially in Mg-elevated dolomitic rocks due to gold in unfractured ground mass.
3. Preg-borrowing clays are present in local areas that exhibit lower cyanide solubility in the laboratory assay procedure that does not show up in column leach testing. This is due to the clays giving back their preg-borrowed gold as the leach solution grade declines during the column leach cycle.
4. Preg-borrowing residual "smoky" carbon exists at Black Pine that acts very similar to the preg-borrowing clays.
5. Very minor sulfides, containing gold in solid-solution, are present that inhibit gold cyanide solubility.
6. Acid-base accounting show ores to be non-acid generating, therefore any residual sulfides making it onto the heap will not cause acid formation or significant reduction in pH during processing.

ROM recovery models have been developed for heap leach P_{80} feed size ranging up to 250 mm (10 inches). To assist in estimating the commercial ROM feed size, for each lithologic unit, all metallurgical core has been visually logged to estimate the percentage of +150 mm (+6 inches) core in all 174 variability composites that were tested. Refer to Table 13-29 for a summary of the P_{80} logging and recommended ROM feed size to be used for developing the gold recovery models for each unit, or combination of units.

Note: The +150 mm (+6 inch) estimate is not an RQD measurement, but a metallurgical representation of the dense, unfragmented solid core that will not be easily broken during conventional blasting and mechanical mining work. There are instances where +150 mm core is logged as -150 mm due to it being identified as having one or more of the following characteristics:

- Strongly decalcified core that easily absorbs water.
- Brecciated and decalcified (vuggy) core.
- Weakly cemented brecciated core that is easily broken up into its natural gravel fragments via either mild blasting and or simple load/haul/dump mechanical work

Black Pine is noted for its consistently flat gold extraction vs. P_{80} feed size response. This is mainly due to gold being deposited in fractures, and in strongly decalcified and/or brecciated carbonate rocks. The one exception being dolomitic rocks with Mg content >2.0% are proven to be harder to break and have fewer natural fractures as evident in Table 13-28 below. Mining drill/blast costs have been increased to put more blast energy into the dolomitic rocks to assist in finer breakage to meet the design ROM P_{80}.

Table 13-28: ROM P_{80} Modelling

Ore Type	Feet	Ft(%)	Mg %	+6" (%)	Modeled P80
Ppos	**606.8**	**7.02**		**8.5**	
Met-1 Rangefront Ppos: AuCN >65%	606.8	7.02	0.35	8.5	**100mm**
Pola	**2033.8**	**23.5**		**19.3**	
Met-1 Discovery Zone Pola: AuCN >65%	545.9	6.32	1.05	19.7	**150mm**
Met-1 E Pit Zone Pola: AuCN >65%	143.7	1.66	0.55	8.6	
Met-1 J Zone Pola: AuCN >65%	64.3	0.74	1.35	28.4	
Met-1 Rangefront Pola: AuCN >65%	**1279.9**	14.81	0.79	19.8	
Polb	**820.2**	**9.5**		**15.6**	
Met-1 Discovery Zone Polb: AuCN >65%	458.8	5.31	1.25	18.9	**125mm**
Met-1 E Pit Zone Polb: AuCN >65%	107.5	1.24	0.28	8.0	
Met-1 Backrange Polb: AuCN >65%	222.9	2.58	0.13	12.6	
Met-1 M Zone Polb: AuCN >65%	31.0	0.36	0.25	14.0	
Polc	**1576.3**	**18.2**		**37.3**	
Met-1 Discovery Zone Polc: AuCN >65%	**1313.4**	15.20	1.70	38.2	**200mm**
Met-1 M Zone Polc: AuCN >65%	262.9	3.04	0.89	32.8	
Pold	**953.3**	**11.0**		**43.4**	
Met-1 Discovery Zone Pold: AuCN >65%	507.6	5.87	3.63	49.9	**250mm**
Met-1 F Zone Pold: AuCN >65%	52.5	0.61	4.82	57.0	
Met-1 C/D Pit Pold: AuCN >65%	393.2	4.55	5.63	33.1	
Pols	**1271.7**	**14.7**		**28.6**	
Met-1 Discovery Zone Pols: AuCN >65%	540.9	6.26	2.12	32.0	**200mm**
Met-1 F Zone Pols: AuCN >65%	316.1	3.66	1.40	23.1	
Met-1 C/D Pit Pols: AuCN >65%	414.7	4.80	1.16	28.4	
PmMx	**163.8**	**1.9**		**22.5**	
Met-1 All Zones PmMx OXIDE: AuCN >65%	163.8	1.90	0.64	22.5	**150mm**
Met-2 All Zones	**612.8**	**7.1**		**23.1**	
Met-2 J Zone Pola: AuCN >50%, <65%	49.7	0.58	5.15	63.0	**175mm**
Met-2 Rangefront Pola: AuCN >50%, <65%	46.8	0.54	0.13	2.0	
Met-2 Discovery Zone Polb: AuCN >50%, <65%	257.4	2.98	0.84	17.1	
Met-2 Discovery Zone Polc: AuCN >50%, <65%	179.8	2.08	3.02	29.1	
Met-2 M Zone Pold: AuCN >50%, <65%	79.1	0.92	1.97	16.7	
Met-3 All Zones	**601.9**	**7.0**		**15.9**	
Met-3 Discovery Zone Pola: AuCN >25%, <50%	124.2	1.44	0.47	5.1	**125mm**
Met-3 Discovery Zone Polb: AuCN > 25%, <50%	231.4	2.68	1.10	18.0	
Met-3 Backrange Polb: AuCN >25%, <50%	179.8	2.08	0.29	13.5	
Met-3 Discovery Zone Polc: AuCN >25%, <50%	66.5	0.77	1.51	35.0	

13.10.3 Gold and Silver Recovery Model Equations for ROM Heap Leaching

13.10.3.1 M-1, M-2, and M-3 Gold and Silver Recovery Equations

Gold recovery equations for M-1, M-2, and M-3 oxide ore types are show in Table 13-29. There are two equations for each material type, one for gold head grades <0.40 g/t and one for head grades >0.40 g/t.

Table 13-29: Black Pine – ROM Gold Recovery Equations

Geo met Recovery Zone	P_{80}	Gold Recovery, %	Range
Met 1: Pola (DZ + J-Zone	150 mm	=16.822*ln(HG) + 84.288	Au HG < 0.40 g/t
		=8.074*ln(HG) + 76.716	Au HG > 0.40 g/t
Met-1: Pola (E-Pit)	150 mm	=8.951*ln(HG) + 84.169	Au HG < 0.40 g/t
		=1.709*ln(HG) + 77.409	Au HG < 0.40 g/t
Met-1:Pola (Rangefront)	150 mm	=12.250*ln(HG) + 91.377	Au HG > 0.40 g/t
		=5.270*ln(HG) + 85.309	Au HG < 0.40 g/t
Met-1: Polb (DZ+E-Pit+M-Zone+BR)	125 mm	=4.916*ln(HG) + 76.971	Au HG < 0.40 g/t
		=3.213*ln(HG) + 75.484	Au HG > 0.40 g/t
Met-1: Polc (DZ + M-Zone)	200 mm	=9.669*ln(HG) + 79.435	Au HG < 0.40 g/t
		=5.221*ln(HG) + 75.915	Au HG > 0.40 g/t
Met-1: Pold (DZ + M-Zone)	250 mm	=6.654*ln(HG) + 80.969	Au HG < 0.40 g/t
		=3.790*ln(HG) + 78.679	Au HG > 0.40 g/t
Met-1: Pold (C/D Pit)	250 mm	= 2.129*ln(HG) + 56.945	Au HG < 0.40 g/t
		=0.326*ln(HG) + 55.320	Au HG > 0.40 g/t
Met-1: Pols (DZ+F-Zone+C/D Pit)	200 mm	= 20.799*ln(HG) + 87.213	Au HG < 0.40 g/t
		=7.994*ln(HG) + 76.807	Au HG > 0.40 g/t
Met-1: Ppos (Rangefront)	100 mm	=4.228*ln(HG) + 78.391	Au HG < 0.40 g/t
		=0.807*ln(HG) + 75.198	Au HG > 0.40 g/t
Met-1: PmMx (ALL)	150 mm	=6.717*ln(HG) + 82.233	Au HG < 0.40 g/t
		=4.001*ln(HG) + 79.909	Au HG > 0.40 g/t
Met-2: (ALL Zones + Pits)	175 mm	=6.401*ln(HG) + 63.622	Au HG < 0.40 g/t
		=4.472*ln(HG) + 61.956	Au HG > 0.40 g/t
Met-3: (ALL Zones + Pits)	125 mm	=3.972*ln(HG) + 50.521	Au HG < 0.40 g/t
		=0.758*ln(HG) + 47.521	Au HG > 0.40 g/t

Black Pine silver head grades are low, except for M-Zone. All Zones and Pits have similar silver head grade vs tails grade response except for M-Zone and Back Range, which were modelled separately. Oxide silver recovery equations are shown in Table 13-30. There are two equations for each material type, one for silver head grade <3.0 g/t and one for silver head grade >3.0 g/t.

Table 13-30: Black Pine – ROM Silver Recovery Equations

Geo-met Recovery Zone	P_{80}	Silver Recovery, %	Range
Met-1,2,3: All Zones, Except M-Zone and BR	All P_{80}'s	=5.503*ln(HG) + 6.698	Au HG < 3.0 g/t
		=3.696*ln(HG) + 8.536	Au HG > 3.0 g/t
Met-1,2,3: M-Zone	All P_{80}'s	=5.733*ln(HG) + 11.371	Au HG < 3.0 g/t
		=3.863*ln(HG) + 13.283	Au HG > 3.0 g/t
Met-1,2,3: Backrange	All P_{80}'s	=2.574*ln(HG) + 35.714	Au HG < 3.0 g/t
		=1.076*ln(HG) + 37.193	Au HG > 3.0 g/t

13.11 Reagent Consumptions

Reagent consumptions and requirements, including cyanide, lime, and cement were estimated by M3 based on metallurgical test work completed to date for the Black Pine materials and are summarized below.

13.11.1 Cyanide Consumption

Cyanide consumptions were averaged for the laboratory column leach tests and ranged from 0.29 to 1.60 kg/t and averaged 0.65 kg/t. Crushed rock laboratory column leach cyanide consumptions are much higher than experienced in commercial scale ROM heap leach operations. Standard cyanide consumption adjustment from the laboratory to ROM heap leaching ranges from 25% (0.163 kg/t) to 33% (0.216 kg/t) of laboratory consumption.

Cyanide consumption for ROM ore is provisionally estimated at 0.18 kg/t.

13.11.2 Lime Consumption

Hydrated lime requirement for pH control ranged from 0.50 to 1.52 kg/t and averaged 0.97 kg/t. Converting to commercial quicklime requirements, 0.88 kg/t is estimated for ROM ore pH control.

13.12 Summary

The QP is of the opinion that samples tested are sufficiently representative to support the conclusions summarized herein. Metallurgical testing is ongoing and is designed in part to continue to evaluate all types and styles of mineralization.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

Mineral Resources for the Black Pine Project were estimated in accordance with NI 43-101 guidelines. The modelling and estimation of the Mineral Resources was completed between October 2023 and June 2024 by or under the supervision of the SLR QP. The Effective Date of the Resource Estimate is June 1, 2024. The Mineral Resource estimate presented here supersedes any previously stated Mineral Resources for the Black Pine Property.

For each of the eight contiguous areas representing the deposit, domains representing gold mineralization were defined in Leapfrog Geo software and sub-block model estimates were completed within Leapfrog Edge software, using 3.048 m (10 ft) capped composites and a multi-pass inverse distance cubed (ID^3) interpolation approach. Blocks were classified considering local drill hole spacing and age. Class groupings were based on criteria developed using continuity models (variograms) and modified to reflect geological understanding and to ensure cohesive classification shapes. Area block models were stitched together and reblocked prior to pit optimization in whittle.

Wireframe and block model validation procedures including wireframe to block volume confirmation, statistical comparisons with composite and nearest neighbor (NN) estimates, swath plots, visual reviews in 3D, longitudinal, cross section, and plan views were completed for all zones.

The Black Pine Project Mineral Resource estimate is presented in Table 14-1.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 14-1: Summary of Mineral Resources for the Black Pine Project – January 21, 2023

Classification	Tonnage Mt	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	402.6	0.32	4,163
Inferred	97.7	0.23	712

Notes:

1. CIM Standards definitions were followed for Mineral Resources.
2. Bulk density is variable by rock type.
3. Mineral Resources are reported within conceptual open pits estimated at a gold cut-off grade of 0.10 g/t, using the PFS pit slope parameters, a long-term gold price of US$2,000 per ounce and the PFS variable gold leach recovery model derived from extensive metallurgical studies.
4. All gold mineralized material falling outside the conceptual open pits and carbonaceous material is considered waste rock and is excluded from resource classification.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6. Mineral Resources are inclusive of Mineral Reserves.
7. Rounding may result in apparent discrepancies between tonnes, grades, and contained gold content.

14.2 Comparison to Previous Resources

While the gold estimate is unchanged from the previous Mineral Resource estimate (effective February 15, 2024), SLR has updated several economic and technical considerations following studies associated with the PFS work. Key changes relative to the previous Mineral Resource estimate are:

- Updated metallurgical recovery model for gold.
- A decrease in the gold cut-off grade from 0.2 g/t to 0.1 g/t and an increase in gold price from $1,800/oz to $2,000/oz.
- Small changes to mining cost inputs during pit optimization to align with PFS work

- Small class revisions to low-grade (<0.2 g/t Au) material.

Changes are presented in Table 14-2.

Table 14-2: Comparison of Previous and Current Mineral Resource Estimates

Class	February 15, 2024			June 1, 2024			Difference		
	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (koz Au)	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (koz Au)	% Difference Tonnage	% Difference Grade	% Difference Contained Metal
Indicated	203,771	0.49	3,206	402,564	0.32	4,163	97.56%	-34.69%	29.85%
Inferred	24,085	0.42	325	97,680	0.23	712	305.56%	-45.24%	119.08%

14.3 Mineral Resource Cut-Off Grade

A gold cut-off grade of 0.1 g/t was developed for the Black Pine deposit and reflects assumed mining costs of a conventional open pit mine with typical heap leach processing costs. The full operating cost, including mining, processing, and G&A costs have been used in the calculations. Capital costs, including sustaining capital, have been excluded. Table 14-3 lists the parameters used to calculate the cut-off grade.

Table 14-3: Black Pine Resource Cut-Off Grade Summary

Parameter	Units	Black Pine
Gold Price	US$/oz	2,000
Dore Freight, Security & Insurance	$/oz produced	2.2
Royalties (0.5% of Au Price)	$/oz produced	50
Total Selling Cost	$/oz produced	47.2
Processing Gold Recovery[1]	%	varies
Process Cost	$/t leached	1.83
Site General Cost	$/t leached	0.8
Total Process and Site Cost	$/t leached	2.63
Mining Cost	$/t mined	2.35
Revenue Factor		1.053

Notes:

1. The metallurgical recoveries vary from 63% - 83% based on the rock type and Au grade.
2. For all reporting purposes, a marginal cut-off grade of 0.10 g/t Au is used
3. The Selling cost of $47.2/oz used assumes a Gold price of $1800/oz. The incremental change in selling cost of $5.0/oz at a Gold price of $2,000/oz does not have any material impact in the resource declaration.

14.4 Resource Database

Within the Black Pine property, Liberty Gold and its predecessors have completed 2,883 holes (1,854 historical RC holes, 26 historical core holes, and 970 RC and 33 core holes drilled by Liberty Gold). Various data and interpretations derived from these holes, as well as digital surfaces of the Project area, were provided to SLR by Liberty Gold. The Project drill-hole database is in UTM Zone 12 NAD83 coordinates (in meters). Drill hole spacing ranges from an average of 30 meters in the center of the Discovery, CD, E, and F areas, to an average of 50 meters in the Backrange, J, M, and Rangefront areas. Drill hole spacing is up to 200 meters in the peripheral areas of the Project. Historical drill holes are generally vertical in orientation, with many of the Liberty Gold holes angled in a fanned arrangement to minimize ground disturbance from drill pads.

A total of 279 historical drill holes were identified as including some composited sample intervals of up to 6.096 m (20 ft). This practice is understood to have been executed as a cost-saving measure whereby composited 20 ft long samples (from four, 5 ft samples) were submitted for gold cyanide bottle roll leach tests and selectively replaced with the original 1.524 m (5 ft) samples analyzed by gold fire assay upon return of the composited gold grade above a certain threshold. Where maintained in the database, long samples are therefore typically low grade. More thorough discussion regarding these assays can be found in Gustin, et. al. (2021).

To reduce the influence of these long, lower precision, sample intervals in areas characterized by new drilling, but acknowledging that where present, they distinguish areas of lower grade, SLR has adopted an approach in which the original values are maintained in the supporting assay database when further than 25 m from a modern drill hole. Where within 25 m of a modern drill hole, these long, composited samples have been ignored. This has resulted in 1,061 assays being ignored from the compositing (and estimation) process. Table 14-4 describes the drilling and meterage used for the Mineral Resource estimate for gold.

Table 14-4: Summary of Drill Holes supporting the Block Model Estimation by Company

Company	Core Holes	Meters (m)	RC Holes	Meters (m)
Gold Resources	3	135	13	1,083
Newmont Mining			37	3,119
Noranda			520	50,243
Pegasus Mining	15	1,062	1,122	115,522
Pioneer Nuclear			28	2,458
Western Pacific Resources			35	7,219
Historic Total	18	1,197	1,755	179,644
Liberty Gold	31	6,864	966	231,367
Project Total	49	8,061	2,721	411,011

To historical entries returning gold values at detection limit, SLR assigned a value of 0.017 g/t Au regardless of the detection limit for the analysis at the time, reducing the gold grade in approximately 43,000 entries (see section 9.1). Unsampled intervals deemed barren by visual inspection were assigned a gold grade of 0 g/t. Missing intervals were ignored.

14.4.1 Silver

Silver mineralization is present in low concentrations at Black Pine from at least two silver mineralizing events. The dominant, and more extensive silver event correlates with the stratabound gold event while an earlier silver event correlates well with base metals such as Pb, Zn, Cd and appears to be at higher angles to the stratabound silver.

The density of samples submitted for silver analysis at the Project is lower than for gold, and while many samples over the Project were analyzed using a 30 g fire assay (“FA”) test, the common analytical technique applied, aqua-regia digestion with AA finish, is not considered a total digestion for silver, nor sufficiently precise for use in Mineral Resource estimation at the concentration levels typical for Black Pine. A comparison of fire assay and aqua-regia sample results are presented in Section 24.1 and show a low bias for the aqua-regia results. For these reasons, silver has not been included in the Mineral Resource estimate, however, silver was produced as a by-product historically, and there is potential for recovery of small concentrations of silver alongside gold during mining. To help understand the potential for silver at Black Pine, silver has been estimated alongside gold, using the gold estimation domains, which are stratabound. Table 14-5 lists the drilling and meterage used to inform the silver estimate.

Table 14-5: Summary of Drill Holes Supporting the Block Model Silver Estimation by Company

Company	Core Holes	Meters (m)	RC Holes	Meters (m)	Ag Analytical Technique
Gold Resources	3	135	12	1,065	4-acid FA
Newmont Mining			32	2,767	4-acid FA
Noranda			1	91	4-acid FA
Pegasus Mining	9	592	851	87,702	4-acid FA
Pioneer Nuclear			3	271	4-acid FA
Western Pacific Resources			8	2,056	2-acid AA
Historic Total	12	727	907	93,952	
Liberty Gold	31	6,864	966	231,367	2-acid AA
Project Total	43	7,591	1,873	325,319	

14.5 Material Type, Geological, and Mineralization Models

14.5.1 Geological Model

Gold at Black Pine occurs primarily as stratabound mineralization that almost exclusively occurs within the middle structural-plate units, except for Rangefront, which hosts mineralization in the upper plate. In aggregate, favorable host stratigraphy of the middle plate comprise a gently east-dipping section of Pennsylvanian carbonate rocks up to 400 m thick that is extensively folded and faulted. The mineralization often occurs at, or subparallel to, stratigraphic contacts, along which strata-parallel structural movement of uncertain extent is evident. While less common, local examples of solution breccia-hosted mineralization are also considered a mineralization control.

Gold is distributed throughout the middle structural plate, but the most extensive mineralization is focused within more favorable stratigraphic units, such as calcareous siltstones in association with decalcification and/or low- to moderately dipping faults.

Liberty Gold provided SLR with digital fault surfaces and lithologic wireframe solids of the various units in the upper, lower, and middle structural plates, all of which were created using Leapfrog software with extensive explicit controls applied. This geological modelling covers the full extents of the mineralization at Black Pine. The digital surfaces and solids were used extensively as guides for the modelling of the mineral domains that served as the primary constraint on the estimation of project Mineral Resources. SLR reviewed and accepted the geologic model provided.

The geological model forms the backdrop and context for the mineralization domains discussed in Section 14.5.2 below. Three units within the geological model are used as domains for gold estimation:

1. Backfill and Waste Dumps (see Section 14.5.6) which are insitu material remobilized during historical mining activities.
2. Quaternary (“QAL”) Unit: This unit comprises both original insitu overburden material, and remobilized material from road and pad creation during operations.

While gold has been interpolated within these units, backfill and waste dumps were downgraded to an Inferred Mineral Resource classification, and the Quaternary unit is currently excluded from the Mineral Resource estimate and is maintained for internal studies.

14.5.2 Mineral Domains

The Project was divided into a total of eight different areas to allow for easier handling of the mineralization domain building process. The areas were chosen based on historical activities and concentrations of drillholes and mineralization. Figure 14-1 shows the different areas across the Project. The 2023 Mineral Resource estimate (Rodney et. al., 2023) had seven areas; in this update, CDF has been subdivided into CD and F. All other area boundaries have remained the same.

Initial mineralization domains were prepared by Liberty Gold geologists, which were reviewed and adopted by SLR, with some minor edits. Within each area, medium grade mineralized domains were developed considering the principal mineralization controls: lithology and structure, and a targeted gold grade of 0.1 g/t. Enclosed within the medium grade domains, high grade domains were built targeting material above 0.5 g/t Au; except Rangefront, which targeted higher grade material above 0.3 g/t Au. These modelling cut-off grades were chosen to reflect the natural grade boundary above which continuous higher-grade material could be captured. Low grade material (just below, equal to, or just above the targeted gold grade) was modelled using Leapfrog Geo's Indicator Interpolant function and overprint the medium grade domains. More information about the low-grade domains is in Section 14.5.5.

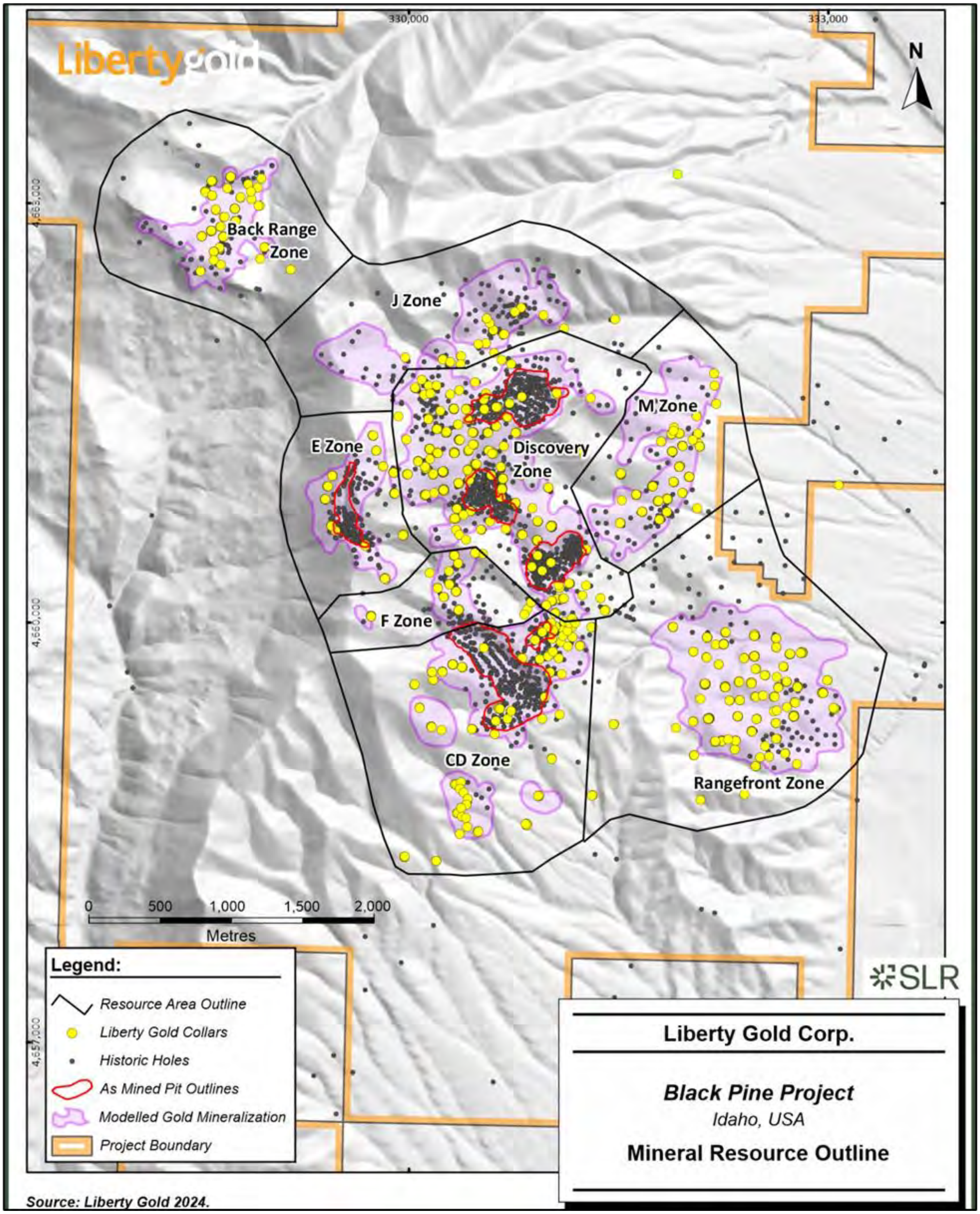


Figure 14-1: Black Pine Project Mineral Resource Outline

While the modelled mineralization overwhelmingly lies within the bounds of the middle structural-lithological plate, minor volumes of the mineral domains were also modelled within the upper and lower plates, close to their structural contacts with the middle plate.

The high-grade domains form relatively thin, elongated zones that typically lie along or close to lithologic contacts and/or low-angle structures that are parallel or transgress lithologic boundaries at acute angles. In strongly mineralized areas, this domain has strong continuity both along strike and dip. Medium-grade mineralization both encompasses the higher-grade domains and extends outwards and terminates along the same low-angle structures and lithologic contacts.

The modelled mineralization at Black Pine extends discontinuously over a northwest extent of about 6,300 m, a maximum northeast-southwest extent of 3,700 m, and an elevation range of 1,100 m, although the maximum true width of the mineralization is approximately 400 m.

A total of 269 individual mineralized domain wireframes were created for the Project (Table 14-6). Cross-sections showing examples of the gold mineral-domain modelling are shown in Figure 14-2 and Figure 14-3. In addition to these, domains, low grade domains described in Section 14.5.5, wireframes representing the Qal unit from the geological model described in Section 7, and wireframes representing the backfill and waste dumps described in Section 14.5.6 were also used as domains in the estimation process.

Table 14-6: Black Pine Project Mineralization Domain Wireframes

Area	Number of Medium Grade Wireframes	Number of High-Grade Wireframes
Disco	13	43
Rangefront	8	15
CD	17	54
F	3	8
E Pit	15	15
M Zone	11	14
J Zone	12	9
Backrange	16	16
Total Medium and High-Grade Wireframes	269	

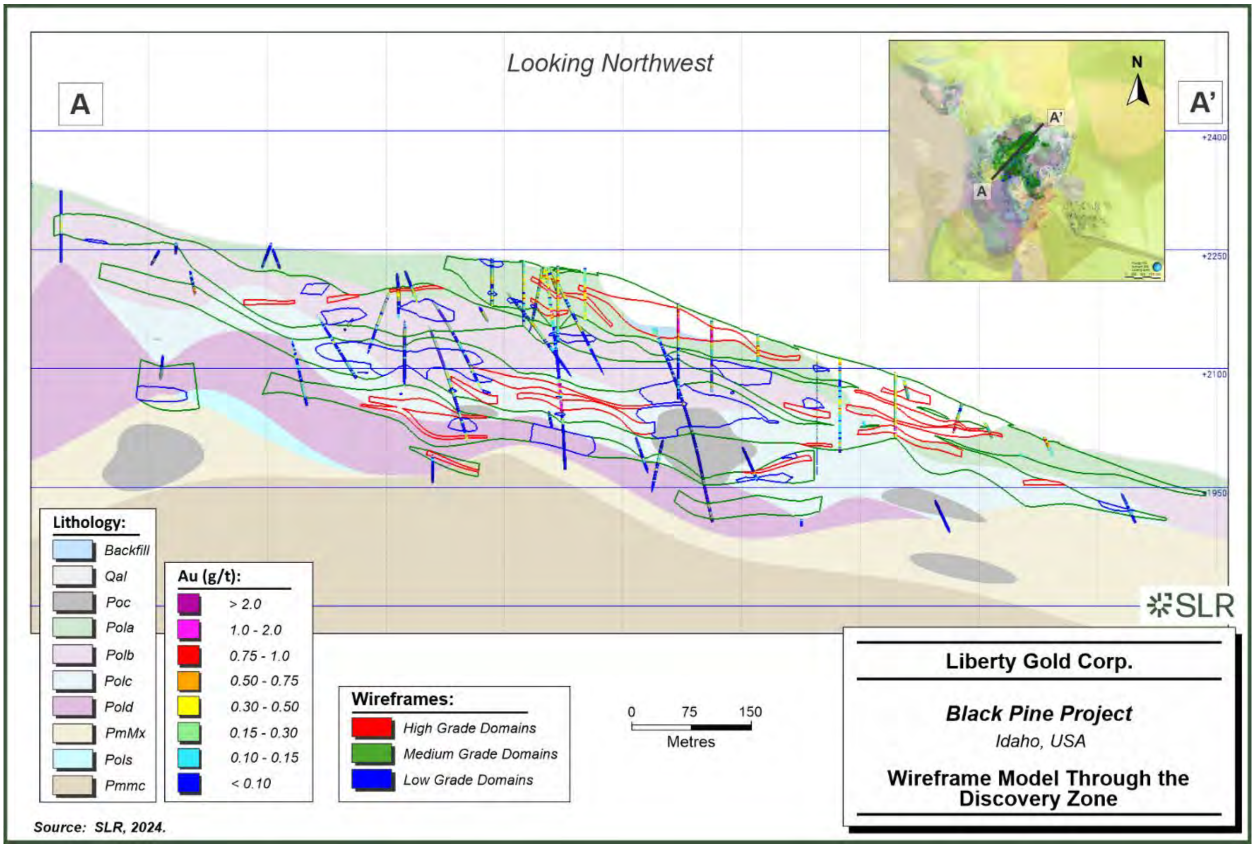


Figure 14-2: Wireframe Model through Discovery Zone

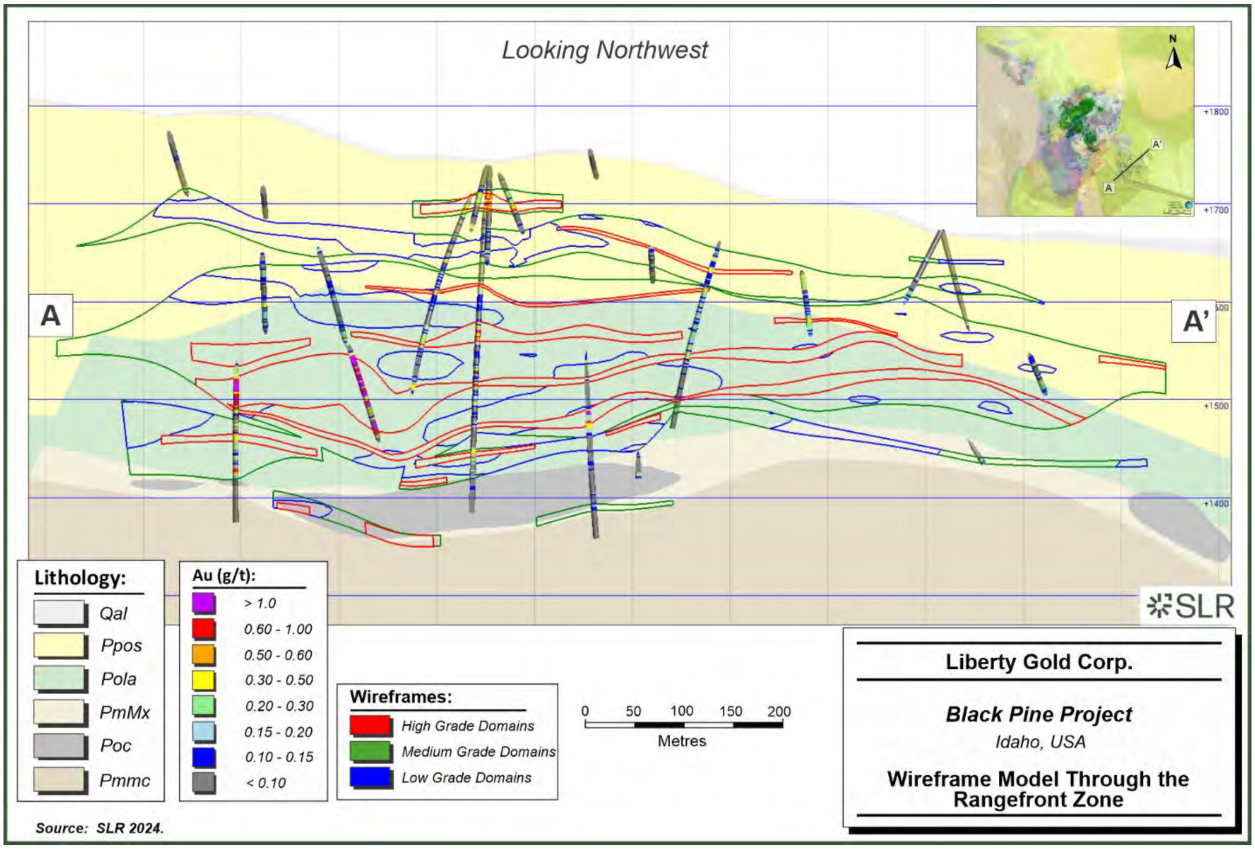


Figure 14-3: Wireframe Model through Rangefront Zone

14.5.3 Oxidation and Modeling of Carbonaceous Material

Gold mineralization intersected by drilling is overwhelmingly oxidized, although irregularly distributed bodies of unoxidized mineralization do occur in various middle-plate stratigraphic units and are typically associated with zones of black carbonaceous shale or siltstone. These zones are characterized by very low cyanide-soluble to fire-assay gold ratios and current metallurgical test work confirms that these materials are preg-robbing, consistent with historical reports from Pegasus' mining operation. While typically of limited extent within mineralized zones, much larger bodies of unoxidized carbonaceous rock have been intersected by drillholes in areas adjacent to mineralized zones. Exposures of carbonaceous rock can be seen in the walls of some of the historical open pits.

The unoxidized carbonaceous zones were modelled by Liberty Gold as part of the geological model created in Leapfrog as solids that were used to code recovery in the resource block model. All areas identified by these solids were coded with 0.001% recovery prior to pit optimization and the areas are excluded from the Mineral Resource statement.

Some areas of mineralization, most notably in the Polb unit, show cyanide gold solubilities that are somewhat lower than the adjacent Pola and Polc units and have little to no associated sulphides or carbonaceous materials. At present, these low-cyanide-solubility occurrences are hypothesized to be related to the presence of clays that inhibit the extraction of gold, indicated by the cyanide shake-leach analyses performed routinely by ALS on the drill samples. While these occurrences are not explicitly modelled, they do influence the metallurgical recoveries of the Polb unit that are coded into the model. Figure 14-4 shows the carbon (Poc, modelled in gray) along with the high- and low-grade domains, and the lithology model.

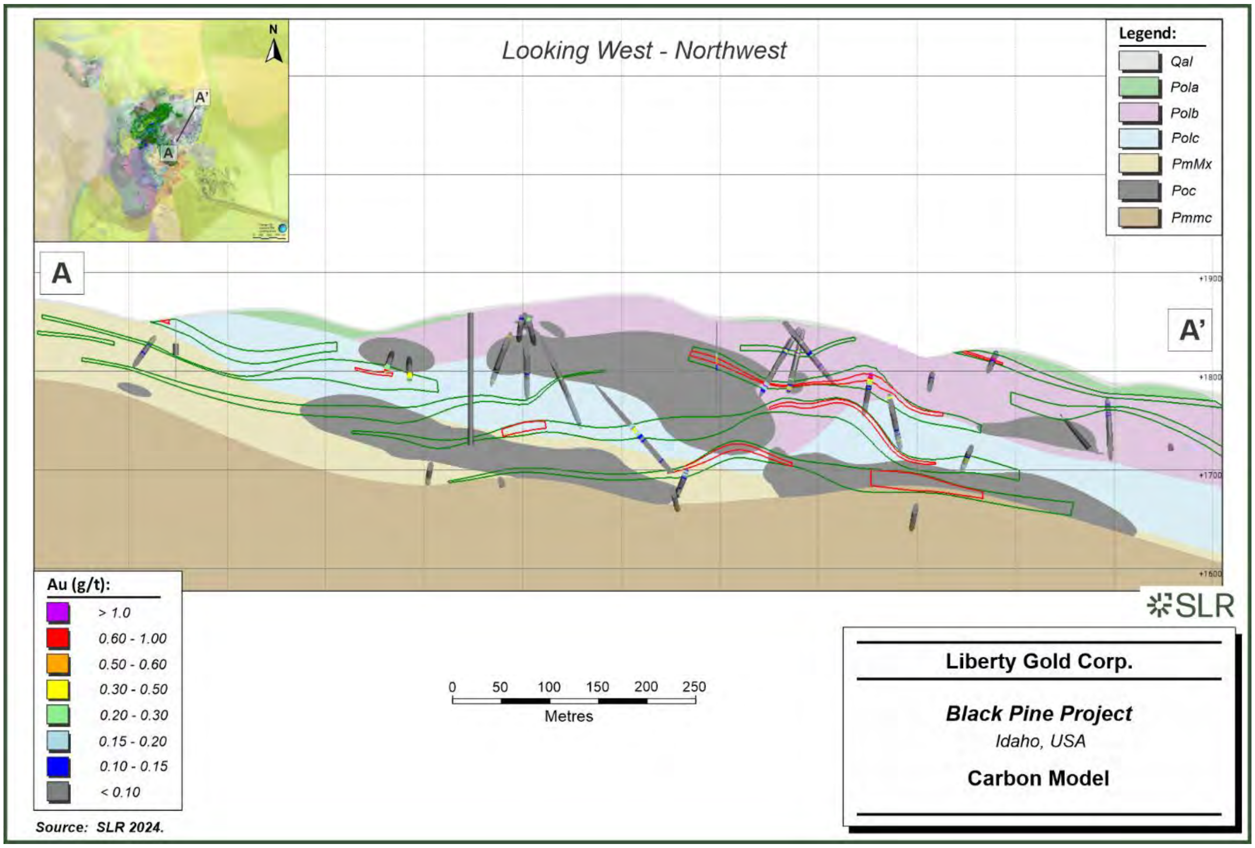


Figure 14-4: Lithology Model Showing Carbonaceous Zones (Poc)

14.5.4 Geometallurgical Model

At Black Pine, 8 historic and 176 Liberty Gold column tests across the deposit inform recovery equations partitioned by deposit area, formation, gold grade and Met domain (Met 1-4) and defined by gold cyanide solubility or the ratio of drill assay gold cyanide leach to gold fire assay. Liberty Gold drilling at Black Pine has generated approximately 267,000 cyanide leach assay and fire assay pairs from over 1,000 drill holes. However, due to a handling error, approximately 69,700 cyanide leach assays were contaminated, resulting in database gaps and reduced confidence in metallurgical domain modelling. SLR worked closely with Liberty Gold geologists and metallurgists to use machine learning to predict gold cyanide solubilities in drill holes with contaminated or missing data. A Neural Network and Gradient Boosting algorithm was iteratively trained on select drill hole data parameters, primarily adjacent valid gold cyanide leach assays, lithology, and logged carbon intensity. The machine learning model was able to predict the metallurgical domain with a 91% weighted accuracy. The resulting infilled database allowed for metallurgical domain modelling using Leapfrog Geo. These Met domains were used alongside area, gold grade and carbonaceous zone coding in the block model to assign recovery equations developed by the Liberty Gold metallurgist.

Figure 14-5 shows the geometallurgical domains and the cyanide solubility ranges for each domain are listed in Table 14-7.

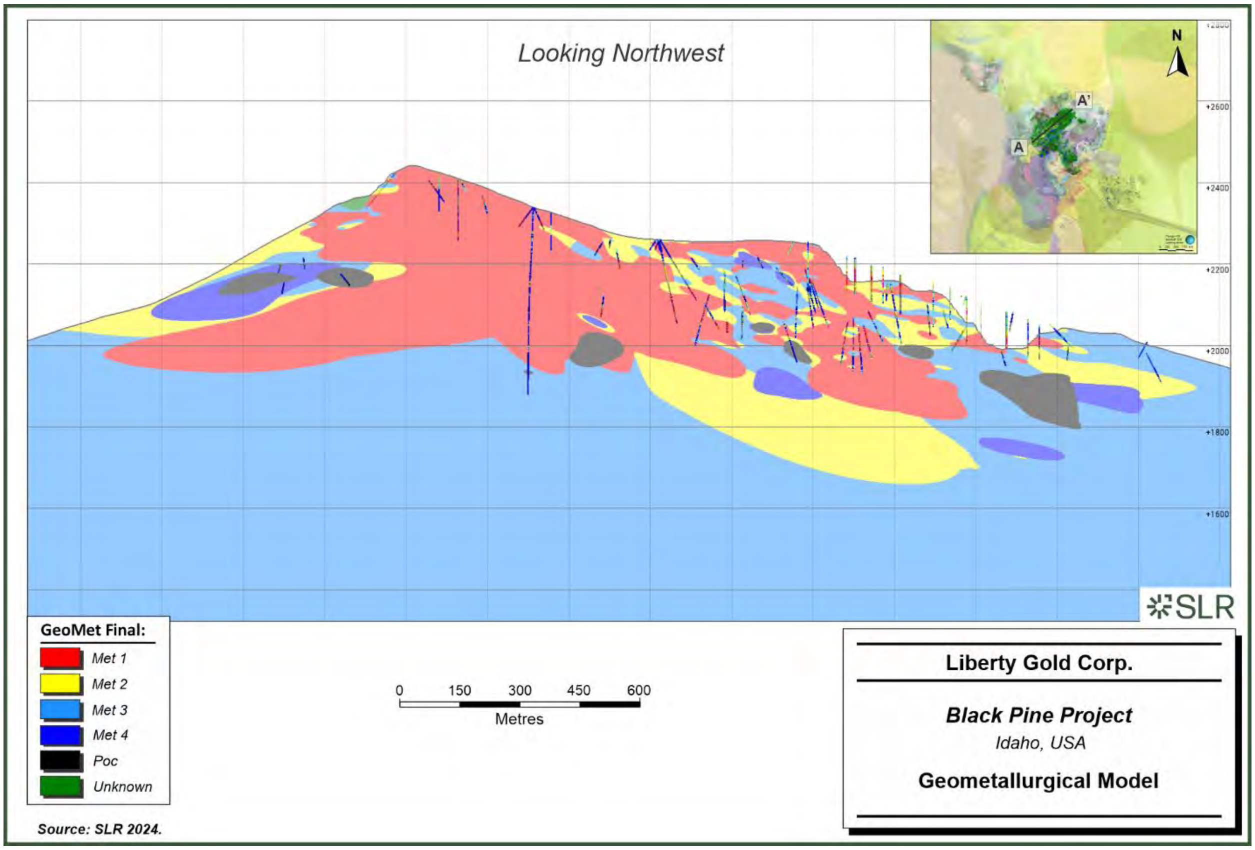


Figure 14-5: Geometallurgical Model

Table 14-7: Black Pine Project Geometallurgical Domains

Domain Name	Category	Cyanide Solubility (% AuCN)
Met 1	Oxide	65 > %AuCN
Met 2	High Transition	50 > % AuCN < 65
Met 3	Low Transition	25 > % AuCN < 50
Met 4	Non-Leach	% AuCN < 25

14.5.5 Low Grade Indicator Domains

A common concern with low grade disseminated gold deposits is the risk of samples slightly above or slightly below the cut-off grade being used together during interpolation. This can cause the amount of material that is equal to, or slightly above the cut-off, to be overstated in the estimate. To mitigate against this risk, SLR used the ‘Indicator RBF Interpolant’ tool within Leapfrog Geo software to create low grade domains overprinting the medium (above cut-off) grade domains. The resultant shapes were estimated separately from the mineralized domains using composites with very low gold and silver cap values. These low-grade domains further segregate the just-below cut-off grade samples and volumes and prevent smearing of the higher grades. Figure 14-6 shows an example of the low-grade indicator domains contained within the medium grade domains.

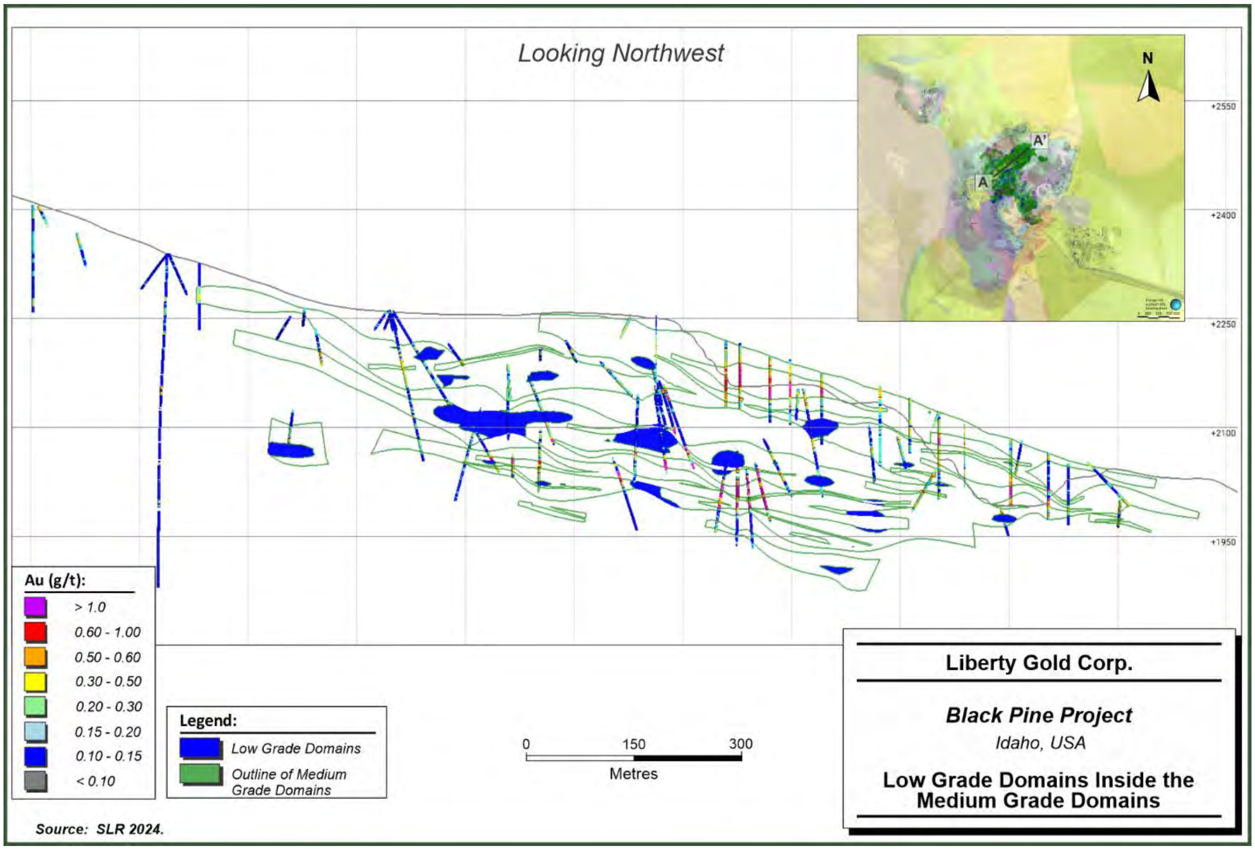


Figure 14-6: Low Grade Indicator Domains within the Medium Grade Domains

14.5.6 Backfill and Waste Dumps

Three digital topographic surfaces were provided to SLR: a pre-mining surface; a surface that represents the 'as mined' topography; and a present-day topography. SLR reviewed the topographies with Liberty Gold geologists and agreed that they represent the property well. The as mined topography was used with the present-day topography to accurately create solid volumes which represent the backfill dumps and waste dumps placed by previous operators of the Black Pine mine. The topographies were used to code the block models so that the Mineral Resources were limited to in-situ material, or remobilized material within the backfill dumps and waste dumps.

A drilling program was designed and executed by Liberty Gold to better understand the grade of the material contained withing these remnant dumps. A total of 137 drill holes (115 waste dumps, 60 backfill dumps) were drilled. These drill holes allowed for a grade estimation into the blocks contained within the volumes of the solids provided and coded into the block model. Figure 14-7 shows the waste dumps.

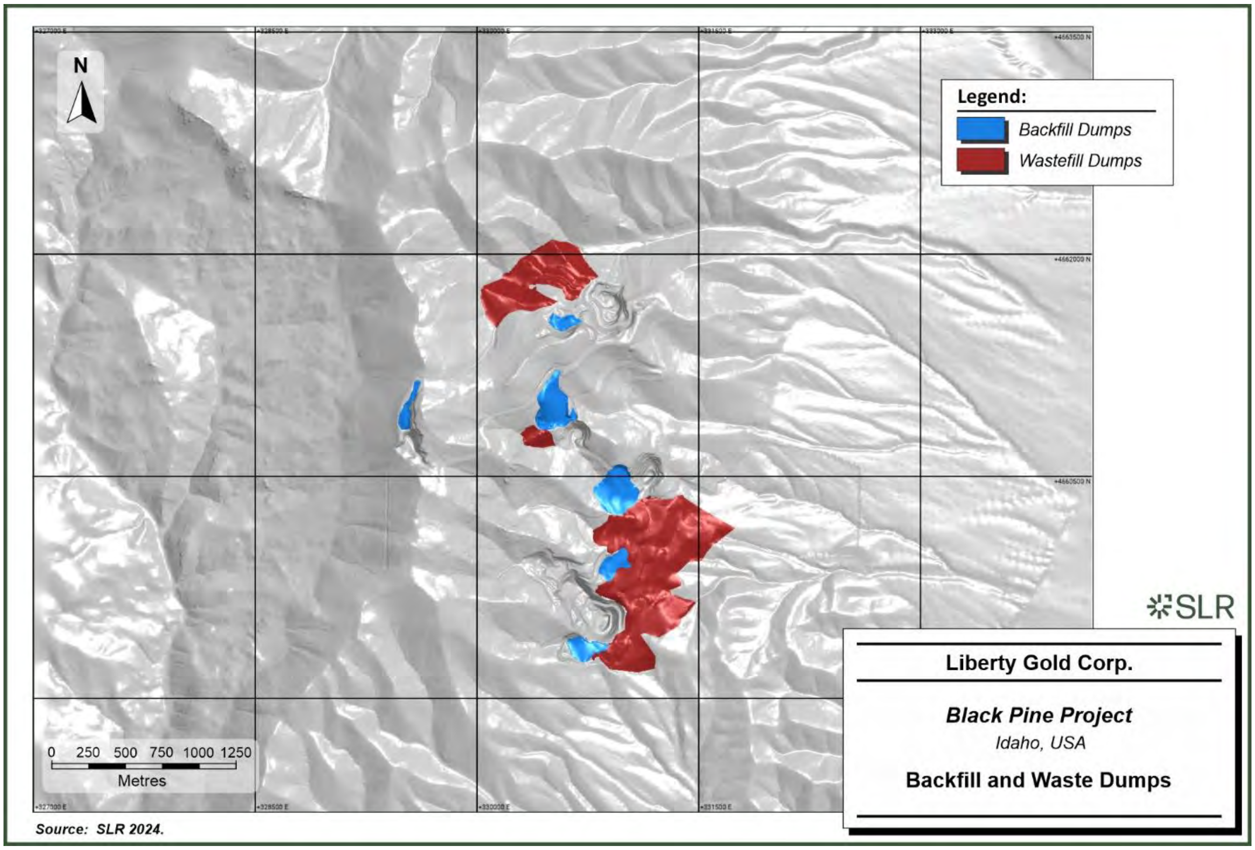


Figure 14-7: Backfill and Waste Dumps

14.6 Capping

Where the assay distribution is skewed positively or approaches log-normal, erratic high grade assay values can have a disproportionate effect on the average grade of a deposit. One method of treating these outliers to reduce their influence on the average grade is to cut or cap them at a specific grade level.

The SLR QP is of the opinion that the influence of high-grade gold and silver assays must be reduced or controlled and uses a number of industry best practice methods to achieve this goal, including capping of high-grade values. SLR employs several statistical analytical methods to determine an appropriate capping value including frequency histograms, probability plots, decile analyses, and capping curves. Using these methodologies, the SLR QP examined the selected capping values for the mineralized domains for the Project.

During this process it was noted that there were two high-grade domains (3200 and 3201) from the E Pit zone which had significantly higher gold assay populations and were analyzed separately from the other high-grade domains within the E Pit zone. Separate high grade gold cap values were applied to domains 3200 and 3201.

Examples of the capping analysis are shown in Figure 14-8, Figure 14-9, and Figure 14-10 and as applied to the data set for the mineralized domains. Capped assay thresholds by high-grade, medium-grade, and low-grade zones within the respective areas are summarized in Table 14-8.

Capped assay statistics by zones are summarized in Table 14-9 and compared with uncapped assay statistics.

In the SLR QP's opinion, the selected capping values are reasonable and have been correctly applied to the raw assay values for the Black Pine Mineral Resource estimate.

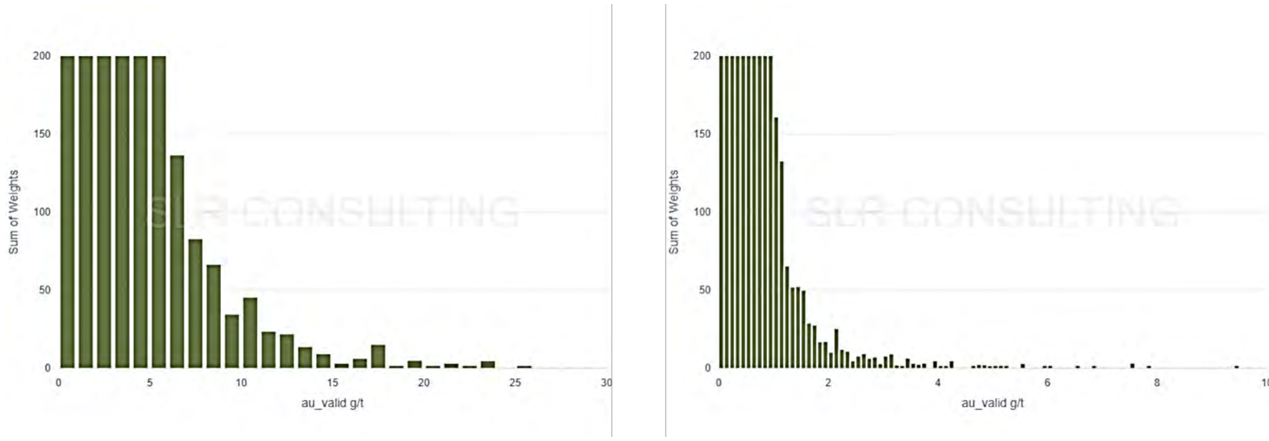


Figure 14-8: Histogram for Discovery Gold Assays (zoomed in on y-axis)

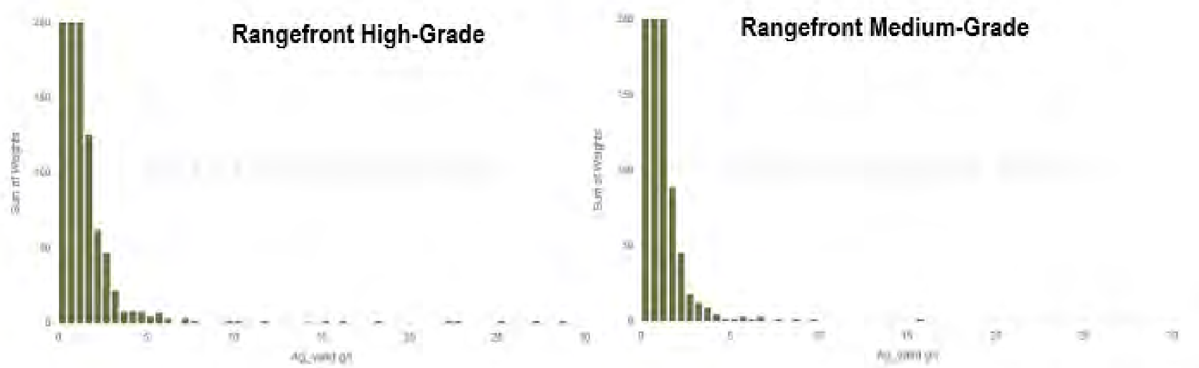


Figure 14-9: Histogram for Rangefront Silver Assay (zoomed in on y-axis)

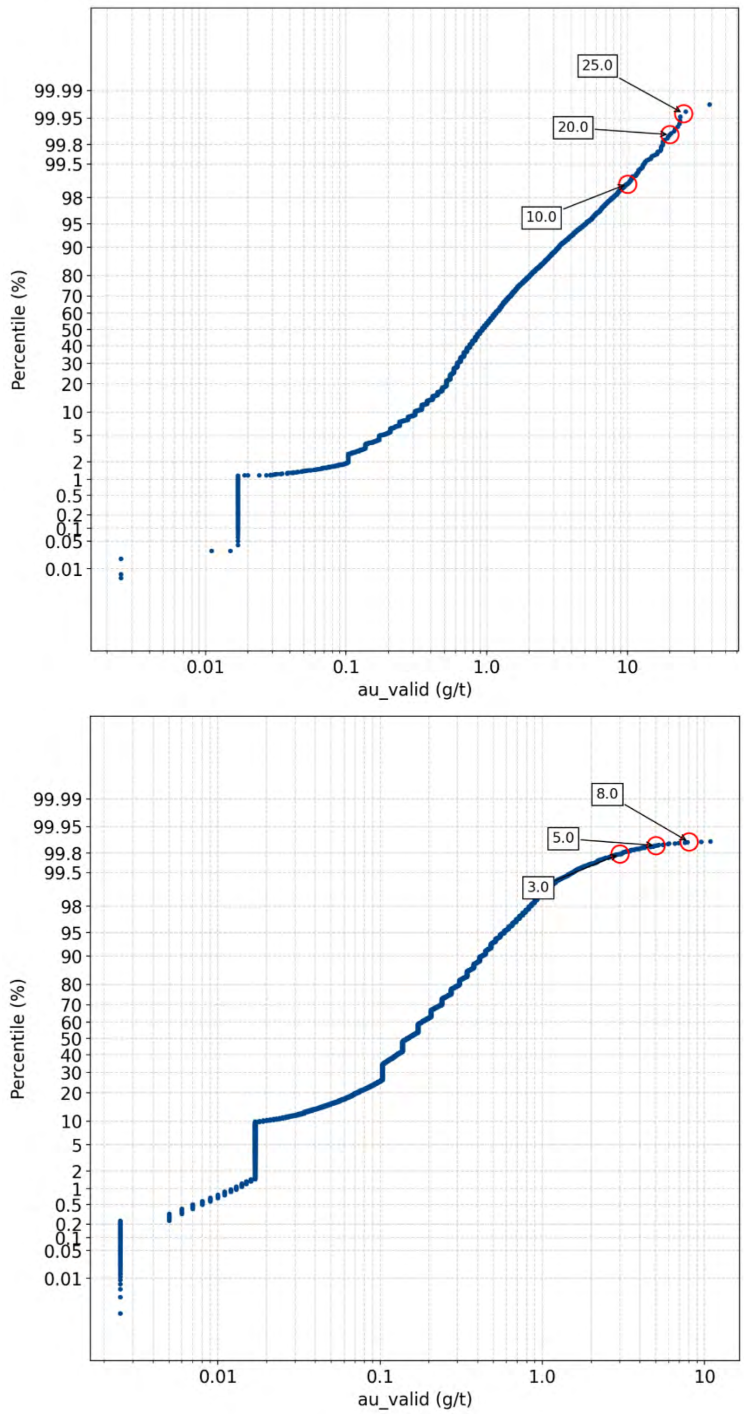


Figure 14-10: Log Probability Plots for Discovery Gold Assays

Table 14-8: Gold and Silver Assay Caps by Domain Group

Area	Au High Grade Cap (g/t)	Ag High Grade Cap (g/t)	Au Medium Grade Cap (g/t)	Ag Medium Grade Cap (g/t)	Au Low Grade Cap (g/t)	Ag Low Grade Cap (g/t)
Back Range	8	25	-	10	1	1
CD	8	25	2	10	1	1
F	8	25	-	10	1	1
Discovery	20	20	2	10	1.5	1
E Pit	5	25	2	10	1	1
E Pit_3201	25	25	5	-	1	1
E Pit_3200	25	25	3	-	1	1
J Zone	-	25	-	10	1	1
M Zone	8	100	-	10	1	1
Rangefront	5	25	2	10	1	1

Notes:

1. 'Qal' was assigned the Low-Grade Cap value for all areas.

Table 14-9: Descriptive Statistics of Uncapped vs Capped Gold Assays

Zone	Min Group	Descriptive Statistics							
		Assays	Number of Samples	Mean (g/t)	CV	Min (g/t)	Median (g/t)	Max (g/t)	Number of Caps
Backrange	HG	Au	640	1.29	1.17	0.02	0.82	12.89	
		Au Cap		1.26	1.06	0.02	0.82	8.00	6
	MG	Au	1,883	0.27	0.93	0.00	0.20	2.20	
		Au Cap		0.27	0.93	0.00	0.20	2.20	-
	LG	Au	393	0.06	1.37	0.01	0.03	0.82	
		Au Cap		0.06	1.37	0.01	0.03	0.82	-
CD	HG	Au	3,287	0.87	1.03	0.00	0.69	25.27	
		Au Cap		0.86	0.85	0.00	0.69	8.00	3
	MG	Au	12,181	0.22	0.60	0.00	0.20	2.43	
		Au Cap		0.22	0.60	0.00	0.20	2.00	2
	LG	Au	3,818	0.05	1.04	0.00	0.02	1.00	
		Au Cap		0.05	1.08	0.00	0.02	2.43	1
F	HG	Au	375	1.15	1.68	0.02	0.66	17.35	
		Au Cap		1.05	1.25	0.02	0.66	8.00	7
	MG	Au	1,595	0.23	0.71	0.01	0.17	1.75	
		Au Cap		0.23	0.71	0.01	0.17	1.75	-
	LG	Au	430	0.06	0.85	0.00	0.04	0.41	
		Au Cap		0.06	0.85	0.00	0.04	0.41	-
Disco	HG	Au	9,085	1.54	1.31	0.00	0.93	38.26	
		Au Cap		1.54	1.28	0.00	0.93	20.00	9
	MG	Au	41,399	0.24	1.10	0.00	0.17	10.83	
		Au Cap		0.24	1.04	0.00	0.17	5.00	1
	LG	Au	9,742	0.07	1.83	0.00	0.04	5.21	
		Au Cap		0.07	1.40	0.00	0.04	1.50	8

Zone	Min Group	Descriptive Statistics							
		Assays	Number of Samples	Mean (g/t)	CV	Min (g/t)	Median (g/t)	Max (g/t)	Number of Caps
E Pit	HG	Au	382	1.13	1.27	0.02	0.76	21.12	
		Au Cap		1.07	0.87	0.02	0.76	5.00	5
E Pit_3200	HG	Au	174	4.56	1.49	0.02	2.19	46.70	
		Au Cap		4.31	1.31	0.02	2.19	25.00	3
E Pit_3201	HG	Au	400	2.96	1.60	0.02	1.20	35.52	
		Au Cap		2.91	1.53	0.02	1.20	25.00	3
E Pit	MG	Au	4,708	0.23	1.09	0.02	0.17	5.86	
		Au Cap		0.23	1.00	0.02	0.17	3.00	8
	LG	Au	1,438	0.05	1.36	0.00	0.02	1.53	
		Au Cap		0.05	1.28	0.00	0.02	1.00	2
J Zone	HG	Au	481	0.96	1.03	0.02	0.72	9.12	
		Au Cap		0.96	1.03	0.02	0.72	9.12	-
	MG	Au	2,219	0.23	1.05	0.02	0.17	4.56	
		Au Cap		0.23	0.94	0.02	0.17	2.00	4
	LG	Au	556	0.05	1.13	0.01	0.02	1.06	
		Au Cap		0.05	1.13	0.01	0.02	1.00	1
M Zone	HG	Au	662	1.55	1.63	0.01	0.98	46.70	
		Au Cap		1.43	1.00	0.01	0.98	8.00	9
	MG	Au	2,389	0.24	1.18	0.01	0.17	5.14	
		Au Cap		0.24	1.11	0.01	0.17	3.00	3
	LG	Au	522	0.07	1.69	0.00	0.04	2.73	
		Au Cap		0.07	1.22	0.00	0.04	1.00	1
Rangefront	HG	Au	4,910	0.63	1.06	0.00	0.43	5.00	
		Au Cap		0.64	1.19	0.00	0.43	11.15	27
	MG	Au	14,789	0.18	0.57	0.00	0.15	1.00	
		Au Cap		0.18	0.59	0.00	0.15	2.38	17
	LG	Au	8,336	0.07	0.70	0.00	0.06	0.97	
		Au Cap		0.07	0.70	0.00	0.06	0.97	-

Table 14-10: Descriptive Statistics of Uncapped vs Capped Silver Assays

Zone	Min Group	Descriptive Statistics							
		Assays	Number of Samples	Mean (g/t)	CV	Min (g/t)	Median (g/t)	Max (g/t)	Number of Caps
Backrange	HG	Ag	624	2.07	1.00	0.32	1.70	39.60	
		Ag Cap		2.04	0.82	0.32	1.70	25.00	1
	MG	Ag	1,756	1.33	0.79	0.10	1.08	18.85	
		Ag Cap		1.32	0.70	0.10	1.08	10.00	2
	LG	Ag	388	1.09	1.44	0.13	0.79	29.30	
		Ag Cap		0.75	0.34	0.13	0.79	1.00	151
CD	HG	Ag	2,678	0.38	3.31	0.00	0.21	36.80	
		Ag Cap		0.37	2.95	0.00	0.21	25.00	3
	MG	Ag	7,827	0.33	4.41	0.00	0.17	68.57	
		Ag Cap		0.31	1.80	0.00	0.17	10.00	4

Zone	Min Group	Descriptive Statistics							
		Assays	Number of Samples	Mean (g/t)	CV	Min (g/t)	Median (g/t)	Max (g/t)	Number of Caps
	LG	Ag	1,992	0.25	0.87	0.00	0.15	1.00	
		Ag Cap		0.35	6.90	0.00	0.15	68.57	36
F	HG	Ag	269	0.91	1.74	0.16	0.51	15.43	
		Ag Cap		0.91	1.74	0.16	0.51	15.43	-
	MG	Ag	1,171	0.70	3.10	0.04	0.40	68.57	
		Ag Cap		0.65	1.27	0.04	0.40	10.00	1
	LG	Ag	321	0.45	0.84	0.08	0.34	2.06	
		Ag Cap		0.41	0.64	0.08	0.34	1.00	39
Disco	HG	Ag	5,152	1.32	3.53	0.05	0.85	241.71	
		Ag Cap		1.24	1.18	0.05	0.85	20.00	5
	MG	Ag	24,055	1.05	5.08	0.01	0.64	458.00	
		Ag Cap		0.93	1.13	0.01	0.64	10.00	99
	LG	Ag	6,415	0.82	2.14	0.02	0.53	229.00	
		Ag Cap		0.58	0.52	0.02	0.53	1.00	1362
E Pit	HG	Ag	256	0.87	1.28	0.03	0.69	12.25	
		Ag Cap		0.87	1.28	0.03	0.69	12.25	-
	MG	Ag	1,778	1.01	2.05	0.03	0.69	68.57	
		Ag Cap		0.95	1.07	0.03	0.69	10.00	7
	LG	Ag	388	0.54	0.95	0.02	0.41	3.43	
		Ag Cap		0.47	0.64	0.02	0.41	1.00	46
J Zone	HG	Ag	346	1.11	1.02	0.10	0.79	12.41	
		Ag Cap		1.11	1.02	0.10	0.79	12.41	-
	MG	Ag	1,292	1.05	1.75	0.06	0.69	45.84	
		Ag Cap		1.01	1.21	0.06	0.69	10.00	6
	LG	Ag	339	0.71	1.18	0.02	0.44	7.61	
		Ag Cap		0.51	0.66	0.02	0.44	1.00	73
M Zone	HG	Ag	648	7.17	3.57	0.10	0.99	369.26	
		Ag Cap		6.49	2.97	0.10	0.99	100.00	2
	MG	Ag	2,203	2.32	2.97	0.01	0.93	100.00	
		Ag Cap		1.65	1.32	0.01	0.93	10.00	8
	LG	Ag	490	0.96	1.13	0.06	0.62	9.60	
		Ag Cap		0.62	0.53	0.06	0.62	1.00	158
Rangefront	HG	Ag	4,702	0.61	2.01	0.02	0.41	25.00	
		Ag Cap		0.65	3.45	0.02	0.41	97.60	14
	MG	Ag	13,881	0.32	1.22	0.01	0.25	10.00	
		Ag Cap		0.53	41.01	0.01	0.25	2468.61	7
	LG	Ag	7,909	0.27	1.32	0.01	0.20	6.86	
		Ag Cap		0.25	0.75	0.01	0.20	1.00	152

14.7 Compositing

Composites were created from the capped, raw assay values using the downhole compositing function of Leapfrog's modelling software package. The composite lengths used during interpolation were chosen considering the predominant sampling length, block height, style of mineralization, and continuity of grade.

The capped assays were composited at 3.048 m (10-foot) down-hole intervals, respecting the mineral domain boundaries. The composite length is equal to twice the average sample length of the assays and is approximately one third of the parent block height (10 m). A small number of unsampled and missing sample intervals were ignored. Residual composites were maintained in the dataset. The gold composite statistics by domain group are shown in Table 14-11 and the silver composite statistics are in Table 14-12.

Table 14-11: Summary of Gold Composite Statistics by Domain Group

Zone	Domain Group	Count	Mean (g/t Au)	CV	Min (g/t Au)	Median (g/t Au)	Max (g/t Au)
Back Range	HG	339	1.27	0.93	0.02	0.89	7.06
	MG	1004	0.27	0.77	0.02	0.21	1.97
	LG	184	0.06	1.02	0.01	0.04	0.42
	Qal	142	0.16	1.55	0.01	0.07	1.71
CD	HG	1,821	0.88	0.73	0.02	0.70	6.10
	MG	6,000	0.23	0.49	0.02	0.21	1.51
	LG	1648	0.05	0.66	0.00	0.06	0.45
	Qal	548	0.14	1.32	0.00	0.08	1.85
F	HG	208	1.05	1.14	0.08	0.69	8.00
	MG	847	0.23	0.59	0.02	0.19	1.15
	LG	198	0.06	0.58	0.01	0.06	0.21
	Qal	75	0.14	1.00	0.01	0.08	1.19
Disco	HG	4,791	1.55	1.16	0.00	0.99	20.00
	MG	21,189	0.24	0.90	0.00	0.19	5.00
	LG	4,510	0.06	1.08	0.00	0.05	1.50
	Qal	538	0.15	1.54	0.01	0.10	3.67
E Pit	HG	209	1.08	0.74	0.02	0.79	4.35
	MG	2413	0.23	0.83	0.02	0.17	3.00
	LG	685	0.05	0.84	0.01	0.04	0.42
	Qal	222	0.18	1.97	0.01	0.10	3.00
E Pit_3200	HG	84	4.28	1.24	0.02	2.37	24.33
E Pit_3201	HG	212	2.87	1.39	0.02	1.35	25.00
J Zone	HG	254	0.96	0.82	0.07	0.75	5.68
	MG	1,166	0.23	0.80	0.02	0.17	1.88
	LG	278	0.05	0.70	0.01	0.05	0.26
	Qal	150	0.13	1.14	0.01	0.09	2.00
M Zone	HG	363	1.43	0.86	0.03	1.03	8.00
	MG	1,269	0.24	0.90	0.02	0.18	2.22
	LG	228	0.07	0.86	0.01	0.06	0.59
	Qal	321	0.06	0.70	0.00	0.05	0.45
Rangefront	HG	2589	0.63	0.96	0.01	0.45	5.00
	MG	7315	0.18	0.45	0.02	0.16	1.00
	LG	3833	0.07	0.52	0.00	0.07	0.35
	Qal	600	0.07	1.06	0.00	0.06	0.95

Table 14-12: Summary of Capped Silver Composite Statistics by Domain Group

Zone	Domain Group	Count	Mean (g/t Ag)	CV	Min (g/t Ag)	Median (g/t Ag)	Max (g/t Ag)
Back Range	HG	331	2.05	0.73	0.34	1.66	25.00
	MG	930	1.32	0.62	0.24	1.11	7.20
	LG	174	0.75	0.30	0.26	0.81	1.00
	Qal	139	1.22	0.73	0.15	0.98	7.13
CD	HG	1,491	0.38	2.69	0.00	0.21	25.00
	MG	4,303	0.31	1.69	0.00	0.19	10.00
	LG	910	0.25	0.84	0.00	0.17	1.00
	Qal	377	0.36	2.22	0.00	0.22	10.00
F	HG	148	0.96	1.76	0.16	0.50	15.43
	MG	592	0.67	1.27	0.04	0.42	10.00
	LG	133	0.41	0.58	0.10	0.36	1.00
	Qal	54	0.44	0.93	0.00	0.31	2.06
Disco	HG	2,716	1.24	1.12	0.08	0.87	20.00
	MG	12,326	0.93	1.06	0.01	0.65	10.00
	LG	2,898	0.58	0.49	0.04	0.55	1.00
	Qal	294	0.75	0.83	0.08	0.58	5.83
E Pit	HG	285	0.92	1.06	0.07	0.69	12.25
	MG	1007	0.94	1.11	0.07	0.70	10.00
	LG	176	0.49	0.58	0.03	0.47	1.00
	Qal	83	0.97	0.69	0.09	0.79	3.50
J Zone	HG	185	1.10	0.85	0.17	0.84	7.61
	MG	659	1.00	1.13	0.07	0.69	10.00
	LG	163	0.51	0.62	0.04	0.46	1.00
	Qal	93	0.81	1.10	0.10	0.63	5.45
M Zone	HG	353	6.46	2.90	0.10	1.03	100.00
	MG	1,163	1.64	1.27	0.09	0.96	10.00
	LG	213	0.62	0.51	0.10	0.63	1.00
	Qal	255	0.58	1.66	0.10	0.34	10.00
Rangefront	HG	2461	0.61	1.82	0.06	0.43	25.00
	MG	6814	0.32	1.17	0.03	0.26	10.00
	LG	3627	0.25	0.70	0.03	0.20	1.00
	Qal	460	0.25	1.32	0.02	0.21	5.07

14.8 Trend Analysis

SLR generated experimental semi-variograms using the 3.048 m gold composited values located within selected individual mineralized domains. The major and semi-major directions were fit in the plane of the mineralization, which was defined with consideration to a series of progressively higher gold grade shells generated within each domain. Experimental semi-variograms were fit with a nugget and one to two structures as required. Downhole variograms were used to model the nugget effect and to fit the across-strike variogram models. The variograms were used to support search ellipsoid distances, trend analysis, and Mineral Resource classification decisions.

Figure 14-11 shows the experimental variogram and model for the “203” low-grade domain within the Discovery Zone.

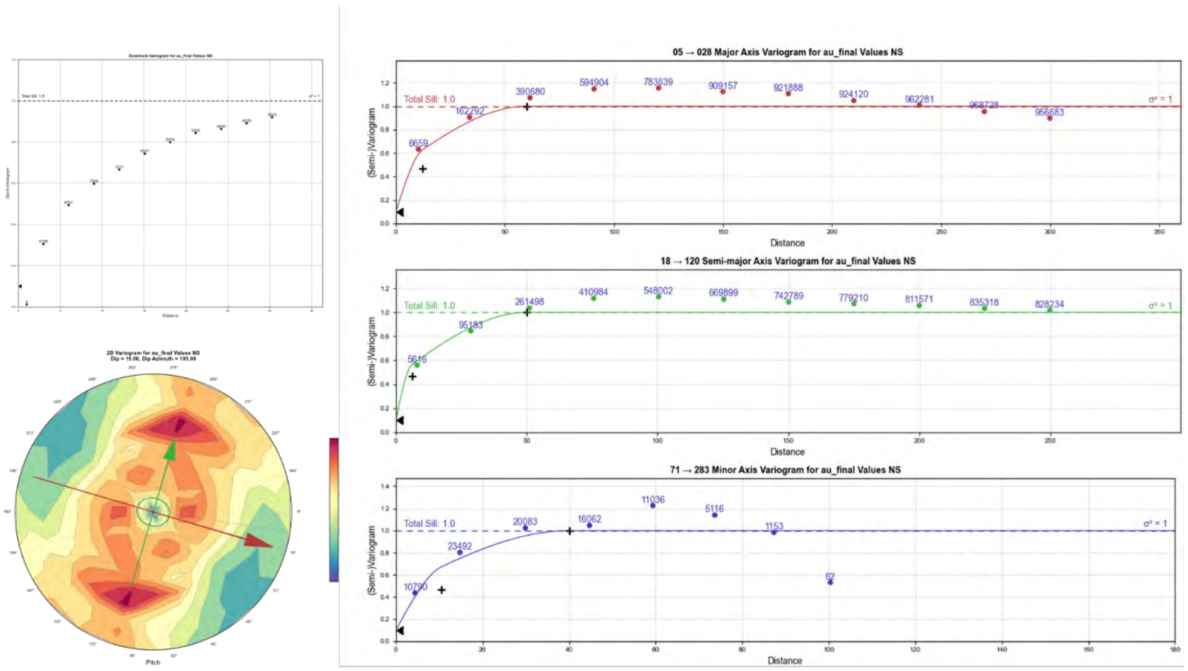


Figure 14-11: Variogram Model – Discovery Low Grade Domain 203

14.9 Search Strategy and Grade Interpolation

The grade interpolation approach used ID^3 and three passes of increasing search ellipse dimensions and relaxed composite restrictions to estimate gold and silver grades into the block models. The medium grade domains used the same interpolation parameters as the medium and high-grade domains, except a soft boundary with a distance of 5 meters was used for the low grade, while hard boundaries were used for the medium and high grade.

Search ellipses for grade interpolation were oriented using dynamic anisotropy, with the longest axis parallel to the strike of gold mineralization and in line with grade trends, and a narrow across strike dimension to limit lateral smoothing Search distances increased with each pass and are described in Table 14-13. An NN estimation (using 5 m composites) as well as an ID^2 were prepared for comparison purposes. In SLR's opinion, the estimation strategies are appropriate for this type of deposit.

Table 14-13: Grade Interpolation Parameters

Type	Pass	Ellipse Size (m)	Min Comps	Max. Comps	Max per DH
Mineralized Domain	1	80x80x10	3	12	2
	2	160x160x15	3	12	2
	3	320x320x20	1	12	2
Backfill/Waste Dump	1	60x60x10	3	20	2
	2	90x90x15	3	20	2
	3	90x90x15	1	20	2

14.10 Block Models

A total of seven sub-blocked block models were created across the Black Pine Project to allow for easier data handling. CD and F are separate areas, but are adjacent, and were estimated within the same block model. After the models were finalized, one large sub-blocked model was created through the combining of the models using Geovia Surpac software. The output from Surpac was imported to Maptek Vulcan software where it was reblocked and exported as a Vulcan block model in CSV format. The regularized block model was imported to Whittle software for pit optimization, and then back into Leapfrog Geo software for reporting. Table 14-14 describes the model dimensions for all the models, including the combined final model. SLR considers the block model sizes to be appropriate for the mining methods and dip of the mineralized zones.

Table 14-14: Black Pine Block Dimensions and Origins

Deposit	Block Model Type	Parent (sub-block) Block Size			Origin			Rotation Z-axis (°)
		X-axis (m)	Y-axis (m)	Z-axis (m)	X-axis (m)	Y-axis (m)	Z-axis (m)	
Back Range	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,662,300	327,900	1,900	0
CD, F	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,658,400	329,500	1,500	0
Discovery	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,659,900	329,800	1,700	0
E Pit	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,660,200	329,100	2,100	0
J Zone	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,661,500	329,100	1,700	0
M Zone	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,660,400	331,200	1,500	0
Rangefront	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,658,800	331,500	1,250	0
Combined Final	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	4,658,400	327,900	1,250	0
Combined Final	Re-Blocked	10	10	5	4,658,400	327,900	1,250	0

14.11 Bulk Density

Bulk density measurements were performed on a total of 1,293 core samples to understand the various densities of the Project. Samples were taken from waste and mineralized intervals in diamond drill holes drilled in 2019 through 2023 from across the project. The data indicates that the densities are influenced primarily by lithology, with secondary effects related to the intensity of gold mineralization. Drill hole logging entries indicate that gold grades tend to increase as porosity increases due to decalcification of receptive lithologies, and decalcification can lead to measurable decreases in densities.

Table 14-15 lists the lithologic units coded into the resource block model and their assigned density values.

Table 14-15: Block Model Densities by Lithology and Gold Domain

Lith Unit	Waste Domain (g/cm^3)	Waste Domain Count	Mineralized Domain (g/cm^3)	Mineralized Domain Count
Poc	2.43	44	2.45	16
PPos	2.46	29	2.44	65
Pola	2.53	35	2.48	238
Polb	2.43	86	2.43	139
Polc	2.53	51	2.44	180
Pold	2.62	83	2.57	148
Pols	2.59	20	2.50	114
PmMx	2.59	29	2.53	13
PMmc	2.63	2	2.6	-
QAL	1.8	-	1.8	-
Waste	1.8	-	1.8	-
Backfill	1.8	-	1.8	-

14.12 Reasonable Prospects of Eventual Economic Extraction

A cut-off grade of 0.10 g/t Au was selected for reporting Mineral Resources to account for uncertainties in the inputs to the cut-off grade calculation. Figure 14-12 shows the conceptual pit shells with a 0.10 g/t Au grade cut-off in relation to the block model.

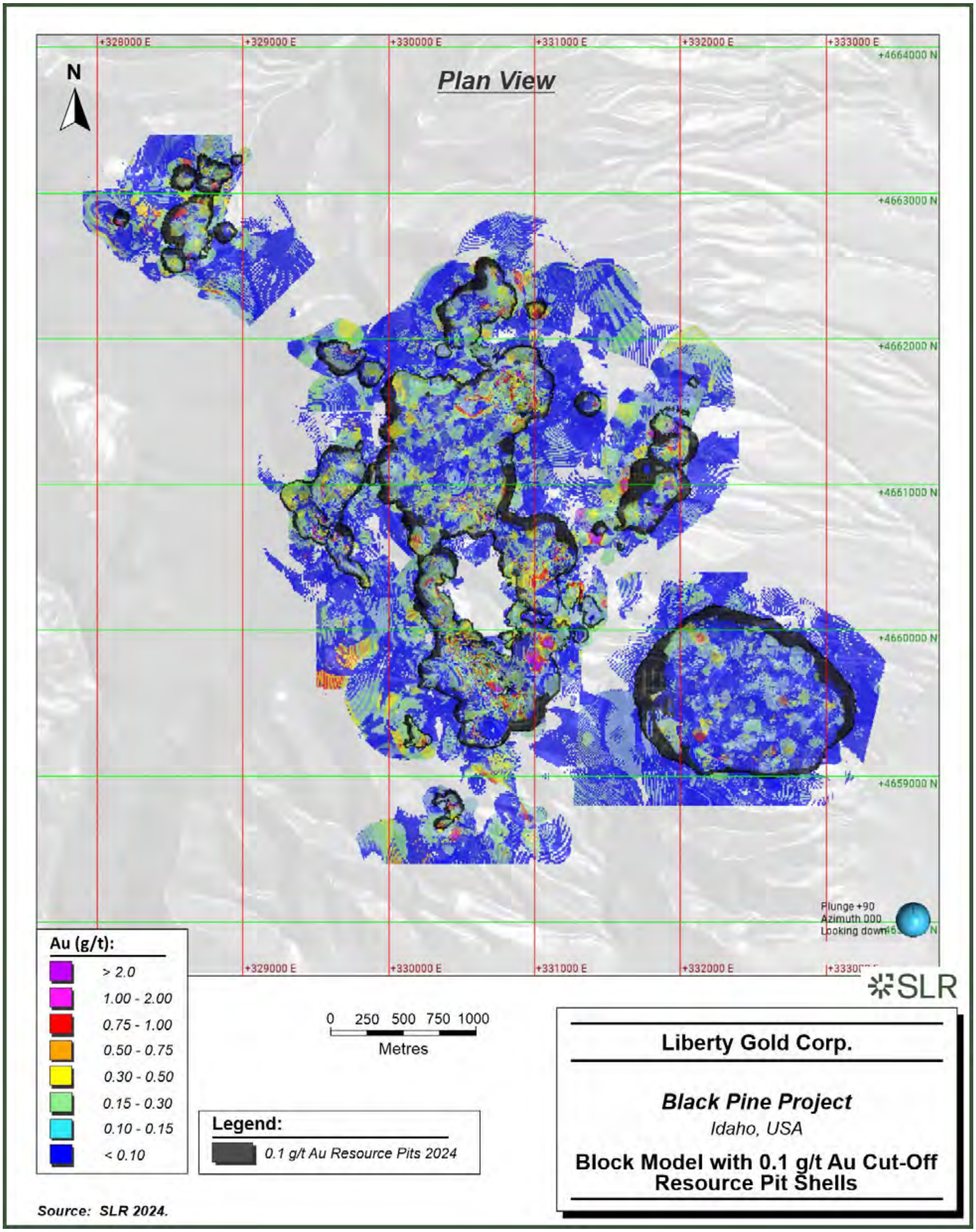


Figure 14-12: Block Model with 0.10 g/t Au Cut-Off Resource Pit Shells

14.12.1 Pit Optimization

To fulfill the CIM requirement of “reasonable prospects for eventual economic extraction”, or RPEEE, SLR prepared conceptual open pit shells for the Project to constrain the block model for Mineral Resource reporting purposes.

Pit optimization was conducted in Whittle software utilizing the Lerchs-Grossmann (“LG”) algorithm to generate a pit shell based on a regular block model (10 m by 10 m by 5 m in size) and a set of input economic and technical parameters summarized in Table 14-17. The Overall Slope Angle (“OSA”) used in the optimization ranged from 45° to 47° and is based on Knight Piesold’s report titled “1-Report_Pit Slope Design_Rev A.pdf” dated October 20, 2022.

Whittle uses the LG algorithm to define the blocks that can be mined at a profit and creates an RPEEE shell (LG shell) based on the following information:

- Initial topography
- Overall slope angles by geotechnical zone
- Metallurgical recoveries by mineralization and rock type
- Geologic grade model with gold and silver grades, density, lithology, and mineral types
- Process and mining costs
- Incremental vertical bench mining costs
- Downstream costs, such as gold refining, royalties, freight, and marketing

The results of the pit optimization partially form the basis of the Mineral Resource statement and are used to constrain the Mineral Resource with respect to the CIM Standards. Pit optimization does not constitute an attempt to estimate reserves.

The resource pit shell input parameters are summarized in Table 14-16. Whittle treated blocks below the cut-off grade of 0.10 g/t Au as waste in preparing the pit shell.

Table 14-16: Black Pine Pit Optimization Parameters

Parameter	Units	Black Pine
Gold Price	US$/oz	2,000
Base Mining Cost	$/t mined	2.35
Process Cost		
Process Cost	$/t leached	1.83
Site General Cost	$/t leached	0.80
Recoveries		
Processing Gold Recovery	%	Variable: 63% - 83%
Other Costs		
Dore Freight, Security & Insurance	$/oz produced	2.20
Royalties (0.5% of Au Price)	$/oz produced	50.00

Table 14-17 presents the Black Pine Mineral Resources tabulated within conceptual pit shells developed using increasing cut-off grades. This is presented to provide grade-distribution data that allows for an assessment of the sensitivity to different pit shells and cut-off grades.

Table 14-17: Black Pine Pit Shell and Cut-Off Grade Sensitivity

Cut-off (g/t Au)	Indicated			Inferred		
	Tonnes (000 t)	Grade (g/t Au)	Contained Metal (koz)	Tonnes (000 t)	Grade (g/t Au)	Contained Metal (koz)
0.10	402,564	0.32	4,163	97,680	0.23	712
0.17	250,010	0.43	3,449	40,937	0.34	445
0.20	197,175	0.49	3,110	27,729	0.39	348
0.50	39,731	1.09	1,388	3,038	0.91	89

Notes:

1. The Project Mineral Resources are shown in bold and comprise all model blocks at a 0.10 g/t Au cut-off that lie within optimized resource pit shells.
2. Tabulations at higher cut-offs than used to define the Mineral Resources represent subsets of the Mineral Resources.
3. The Effective Date of the resource estimations is June 1, 2024.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
5. Rounding as required by reporting guidelines may result in apparent discrepancies between tonnes, grade, and contained gold content.

14.13 Classification

Definitions for Mineral Resource categories used in this Technical Report are consistent with those defined by CIM Standards and adopted by NI 43-101.

A Mineral Resource is defined as a concentration or occurrence of material of economic interest in or on the Earth's crust in such form, grade or quality, and quantity that there are reasonable prospects for economic extraction. A Mineral Resource is a reasonable estimate of mineralization, considering relevant factors such as cut-off grade, likely mining dimensions, location, or continuity, that with the assumed and justifiable technical and economic conditions, is likely to, in whole or in part, become economically extractable. It is not merely an inventory of all mineralization drilled or sampled.

Based on this definition of Mineral Resources, the Mineral Resources estimated in this Technical Report have been classified according to the definitions below based on geology, grade continuity, and drill hole spacing.

Measured Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of conclusive geological evidence and sampling. The level of geological certainty associated with a Measured Mineral Resource is sufficient to allow a qualified person to apply modifying factors, as defined in this section, in sufficient detail to support detailed mine planning and final evaluation of the economic viability of the deposit. Because a Measured Mineral Resource has a higher level of confidence than the level of confidence of either an Indicated Mineral Resource or an Inferred Mineral Resource, a Measured Mineral Resource may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Indicated Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of adequate geological evidence and sampling. The level of geological certainty associated with an Indicated Mineral Resource is sufficient to allow a qualified person to apply modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Because an Indicated Mineral Resource has a lower level of confidence than the level of confidence of a Measured Mineral Resource, an Indicated Mineral Resource may only be converted to a Probable Mineral Reserve.

Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. The level of geological uncertainty associated with an Inferred Mineral Resource is too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability. Because an Inferred Mineral Resource has the lowest level of geological confidence of all Mineral Resources, which prevents the application of the modifying factors in a manner useful for evaluation of economic viability, an Inferred Mineral Resource may not be considered when assessing the economic viability of a mining project and may not be converted to a Mineral Reserve.

The SLR QP has considered the following factors that can affect the uncertainty associated with each class of Mineral Resources:

- Reliability of sampling data.
- Drilling, sampling, sample preparation, and assay procedures follow industry standards.
- Data verification and validation work confirm drill hole sample databases are reliable.
- No significant biases were observed in the QA/QC analysis results.

Confidence in interpretation and modelling of geological and estimation domains:

- Mineralization domains are interpreted manually in cross-sections and refined in longitudinal sections by an experienced resource geologist.
- There is good agreement between the drill holes and mineralization wireframe shapes.

The mineralization wireframe shapes are well defined by sample data in areas classified as Indicated.

Confidence in block grade estimates:

- Indicated block grades correlate well with composite data, statistically and spatially and locally and globally.

Blocks were classified as Indicated or Inferred based on drill hole spacing, confidence in the geological interpretation, and apparent continuity of mineralization.

14.13.1 Measured Mineral Resources

There are no Measured Mineral Resources at the Black Pine Project.

14.13.2 Indicated Mineral Resources

Indicated Mineral Resources were defined where drill hole spacing of 50 m to 60 m was achieved. The drill holes spacing for Indicated classification is supported with the experimental variogram ranges.

SLR notes that a few isolated areas met the criteria for Indicated classification but were supported wholly or largely by historic drilling. These areas have been downgraded to Inferred classification until results are supported by infilled drilling by Liberty Gold.

14.13.3 Inferred Mineral Resources

Any blocks within the low-grade Indicator domains were downgraded to Inferred classification due to limited geological control, very low grade within the domains, and uncertainty in the tonnage. All remaining blocks contained within the wireframe model and estimated within the block model were limited to an Inferred classification.

The backfill dumps and waste dumps that contain mineralization above the cut-off grade were also classified as Inferred Mineral Resources.

14.13.4 Classification Model

The final classification for the Project is shown in Figure 14-13.

In the SLR QP's opinion the classification of Mineral Resources is reasonable and appropriate for disclosure.

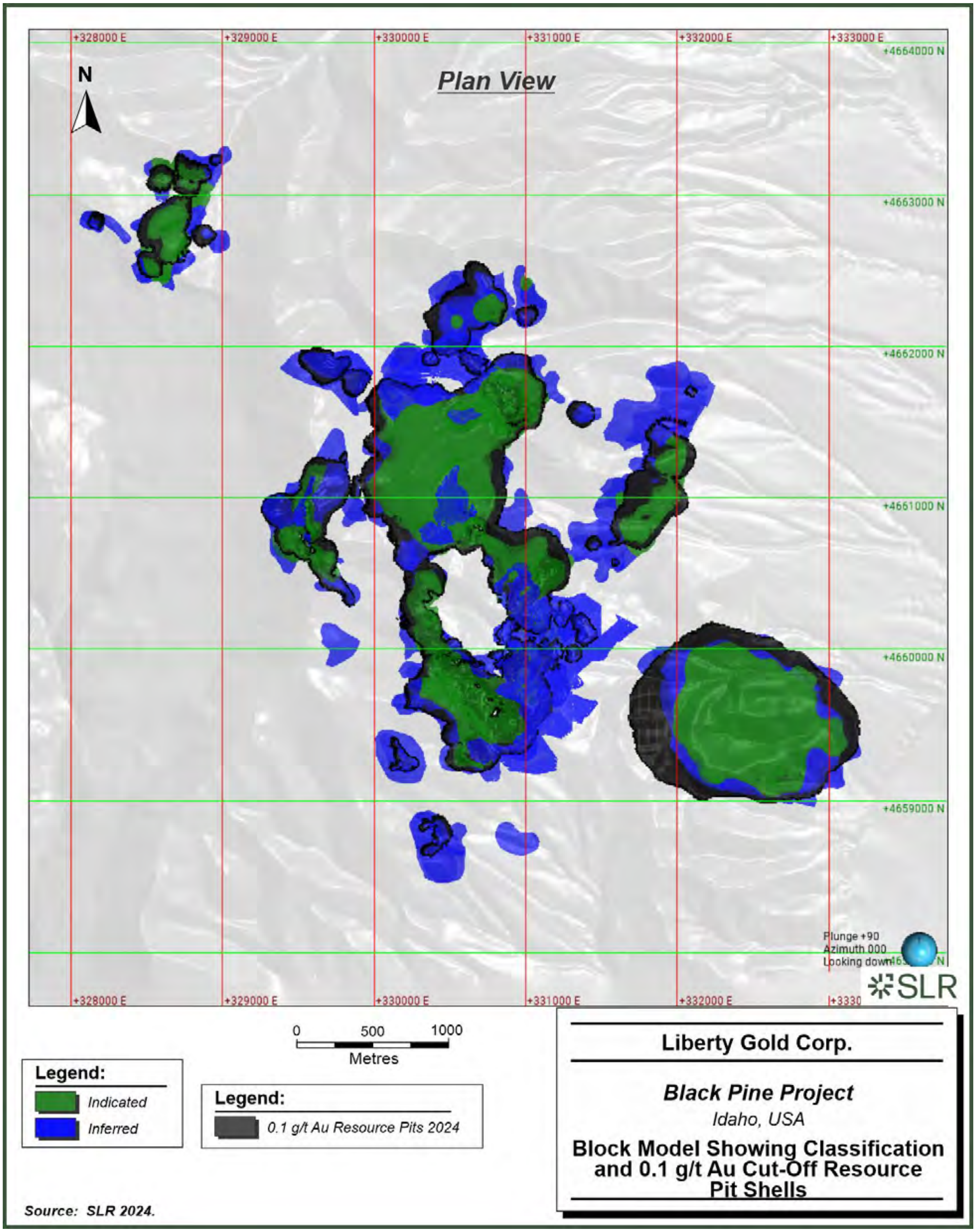


Figure 14-13: Classification Model

14.14 Block Model Validation

14.14.1 Statistics

The SLR QP reviewed and validated the block model using various modelling and interpolation aspects of the Black Pine model. Observations and comments from the model validation are provided below.

The SLR QP reviewed gold and silver grades and proportions relative to the blocks and composite samples. SLR observed that the block grades compared well with drilling and sampling and did not appear to smear significantly across sampled grades.

A statistical comparison of the estimated block gold grades with the 3.048 m composites is shown in Table 14-18. The block results compare well with the composites, indicating a reasonable overall representation of the gold grades in the block model. Note that spatial clustering of composite samples can affect the representativeness of the values, such as the mean and CV.

Table 14-18: Comparison of Block and Composite Gold Grades

Domain	HG/MG/LG	Type	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	CV
Backrange	HG	Blocks	100,278	0.04	6.11	1.22	0.68
		Comps	339	0.02	7.06	1.27	0.93
Backrange	MG	Blocks	462,732	0.00	1.84	0.23	0.62
		Comps	1,004	0.02	1.97	0.27	0.77
Backrange	LG	Blocks	24,710	0.01	0.83	0.09	0.79
		Comps	184	0.01	0.42	0.06	1.02
CD	HG	Blocks	294,988	0.00	5.50	0.80	0.58
		Comps	1,860	0.00	6.10	0.86	0.74
CD	MG	Blocks	884,047	0.00	1.43	0.20	0.41
		Comps	6,079	0.00	1.51	0.22	0.50
CD	LG	Blocks	95,777	0.00	1.09	0.10	0.72
		Comps	1,657	0.00	0.45	0.05	0.66
F	HG	Blocks	28,058	0.12	7.16	0.93	0.64
		Comps	208	0.08	8.00	1.05	1.14
F	MG	Blocks	146,284	0.02	1.11	0.20	0.42
		Comps	847	0.02	1.15	0.23	0.59
F	LG	Blocks	17,525	0.02	0.78	0.10	0.51
		Comps	198	0.01	0.21	0.06	0.58
Disco	HG	Blocks	656,935	0.00	17.59	1.25	0.85
		Comps	4,834	0.00	20.00	1.54	1.17
Disco	MG	Blocks	2,225,060	0.00	4.83	0.22	0.62
		Comps	21,278	0.00	5.00	0.24	0.91
Disco	LG	Blocks	225,637	0.00	4.33	0.10	0.94
		Comps	4,564	0.00	1.50	0.07	1.13
E Pit	HG	Blocks	23,660	0.13	3.61	1.06	0.50
		Comps	209	0.02	4.35	1.08	0.74
E Pit_3200	HG	Blocks	4,063	0.59	23.28	4.38	0.87
		Comps	84	0.02	24.33	4.28	1.24
E Pit_3201	HG	Blocks	6,669	0.14	19.93	2.64	0.93

Domain	HG/MG/LG	Type	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	CV
		Comps	212	0.02	25.00	2.87	1.39
E Pit	MG	Blocks	282,659	0.02	2.85	0.21	0.57
		Comps	2,413	0.02	3.00	0.23	0.83
E Pit	LG	Blocks	25,639	0.01	4.91	0.11	1.25
		Comps	685	0.01	0.42	0.05	0.84
J Zone	HG	Blocks	48,777	0.10	4.27	0.92	0.54
		Comps	254	0.07	5.68	0.96	0.82
J Zone	MG	Blocks	477,972	0.00	1.82	0.20	0.60
		Comps	1,166	0.02	1.88	0.23	0.80
J Zone	LG	Blocks	19,143	0.02	0.22	0.04	0.50
		Comps	278	0.01	0.26	0.05	0.70
M Zone	HG	Blocks	97,507	0.09	7.99	1.44	0.62
		Comps	363	0.03	8.00	1.43	0.86
M Zone	MG	Blocks	545,961	0.02	2.18	0.20	0.69
		Comps	1,269	0.02	2.22	0.24	0.90
M Zone	LG	Blocks	31,284	0.01	1.61	0.13	1.22
		Comps	228	0.01	0.6	0.07	0.86
Rangefront	HG	Blocks	926,327	0.02	5.00	0.62	0.72
		Comps	2,589	0.01	5.00	0.63	0.96
Rangefront	MG	Blocks	1,885,540	0.02	0.94	0.16	0.36
		Comps	7,315	0.02	1.00	0.18	0.45
Rangefront	LG	Blocks	416,801	0.01	0.53	0.09	0.47
		Comps	3,833	0.00	0.35	0.07	0.52

14.14.2 Swath Plots

SLR produced comparative statistics, including NN and ID2 estimations, and swath plots for all deposits. Swath plots generally demonstrated good correlation, with domain and the high-grade zones within it.

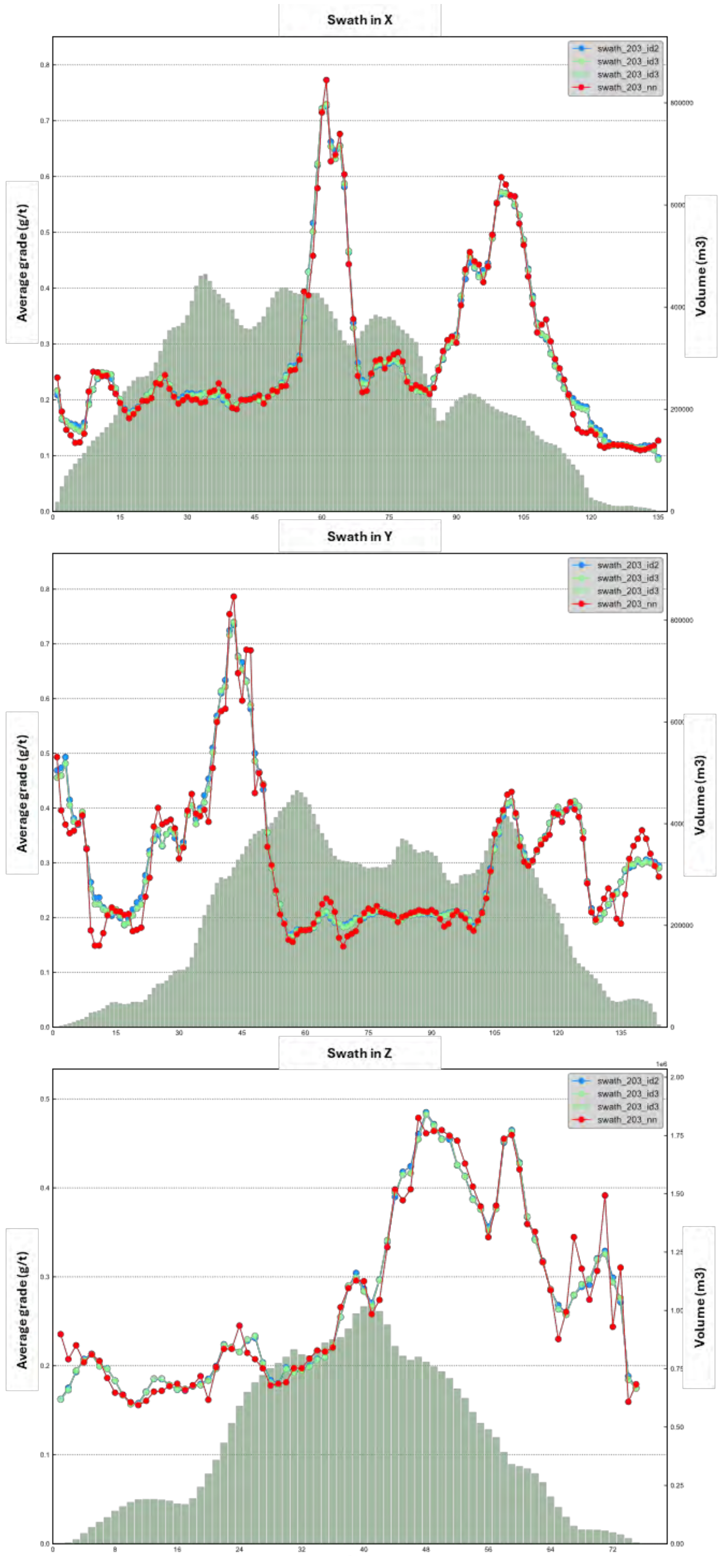


Figure 14-14: Swath Plots Through Discovery 203 Zone

14.14.3 Visual Validation

SLR used visual validation tools including cross sections and plan views to compare the block model with the drilling data. Figure 14-15 through Figure 14-20 show various cross sections through different parts of the Project.

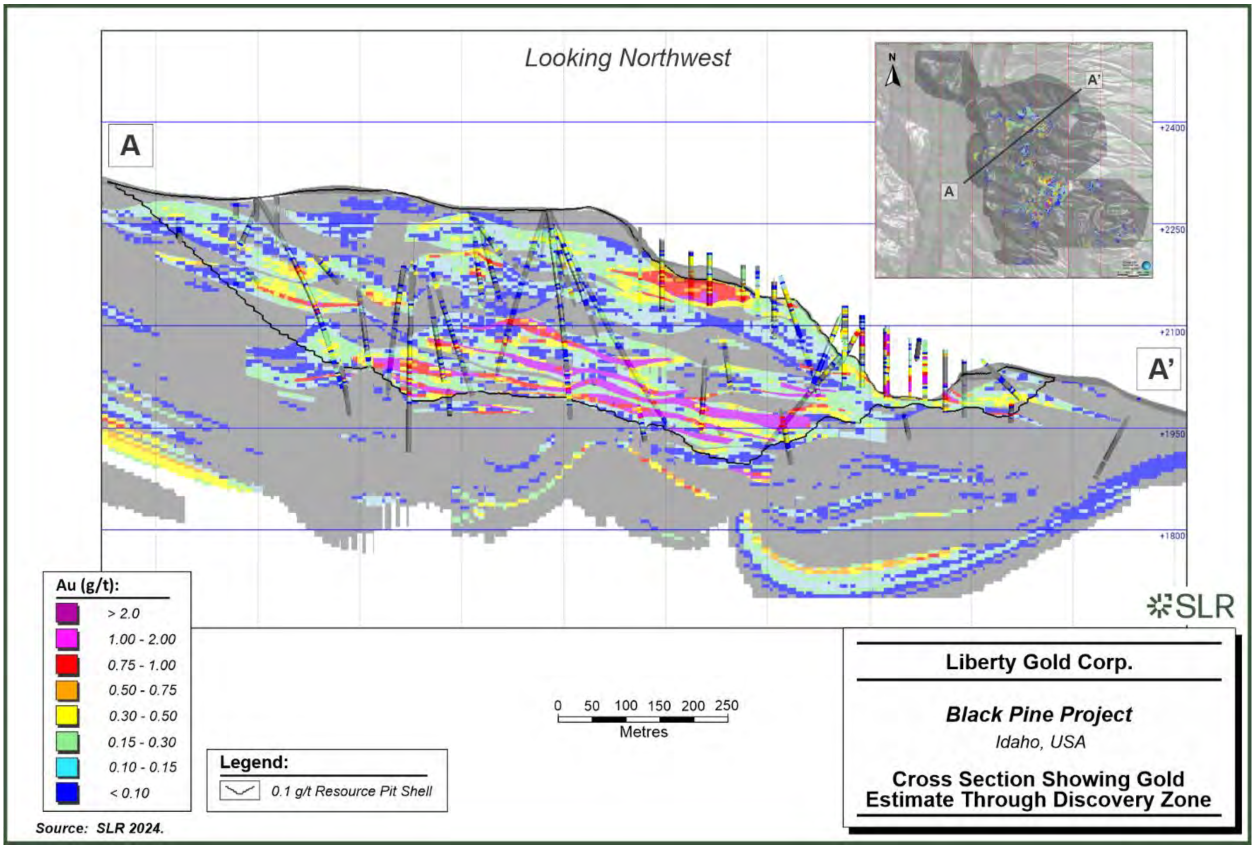


Figure 14-15: Cross Section Showing Gold Estimate Through Discovery Zone

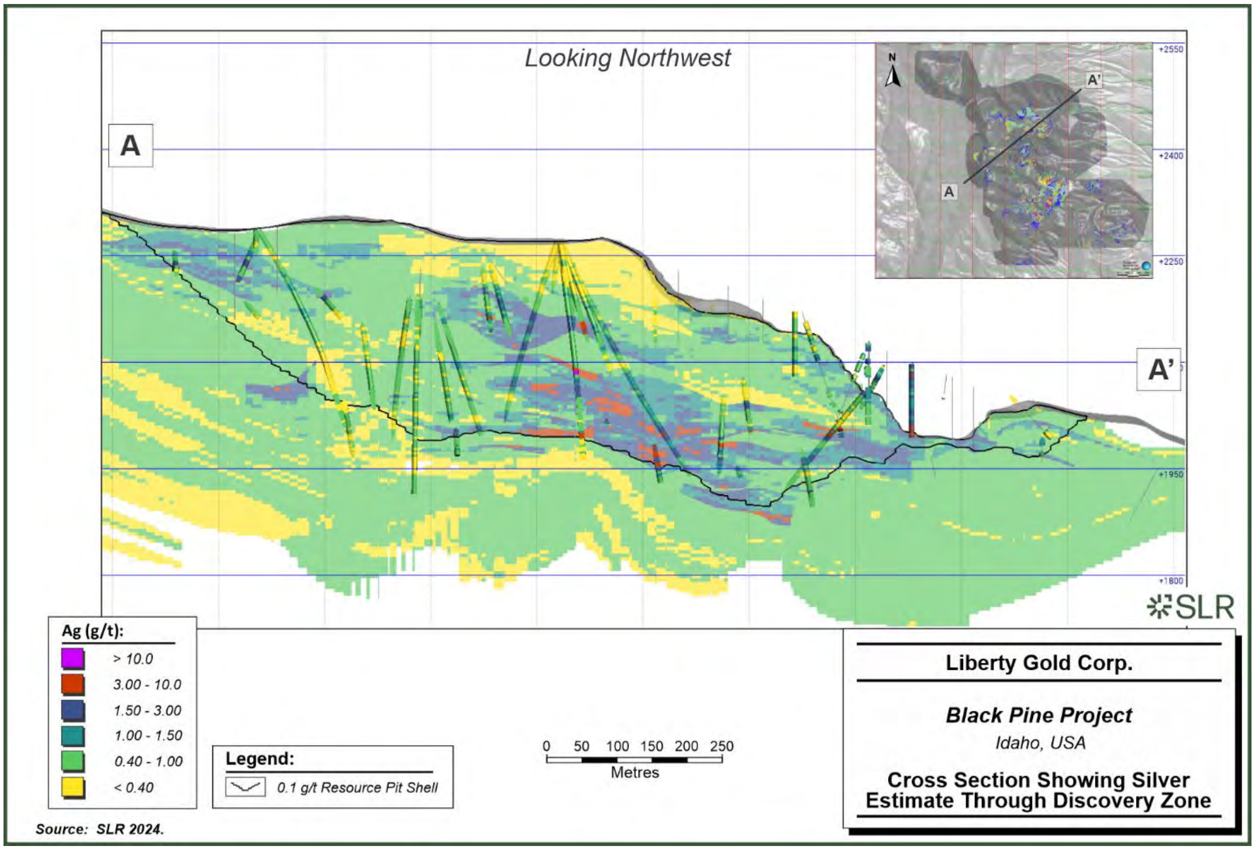


Figure 14-16: Cross Section Showing Silver Estimate Through Discovery Zone

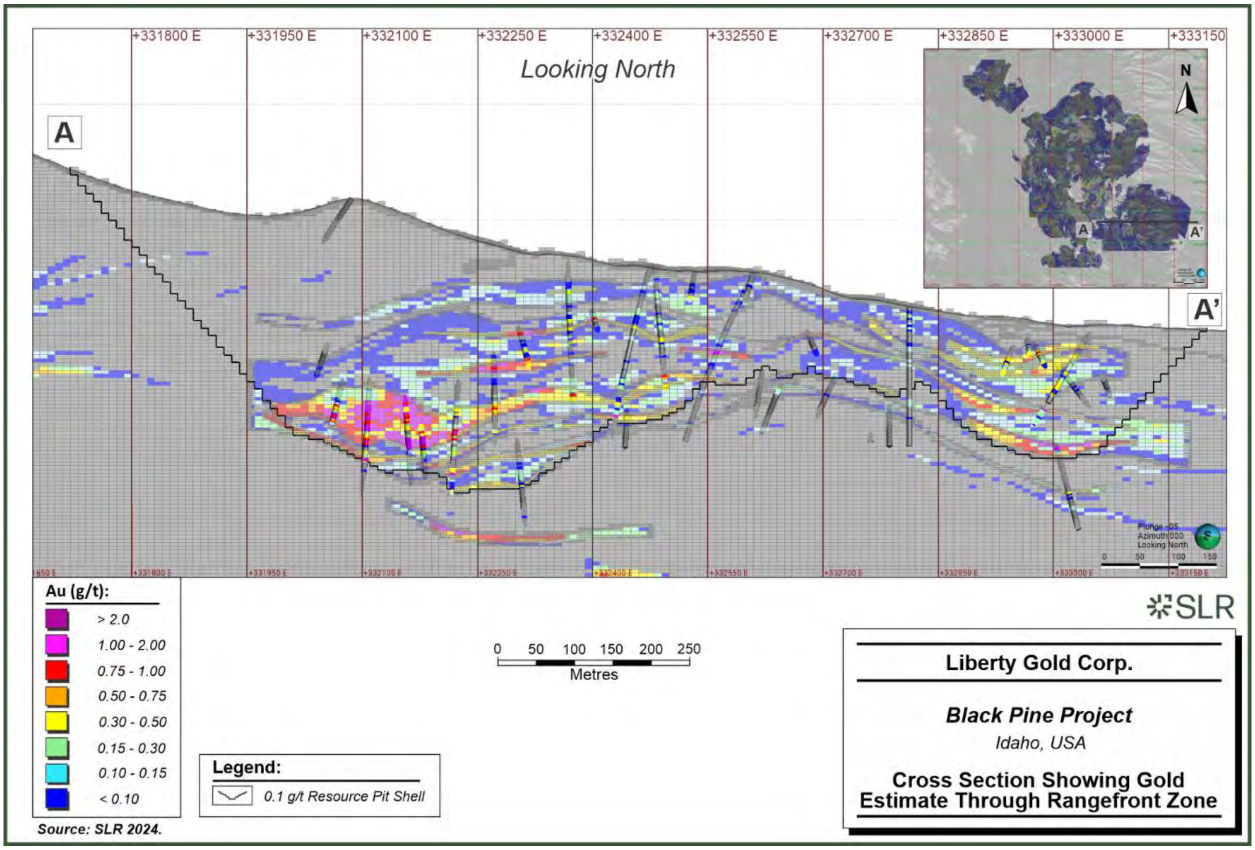


Figure 14-17: Cross Section Showing Gold Estimate Through Rangefront Zone

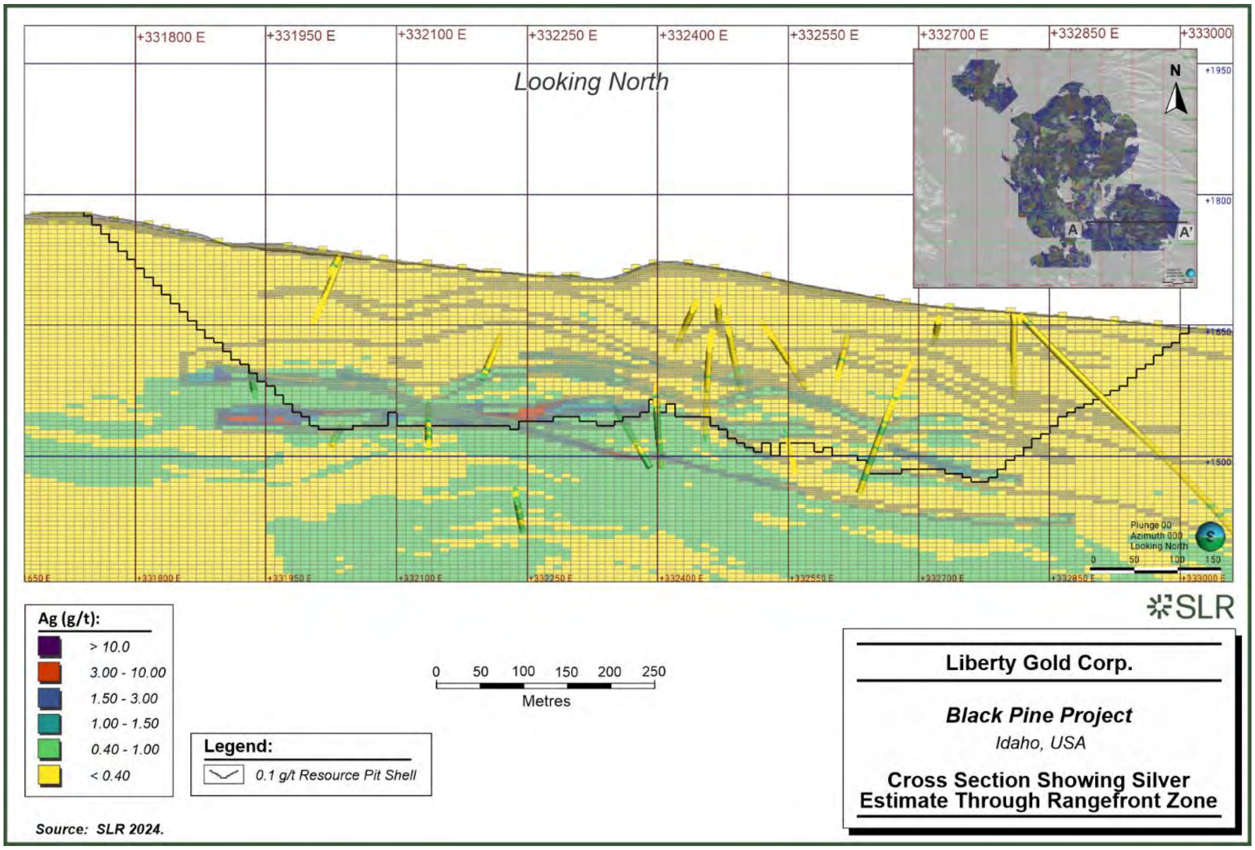


Figure 14-18: Cross Section Showing Silver Estimate Through Rangefront Zone

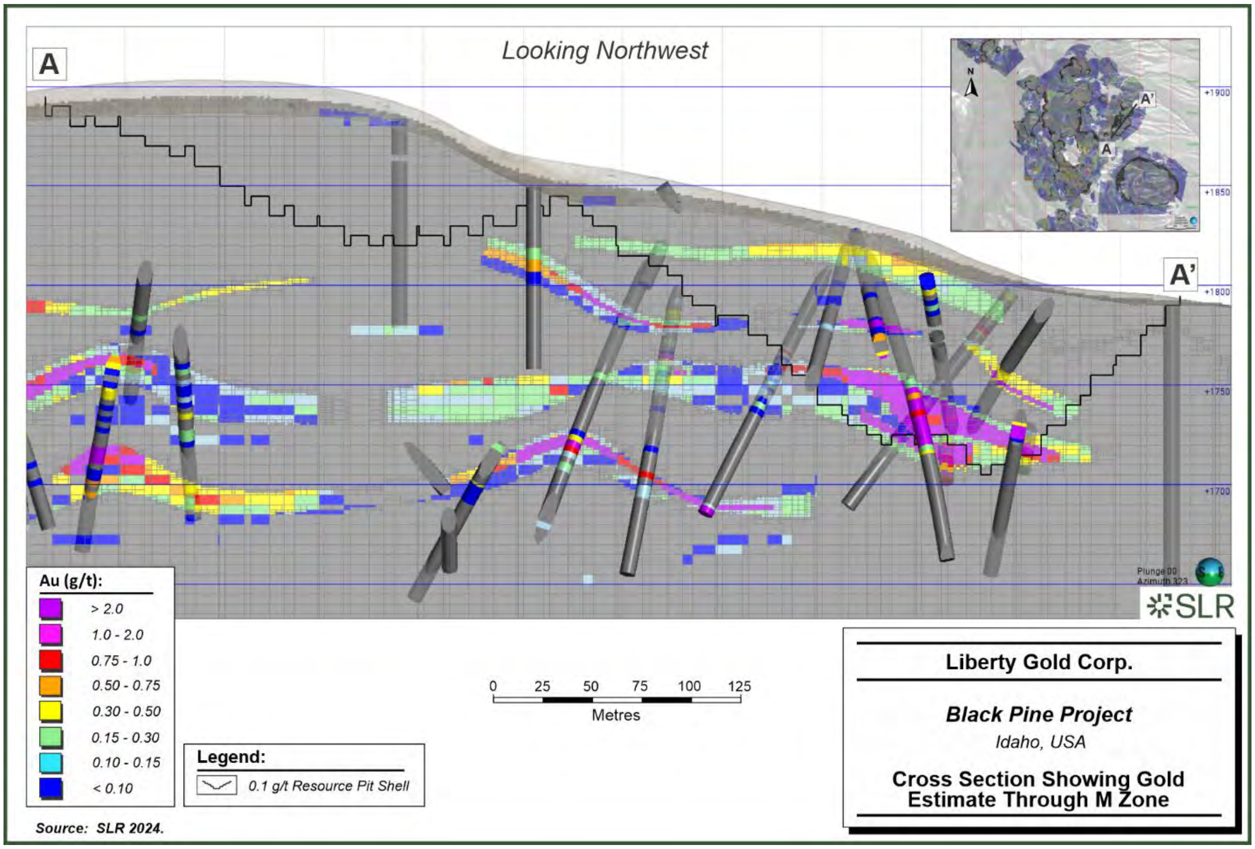


Figure 14-19: Cross Section Showing Gold Estimate Through M zone

14.14.3.1 Blast Data Visual Reconciliation

A lack of production records by pits and zones means a direct reconciliation cannot be performed. However, blast hole records do exist for some of the historic pits. SLR visually compared gold assay results from blast hole data to the updated block model and found the comparisons to be favorable. Figure 14-20 is a cross section through the historic CD pit comparing gold values from historic blast holes with the block model.

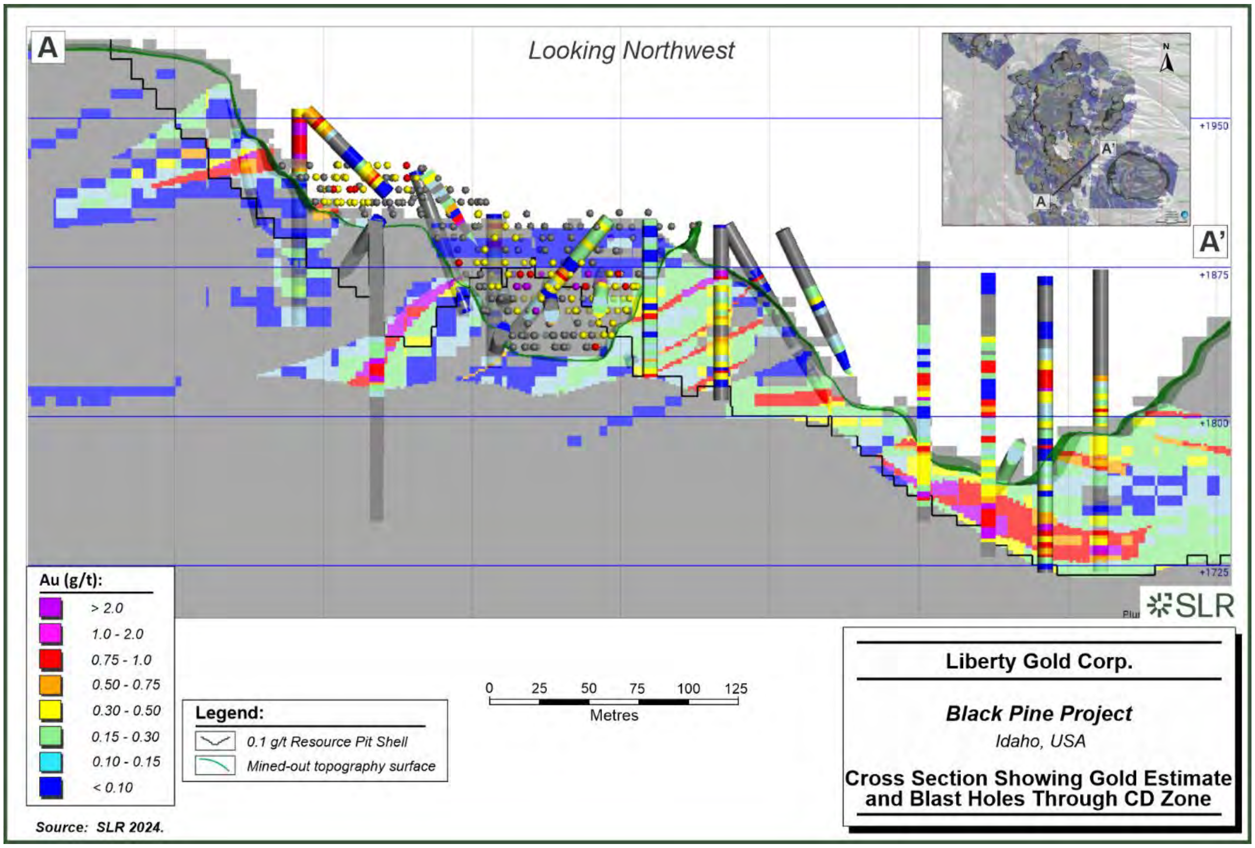


Figure 14-20: Cross Section through CD Zone with Blast Holes

14.15 Grade Tonnage Sensitivity

Figure 14-21 presents the sensitivity of the pit constrained Indicated and Inferred Mineral Resources for the Black Pine Mineral Resource model to various cut-off grades. The total tonnes (above 0 g/t Au) within the conceptual resource pit are 599 Mt.

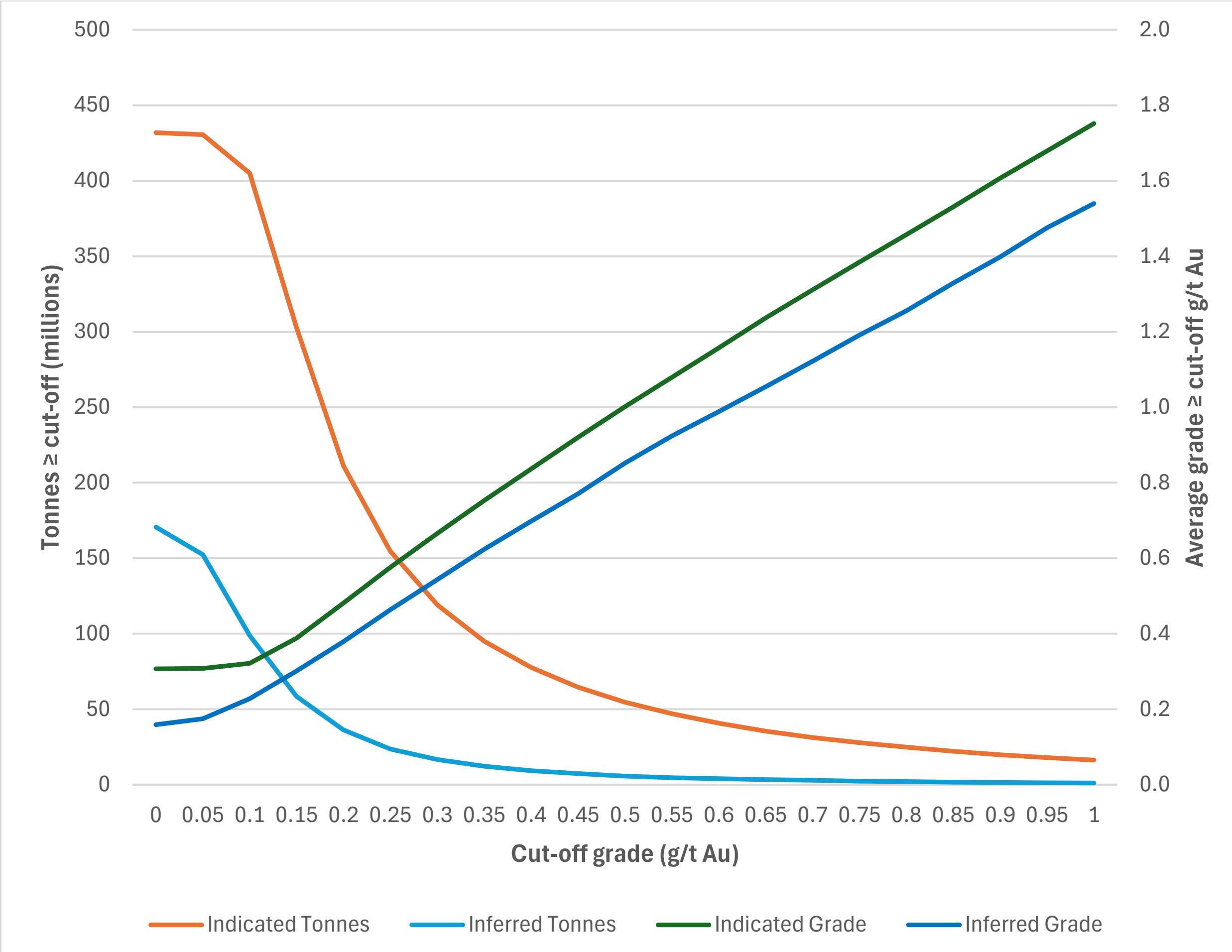


Figure 14-21: Indicated and Inferred Gold Mineral Resource Grade Tonnage Curves

14.16 Mineral Resource Estimate Reporting

The Project resource estimate is summarized by area at a cut-off grade of 0.1 g/t Au in Table 14-19. In the SLR QP's opinion, the assumptions, parameters, and methodology used for the Project Mineral Resource estimate are appropriate for the style of mineralization. The effective date of the Mineral Resource estimate is June 1, 2024. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The SLR QP is of the opinion that with consideration of the recommendations summarized in Section 1 and Section 23, any issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

The SLR QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 14-19: Summary by Area of Mineral Resources as of June 1, 2024

Classification	Area	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Indicated	Backrange	4,875	0.52	82
	Disco	164,047	0.37	1,930
	CD	29,392	0.30	281
	F	5,320	0.34	58
	J	2,860	0.35	32
	M	5,649	0.54	97
	E	6,971	0.29	65
	Rangefront	183,449	0.27	1,619
Total Indicated		402,564	0.32	4,163
Inferred	Backrange	456	0.37	5
	Disco	22,226	0.24	172
	CD	9,004	0.24	69
	F	735	0.22	5
	J	8,171	0.27	72
	M	942	0.27	8
	E	10,790	0.24	84
	Rangefront	45,355	0.20	296
Total Inferred		97,680	0.23	712

Notes:

1. CIM Standards definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at a gold cut-off grade of 0.10 g/t using a long-term gold price of US$2,000 per ounce.
3. Mineral Resources are estimated using a variable recovery derived from metallurgical studies.
4. Bulk density is variable by rock type.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6. Mineral Resources are reported within conceptual open pit shells.
7. Rounding as required by reporting guidelines may result in apparent discrepancies between tonnes, grades, and contained gold content.
8. The effective date of the Mineral Resource estimate is June 1, 2024.

15 MINERAL RESERVE ESTIMATES

The Black Pine Gold Project is planned to be an open pit operation using conventional mining equipment. No underground mining is considered.

The Mineral Reserve estimate is based on the mine designs and mine plans generated by AGP Mining Consultants, Inc. (AGP).

The Mineral Reserves consist of Indicated blocks above a cut-off of 0.100 Au g/t and contained within the ultimate pit design. No Measured Mineral Resources are contained in the Black Pine deposits, so there are no Proven Reserves. Pits were designed in accordance with geotechnical recommendations, and based on economic calculations using metal prices, costs, and recoveries. A waste contact dilution was applied to the final reserve estimate.

15.1 Black Pine Mineral Reserve Estimate

The Mineral Reserves for the Black Pine are listed in Table 15-1 with the gold grade (Au) estimates based on the mine diluted grades in the block model.

Table 15-1: Proven and Probable Reserves – Black Pine Gold Project

Reserve Class	Tonnes (Mt)	Au (g/t)	Contained Ounces (Mozs)
Proven	0.0	-	-
Probable	299.4	0.323	3.11
Total Capacity	299.4	0.323	3.11

Notes:

1. The Mineral Reserve estimate was prepared by AGP, Barrie ON, Canada and has an effective date of June 1, 2024. The Qualified Person responsible as defined under NI 43-101 for the Mineral Reserve estimate is Todd Carstensen RM-SME, Principal Mine Engineer and independent of Liberty Gold
2. Mineral Reserves reported are consistent with the CIM Standards.
3. Mineral Reserves are reported to a cut-off grade of 0.10 Au g/t.
4. The cut-off grades are based on a gold price of US$1,650 Au oz.
5. Metallurgical recovery of gold is based on a variable gold leach recovery model derived from extensive metallurgical studies.
6. All carbonaceous materials have been treated as waste Overall leach recovery averages 70.4%.
7. Mine dilution was estimated based on a 1.0 m skin applied to ore to waste contacts.
8. Units are metric tonnes, metric grams & troy ounces; “Au” = gold.
9. The estimate of Mineral Reserves may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issue.

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves. Areas of uncertainty that may materially impact Mineral Reserve estimation include:

- Commodity price and exchange rate assumptions;
- Capital and operating cost estimates;
- Geotechnical slope designs for pit walls.
- Mining selectivity near the ore contacts;
- Gold leach recovery.

15.2 Block Model

The resource model for the Black Pine deposit was received from Liberty Gold and is used for all mine design work. The model block size is 10 m x 10 m x 5 m.

Relevant information from the resource block models includes the following.

- Gold grade (g/t) (diluted);
- Gold recovery (%);
- Mineral Resource classification (Measured, Indicated, Inferred);
- Block density (t/m^3);
- Metallurgical Zones (gold cyanide solubility zones).

The reserve block model for Black Pine includes relevant information from the resource model with items added for mine planning purposes including geotechnical slope sectors, waste contact dilution, mining cost, and incremental ore haulage cost.

15.3 Mine Recovery and Dilution

AGP applied a mining dilution skin at Black Pine which combines 1-meter of an adjacent waste block with the ore mined. This dilution skin thickness was selected by considering the spatial nature of the mineralization, proposed grade control methods, digging accuracy, and blast heave. The mining dilution is applied to the model gold grades for scheduling and reserve estimate. No ore loss is applied to the Mineral Reserves.

15.4 Wall Slope Angles and Bench Configuration

A pre-feasibility geotechnical investigation for the Black Pine Gold Project has been conducted by Knight Piésold Ltd. The wall slope design parameters used to determine the overall slopes for the pit optimization are based on the recommendations in the geotechnical study. Parameters for the pit designs include bench height, stack height, face angles, berm widths, inter-ramp slope angles, and geotechnical berms, with adjustments as required.

15.5 Cutoff Grade

The Mineral Reserve cut-off grade for Black Pine Gold Project is 0.10 g/t Au.

15.6 Economic Pit Shell Development and Detailed Design

The final pit designs are based on optimized pit shells using Datamine's Studio NPVS software. The optimization software is used to define blocks that can be mined at a profit. The initial step is to develop an economic value for each Indicated open pit resource block. The value calculation is based on the parameters in Table 15-1. Geotechnical parameters for slope design sectors were coded into the reserve model based on a recommended pit slope design criteria from Knight Piésold Ltd. consulting. Pit restrictions include a 50-meter offset from the historic heap leach facility and the Tallman historic tailings. Optimized pit shells were generated using the block values, pit slope criteria, and associated mining, process, and G&A costs. For the purposes of the pit limit determination and design, only blocks classified as Indicated were used. Inferred blocks were considered as waste.

Pit designs developed for Black Pine are based on the $1,650/Au oz optimization shells. A 10 m operating bench and double benching with a berm every 20 m of bench height is used for the Black Pine pit designs. The detailed pit and interim phase designs use the recommended slope criteria from Knight Piesold. The ramp gradient is 10% with 30.2 m or 20.8 m ramp widths dependent on the size haul truck that is proposed for that pit. The Black Pine project consists of three larger deposits and several smaller deposits that are planned to be developed using conventional open pit mining methods. The Black Pine open pits are split into nine areas. The larger Discovery, Tallman, and Rangefront pits, plus M-zone are developed in three phases. J-zone, E-zone and Backrange have two phases and F-zone, and CD are developed with a single phase.

Table 15-2: Pit Shell Cost Assumptions and Parameters

Description		Units	Black Pine
Conversions	Grams per Troy oz		31.1035
Metal Prices	Gold Price	USD/oz	$1,650
	Silver Price	USD/oz	n/a
	Royalty	%	0.25%
Dore	Payable	%	100
	Selling Cost	USD/oz	$2.20
Metallurgical	Gold Recovery	%	Varies AUDLR[1]
	Silver Recovery	%	n./a
Mining	Base to Pit Exit	$/t mined	$1.54
	Haulage	$/t mined	varies
	Sustaining Capital	$/t mined	$0.25
	Incremental Cost below Exit	$/t mined/ 10m	$0.022
	Incremental Cost above Exit	$/t mined/ 10m	$0.020
Processing	Leaching Cost	$/t ore	$1.90
	Incremental Haul	$/t ore	OMCST[2]-WMCST[3]
G&A	General and Administrative	$/t ore	$0.80
Production Assumptions	Leach Feed Rate	t/year	18.25 Mt
	Mining Dilution (for optimization)	%	0.0%
	Mining Loss	%	0.0%
Planning Model	Block classification used		M+I
	Block Model Height	m	5 m
	Mining Bench Height	m	10 m

[1]AUDLR - Gold Recovery assigned in Block Model
[2]OMCST -Haulage cost from pit area to leach pad (from scoping study)
[3]WMCST – Haulage cost from pit area to waste destination (from scoping study)

Table 15-3: Haulage Cost Estimate by Area

Area	Cycle Times (min)		Haulage Cost ($/t)		Pit Exit Elevation
	Waste	Leach	Waste	Leach	(masl)
CD	9.3	26.9	0.28	0.90	1850
F	9.3	26.9	0.28	0.90	1990
E	23.2	69.7	0.70	2.09	2300
Discovery/Tallman	15.3	34.4	0.46	1.03	1990
J	8.7	29.3	0.26	0.88	2000
Rangefront	29.6	19.4	0.89	0.58	1630
M	7.0	17.4	0.21	0.52	1790
Backrange	12.6	71.0	0.48	2.13	2110
Default	22.4	26.9	0.67	0.81	1990

15.7 Mine Schedule

The mine schedule plans to deliver 299 Mt of leach feed grading 0.323 g/t gold to the leach pad and ore stockpiles. The mine life includes one year of pre-production stripping and 15 years of mining. The waste tonnage from all pits totaling 394 Mt will be placed into waste storage facilities or backfilled into mined out pits. The overall strip ratio is 1.3:1. Processing of stockpiled ore will continue for 2 years after mining is complete.

The leach production schedule targets a rate of 18.25 Mt/a (50 kt/d) starting in year 1 and maintains this rate for the project life.

- A detailed mine production schedule was developed using diluted reserves and costed to support the financial evaluation of the Project.

16 MINING METHODS

16.1 Overview

The Black Pine Project is located in southern Idaho, approximately 29 km northwest of the town of Snowville, Utah, and 13 km north-northeast of Curlew Junction, the intersection of Utah State Highways 30 and 42. The Black Pine property straddles the eastern margin of the northerly-trending Black Pine Mountains. Elevations within the property range from a low of 1,650 masl along the eastern edge, to a maximum of 2,440 masl in the western part of the property.

The Black Pine Project consists of two large deposits and several smaller deposits that are planned to be developed using conventional open pit mining methods. The leach process facility will be located at lower elevations on the east side of the property where the topography begins to flatten out. Ore from the pits will be hauled to the leach facility and end dumped on to the pad. Underground mining was not considered for this study.

The mine schedule for open pit mining consists of 299 Mt of leach feed at a 0.323 g/t mine diluted gold grade over a processing life of 17 years. The diluted gold grade for the first five years of production averages 0.450 g/t gold. The open pit waste tonnage totals 394 Mt and will be placed into waste storage areas. The current mine life includes one-year of pre-preproduction followed by 15 years of production mining. The remaining stockpile ore is sent to the leach pad in Years 16 and 17.

The leach production rate is 18.25 Mt/a starting in Year 1. Ore stockpiling begins during the pre-production year and continues throughout the mine life as low grade and lower recovery leach material is deferred when better material is available. A total of 92 Mt of ore is stockpiled with a peak stockpile capacity of 33.3 Mt reached at the end of Year 6.

16.2 Planning Model

16.2.1 Geological Model Importation

The resource block model was provided to AGP from Liberty Gold in ascii format and imported into Hexagon MinePlan® block model format for open pit mine planning. Items imported from the Liberty Gold resource model are shown in Table 16-1.

Table 16-1: Imported Resource Model Items

Field Name	Min	Max	Precision	Units	Comments
AUDLR	0	1	0.0001	factor	Leach Recovery Gold
AUDL	0	20	0.0001	g/t	Gold Grade Diluted
AGDR	0	1	0.0001	factor	Leach Recovery Silver
AGD	0	100	0.0001	g/t	Silver Grade Diluted
DENSE	0	4	0.0001	g/cc	Density
AREA	0	10	1	-	Area Region
CLASS	0	5	1	-	Resource Classification
CLSIW	0	5	1	-	Resource Class Internal Waste
LITH	0	20	1	-	Lithology
TOPO	0	1	0.01	-	Topography
GEOMT	0	20	1	-	Metallurgical Data

Framework details for the open pit block model are provided in Table 16-2.

Table 16-2: Black Pine Planning Model Properties

Framework Description	Black Pine Open Pit Planning model
MinePlan file 10 (control file)	bp2410.dat
MinePlan file 15 (model file)	bp2415.pl2
X origin (m)	327900
Y origin (m)	4658400
Z origin (m) (min)	1250
Rotation (degrees clockwise)	0
Number of blocks in X direction	557
Number of blocks in Y direction	500
Number of blocks in Z direction	238
X block size (m)	10
Y block size (m)	10
Z block size (m)	5

The mine planning block model created by AGP in MinePlan includes additional items for mine planning purposes. The final open pit mine planning model items are displayed in Table 16-3.

Table 16-3: Planning Model Item Descriptions

Item	Min	Max	Precision	Units	Description
AUDR	0	1	0.0001	factor	Leach Recovery Gold
AUD	0	20	0.0001	g/t	Gold Grade
AGDR	0	1	0.0001	factor	Leach Recovery Silver
AGD	0	100	0.0001	g/t	Silver Grade
RAUD	0	20	0.0001	g/t	Recovered Gold Grade AUD*AUDLR
RAGD	0	100	0.0001	g/t	Recovered Silver Grade AGD*AGDLR
DENSE	0	4	0.0001	g/cc	Density
AREA	0	10	1	-	Area Region
CLASS	0	5	1	-	Resource Classification
CLSIW	0	5	1	-	Resource Classification Internal Waste
LITH	0	20	1	-	Lithology
TOPO	0	1	0.0001	masl	Topography
GEOMT	0	10	1	-	Met Data
SLPCD	0	25	1	-	Slope Region
OSA	0	90	0.1	degrees	Overall slope angle for pit limits analysis
IRA	0	90	0.1	degrees	Inter-ramp angle for pit design
FACE	0	90	0.1	degrees	Face slope angle for pit design
BERM	0	60	0.1	m	Catchment berm width
OMCST	0	10	0.001	$/t	Ore Mining Cost - scoping study (AGP)
WMCST	0	10	0.001	$/t	Waste Mining - scoping study (AGP)
INCRC	0	6	0.001	$/t	Bench Incremental Mining Cost
DEPTH	-50	70	1	m	Depth from Base Bench (pit exit)
OMCYC	0	100	0.1	min	Ore haul time from scoping study (AGP)
WMCYC	0	100	0.1	min	Waste haul time from scoping study (AGP)
NDEST	1	5	1	-	Destination from optimization program NPVS
DEST1	1	5	1	-	Destination calculated

Item	Min	Max	Precision	Units	Description
CFPC1	-25	100	0.01	$/t	Cash Flow per tonne
DEST2	1	5	1		Destination calculated
CFPC2	-25	100	0.01	$/t	Cash Flow per tonne
MTYPE	0	10	1	-	Scheduling Geo Met Code
PIT	0	10	1	-	Pit/Phase holder
MINED	0	5	1	-	Code for dilution script, air, or rock
BLOKT	0	2000	0.001	tonnes	Non-diluted block tonnes
OWFG	0	2	1	-	Ore: waste Flag; if >0.1g/t Au cut-off
DTON	0	2000	0.001	tonnes	Diluted block tonnes from dilution script
DDEN	0	5	0.0001	g/cc	Density from dilution script
MAUD	0	20	0.0001	g/t	Mining dilution applied to AUD
MRAUD	0	20	0.0001	g/t	Recoverable gold grade (mine diluted grade MAUD)
MAGD	0	100	0.0001	g/t	Mining dilution applied to AGD
MRAGD	0	100	0.0001	g/t	Recoverable Silver grade (mine diluted grade MAGD)

16.3 Mining Geotechnical

Open pit designs are configured on 10 m bench heights, with berms placed every two benches, or double benching. Bench Face Angles (“BFAs”), inter-ramp angles (“IRAs”) and bench widths are unique to each prescribed geotechnical domain.

Knight Piésold Ltd. (“KP”) has completed a PFS level open pit slope design (KP 2024). Site-specific geology, structural geology, rock mass, and hydrogeological information obtained from various sources were reviewed and compiled in conjunction with the updated geology and fault structure models for geotechnical characterization, slope stability analysis, and pit slope design. The study focused on three major mineral development zones, namely, the Discovery (Main and Tallman), CD, and Rangefront Pits.

A simplified rock mass model was created for geotechnical characterization and includes the following geotechnical units:

- Sandstone – consists of the Upper Plate sandstone and siltstone member (Ppos)
- Siltstone – consists of the Middle Plate non-clastic siltstone and limestone members (Pola, Polb, and Polc)
- Limestone – consists of the Middle Plate clastic limestone and dolomite member (Pold) and limestone, sandstone, and quartzite member (Pols)
- Shale – consists of the Lower Plate shale and siltstone member (PMmc)
- Fault Zone

The overburden and other minor lithology units that are not correlated with the discrete stratigraphic units are considered negligible for the PFS.

Intact rock strength of the sedimentary package ranges from Medium Strong to Strong, except in the Fault Zone unit. Overall rock mass ranges from Poor to Good quality but is largely competent despite being weathered and oxidized. Flatly dipping to subhorizontal bedding and steeply dipping fracture sets were identified in the deposit area. Localized high-angle faults, lithology contacts, and bedding were identified but will require further verification. The groundwater table is expected to be below the proposed ultimate pit bottoms with the exception of the Rangefront Pit.

The site-specific geotechnical information and slope stability analysis results have been utilized to identify specific design sectors for the proposed pits. The geotechnical slope design criteria for the proposed open pits were developed based on the results of kinematic and limit equilibrium slope stability analyses.

The maximum depths of the proposed Discovery and Rangefront Pits are in the order of over 400 m, while the CD Pit is around 300 m deep. Design sectors were delineated for the proposed Discovery (Main and W), CD, and Rangefront Pits following geological and structural features, rock mass characteristics, and orientations of the proposed ultimate pit walls. Recommended pit slope configurations for the ultimate Discovery, CD, and Rangefront pits, including bench heights, BFAs, IRAs, and OSAs, are summarized in Table 16-4. The recommended IRAs are also illustrated in Figure 16-1 for all proposed pits, including those are not listed in Table 16-4.

Table 16-4: Recommended Pit Slope Angles

Pit	Design Sector	Maximum Wall Height (m)	Bench Height (m)	Bench Face Angle (º)	Bench Width (m)	Inter-ramp Angle (º)	Overall Slope Angle (º)
Discovery Main	West	370	20	70	10	49	45
	North	235	20	70	9	51	
	East	210	20	75	9	54	
Discovery Tallman	West	170	20	70	9	51	
	East	250	20	75	9	54	
CD	West	290	20	70	9	51	48
	East	130	20	75	9	54	
	South	150	20	70	10	49	
Rangefront	West	400	20	70	10	49	44
	North	225	20	70	9	51	
	South	265	20	70	9	51	

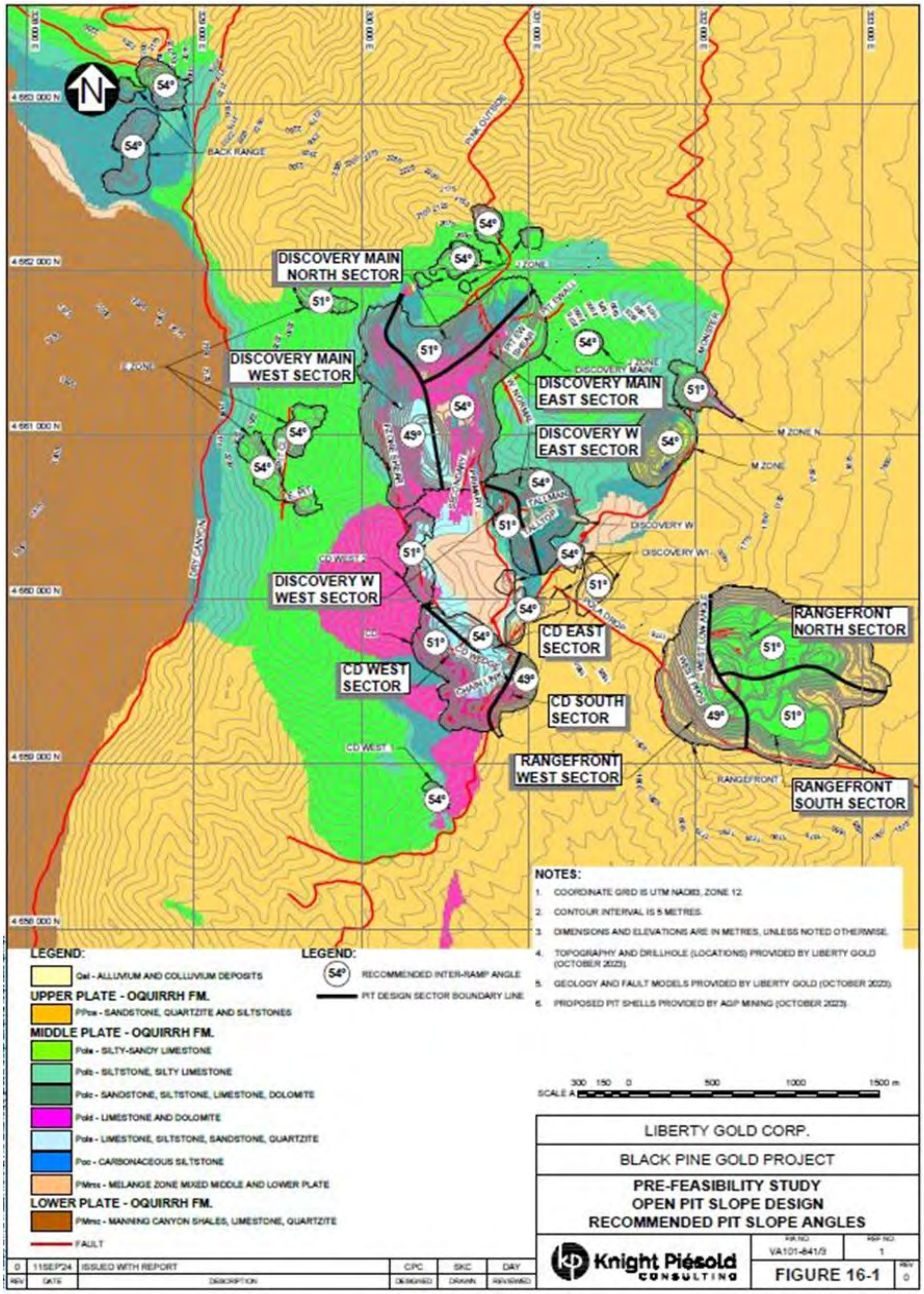
It is recommended that the maximum height of the inter-ramp slopes not exceed 160 m due to the uncertainty of faults and contacts and the variability in rock mass competency. A geotechnical stepout (wider bench) or haul ramp should be placed on inter-ramp slopes exceeding 160 m in height to effectively flatten the overall slope angle and to provide additional capacity for rockfall containment and access for debris cleanout.

The maximum OSAs represent the highest slopes in the Discovery Main, CD, and Rangefront Pits. The OSAs may vary with the slope height and stepout/haul ramp locations in the final pit walls. The OSAs in other smaller pits are expected to be equal to or a few degrees flatter than the applied IRAs.

The recommended pit slope design criteria shown in Table 16-4 are considered for final pit walls. The following design criteria can be adopted for phased interim pit walls of the Discovery and Rangefront Pits.

- Bench Height: 2 x 10 m
- BFA: 70°
- Catch Bench Width: 9 m
- IRA: 51°

The implementation of the slope design in accordance with the mine plan not only includes specifications for slope geometry, but also requires low-damage good-controlled blasting and excavation practices and systematic slope monitoring throughout pit operations. Pit dewatering may be required in the later phases of the Rangefront Pit development when potential groundwater table is encountered near the pit bottom.



Source: Knight Piésold Ltd, 2024

Figure 16-1: Pit Design Sectors and Recommended Inter-ramp Angles

16.4 Economic Pit Shell Development

The open pit ultimate size and phasing requirements were determined with various input parameters including estimates of the expected mining, processing and general and administrative ("G&A") costs, as well as metallurgical recoveries, pit slopes and a long-term metal price assumption. Mining costs vary by area as there are significant ore haul cost differences from pits located at higher elevations and further from the heap leach pad. Process and Administrative costs were provided by M3 based on the proposed throughput assumptions and preliminary costing estimates. Metal pricing and selling cost were provided by Liberty Gold.

The parameters used are shown in Table 16-5. Values are in metric units and United States dollars (US$) unless otherwise noted. The mining cost estimates are based on the use of 144 tonne haul trucks, planned waste rock storage areas, and the proposed leach pad location. Incremental haulage costs for leach and waste material were derived from the scoping study haulage estimates performed by AGP in 2023.

Table 16-5: Pit Shell Cost Assumptions and Parameters

Description		Units	Black Pine
Conversions	Grams per Troy oz		31.1035
Metal Prices	Gold Price	USD/oz	$1650
	Silver Price	USD/oz	n/a
	Royalty	%	0.25%
Dore	Payable	%	100
	Selling Cost	USD/oz	$2.20
Metallurgical	Gold Recovery	%	Varies AUDLR[1]
	Silver Recovery	%	n./a
Mining	Base	$/t mined	$1.54
	Haulage	$/t mined	varies
	Sustaining Capital	$/t mined	$0.25
Processing	Leaching Cost	$/t ore	$1.90
	Incremental Haul	$/t ore	OMCST[2]-WMCST[3]
G&A	General and Administrative	$/t ore	$0.80
Production Assumptions	Leach Feed Rate	t/year	18.25Mt
	Mining Dilution	%	0.0%
	Mining Loss	%	0.0%
Planning Model	Block classification used		M+I
	Block Model Height	m	5m
	Mining Bench Height	m	10m
[1]AUDLR - Gold Recovery assigned in Block Model			
[2]OMCST -Haulage cost from pit area to leach pad (from scoping study)			
[3]WMCST – Haulage cost from pit area to waste destination (from scoping study)			

16.4.1 Mining Cost Determination

Mining costs were assigned into the planning block model based on 144 tonne haul trucks and benchmarking from recent studies performed by AGP. The haulage cost component was removed from the benchmarking study and a base mining cost without haulage was determined to be $1.54 /t.

The haulage cost was calculated for each pit area and added to the block model via a Python script that added the base mining cost without haulage, the haulage cost by area from Table 16-6 and an incremental 10 m bench haulage cost of $ 0.0204/t above the pit exit elevation and $0.0220/t below the pit exit. Incremental bench haulage costs were built up from the first principles using a cost per hour for 144 tonne haul trucks and 10% ramp cycle times. The pit exit base elevations were selected based on previous pit designs from the 2023 scoping study.

Table 16-6: Haulage Cost Estimate by Area

Area	Area Code	Cycle Times (min)		Haulage Cost ($/t)		Pit Exit Elevation
		Waste	Leach	Waste	Leach	(masl)
CD	1	9.3	26.9	0.28	0.90	1850
F	2	9.3	26.9	0.28	0.90	1990
E	3	23.2	69.7	0.70	2.09	2300
Disco	4	15.3	34.4	0.46	1.03	1990
J	5	8.7	29.3	0.26	0.88	2000
Rangefront	6	29.6	19.4	0.89	0.58	1630
M	7	7.0	17.4	0.21	0.52	1790
Backrange	8	12.6	71.0	0.48	2.13	2110
Default	9	22.4	26.9	0.67	0.81	1990

16.4.2 Cut-off grade

The ore-waste cut-off is based on metal price, costs, recovery, and the incremental ore haul cost. A model block is considered an ore block when the revenue generated is greater than the processing costs for that block. The revenue is calculated using the gold grade and gold recovery from the block model and the realized gold price. The process cost includes the leaching cost, general and administrative cost and the ore haulage incremental cost. Inferred or unclassified blocks are considered waste. The cut-off grade calculation is shown in Equation 1.

Equation 1: Cutoff Grade Calculation

$$\text{COG (g/t)} = \frac{\textit{Process + G\&A + Incremental Ore Haul}}{\textit{(Gold Price - Selling Cost) x Recovery}}$$

Where:

- Process is the total processing costs (fixed and variable) in $/t treated
- G&A is General and Administration cost in $/t treated
- Incremental Ore Haul is the difference between ore and waste mining cost in $/t treated
- Gold Price is the gold price in $/oz
- Selling Cost is the cost of selling gold (refining and royalties) in $/oz
- Recovery is the metallurgical recovery (%)

The minimum 0.10 g/t cutoff was used for mine planning optimization and for final ore determination.

The average cutoff grade varies by deposit area. For instance, the average gold grade is calculated to be 0.086 g/t using the metal prices and costs from Table 16-5 and a gold recovery of 66.7%. When the incremental ore haulage is added by deposit zone the cutoff varies, see Table 16-7.

Table 16-7: Cut-Off Grade by Pit Zone

Zone	Incremental Ore Haul Cost $/t	Au Cut-Off g/t
CD	$0.62	0.094
E Zone	$1.39	0.116
Discovery	$0.57	0.093
J Zone	$0.62	0.094
Rangefront	$0.00	0.077
M Zone	$0.31	0.085
Back Range	$1.79	0.127
F Zone	$0.62	0.094

16.4.3 Topography

The topographic information for the Black Pine and used for mine planning was received along with the resource block and matched the topography in the block model. The topography is larger than the block model footprint and covers all areas impacted by mining including all pits, waste facilities, stockpiles and surface roads.

16.4.4 Process Cost Determination

The process cost input into the pit optimization economic model includes the $1.90/ t leach cost, $0.80/ t General and Administrative cost, and an incremental ore haul cost. The incremental ore haul cost is the additional cost to haul the leach pad material as shown in the following equation.

$$Incremental\ Ore\ Haul\ Cost = Ore\ Haul\ Cost - Waste\ Haul\ Cost$$

The incremental ore haul cost will impact the cutoff grade of material sent to the process. In cases, such as Rangefront, where the waste haul cost is greater than the ore haul cost, the incremental ore haul cost applied to the process cost is set to zero.

16.4.5 Gold Recovery

Gold recovery assumptions for gold are included in the resource block model received from Liberty Gold and are used to determine the recoverable metal.

16.5 Ore Loss and Dilution Consideration

Two types of dilution are considered. The first dilution is an internal model dilution. Model blocks that do not fall completely inside the interpolation grade shell are diluted considering the portion of the block that falls outside the grade shell. The geologic resource model received from Liberty Gold includes the internal model dilution.

Mining dilution occurs when material below the cut-off grade cannot be selectively separated from the ore during mining and results in the mixing of waste along the perimeter of the ore due to blasting and over mining. AGP applied a dilution skin which combines 1-meter of an adjacent waste block with the ore mined. If one side of a mineralized block above cut-off is in contact with a waste block, then it is estimated that dilution of 10% (1 m * 10 m / 100 m^2) by volume would result. If two sides are contacting a waste block, it would rise to 20%, three sides is 30%, and up to 40% if the ore block is surrounded by waste on four sides. No dilution is applied in the vertical direction. Isolated ore blocks with waste on four sides are uncommon, and left as highly diluted ore blocks rather than considered as waste. If the adjacent waste block contains an estimated gold grade, that grade is used in the dilution calculation, otherwise the grade is treated as zero for dilution purposes.

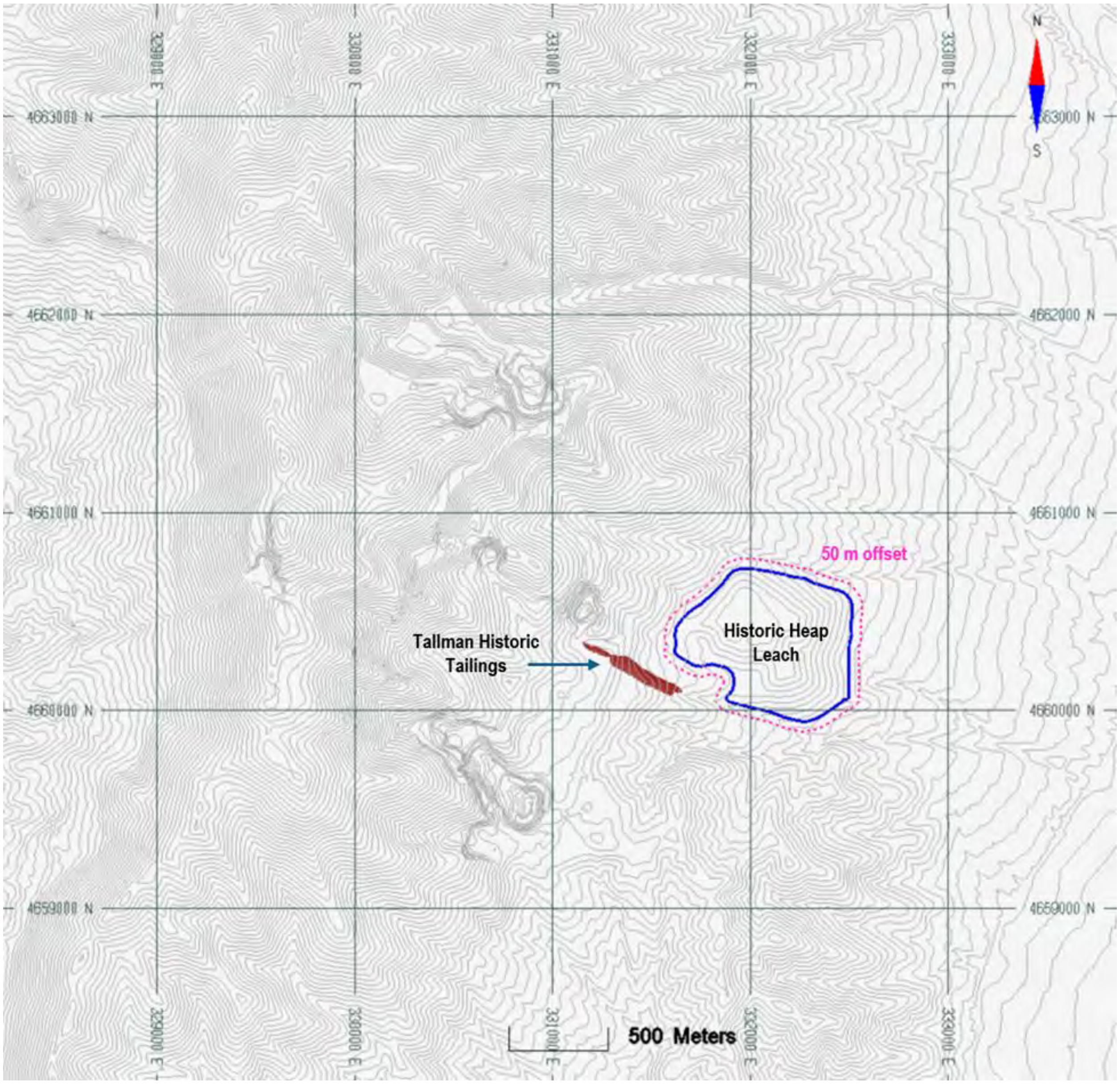
Mining is completed on 10 m benches and the block size within the model is 10 m x 10 m in the horizontal and 5 m high. The skin thickness chosen for dilution is based on the spatial nature of the mineralization, proposed grade control methods, equipment size, digging accuracy, and blast heave. A series of dilution runs varying the skin thickness was evaluated and helped determine the dilution skin thickness.

Comparing the in-situ values to the net diluted values within the final mining limits, the diluted feed contained 3.7% more tonnes and a 1.4% lower gold grade than the in-situ feed summary. AGP considers these dilution percentages to be reasonable

The mining dilution is applied in the final mining schedule but was not used during pit optimization. No ore loss is applied, and recovery is assumed to be 100% for all ore material.

16.5.1 Pit Restrictions

Pit restrictions include a 50-meter offset from the historic heap leach facility and the Tallman historic tailings. The restrictions are shown in Figure 16-2.

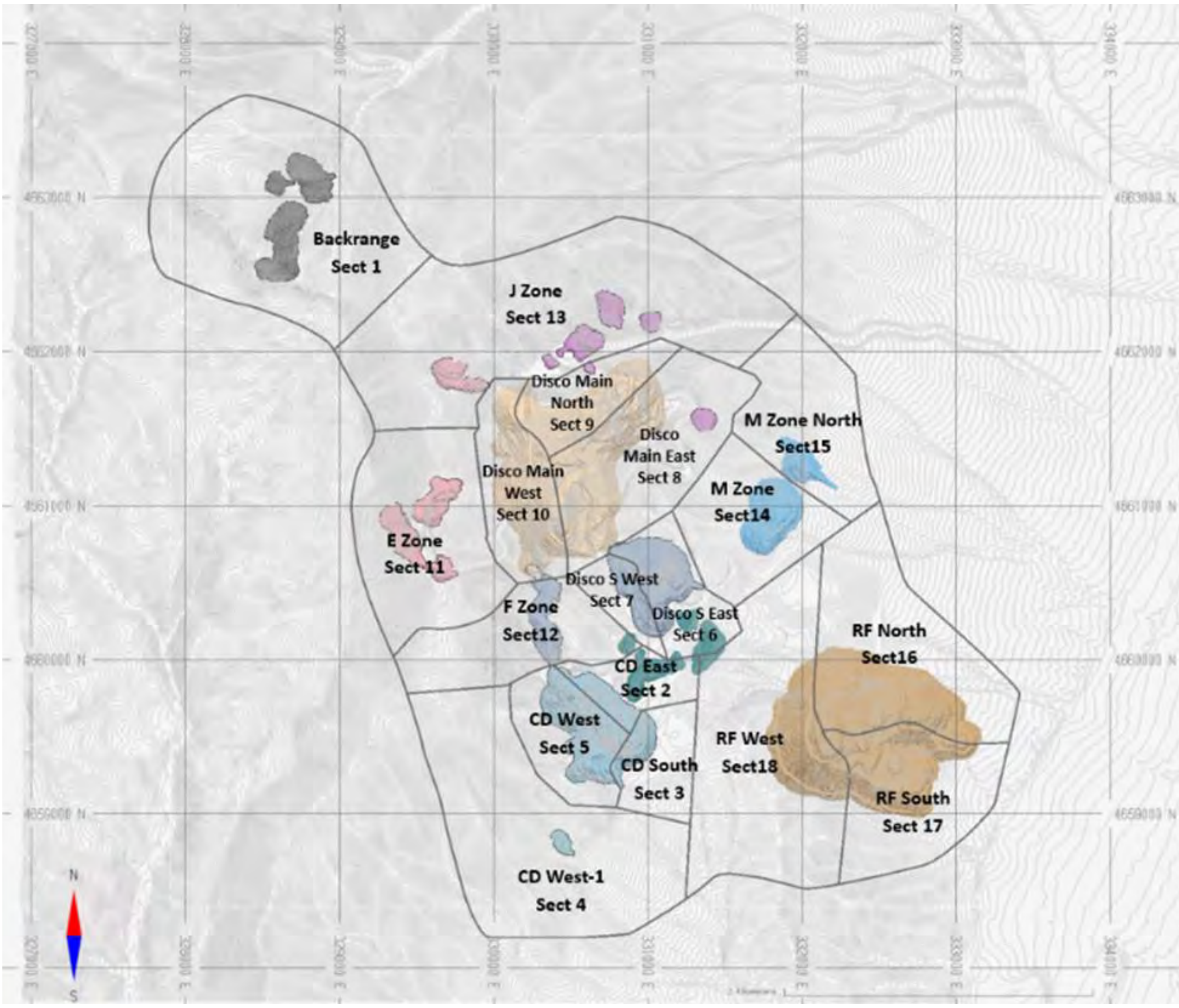


Source: AGP, 2024

Figure 16-2: Pit Restrictions

16.5.2 Slopes Angle Recommendations

Wall slope angles for pit optimization were based on guidance from Knight Piesold Consulting in a draft letter issued January 17, 2024. A design sector map was created which was defined by the scoping study pit designs and dominant geotechnical units. Slope sector areas were used to code the model SLPCD item using the area polygons in Figure 16-3.



Source: AGP, 2024

Figure 16-3: Slope Sector Areas

The overall slope angle is determined by the inter-ramp slope angle, safety berm width, and the projected number of ramps that pass-through a given slope sector. The overall slope includes a geotechnical step out that is recommended in place of the typical berm for every 160 m vertical height not intersected by a haul ramp. A 20 m geotechnical berm was added to the Discovery Main west and the Rangefront west sectors due to these walls exceeding the 160 m of uninterrupted vertical height.

The overall slope parameters used in the optimization package include the necessary ramps and geotechnical berms within each sector to accurately reflect the final wall slope configuration and minimize the variance between the optimized shapes and the actual design. Previous pit designs were used to estimate the number of ramps and geotechnical berms required in each sector. Double benching (2 x 10 m high) is recommended for the pit slope development at Black Pine. Calculated OSA angles by sector are given in Table 16-8.

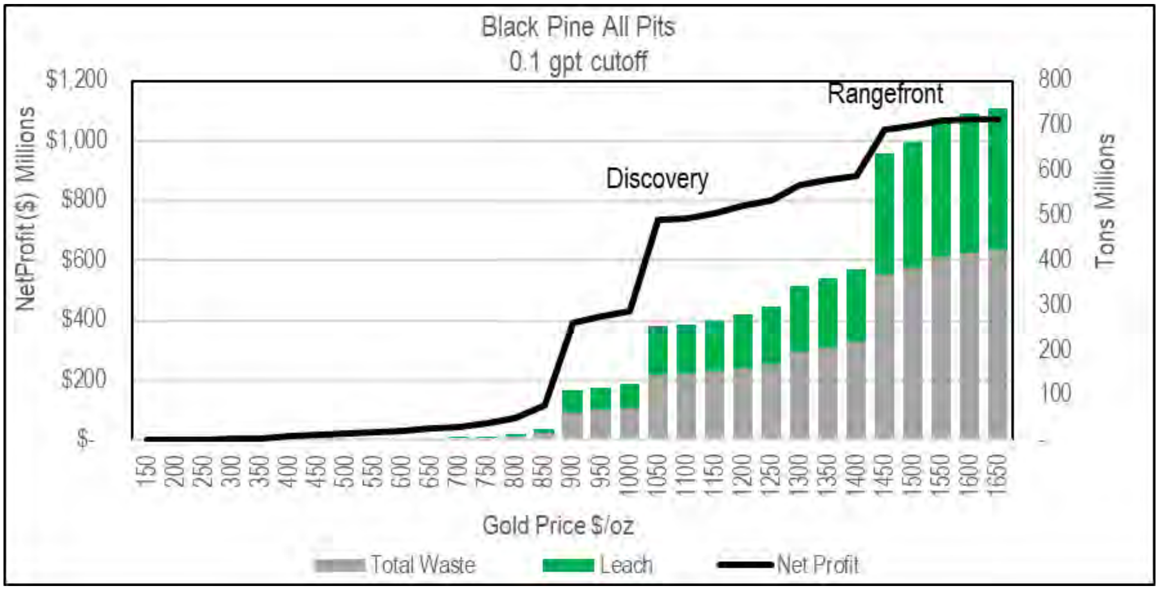
Table 16-8: Overall Slope Angles and Slope Parameters by Sector

Sector	Osa (°)	Ira (°)	Face (°)	Berm (m)	Sector Code	Bench Height (m)	Stacked Height (m)
Backrange	45.0	54.3	75.0	9.0	1	10	20
CD East	54.0	54.3	75.0	9.0	2	10	20
CD South	37.0	49.2	70.0	10.0	3	10	20
CD West	54.0	54.3	75.0	9.0	4	10	20
CD West 1	43.0	50.9	70.0	9.0	5	10	20
Disco Southeast	48.0	54.3	75.0	9.0	6	10	20
Disco Southwest	50.0	50.9	70.0	9.0	7	10	20
Disco Main East	46.0	54.3	75.0	9.0	8	10	20
Disco Main North	46.0	50.9	70.0	9.0	9	10	20
Disco Main West	45.0	49.2	70.0	10.0	10	10	20
E Zone	45.0	54.3	75.0	9.0	11	10	20
F Zone	54.0	54.3	75.0	9.0	12	10	20
J Zone	54.0	54.3	75.0	9.0	13	10	20
M Zone	45.0	54.3	75.0	9.0	14	10	20
M Zone North	50.0	50.9	70.0	9.0	15	10	20
Rangefront North	47.0	50.9	70.0	9.0	16	10	20
Rangefront South	44.0	50.9	70.0	9.0	17	10	20
Rangefront West	43.0	49.2	70.0	10.0	18	10	20
Default	47.0	54.3	75.0	9.0	19	10	20

16.5.3 Price Shells

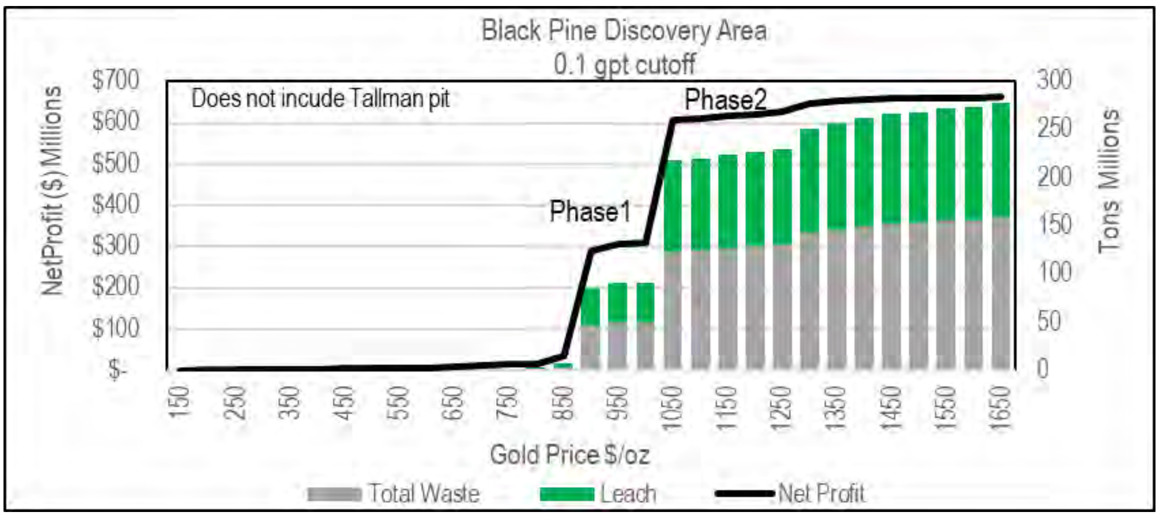
Pit optimization was run using incremental gold prices to generate a set of nested pit shells to US$ 1,650 /oz. gold. The incremental price shells, run at 0.1 g/t cutoff, guide the selection for final pit extents along with pushbacks leading to the final pit design. The initial pit optimization was conducted over the entire Black Pine block model and later runs were constrained to the larger Discovery and Rangefront pits. The constrained runs for Discovery and Rangefront helped inform the phasing of those pits.

Leach tonnages, waste tonnages, and potential net profit at $50/ oz gold price increments for the site wide optimization shells are shown in Figure 16-4. Figure 16-5 and Figure 16-6 show the results of nested pit analyses for the Discovery and Rangefront pits. Figure 16-7 is a plan view of the $1,650/ oz optimization shells.



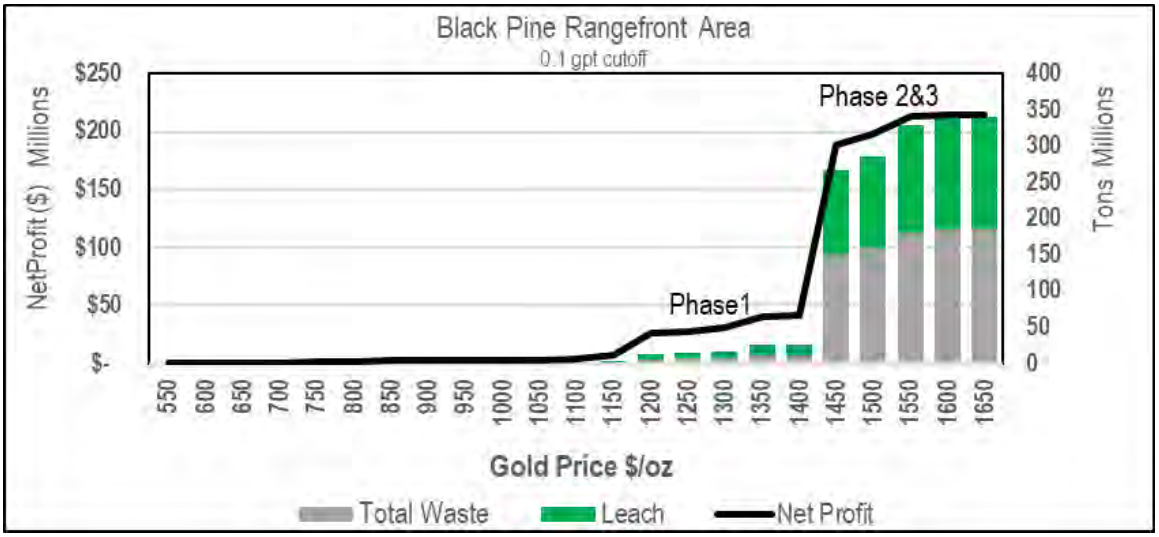
Source: AGP, 2024

Figure 16-4: Gold Price Optimization of All Areas



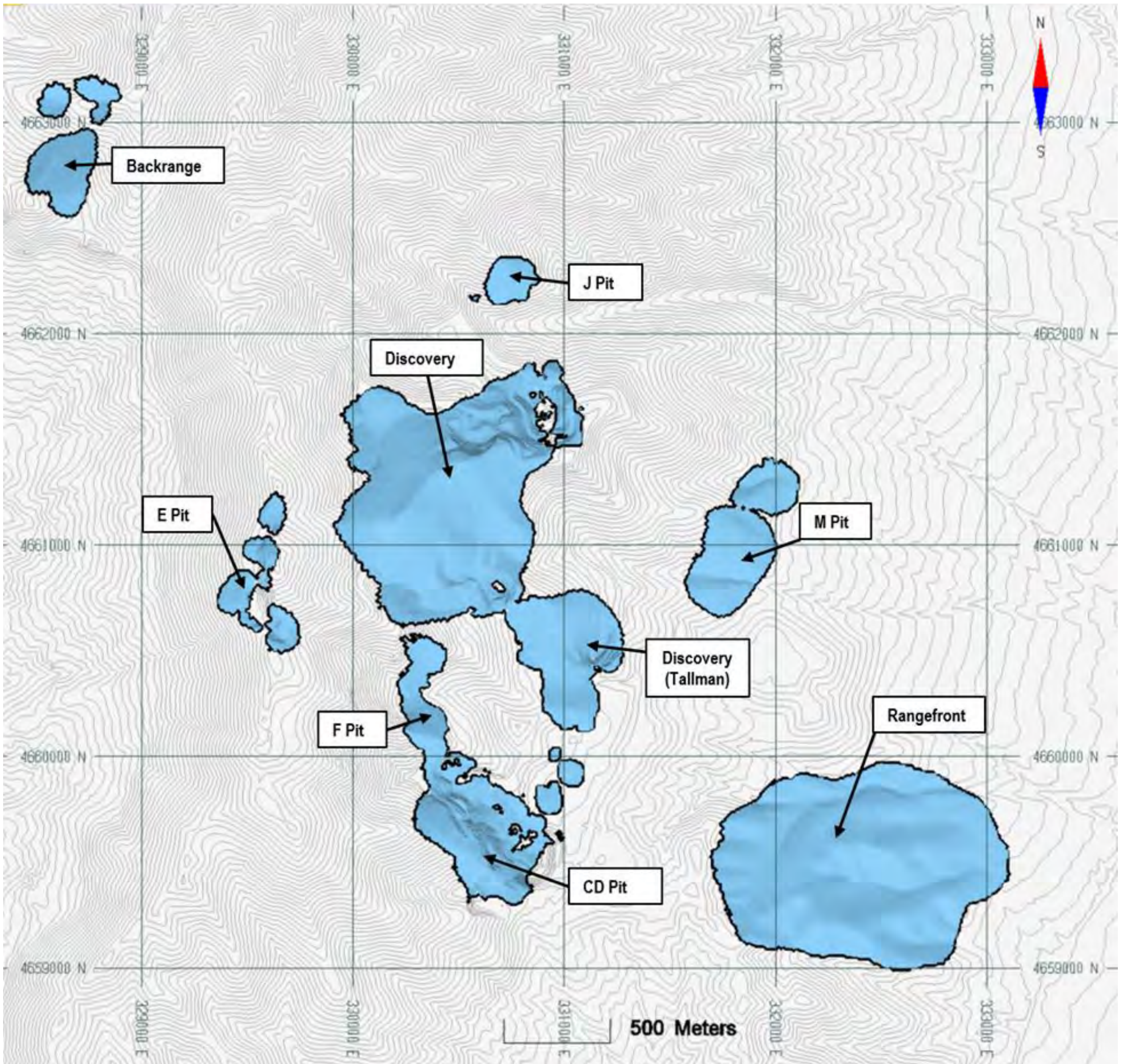
Source: AGP, 2024

Figure 16-5: Gold Price Optimization Discover



Source: AGP, 2024

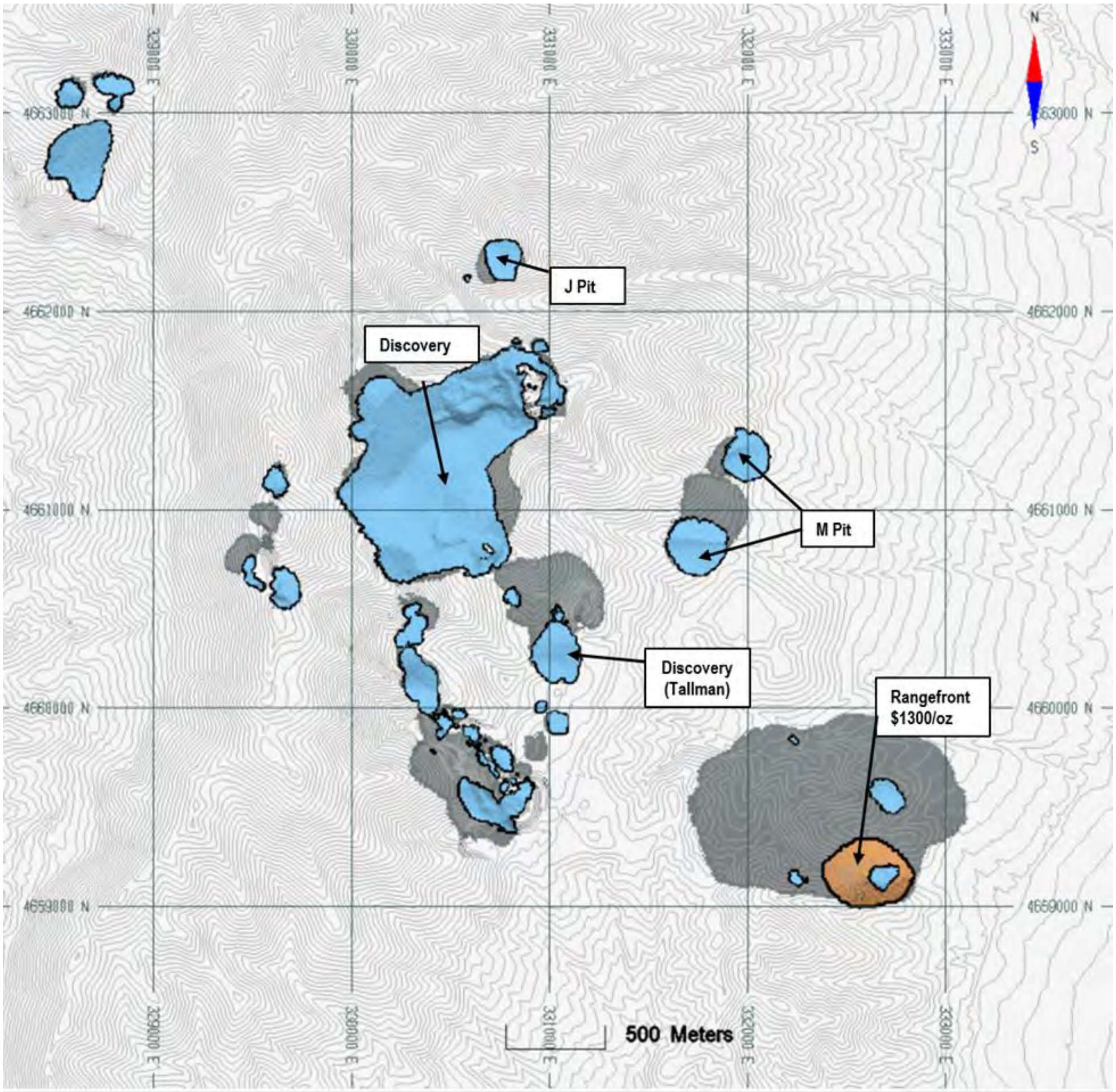
Figure 16-6: Gold Price Optimization Report



Source: AGP, 2024

Figure 16-7: View of the Pit Optimization Solids

A second set of optimization shells were run at an elevated cutoff of 0.2 gpt to further identify pit shells and areas of higher grade and higher value for targeting early mining and phase development. The second set of pit shells led to the selection of the $1,250/oz pit shells at a 0.2 gpt cutoff being the basis for higher value starting pits at Discovery, Tallman, J-Pit, and M pit. A $1,300/oz pit shell was used for the starting phase at Rangefront and the second phase at Discovery. Limited space in the F and CD pits prevented the addition of internal pushbacks and these pits are mined to the ultimate pit limits in one phase. Figure 16-7 shows the 0.2 gpt cutoff pit shells at a $1,250/oz gold price.



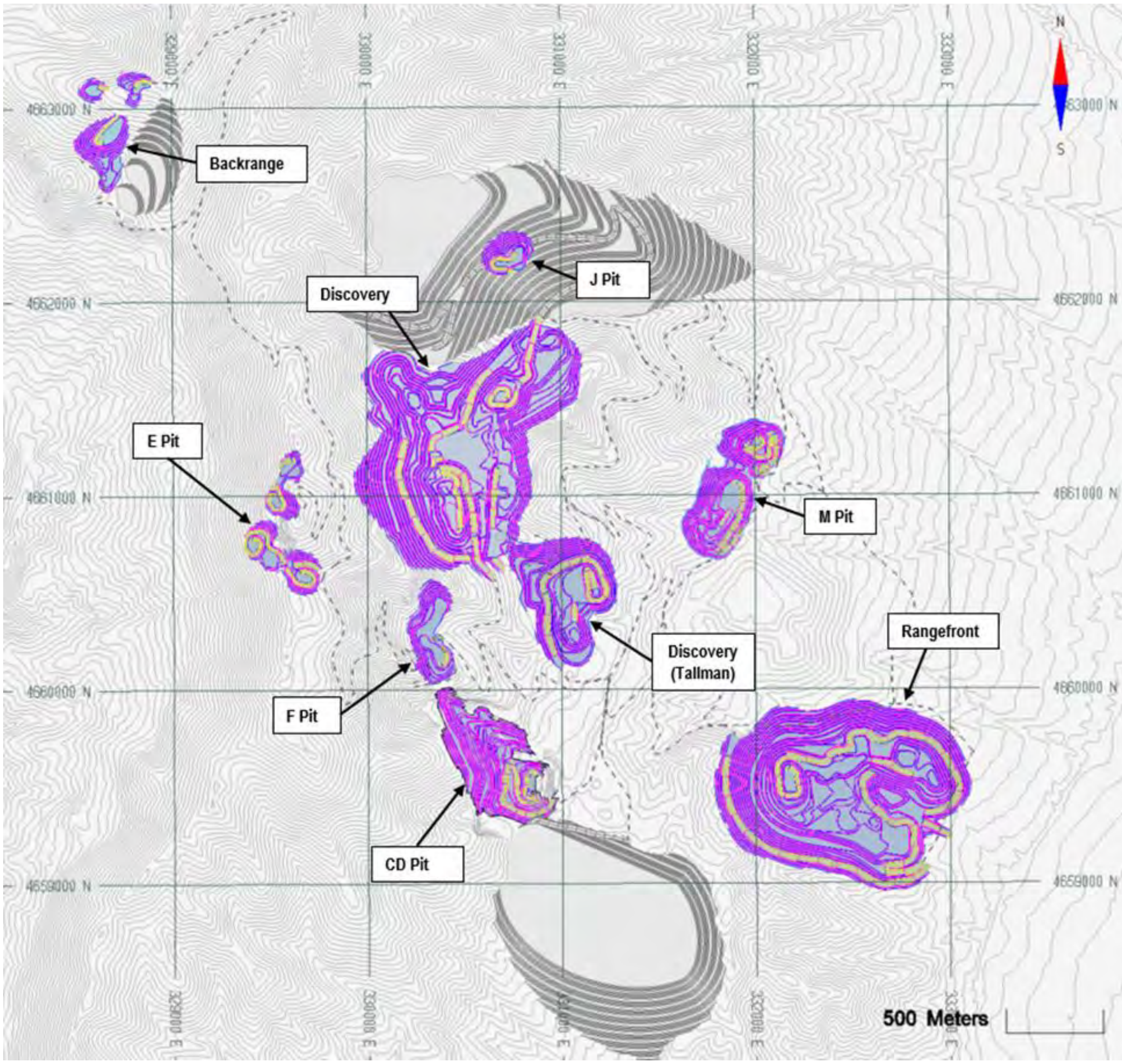
Source: AGP, 2024

Figure 16-8: Pit Optimization Shells for Early Pushbacks

16.6 Pit Designs

A conventional open-pit, truck-and-shovel mining method will be used for the Black Pine Gold Project. Haul truck sizing includes 144 tonne rigid-frame haul trucks and 64 tonne rigid-frame haul trucks. The small pits, pioneering cuts, and mining areas with narrow bench widths and limited mining space will be mined with the smaller mining equipment. The 64 tonne trucks with decreased ramp width will also reduce the construction and maintenance cost on the surface roads that transverse the steep terrain leading to the pits located at the top of the Black Pine mountains.

The Black Pine open pits are split into 9 areas: Discovery, Tallman, M, J, F, E, CD, Backrange, and Rangefront. Discovery, Tallman, M and Rangefront are developed in three phases. J, E, and Backrange have two phases and F, and CD are developed with a single phase. Figure 16-9 shows the location of ultimate pit designs at Black Pine. The ultimate pit and phase designs consist of double benching with an operating bench height of 10 meters.



Source: AGP, 2024

Figure 16-9: Ultimate Pit Designs

The larger pits Discovery, CD, Tallman, and Rangefront will primarily use 144 tonne trucks, while the smaller pits including E and Backrange that are accessed by steep mountain terrain will utilize 64 tonne trucks. The road width is set to 30.2 m for the 144 tonne trucks while the 64 tonne trucks have a 22.8 m width. The haul road width allows 3 times the operating width of the haul truck, a drainage ditch and a safety berm (3/4 tire height). Typical ramps grades are designed at 10%. The haul road design criteria is shown in Table 16-9.

Table 16-9: Haul Road Design Criteria

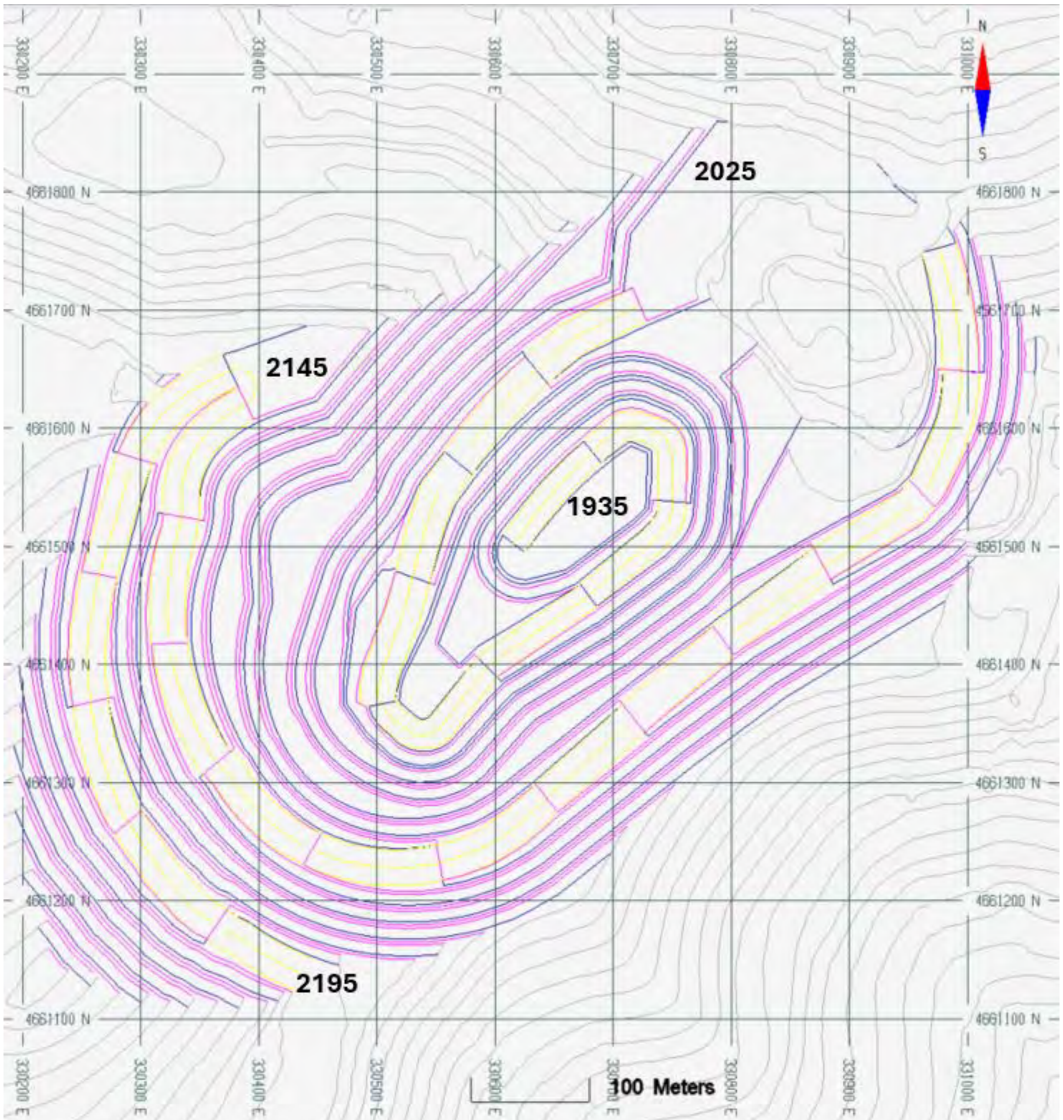
Ramp Attribute	144 t trucks	64 t trucks
Double Ramp width (m)	30.2	22.8
Single Ramp width (m)	27.7	17.1
Ramp Gradient (%)	10%	10%
Switchback design	Flat	Flat

16.6.1 Discovery

The Discovery Main pit is developed in three phases and is the main ore feed source for the first seven years of production at Black Pine.

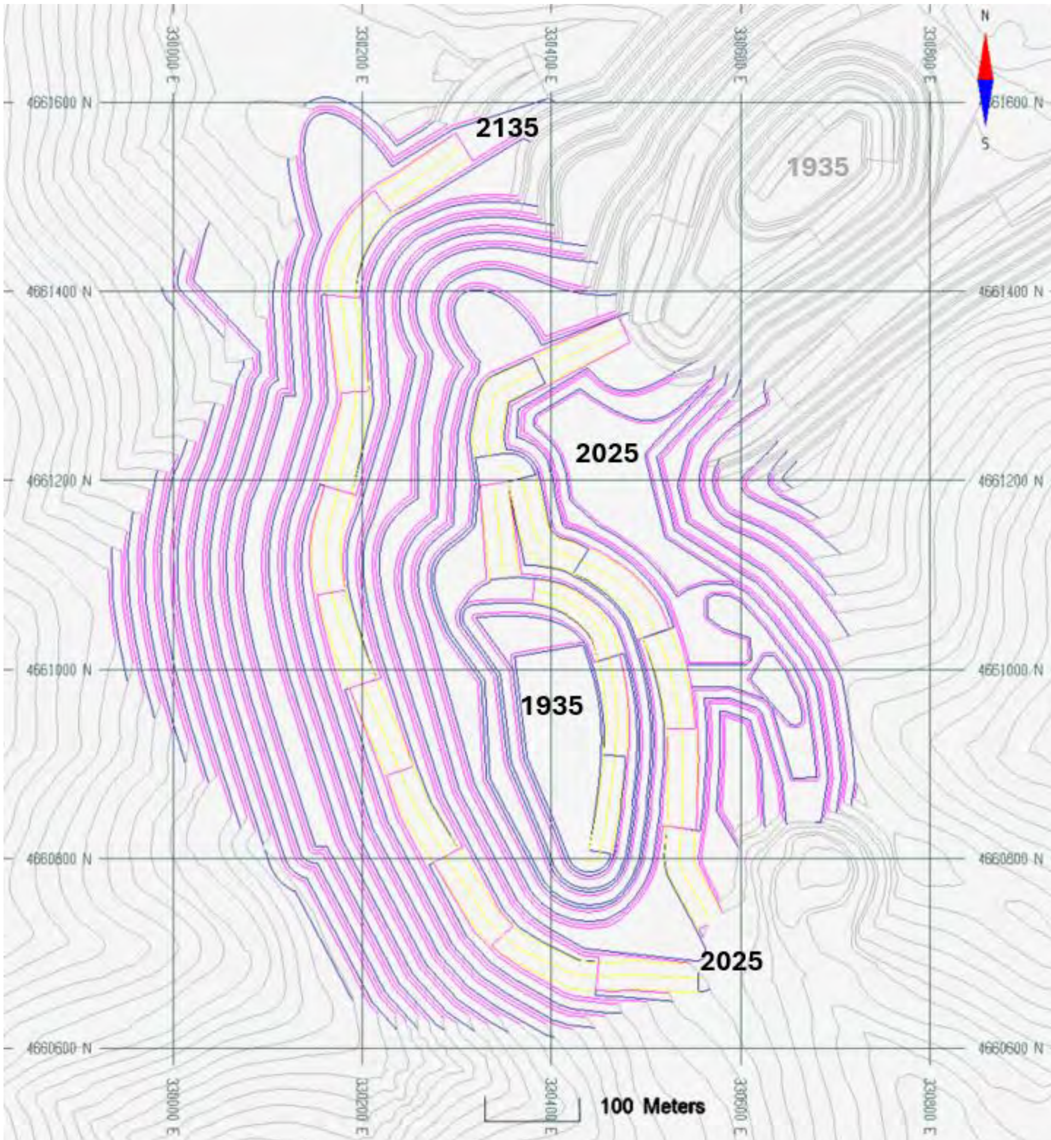
Phase1 is located on the northeast side of the Discovery deposit and will start mining during the pre-production period, Year -1. Bench elevations range from 2,305 meters to 1,935 meters. Waste material will be routed to the Discovery Waste Rock Facility using the 2045 and 2015 pit exit ramps and ore will primarily use the ramp exits located at the 2,015 and 2,195 elevations. Ore mined during pre-production will be sent to the stockpile for processing in later years. See Figure 16-10 for the Discovery Phase 1 design.

Discovery Phase 2 is mined from Year 2 to Year 5 and is located southwest of Phase 1. Bench elevations range from 2,325 meters to 1,935 meters. A 20 m geotechnical berm is located on the west wall at the 2,155 elevation. The main waste exit is adjacent to the Discovery Waste Rock Facility at the 2,135 elevation. Some waste material in the upper benches will be directly hauled to the waste rock facility. Ore will mainly be routed using the south pit exit located near the Tallman pit at the 2025 elevation although the existing ramp system established in Phase 1 will be used when it is more practical. The Discovery Phase 2 design is shown in Figure 16-11.



Source: AGP, 2024

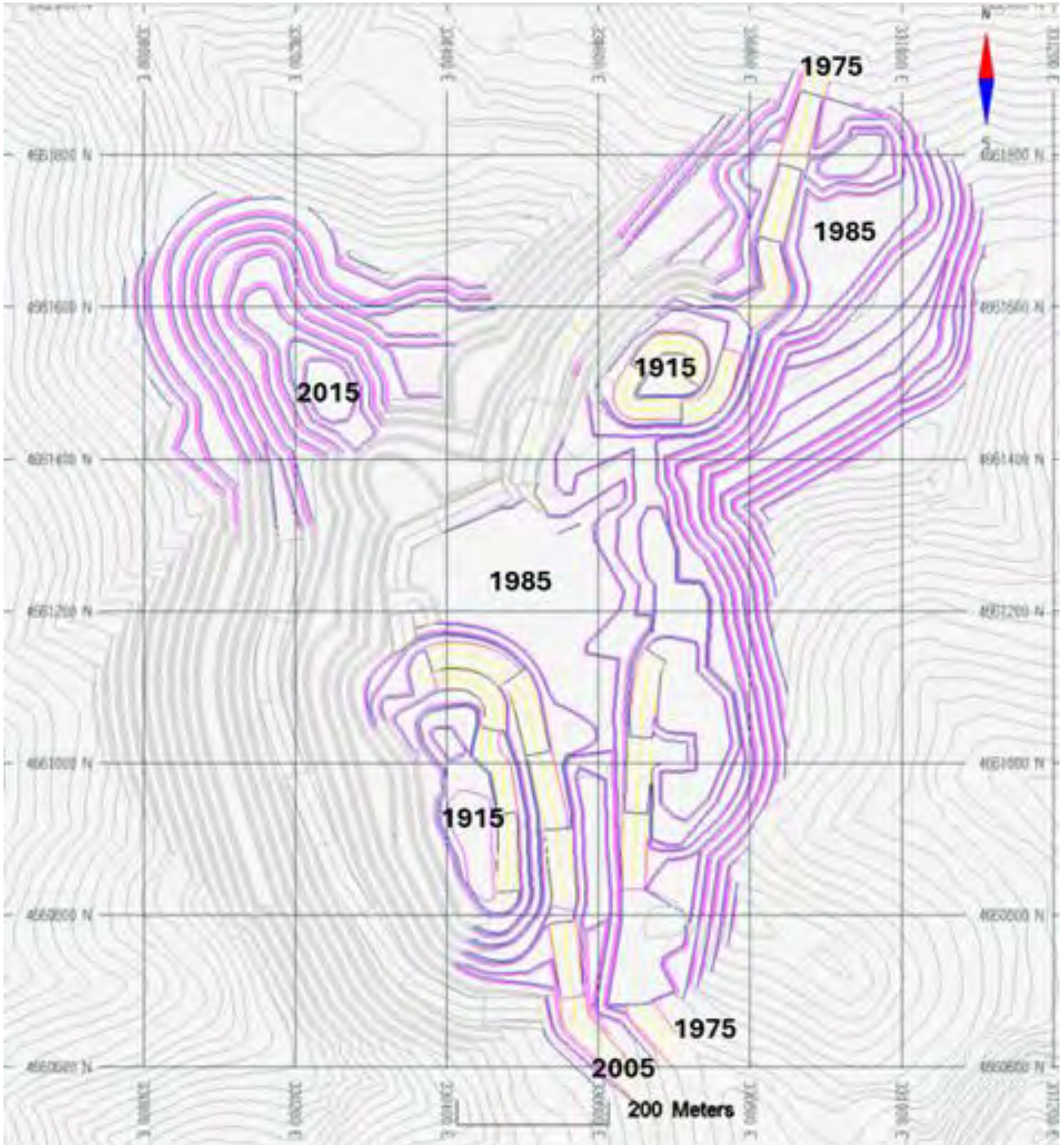
Figure 16-10: Discovery Phase 1



Source: AGP, 2024

Figure 16-11: Discovery Phase 2

The final phase in the Discovery Main pit is Phase 4 which is split into east and northwest sections, Phase 4a and 4b respectively. Phase 4a bench elevations range from 2,155 meters to 1,915 meters. Phase 4b bench elevations range from 2,185 meters to 2,015 meters as shown in Figure 16-12. Phase 4 is mined from Year 5 through Year 7. The pit exit to the north is at the 1,975 elevation while the south ramps exit at the 2,005 and 1,975 elevations. Phase 4b has no ramps designed in the final wall configuration and material will be directly hauled off the benches or utilize the existing haul ramps from Phase 2.

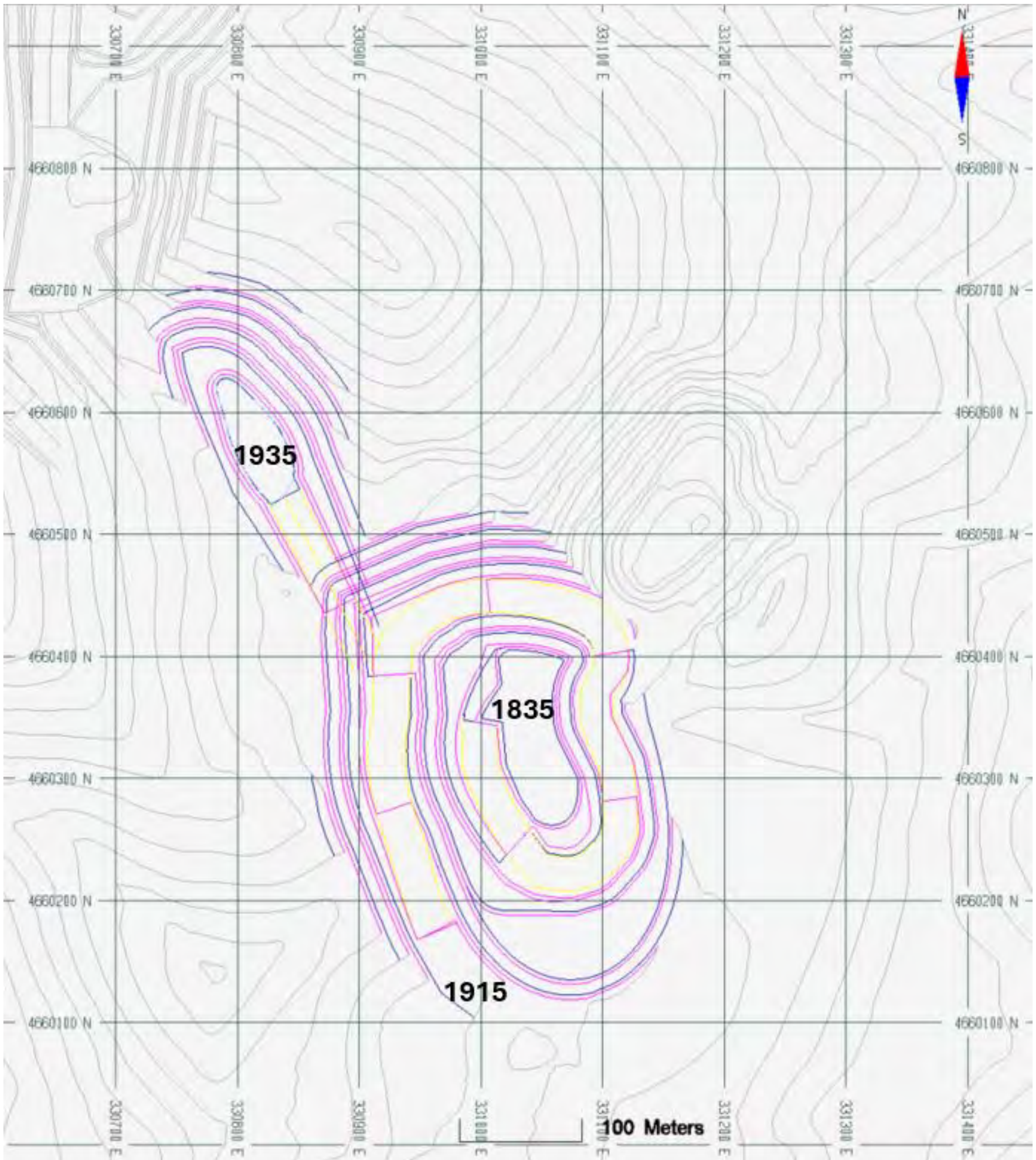


Source: AGP, 2024

Figure 16-12: Discovery Phase 4

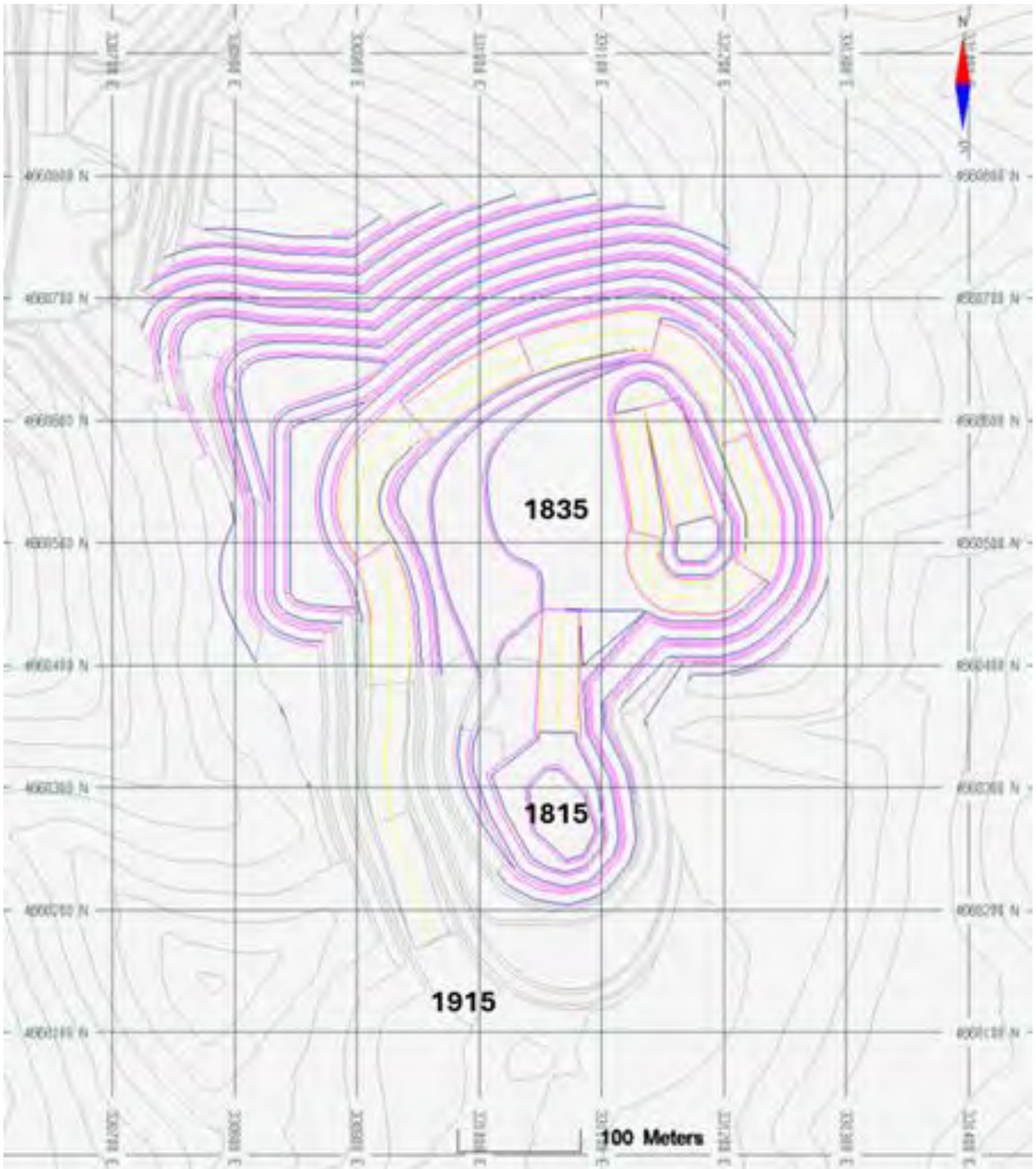
16.6.2 Tallman (Discovery Phase 3)

The Discovery Tallman pit has three phases that are referred to as Discovery Phase 3a, 3b, and 3c in the pit phase naming. The first two phases, 3a and 3b, are small higher-grade pits that are mined in Years 3 and 4. Phase 3c mining is complete in Year 8. The Discovery Phase 3 bench elevations range from 2,065 meters to 1,835 meters. Access to Discovery Phase 3 is to the south at the 1,915 elevation. The Discovery phase designs are in Figure 16-13 and Figure 16-14.



Source: AGP, 2024

Figure 16-13: Discovery Phase 3a-3b (Tallman)



Source: AGP, 2024

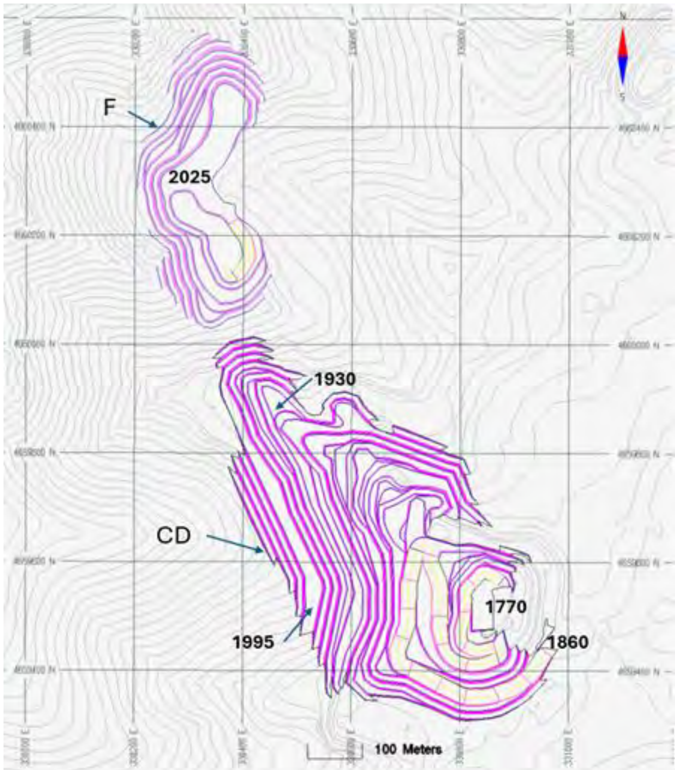
Figure 16-14: Discovery Phase 3c (Tallman)

16.6.3 CD and F Zone pits

CD and F pits are located directly south of the Discovery pit and are each mined with a single phase. CD bench elevations range from 2,065 meters to 1,770 meters and F bench elevations are from 2,145 meters to 2,015 meters. CD mines west of and up the steep terrain located west of the previously mined CD pit. F pit is in an area that was not previously mined. A geotechnical catch bench with a 20 m minimum width is in the west wall of the CD pit at the 1995 elevation. F pit is completely mined during preproduction year -1 with ore sent to stockpiles for later placement on the leach pad. CD is mined starting in Year 3 and is completed in Year 6. The upper benches in CD will require dozing and mining with the smaller excavator and trucks to establish the small ore bearing benches in the steep west wall. A wider ramp is provided from the 1,860 to 1,805 elevation in the CD pit and these benches can accommodate the large mining equipment. CD and F Zone pit designs are shown in Figure 16-15.

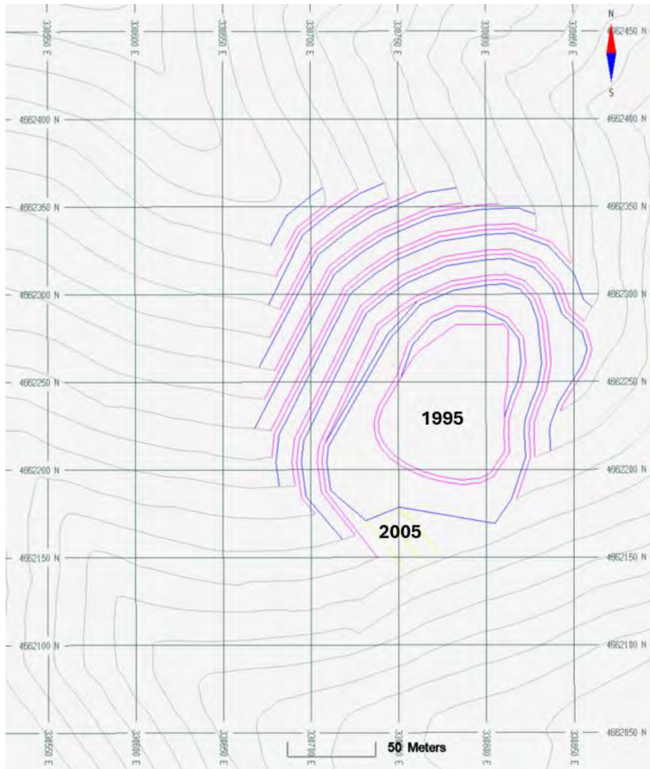
16.6.4 JZone

J pit is a small pit located North of Discovery that is mined in two phases. The first phase, mined in Years 1 and 2, targets higher grade material. Phase 2 is completed in Year 3. By the end of the mine life, J pit is covered by the Discovery Waste Rock Facility ("WRF"). J bench elevations range from 2,125 meters to 1,985 meters. J Zone phase designs are shown in Figure 16-16 and Figure 16-17.



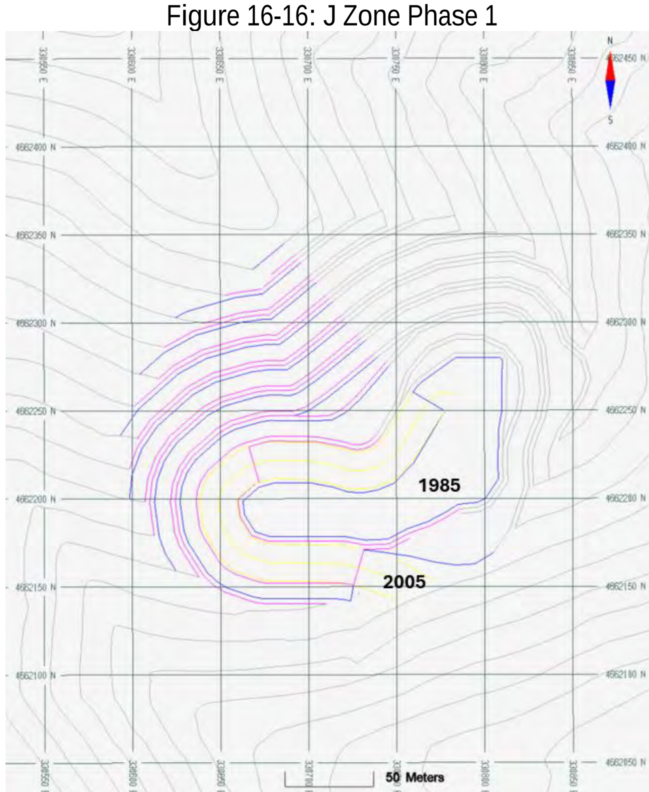
Source: AGP, 2024

Figure 16-15: CD and F Zone Pits



Source: AGP, 2024

Figure 16-16: J Zone Phase 1

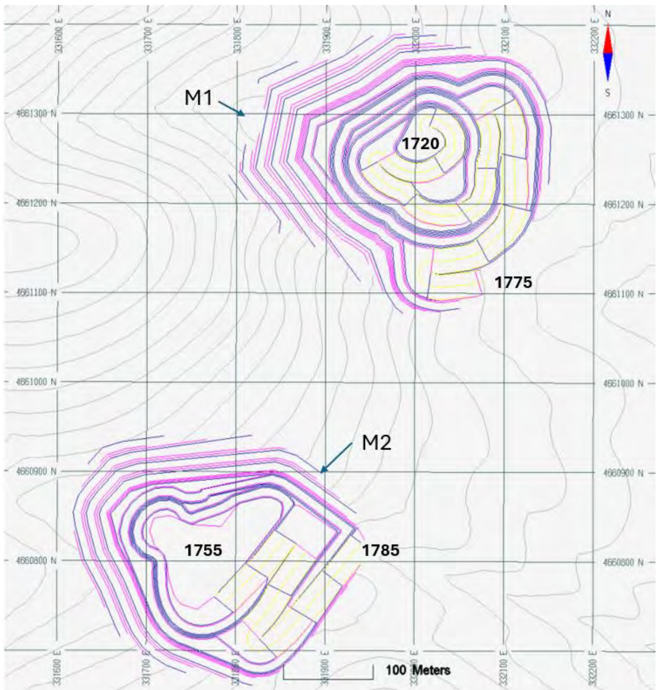


Source: AGP, 2024

Figure 16-17: J Zone Phase 2

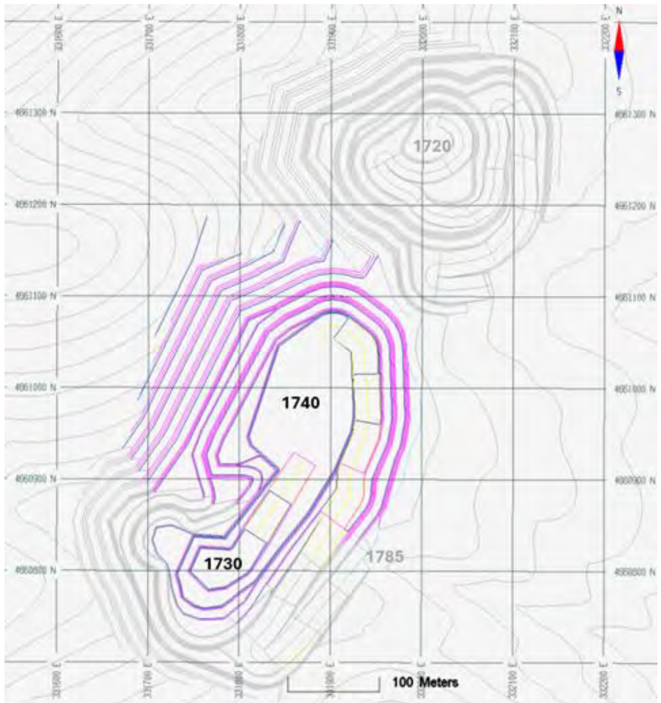
16.6.5 M Zone

M Zone is located east of the Discovery pit and is near the haul road leading from Discovery to the heap leach pad. M is mined in three phases. M pit Phases 1 and 2 are mined in Years 1 through 4. M zone Phase 3 mining starts in Year 7 and completed in Year 8. M has a higher waste to ore ratio than other pits at Black Pine with a 3.9:1 strip ratio. M zone bench elevations range from 1,905 meters to 1,720 meters. M Zone phase designs are shown in Figure 16-18 and Figure 16-19. The M zone pit is backfilled to the 1800m elevation after year 8.



Source: AGP, 2024

Figure 16-18: M Zone Phase 1 and 2



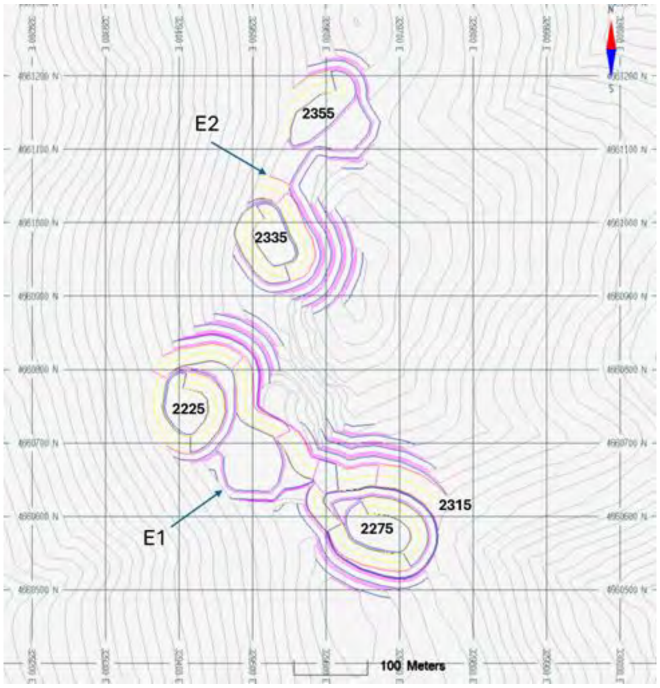
Source: AGP, 2024

Figure 16-19: M Zone Phase 3

16.6.6 E Zone and Backrange

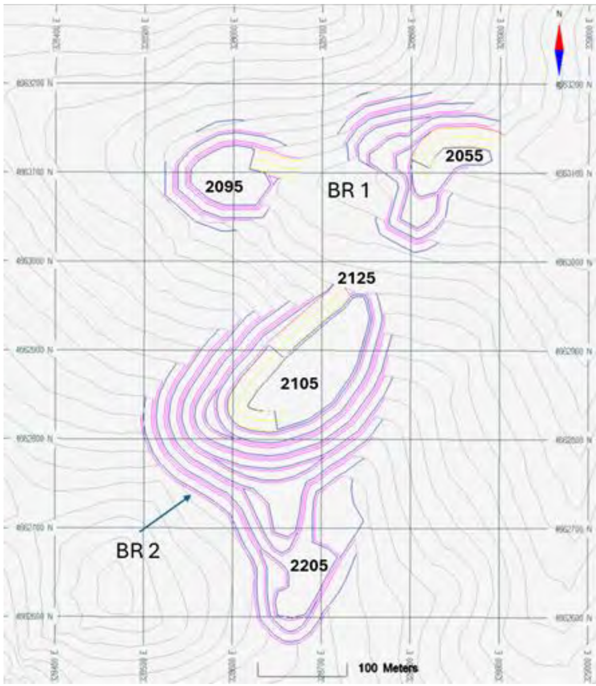
E Zone is on the ridgeline west of the Discovery pit, at an elevation of 2,435 meters to 2,255 meters. There are two independent phases as shown in Figure 16-20. Mining begins at E zone in Year 5 and all mining is complete by Year 8. Access to E zone is a surface road that runs from approximately 1,985 masl to 2,400 masl over 4 km and connects to the F zone surface haul road.

Backrange is a small higher grade deposit northwest of the Discovery pit and located on the back side of the ridgeline from the other Black Pine deposits. Access to back range is by a 4 km surface road that connects to the E zone surface haul road. Backrange has two phases as shown in Figure 16-21. Backrange is mined in Years 5 and 6. The Phase 1 elevation is from 2,135 meters to 2,055 meters, and Phase 2 elevation is from 2,265 meters to 2,105 meters.



Source: AGP, 2024

Figure 16-20: E Zone Phase 1 and 2



Source: AGP, 2024

Figure 16-21: Backrange Phase 1 and 2

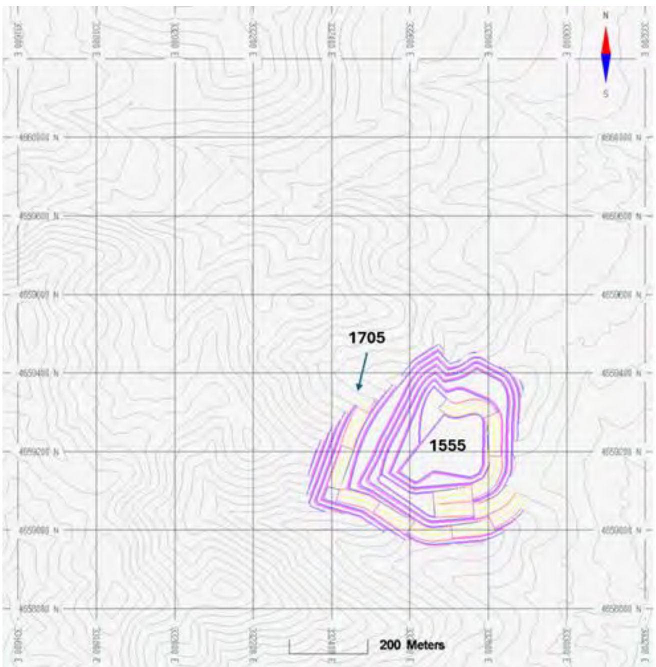
16.6.7 Rangefront

Rangefront is a large pit developed in three phases and located south of the historical and proposed leach pad area.

Phase 1 is a higher value pit that is mined in Years 7 and 8. Bench elevations range from 1,765 meters to 1,555 meters. The Rangefront Phase 1 design is shown in Figure 16-22.

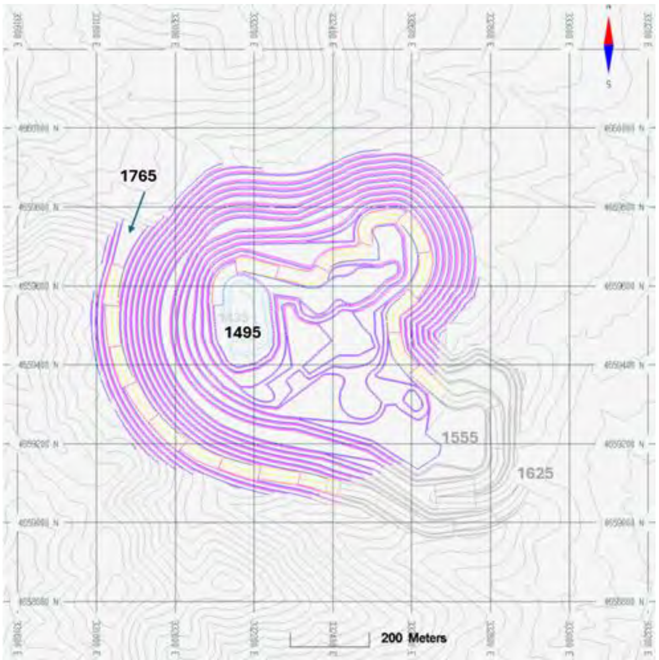
Rangefront Phase 2 extends Phase 1 to the west and is a large pushback that is mined continuously from Year 8 until the end of mining in Year 15. The initial pit design is to the 1,435 elevation, however the mining schedule does not mine below the 1,465 elevation in Phase 2 due to ground water concerns and the pit is truncated at this level. The Rangefront Phase 2 design is shown in Figure 16-23.

Phase 3 mines the northeast part of the Rangefront deposit in Year 10 through Year 15. Bench elevations range from 1,765 meters to 1,465 meters. See Figure 16-24 for the Rangefront phase 3 design.



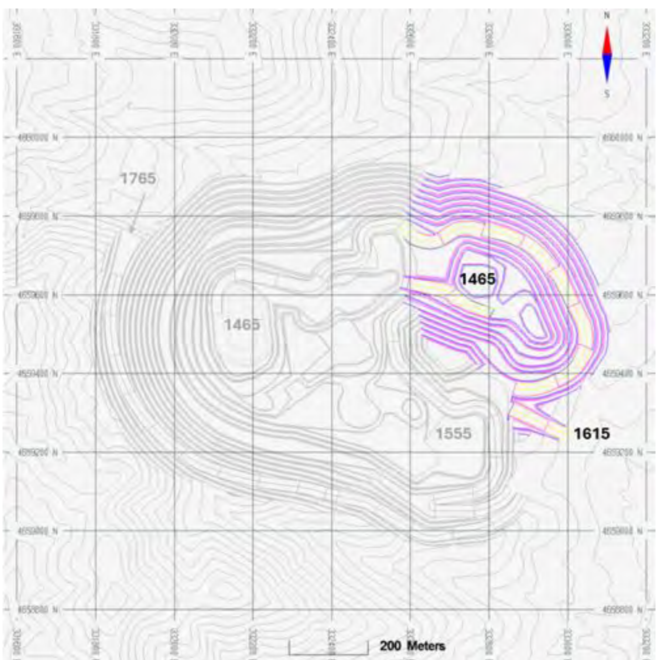
Source: AGP, 2024

Figure 16-22: Rangefront Phase 1



Source: AGP, 2024

Figure 16-23: Rangefront Phase 2



Source: AGP, 2024

Figure 16-24: Rangefront Phase 3

16.7 Waste Rock Storage Facilities

There are two main WRFs. The Discovery WRF is a valley fill located east of the Discovery pit and the Rangefront WRF is located southwest of the Rangefront pit near the CD pit. The smaller Backrange WRF is dedicated to waste rock from the Backrange pits and is located just east of southernmost Backrange pit. CD and M pits are backfilled with waste rock. The WRD capacity is given in Table 16-10. The waste rock facilities are shown in Figure 16-25.

The design of the rock waste storage facilities used a swell factor of 1.25 considering some compaction will occur in the dumps. All waste rock storage facilities were designed with a 37° face slope and overall slopes of 25.2° (2.15:1V). A 16 m catch bench is used for every 20 m of lift. Ramp access on the waste facilities is designed at 30.2 m to accommodate the larger trucks.

Table 16-10: Waste Rock Facility Capacity

Waste Rock Facility	Volume (M^3)	Tonnes (Mt)
Discovery WRF	104.9	205.7
Rangefront WSF	74.4	145.9
Backrange WSF	3.2	6.3
M pit Backfill	6.3	12.4
CD Pit Backfill	26.0	60.0
Total Capacity	214.8	430.0

16.8 Stockpile Facilities

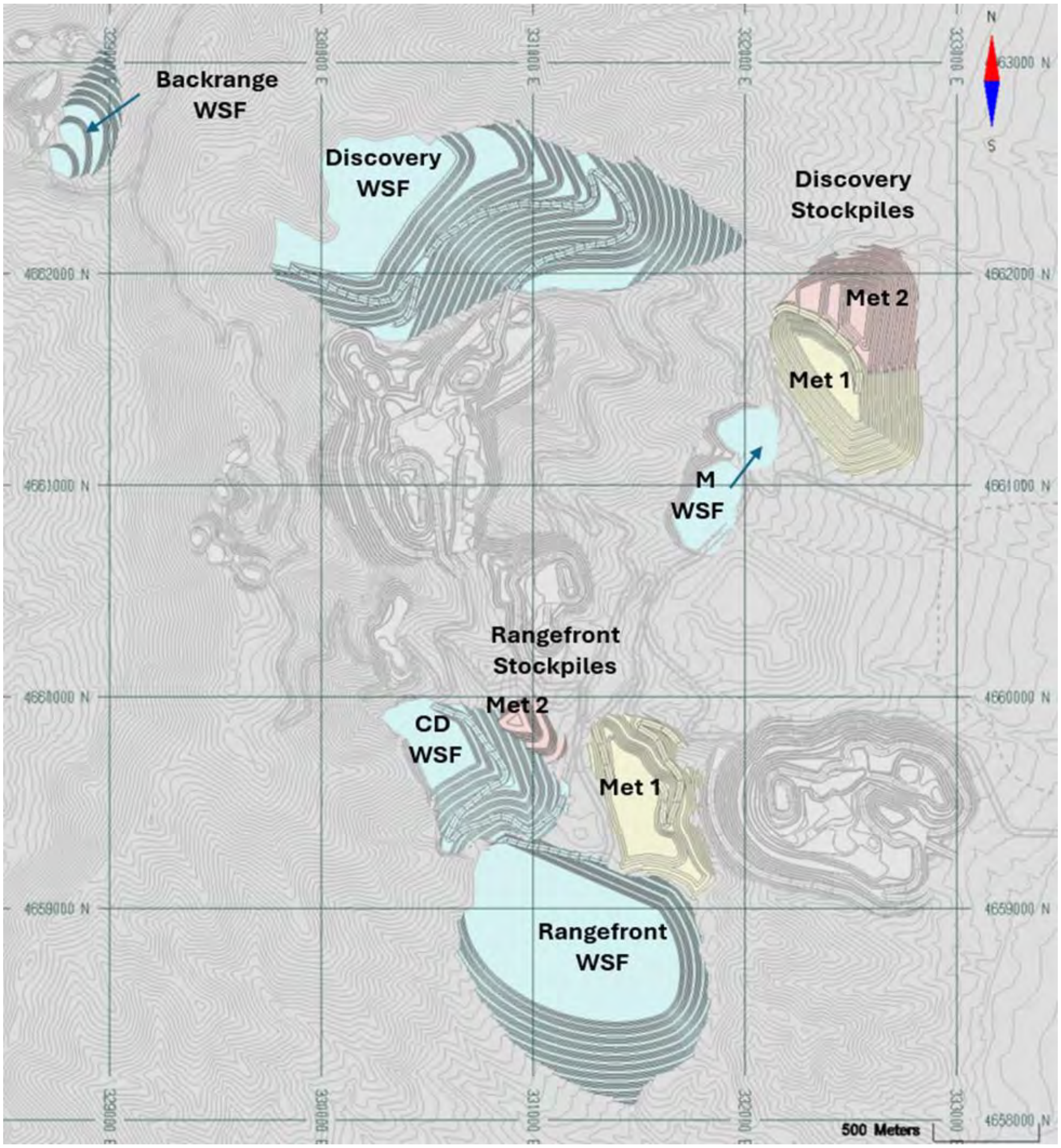
Leach feed is stockpiled during the pre-production period and throughout the mine life. Leach ore is sent to the different stockpiles based on the gold cyanide solubility codes in the resource model.

- Met 1: gold cyanide solubility > 65%.
- Met 2: gold cyanide solubility > 50%, <65%
- Met 3: gold cyanide solubility > 25%, <50%

Met 2 and Met 3 material is sent to a specified area of the heap leach pad apart from the Met 1 ore. Different stockpiles are also established for the Met 1 and the Met 2-3 material. Future studies will consider separating the Met 2 and Met 3 material on the leach pad and distinct stockpiles for each will be required. Stockpile capacity for the Project is shown in Table 16-11 and stockpile locations in Figure 16-25.

Table 16-11: Stockpile Capacity

Stockpile Capacity	Volume(M^3)	Tonnes (Mt)
Discovery Met 1	26.7	52.3
Rangefront Met 1	10.1	19.9
Discovery Met 2-3	9.2	18.0
Rangefront Met 2-3	1.2	2.4
Total Capacity	47.2	92.6

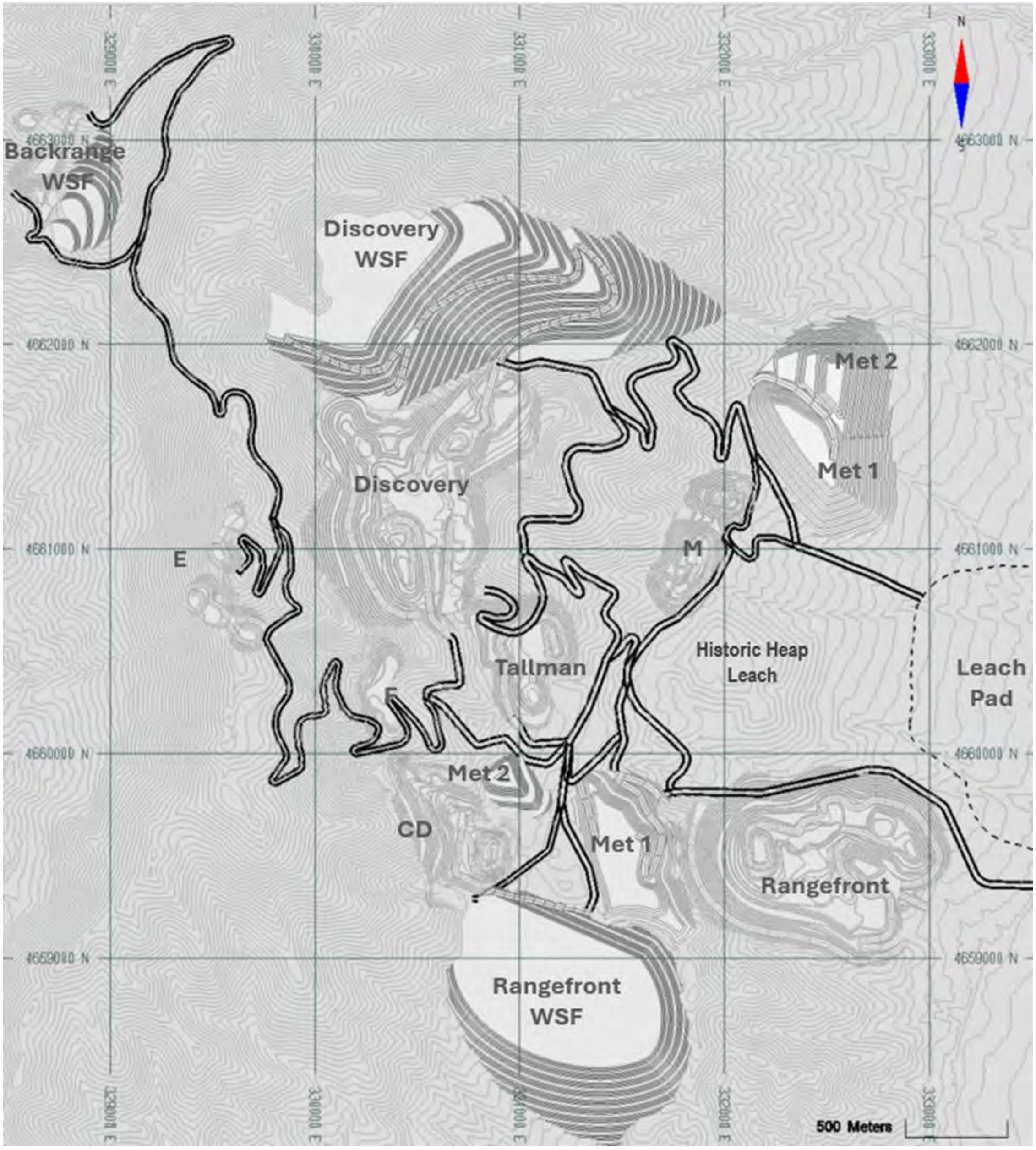


Source: AGP, 2024

Figure 16-25: Waste Rock Facilities and Stockpiles

16.8.1 Surface Haulage – Road Network

An extensive surface haul road network will need to be constructed for access from the pits to the waste rock facilities, stockpile locations, heap leach pad, and other facilities. Figure 16-26 shows the ex-pit haul roads at Black Pine.



Source: AGP, 2024

Figure 16-26: Mine Surface Haul Roads

There are over 24 km of ex-pit surface haul roads that range from an elevation of 2,400 masl at the ridgeline near E pit to 1,515 masl at the lower end of the leach pad. The capital cost for the construction of the haul roads, as well as the road maintenance costs are in Section 21. The surface haul routes, and the in-pit ramps are used to compute truck cycle times that inform operating costs and haul truck requirements.

16.9 Mine Plan

The mine schedule plans to deliver 299 Mt of ore to the leach pad grading 0.323 g/t gold over the 17-year project life including 2 years of processing stockpiled ore after mining has ended. Waste tonnage from all pits totaling 394 Mt is placed into waste storage facilities including the back filling of the CD and M pits. The overall open pit strip ratio is 1.3:1. Total mine production from all pits is stated in Table 16-12.

Table 16-12: Mine Production totals

	Leach Ore Mt	AUD g/t	AUD Recovery %	Waste Mt	Total Tonnes Mt	Contained Gold Moz
All Pits	299.4	0.323	70.4	394.3	693.7	3.11

The mine plan includes one year of pre-production stripping followed by 15 years of mining is shown in Table 16-13. The mining rate reaches 50 Mt/a of material mined in in year 2 and maintains 50-55 Mt/a rate through Year 7, peaking at 54.8 Mt in Year 5. The annual mining rate varies between 30 Mt and 48 Mt from Year 8 onward and is lower in the years that have a large amount of stockpile reclaim in order to manage the mining fleet. Total material moved peaks at 55 Mt in Years 5-8.

Mine production is accelerated throughout the mine schedule to bring the higher-grade ore to the leach pad earlier in the project life, while lower grade and lower recovery material is stockpiled. A total of 92.3 Mt of ore is stockpiled over the mine life with the maximum amount of material in stockpiles reaching 33.3 Mt in Year 6. Stockpiled material is processed for 2 years after the mining stops. The stockpile balance at the end of each year is shown in Table 16-14.

The leach production schedule has a feed rate of 18.25 Mt/a (50 kt/d) starting in Year 1 and remains at this rate for the remainder of the project life.

16.10 Mine Plan Schedule

Prior to mining, proper roads from the open pits to the leach pad, waste destinations, and other facilities will be constructed. Ditching and drains to control surface water will be established for the roads and mine facilities. The initial mining fleet will be utilized for some pre-mining construction activities.

Pre-production mining starts in Discovery Phase 1, Discovery Phase 2, and J pit with the leach ore are sent to the appropriate stockpile. Waste rock will be routed to the nearest waste rock facility, and suitable waste material may be used for construction purposes. During pre-production, a total of 13.1 Mt of leach ore will be sent to the ore stockpiles. The Discovery WRF, Rangefront WRF, and stockpiles will be established during pre-production phase. See Figure 16-27.

Year 1, the first year of gold production has 18.1 Mt/a of ore at a 0.37 g/t diluted gold grade sent to the leach pad for processing including higher grade ore that was stockpiled during pre-production mining. Lower grade and lower recovery material from the mine is stockpiled (7.6 Mt). The mining rate is 48.1 Mt as mining continues in Discovery Phase 1. M pit and J pit are also mined in this period (see Figure 16-28).

Year 2, leach ore placed is 18.25 Mt at a 0.55 g/t diluted gold grade. Mining continues in Discovery Phase 1 and starts in Discovery Phase 2. J Pit is mined out during this period (see Figure 16-29). The amount of ore stockpiled is 5.2 Mt. The total mining rate reaches 50 Mt.

Year 3, ore placement continues at 18.25 Mt with a 0.33 g/t diluted gold grade. Ore sent to stockpiles is 3.0 Mt and the end of year stockpiled ore balance is 20.4 Mt. No stockpiled ore is reclaimed in year 3. Discovery Phase 1 is mined out and mining continues in Discovery Phase 2, and M pit. Mining in the CD and Tallman Phase1 pits start during this period. The mining rate is 54.8 Mt (see Figure 16-30).

Year 4, ore placement continues at 18.25 Mt with a 0.45 g/t diluted gold grade of direct mine feed. No stockpile material is reclaimed. Ore sent to stockpiles is 3.8 Mt. and Discovery Phase 2 is the main mining pit with nearly all mine production coming from this phase. Mining starts in the E pit. The mining rate is 54.6 Mt (see Figure 16-31).

Year 5, ore placement continues at 18.25 Mt. Ore sent to the pad has a 0.55 g/t diluted gold grade with 0.69 Mt from the stockpiles. A total of 8.4 Mt of ore is stockpiled and the stockpiled balance reaches 31.9 Mt. Mining is initiated in Backrange and continues to advance in Discovery Phases 2 and 4, CD and E pit (see Figure 16-32). The mining rate is 54.3 Mt. Total material movement including stockpile reclaim peaks at 55.0 Mt and remains at this peak for the next two years.

Year 6, ore placement continues at 18.25 Mt. Ore sent to the pad has a 0.35 g/t diluted gold grade. Mining is accelerated in Discovery Phase 4 as Discovery Phase 2 is depleted. Backrange, CD, and E Pit are completed in Year 6, and mining starts in Rangefront Phase 1. Ore sent to stockpiles is 4.2 Mt and the stockpiled ore peaks at 33.3 Mt by years end. The mining rate is 52.2 Mt.

Year 7 marks the end of consistent higher-grade material from the pit as the Discovery Main pit is nearly mined out. Ore sent to the pad has a 0.35 g/t diluted gold grade. An additional 3.3 Mt of ore is routed to the stockpiles and the stockpile balance remains high at 32.7 Mt. The mining rate is 51.0 Mt, and mining starts in Rangefront Phase 1.

In Year 8, mining is completed in all pits except Rangefront Phase 2 and Rangefront Phase 3 (see Figure 16-33). A large amount of low-grade material, 10.2Mt, is routed the stockpiles while 10.8 Mt is reclaimed. Ore sent to the pad has a 0.32 g/t diluted gold grade. The mining rate is 44.0 Mt.

Mining in Years 9 through 14 is in Rangefront Phases 2 and 3. Nearly all ore placed on the pad in Years 9 and 10 is from stockpiles as waste is stripped from the upper benches of the Rangefront pits. Ore grade placed on the pad is uneven and varies from 0.19 g/t to 0.48 g/t as mining encounters pockets of high- and low-grade material mining in the Rangefront pit (see Figure 16-34).

Mining is complete in Year 15 (see Figure 16-35) and the remaining 25.6 Mt of stockpile ore will be placed onto the leach pad over the following 2 years (see Figure 16-36).

Table 16-13: Annual Production Summary and Material Movement

	Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Total
Mining Summary	Mined Waste (Mt)	13.9	28.2	29.1	33.6	32.6	28.3	32.5	33.4	26.5	28.7	25.6	29.1	15.0	12.9	15.9	8.9	-	-	394.3
	Leach Ore (Mt)	13.1	19.8	20.9	21.2	22.1	26.0	19.6	17.6	17.6	1.4	9.4	18.8	25.0	27.1	14.1	25.6	-	-	299.4
	Au (g/t)	0.27	0.31	0.50	0.31	0.41	0.46	0.33	0.34	0.32	0.19	0.17	0.18	0.22	0.28	0.20	0.39	-	-	0.32
	Au Recovery (%)	62.5	64.0	68.8	66.5	69.7	70.2	64.9	64.8	72.1	72.1	71.3	71.7	74.1	77.3	72.5	80.3	-	-	70.4
	Total Mined (Mt)	27.0	48.1	50.0	54.8	54.6	54.3	52.2	51.1	44.1	30.1	35.0	48.0	40.0	40.0	30.0	34.5	-	-	693.7
Stockpile Summary	Discovery Met 1	7.1	4.0	2.6	1.5	2.7	4.1	2.6	0.7	0.2	-	-	-	-	-	-	-	-	-	25.5
	Discovery Met 2-3	2.4	3.5	2.6	1.1	1.1	2.1	1.4	0.6	0.1	-	-	-	-	-	-	-	-	-	14.9
	Rangefront Met 1	2.9	0.1	0.0	0.3	0.0	1.5	0.1	2.1	9.6	0.8	6.1	2.4	6.4	8.6	0.8	7.2	-	-	48.9
	Rangefront Met 2-3	0.8	0.0	-	0.1	-	0.8	0.0	0.0	0.4	-	0.1	0.2	0.3	0.2	0.1	0.1	-	-	3.1
	Total Stockpiled (Mt)	13.1	7.6	5.2	3.0	3.8	8.4	4.2	3.3	10.3	0.8	6.2	2.6	6.7	8.8	0.8	7.3	-	-	92.3
	Reclaim (Mt)	-	6.0	2.6	-	-	0.7	2.8	3.9	10.9	17.7	15.0	2.0	-	-	5.0	0.0	18.3	7.4	92.3
	Total Moved (Mt)	27.0	54.0	52.6	54.8	54.6	55.0	55.0	55.0	55.0	47.8	50.0	50.0	40.0	40.0	35.0	34.5	18.3	7.4	786.0
Processed Material	Leach Feed (Mt)	-	18.2	18.3	18.3	18.3	18.3	18.2	18.3	18.3	18.3	18.3	18.3	18.3	18.3	18.3	18.2	18.3	7.4	299.4
	Au (g/t)	-	0.37	0.55	0.33	0.45	0.55	0.35	0.34	0.32	0.20	0.19	0.19	0.25	0.34	0.19	0.48	0.15	0.14	0.32
	Au Recovery (%)	-	65.3	69.6	67.2	70.3	71.9	64.8	64.8	70.1	56.6	68.7	71.8	75.1	78.9	72.2	81.7	69.9	68.7	70.4
	Met 1 (Mt)	-	12.6	10.8	15.3	13.9	15.5	11.2	12.4	17.2	8.4	15.8	17.9	17.9	18.0	18.0	18.1	17.9	6.9	247.9
	Au (g/t)	-	0.40	0.67	0.33	0.46	0.55	0.37	0.35	0.31	0.21	0.18	0.19	0.25	0.34	0.19	0.48	0.15	0.14	0.32
	Au Recovery (%)	-	70.1	76.8	69.9	74.7	74.8	71.0	71.0	72.1	64.7	71.8	72.1	75.5	79.2	72.5	81.9	70.3	69.9	74.2
	Met 2-3 (Mt)	-	5.6	7.4	3.0	4.3	2.7	7.1	5.9	1.1	9.8	2.4	0.3	0.4	0.2	0.3	0.2	0.4	0.5	51.5
	Au (g/t)	-	0.29	0.39	0.32	0.42	0.50	0.31	0.34	0.60	0.19	0.19	0.16	0.19	0.31	0.20	0.48	0.15	0.13	0.32
	Au Recovery (%)	-	50.5	51.6	53.2	54.3	53.4	52.8	51.4	53.9	49.1	48.9	53.1	50.1	50.1	52.8	54.7	52.6	51.1	51.9

Table 16-14: Annual Stockpile Inventory (end of year)

Stockpile	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
Discovery Met 1 (Mt)	7.1	11.1	11.1	12.6	15.3	18.7	21.4	18.6	7.9	-	-	-	-	-	-	-	-	-
Rangefront Met 1 (Mt)	2.4	3.5	6.1	7.2	8.3	10.4	11.8	11.8	11.9	2.4	-	-	-	-	-	-	-	-
Discovery Met 2-3 (Mt)	2.9	0.1	0.1	0.3	0.3	1.8	0.1	2.2	11.8	12.6	6.2	6.8	13.2	21.8	17.5	24.8	6.9	-
Rangefront Met 2-3 (Mt)	0.8	0.1	0.1	0.2	0.2	0.9	0.0	0.1	0.4	0.1	0.2	0.2	0.5	0.7	0.8	0.9	0.5	-
Total Stockpile (Mt)	13.1	14.8	17.4	20.4	24.2	31.9	33.3	32.7	32.1	15.2	6.4	7.0	13.7	22.5	18.4	25.7	7.4	-

Table 16-15: Tonnes Mined by Pit/Phase (Mt)

	Discovery				Rangefront			F Zone	M Zone	J Zone	CD	E Zone	Backrange
	Ph1	Ph2	Ph4	Tallman	Ph1	Ph2	Ph3						
PP-1	20.4	-	-	-	-	-	-	5.7	-	0.9	-	-	-
Y1	39.7	-	-	-	-	-	-	-	6.5	1.9	-	-	-
Y2	28.1	20.0	-	-	-	-	-	-	0.3	1.6	-	-	-
Y3	1.2	31.2	0.1	9.5	-	-	-	-	6.8	-	6.0	-	-
Y4	-	53.9	-	0.3	-	-	-	-	0.3	-	0.0	0.1	-
Y5	-	25.0	11.4	-	-	-	-	-	-	-	8.8	2.0	7.1
Y6	-	-	39.8	6.0	-	-	-	-	-	-	0.9	4.3	1.1
Y7	0.0	0.4	16.8	17.8	8.2	-	-	-	7.8	-	-	-	-
Y8	-	-	0.0	4.3	21.1	15.6	-	-	2.8	-	-	0.2	-
Y9	-	-	-	-	-	30.1	-	-	-	-	-	-	-
Y10	-	-	-	-	-	35.0	0.0	-	-	-	-	-	-
Y11	-	-	-	-	-	47.6	0.3	-	-	-	-	-	-
Y12	-	-	-	-	-	39.9	0.1	-	-	-	-	-	-
Y13	-	-	-	-	-	35.0	5.0	-	-	-	-	-	-
Y14	-	-	-	-	-	4.0	26.0	-	-	-	-	-	-
Y15	-	-	-	-	-	20.8	13.7	-	-	-	-	-	-
Total	89.4	130.5	68.1	37.8	29.3	228.1	45.1	5.7	24.6	4.4	15.7	6.7	8.3

16.11 Mine Equipment Selection

The mine equipment fleet has been sized to achieve up to an annual production rate of 60 Mt/a. The larger mining fleet includes 22 m^3 diesel hydraulic shovels and 144 tonne rigid body trucks. The smaller fleet has 6.7 m^3 diesel hydraulic excavators and 64 tonne rigid body trucks. Additional loading support will be provided by 11.5 m^3 front end wheel loaders.

Blasthole drilling will be completed with a diesel down the hole hammer ("DTH") drills with 165 mm bits. These drills provide the capability to drill patterns for 10 m bench heights.

The support equipment fleet will be responsible for the usual road, pit, and dump maintenance requirements. Due to the number of mining areas and surface haul roads additional graders and dozer are included in the fleet. A mobile crushing and screening plant is planned to provide road surface aggregate and blasthole stemming.

The proposed equipment requirements for the LOM plan are included in Section 21.

16.12 Grade Control

Grade control will be done using the blast hole sampling method. RC drilling is not included for grade control but should be assessed as the Project advances to the next study phase.

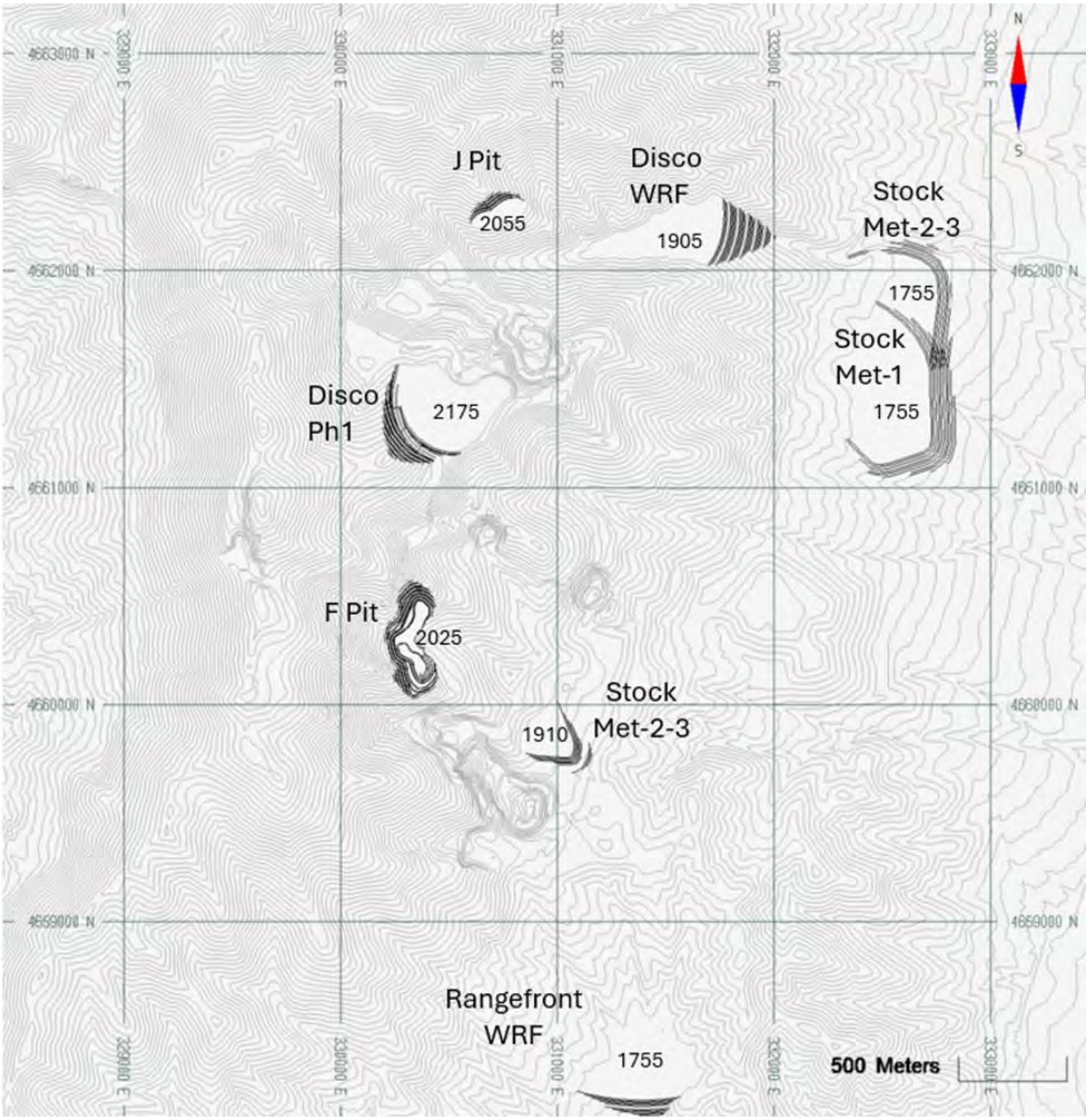
The samples will be collected daily and sent to the assay laboratory for grade determination. The assays inform a short-range model that allows the mine engineering and operations team to guide mining activities to ensure the Project achieves its targets for metal production.

16.13 Pit Dewatering

All pits are considered dry and will not require dewatering. The Rangefront pit is limited to 1465 masl to keep the water level below the working benches elevation in that pit.

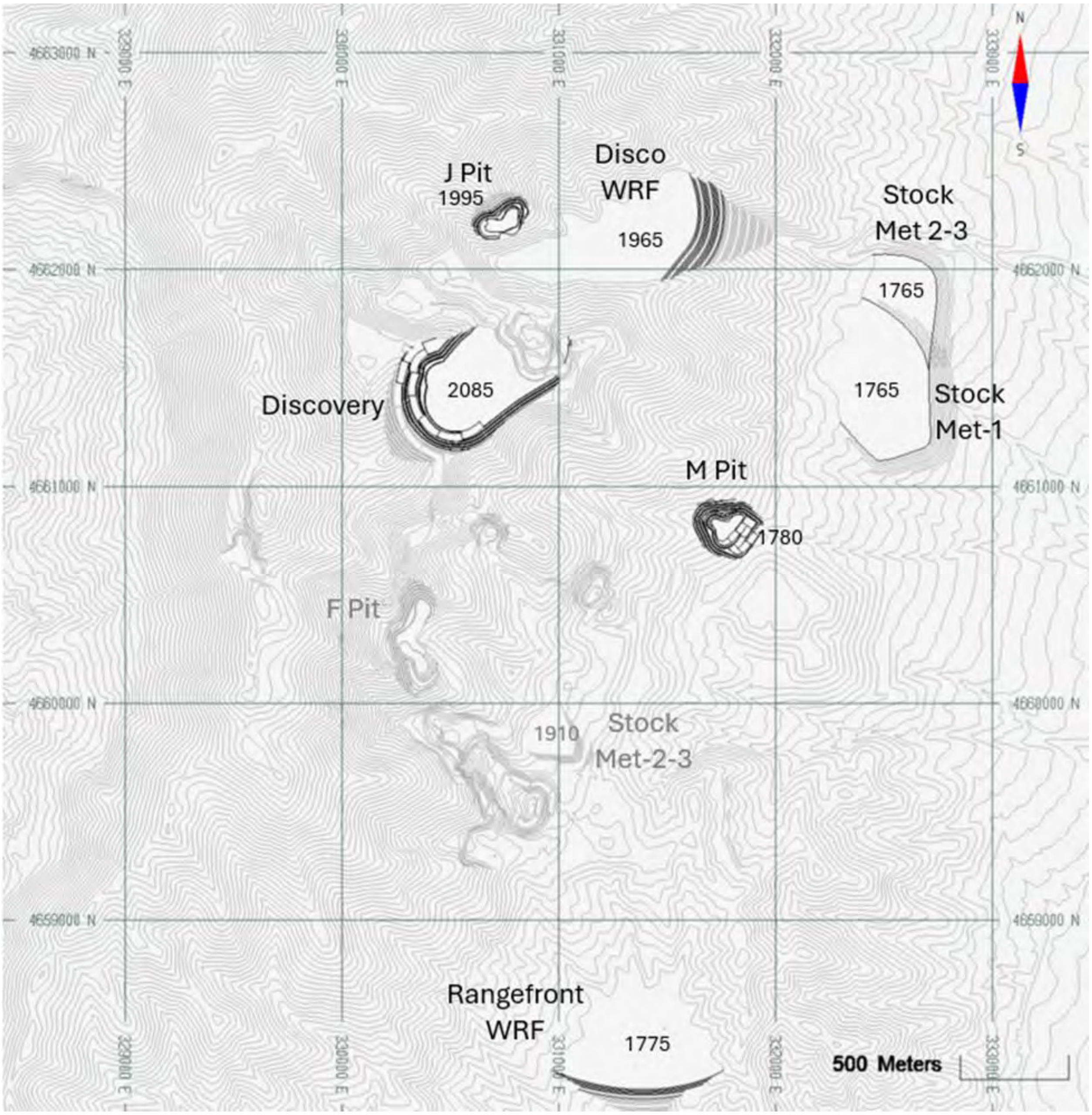
A small dewatering allowance has been included for the accumulation of surface water due to storm events and snow melting. If required, surface water will be pumped by mobile pumps placed in sumps on the mining level.

16.14 End of Period Maps



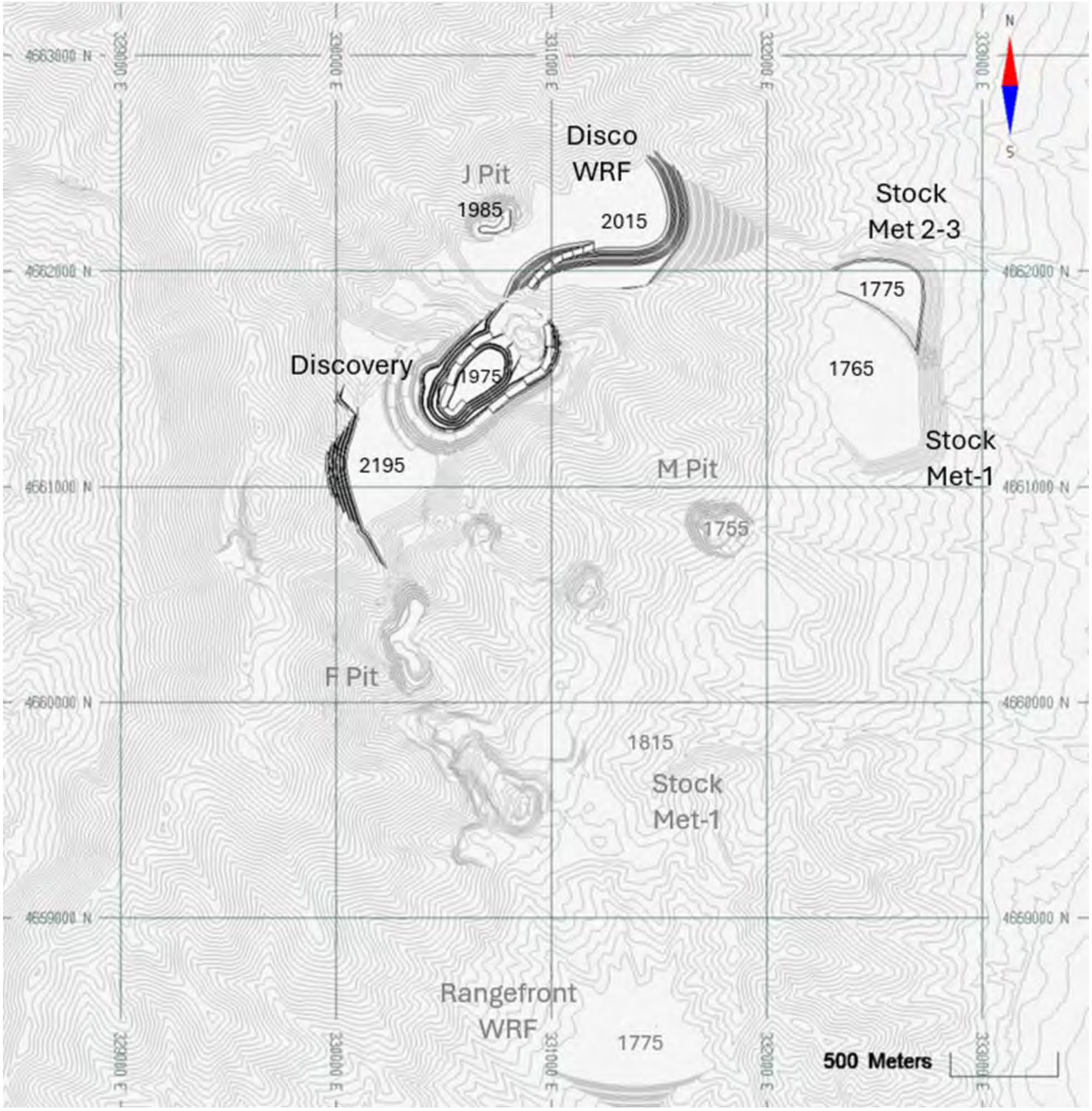
Source: AGP, 2024

Figure 16-27: End of Year -1 (Pre-Production)



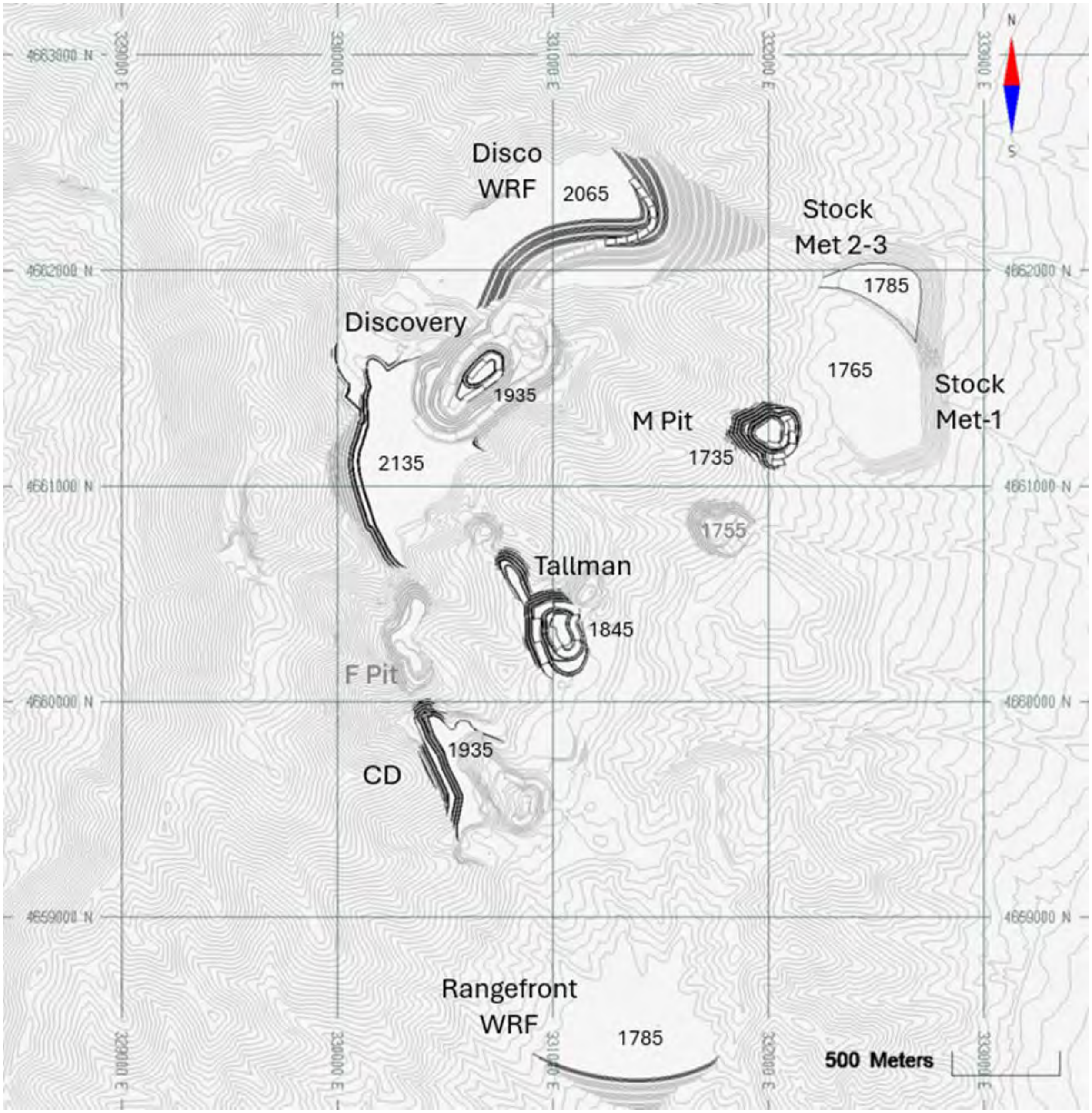
Source: AGP, 2024

Figure 16-28: End of Year 1



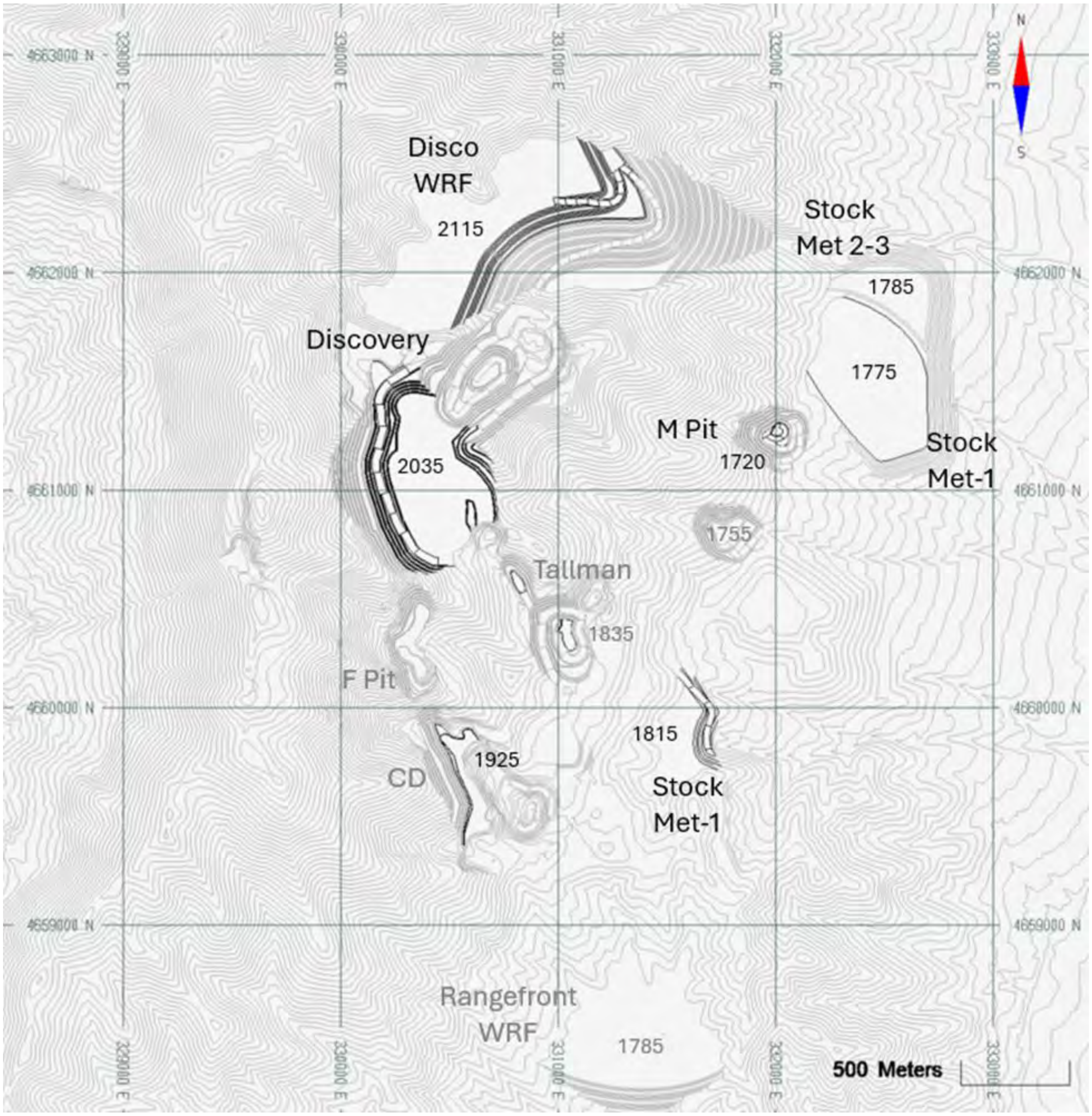
Source: AGP, 2024

Figure 16-29: End of Year 2



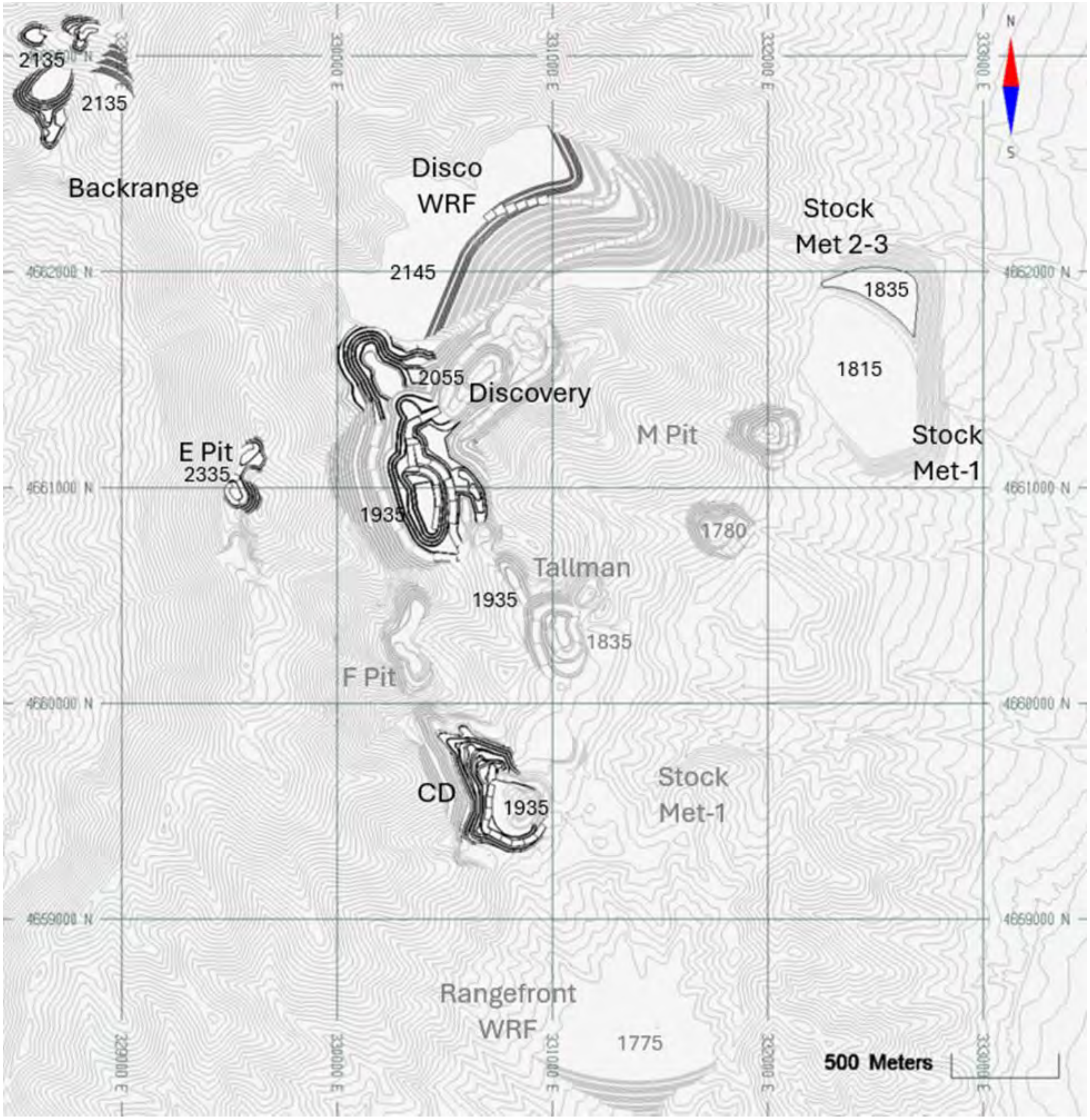
Source: AGP, 2024

Figure 16-30: End of Year 3



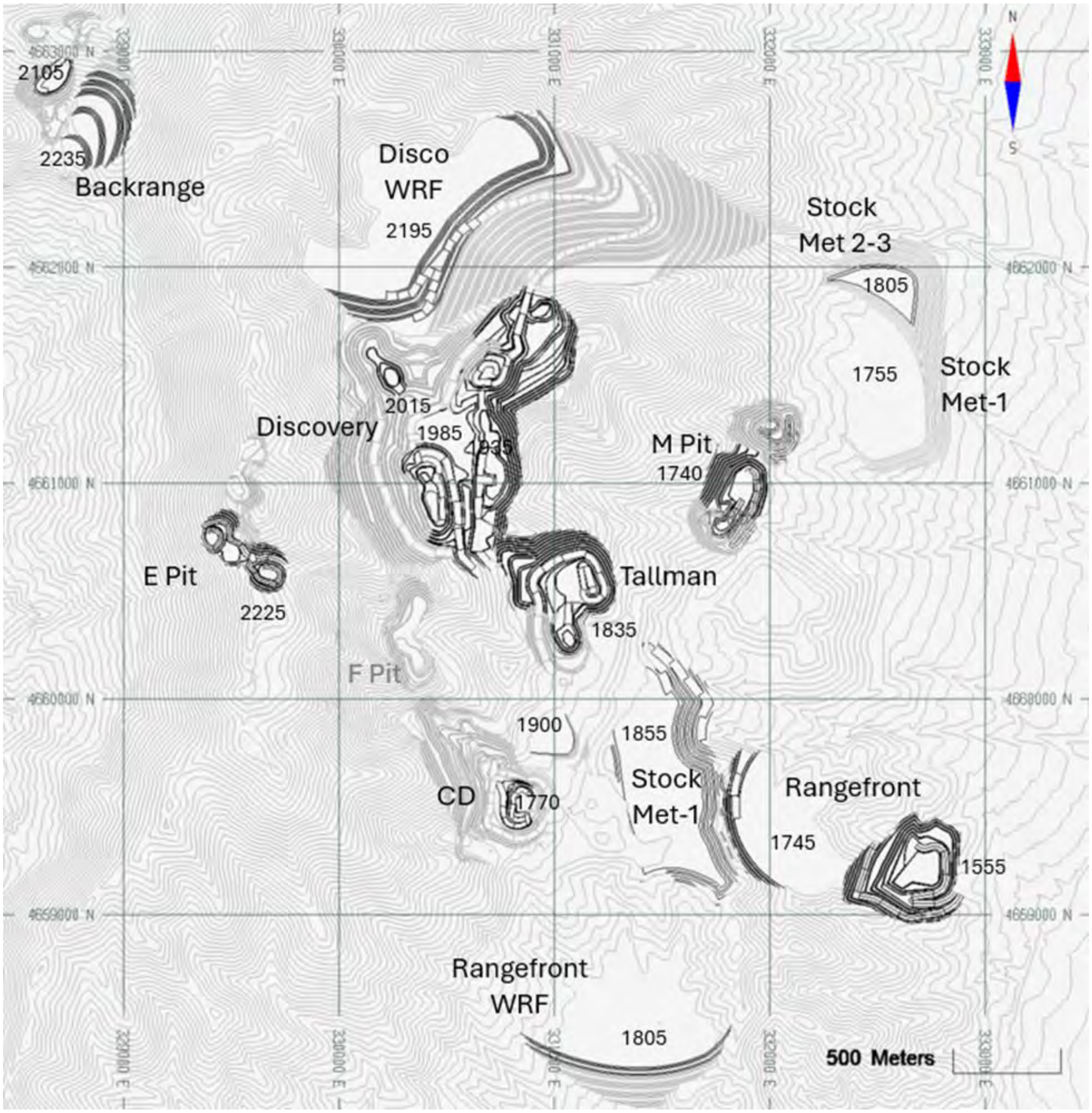
Source: AGP, 2024

Figure 16-31: End of Year 4



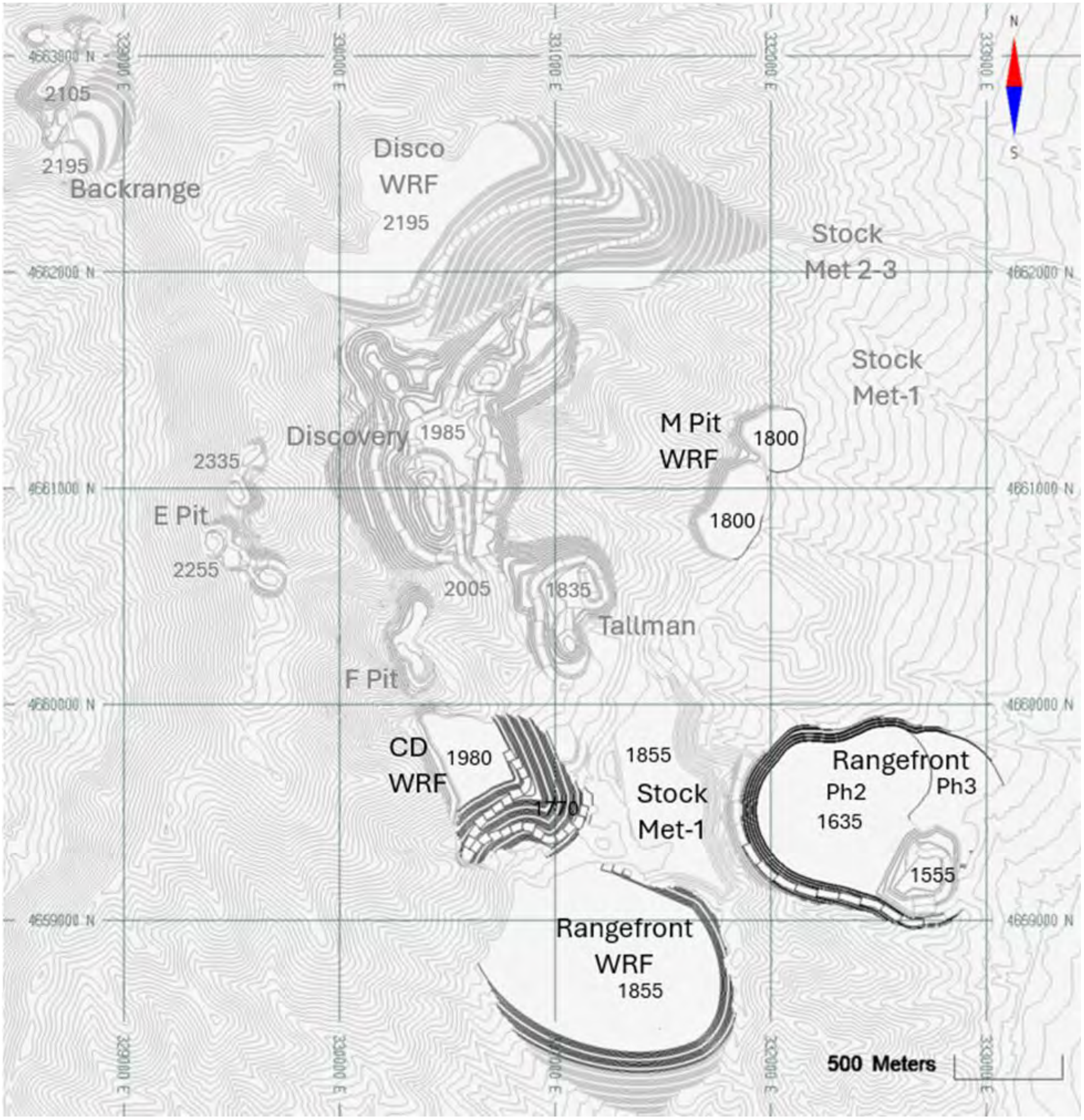
Source: AGP, 2024

Figure 16-32: End of Year 5



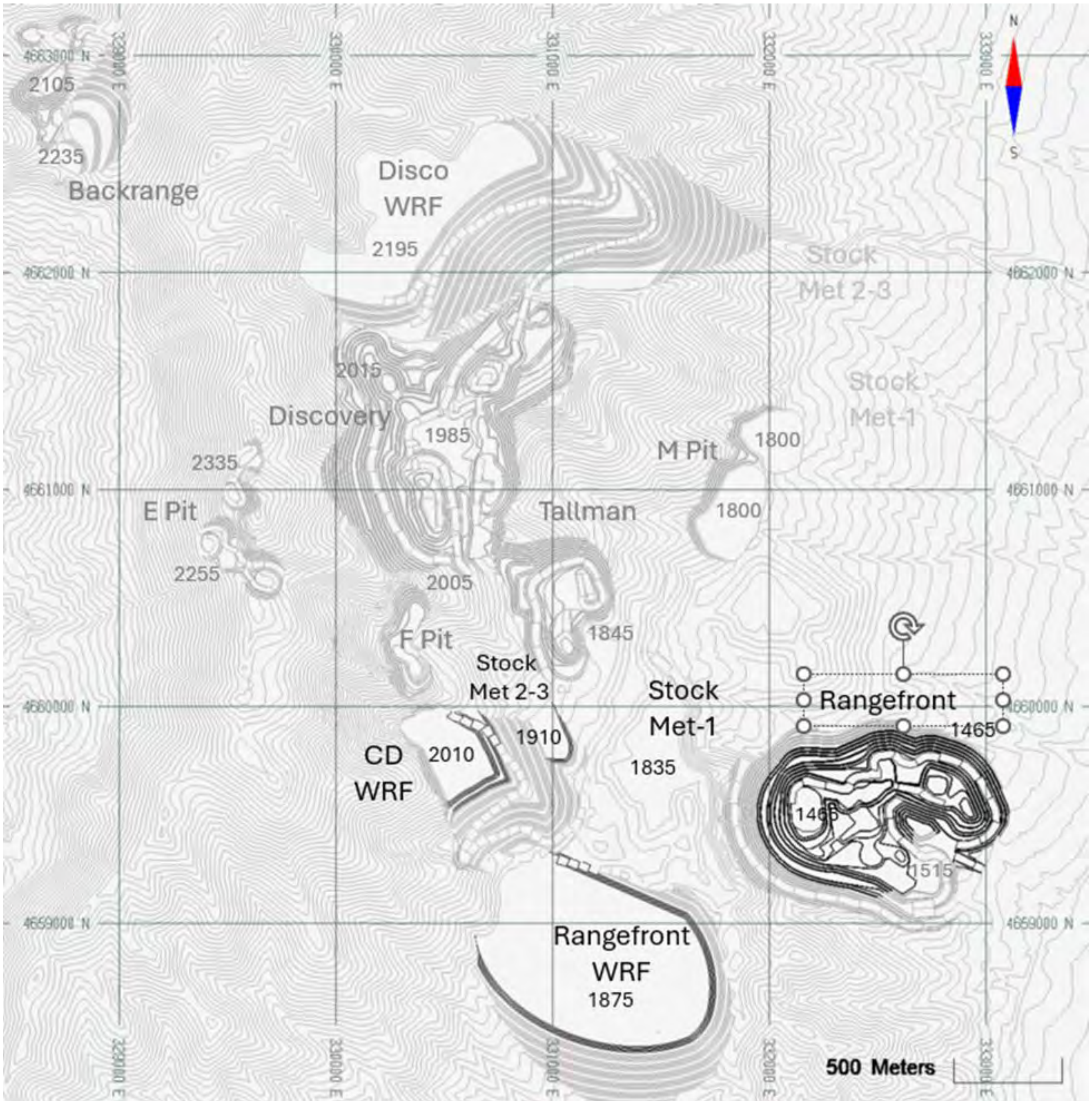
Source: AGP, 2024

Figure 16-33: End of Year 8



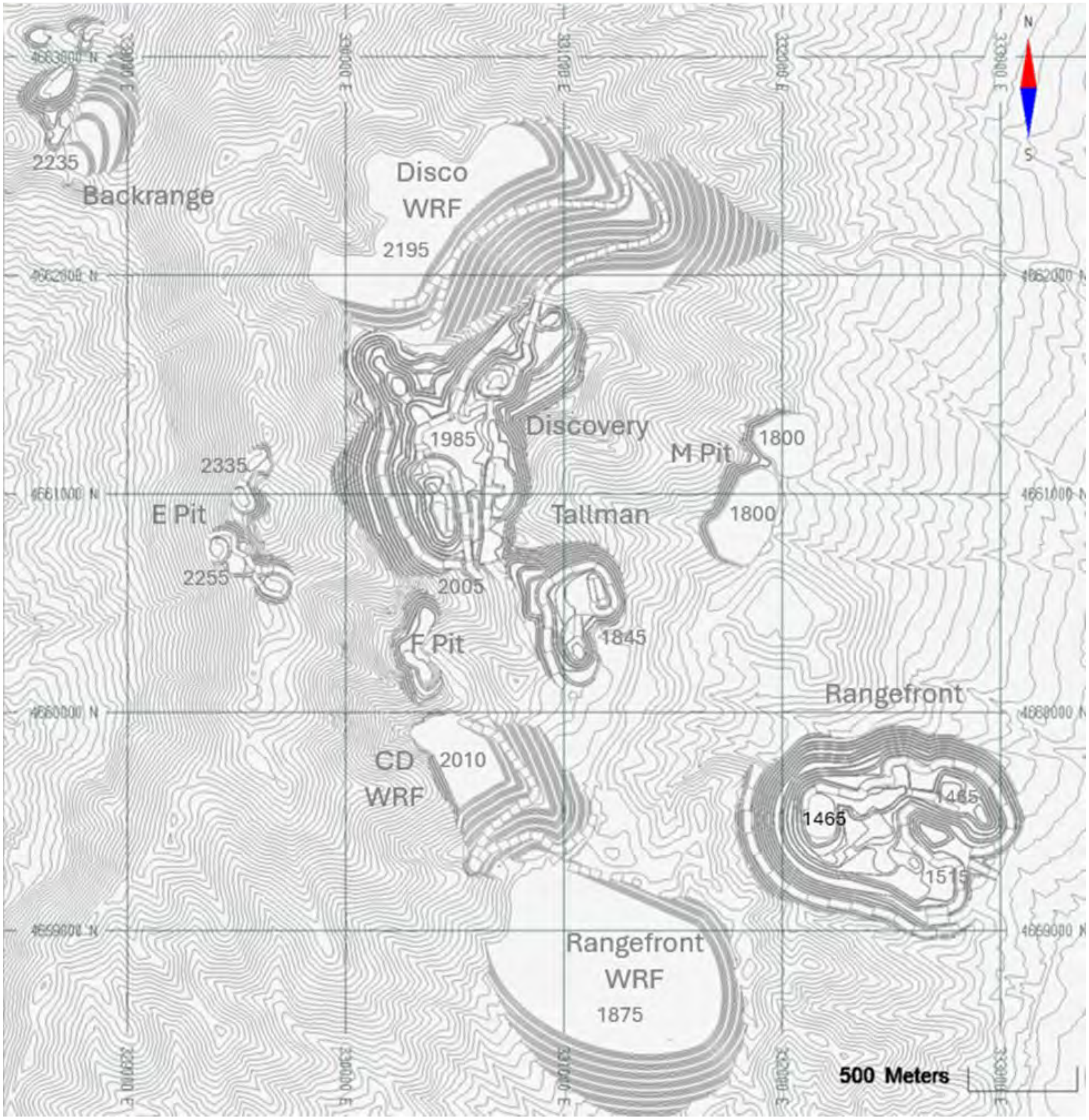
Source: AGP, 2024

Figure 16-34: End of Year 11



Source: AGP, 2024

Figure 16-35: End of Year 15 (Mining Complete) -2 years of stockpile reclaim remaining



Source: AGP, 2024

Figure 16-36: End of Year 17 – End of Mine Life Stockpiles Reclaimed

17 RECOVERY METHODS

The process selected for recovery of gold and silver from the Black Pine deposit is a conventional heap-leach recovery circuit. The ore will be mined by standard open pit mining methods from several pits. Ore will be truck-stacked on the heap leach pad as ROM ore directly, without crushing.

There are three main classifications of ore, each associated with gold recovery as a ratio of cyanide solubility to fire-assay (AuCN:AuFA). Ore with a gold recovery AuCN:AuFA of over 65% is identified as “Met 1”. Ore with a gold recovery of less than 65%, but greater than 50%, is identified as “Met 2”. Ore with a gold recovery of less than 50%, but greater than 25%, is identified as “Met 3”. The heap leach pad is designed to maintain Met 1 and Met 2/3 ores separated; the solution that permeates through the pad is also maintained segregated between Met 1 and Met 2/3. As can be implied, Met 2 and Met 3 are kept together, but isolated from Met 1.

A dilution cyanide solution (Barren Solution) will be applied to the pad for both Met 1 and Met 2/3 ores. Process Solution from the Met 2/3 portion of the heap leach pad will report to the barren solution tank. This will allow the grade in the barren to build-up in order to have higher process solution grades reporting to the carbon columns. Process solution from the Met 1 portion of the heap leach pad will report to a Process Solution Tank. The leached gold and silver will be recovered from solution using a carbon adsorption circuit. Gold and silver will be stripped from carbon using a desorption process, followed by electrowinning to produce a precipitate sludge. The precipitate sludge will be processed using a retort oven for drying and mercury separation and recovery, and then refined in a melting furnace to produce gold and silver doré bars.

The Black Pine deposit has a total estimated Mineral Reserve of 299.4 million tonnes. The total estimated mine life is 17 years; solution application on the heap leach pad typically continues for an additional 2 years after mining operations have ceased to recover additional solubilized metal ounces. The nominal ore placement rate on the pad is an average of 18.25 million tonnes per annum, equivalent to 50,000 tonnes per day.

17.1 Gold and Silver Recoveries

The gold and silver recoveries for heap leaching of the Black Pine ore have been taken from the recommendations detailed in Section 13 of this Technical Report.

For the Met 1 Mineral Resources, the overall life-of-mine average gold recovery for the ore is estimated at 74.2 percent. For the Met 2/3 Mineral Resources, the overall life-of-mine average gold recovery for the ore is estimated at 51.9 percent.

17.2 Reagents and Consumptions

The major reagent consumptions for heap leaching of Black Pine ore have been taken from available metallurgical test results from column leach tests on crushed material. No test data exists at the ROM particle size, so the selected reagent consumptions have been estimated based on test results on the coarsest samples tests.

17.3 Sodium Cyanide

Sodium cyanide (NaCN) will be used in the leaching process and will be delivered in tanker trucks as a liquid at 30% concentration by weight (1.16 SG). Sodium cyanide will be stored in a 25,000-gallon steel tank at the ADR area within concrete containment and will be distributed to the process by a distribution pump with individual control valve stations at each point of use.

All cyanide distribution lines will be double-containment, either by “pipe-within-pipe” or “pipe-overliner” containment systems. Cyanide consumption has been estimated as follows:

- 0.18 kg/tonne (0.36 lb/ST) Overall

17.3.1 Lime

Pebble quicklime (CaO) will be used to treat the ROM ore prior to cyanide leaching to maintain the alkaline pH. Lime will be delivered in bulk by 20-ton trucks, which will be off-loaded pneumatically into a 200-ton storage silo with a variable speed feeder that will meter lime directly onto the ore being carried by haul trucks to the heap leach pad and will be added in proportion to the tonnage of ore in each truck.

Lime will be consumed at an estimated rate of 0.97 kg/tonne (1.94 lb/ST) overall.

17.3.2 Activated Carbon

Activated carbon will be used to adsorb precious metals from the leach solution in the adsorption columns. Make-up carbon will be 6 x 12 mesh and will be delivered in 2,200 lb supersacks. It is estimated that approximately 3-4% of the carbon stripped will have to be replaced due to carbon fines losses.

17.3.3 Sodium Hydroxide (Caustic)

Sodium hydroxide (caustic) will be delivered to site as a liquid at 50% caustic by weight (1.53 SG). Liquid caustic will be stored in a 15,000-gallon steel tank and metered to the strip solution tank and acid wash circuits by a distribution pump with individual control valve stations at each point of use.

17.3.4 Nitric Acid

Nitric acid (7%) will be used in the acid wash section of the elution circuit prior to desorption. Nitric acid will be delivered to site as a liquid at 57% solution strength and diluted to 7% in the dilute acid tank. Acid washing consists of circulating a dilute acid solution through the bed of carbon to dissolve and remove scale from the carbon. Carbon acid washing will be done before each desorption cycle, or as required to maintain carbon activity level.

17.3.5 Fluxes

Various fluxes will be used in the smelting process to remove impurities from the bullion in the form of a glass slag. The normal flux components are a mix of silica sand, borax, and sodium carbonate (soda ash). The flux mix composition is variable and will be adjusted to meet individual project smelting needs: fluorspar and/or potassium nitrate (niter) are sometimes added to the mix. Dry fluxes will be delivered in 55 lb bags. Average consumption of fluxes has been estimated to be 2 lb per lb of gold and silver produced.

17.3.6 Antiscalant

Antiscalant will be used to prevent the build-up of scale in the process solutions and heap irrigation lines. Antiscalant will be added directly into pipelines or tanks, and consumption will vary depending on the concentration of scale-forming species in the process stream. Delivery will be in liquid form in 264-gallon (1 m^3) totes.

Antiscalant will be added directly from the supplier tote bins into the pregnant, barren, and desorption pumping systems using variable speed chemical-metering pumps. On average, antiscalant consumption is expected to be about 6 ppm for leach solutions and 10 ppm for strip solutions to be treated.

17.4 Process Flowsheet

An overall process flowsheet for the Project is presented in Figure 17-1.

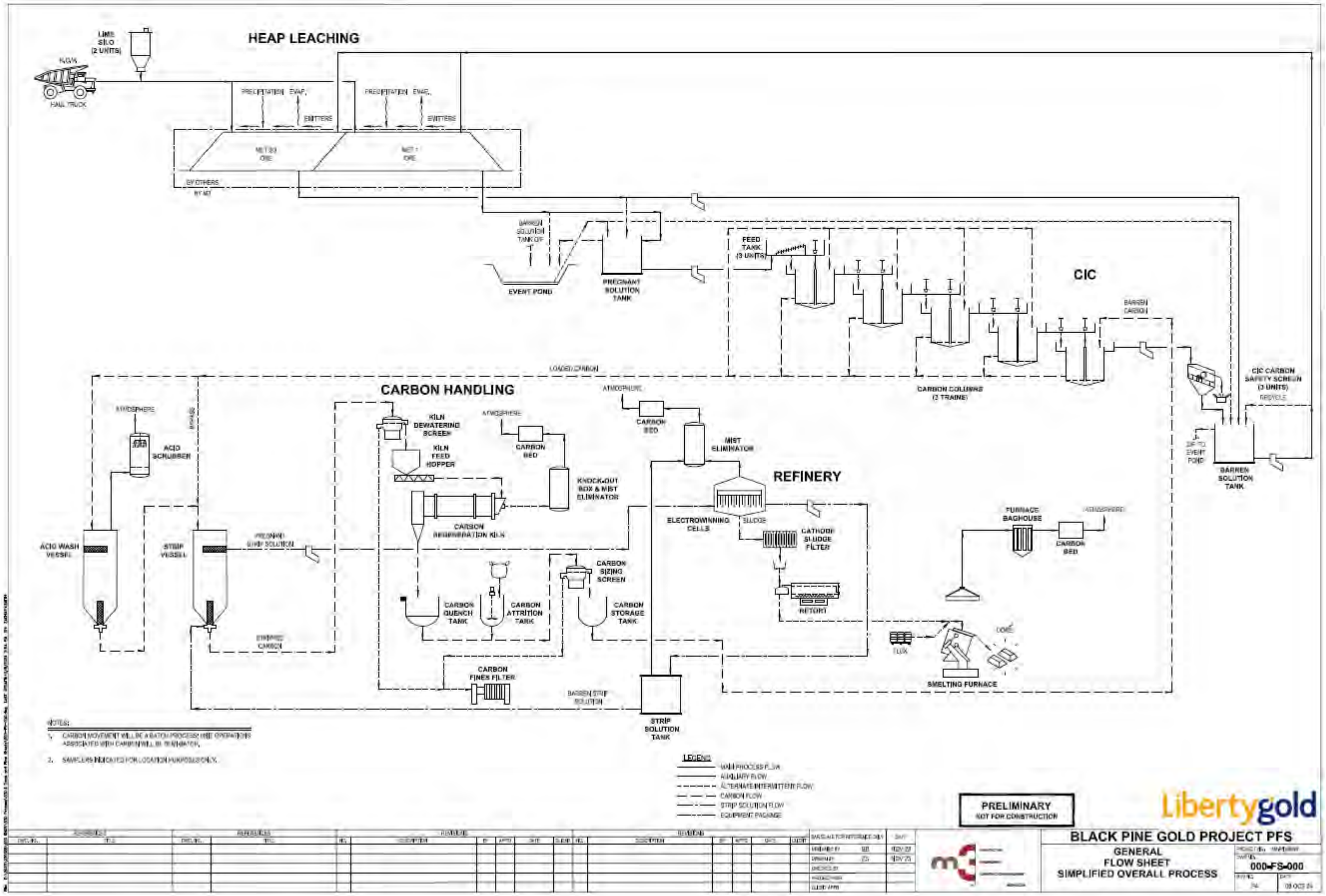


Figure 17-1: Process Flowsheet for the Black Pine Project

17.5 ROM Truck Stacking

Excavation, loading, hauling, and dumping of ROM material will be conducted by the mining fleet. ROM ore will be loaded into haul trucks and transported to the active stacking face at an average rate of 50,000 tonnes/day. ROM production and stacking will vary based on the ore availability from the mine pits.

Quicklime (CaO) will be used for pH control of the process. Pebble quicklime will be stored in a 200-ton silo which will be equipped with a variable speed feed system that will feed a clam gate for lime addition to the trucks. Once the haul trucks have been loaded, the lime will be metered directly into the loaded trucks which will then deliver the ore to the active stacking area. One lime silo will be installed at the haul roads. Lime will be added in proportion to the tonnage of ore being hauled.

The ore haul trucks will operate on top of the lift being constructed. A ramp, or ramps, will be constructed to reach the top of each current lift. The trucks will direct-dump the ore on the current lift and a dozer will push the ore over the edge of the lift to form the expanding heap. The stacked ore will be deep-shank cross-ripped with the dozer prior to leaching. Ore will be stacked in 10 m (32.8 ft) high lifts with a maximum ore heap height of 100 m (328 ft).

Prior to stacking a new lift over the top of an old one, the top of the old lift will be cross-ripped to break up any cemented/compacted sections and to redistribute any fines that may have been stratified by the irrigation solution or rainfall.

Following stacking, the ore will be drip irrigated with dilute cyanide leach solution and the resulting gold-bearing solutions collected in the pregnant solution tank. The leach pad will be a multiple-lift, single-use type pad.

17.6 Leaching and Solution Handling

After each leach cell has been stacked and dozer ripped, the irrigation system will be installed. Dripline emitters will be used to apply a dilute cyanide solution at an application rate of 0.004 gpm/ft^2 for ROM ore. A leach cycle of 60 days has been selected for ROM, based on a review of the leach curves.

Barren leach pH solution will be maintained at a minimum value of 10 and will be controlled by the addition of lime on the fresh ore. Barren solution will be delivered from a barren tank located at the recovery plant, by high-flow high-head pumps at a nominal flow rate of 9,400 gpm (expected range: 8,000-10,000 gpm). This solution will be carried by a steel pipeline to the base of the heap and then to a network of sub-headers and risers to the top of the heap where it is finally applied to the material by drip emitters.

Solution passing through the heap will dissolve the values contained therein and be collected in a network of perforated solution collection pipes, which feed to a common discharge point at the base of the heap. The solution will then be carried by gravity to a process solution tank. Excess solution from the heap will overflow from the process tank to a lined event pond. Process solution is pumped from the process solution tank to the adsorption carbon column circuit at the recovery plant.

The carbon adsorption circuit consists of three trains of cascade-style columns. Process solution flows through the columns to load the soluble gold onto the carbon. Barren solution exiting the columns is directed to the barren tank where make up cyanide is added, and the solution returned to the heap for further leaching. Overflow from the barren tank is directed to the event pond.

17.7 Leach Pad Phasing and Construction

It is assumed the leach pad will be constructed in four phases.

Barren solution containing cyanide will be irrigated onto the ore using drip irrigation. Process solution will be collected at the base of the heap by the leach pad liner and collection system, which will route the process solution to the process plant for gold recovery and reagent reconditioning. Once an area has been leached for the target time or metal recovery, the next lift will be placed on top of the already leached ore and the process repeated.

17.7.1 Solution Management

Process solution from the Met 2/3 section of the heap leach pad will free drain to the barren solution tank. Soluble metal values that are in the Met 2/3 process solution will increase the grade of the barren solution. The barren solution is applied to both the Met 2/3 sections of the heap leach pad as well as the Met 1 sections of the heap leach pad. The values from the Met 2/3 section will be recovered through passing through the Met 1 section and combining with the Met 1 process solution.

Process solution from the Met 1 section of the heap leach pad will free drain to the process solution tank. The design intention is for all process solution that reports to the process solution tank to report to the downstream metal recovery plant. If there is an event which causes the process solution tank to overflow, the overflow will report to an event pond and then be pumped back to the process solution tank when the overflow condition has been addressed.

17.8 ADR Plant

The recovery plant at Black Pine has been designed to recover gold and silver values using an adsorption-desorption-recovery (“ADR”) process. Process leach solution from the heap leach will be pumped to the carbon in column circuit (“CIC”) and adsorbed onto activated carbon (adsorption). Three trains of carbon columns are included in the design, primarily to allow the diameter of the columns to be maintained within the transportation shipping envelope. Loaded carbon from the CIC circuit will be desorbed in a high-temperature elution process coupled to an electrowinning circuit (desorption), followed by retorting to remove mercury and smelting of the resulting sludge to produce doré bullion (recovery). Before elution, each batch of carbon will be acid washed to remove any scale and other inorganic contaminants that might inhibit gold adsorption on carbon. All or a portion of the carbon will be thermally reactivated using a rotary kiln.

The ADR plant General Arrangement is presented in Figure 17-2.

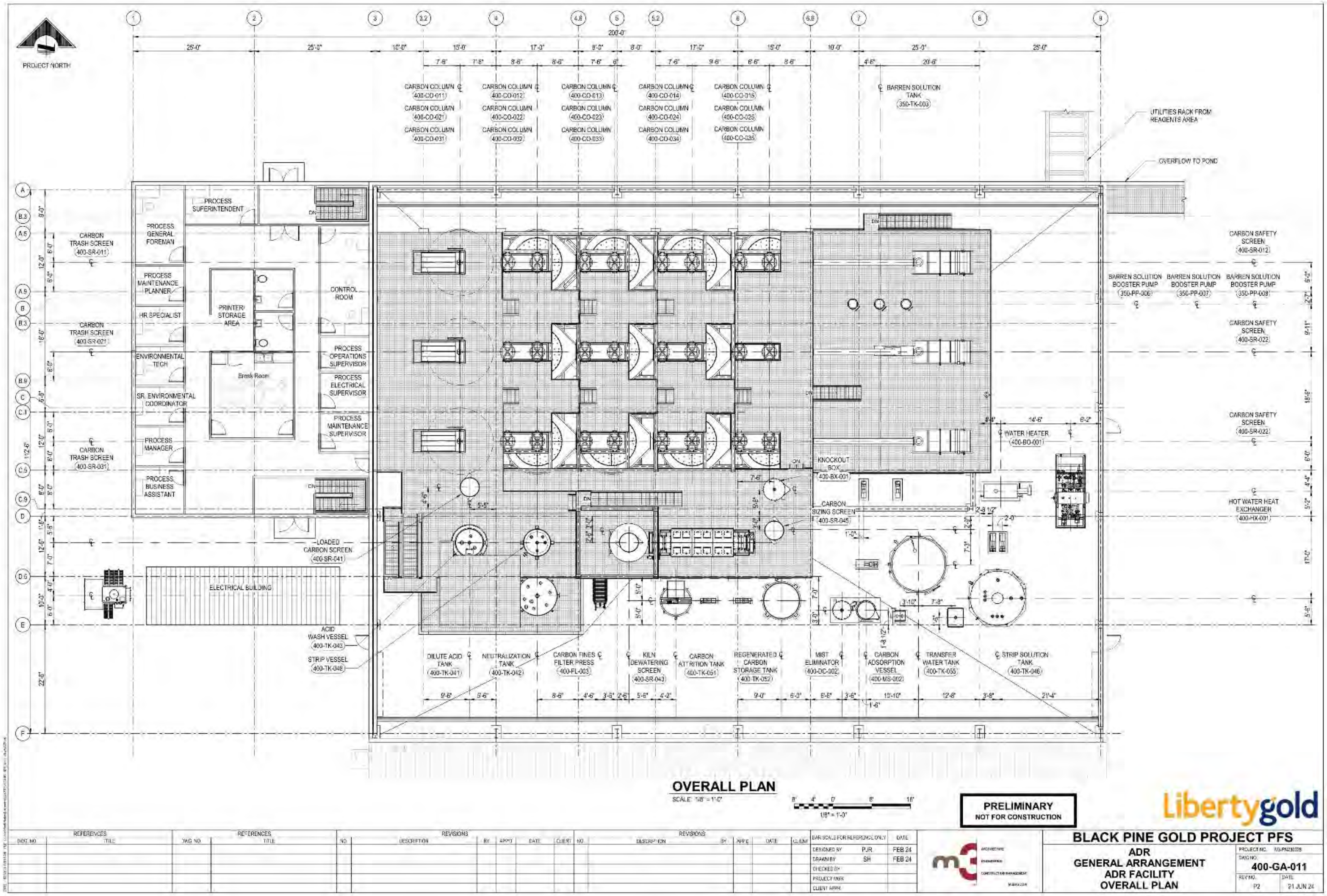


Figure 17-2: ADR Recovery Plant General Arrangement

17.8.1 Adsorption

Adsorption of gold and silver onto carbon will occur in the carbon adsorption circuit. The adsorption circuit will consist of three trains of five, cascade type open-top up-flow mild-steel CICs each. Each of the carbon columns are nominally 12 feet in diameter by 8 feet high and are sized to hold 5.5 tonnes (6 tons) of activated carbon.

The nominal flow to the adsorption circuit will be 3500 gpm max per train. Barren solution exiting the last carbon adsorption column in the train will flow through a vibrating screen to separate any floating carbon from the solution, then flow by gravity into the barren tank.

Antiscalant will be added to the process solution tank to prevent scaling of carbon and reduction of the carbon loading capability. Magnetic flowmeters equipped with totalizers will measure solution flow to the adsorption circuit. Process solution will flow by gravity through each set of five columns in series, exiting the lowest column as barren solution. Process and barren solution continuous samplers will be installed at the feed and discharge end of each carbon column train, respectively. Solution samples will be used to measure pregnant and barren solution gold and silver concentrations.

Adsorption of gold and silver from process leach solutions from the heap circuit will be a continuous process. Once the carbon in the lead column achieves the desired precious metal load it will be advanced to the elution (desorption) circuit using screw type or recessed impellor centrifugal pumps. Carbon in the remaining columns will be advanced counter current to the solution flow to the next column in series. New or acid washed/regenerated carbon will be added to the last column in the train.

The stripping of carbon will occur once per day, on average, once sufficient soluble metal is present on the incoming pregnant solution.

17.8.2 Carbon Acid Wash

Loaded carbon transferred from the CIC circuit will pass through a circular, vibrating screen, which allows for the majority of the elevated pH, cyanide-bearing solution to return to the CIC circuit during carbon transfer. Dewatered carbon reports to the acid wash column. A dilute acid solution will then be prepared in the mix tank, and circulation established between the acid wash vessel and the acid mix tank. Completion of the cycle will be indicated when the pH stabilizes between 1.0 and 2.0 without acid addition for a minimum of thirty minutes of circulation.

The carbon will then be rinsed with raw water followed by rinsing with dilute caustic solution to remove any residual acid. Total time required for acid washing a batch of carbon will be approximately four hours. After acid washing has been completed, a carbon transfer pump will transfer the carbon to the desorption circuit.

17.8.3 Desorption

A pressure Zadra hot caustic desorption circuit for the stripping of metal values from carbon has been selected for Black Pine, which requires 12 hours or less to complete a cycle. During the elution cycle, gold and silver are continuously extracted by electrowinning from the pregnant eluate concurrently with desorption.

The desorption circuit is sized to strip gold and silver from carbon in 5.4 tonne (6- ton) batches and will be equipped with a strip solution tank, strip solution pump, primary (heat up), secondary (heat recovery), and tertiary (cooling) heat exchangers, hot water heater, elution column, and elution column drain pump. After carbon has been transferred to the elution column, barren strip solution (eluant) containing sodium hydroxide and sodium cyanide will be pumped through the heat recovery and primary heat exchangers and introduced to the elution vessel at a nominal temperature of 300°F and a nominal operating pressure of approximately 100 psig for ten hours.

Under normal operating conditions, barren eluant solution from the solution storage tank will pass through the heat recovery exchanger to be preheated by hot pregnant eluate leaving the elution column. The barren eluant solution then passes through the primary heat exchanger to raise the temperature up to 300°F using pressurized hot water (~330°F) from the hot water heater system.

The elution column will contain internal stainless-steel inlet screens to hold carbon in the column and to distribute incoming stripping solution evenly in the column. Pregnant eluate leaving the elution column will pass through two external stainless-steel screens before passing through the heat recovery exchanger and the cooling heat exchanger to reduce the temperature to about 175°F (to prevent boiling). The cooled pregnant eluate solution will flow to the electrowinning cell.

After desorption is complete, the stripped carbon will be transferred to the carbon regeneration circuit by a carbon transfer pump.

17.8.4 Electrowinning

The electrowinning circuit will be operated in series with the elution circuit. Solution will be pumped continuously from the barren strip solution tank through the elution column, then through the electrowinning cells, and back to the strip solution tank in a continuous closed loop process.

The electrowinning circuit will include two electrowinning cells, each equipped with a rectifier. Gold and silver will be won from the eluate in the electrowinning cell using stainless steel cathodes using a current density of approximately 4.5 amperes per square foot of anode surface. Caustic soda (sodium hydroxide) in the eluate solution will act as an electrolyte to encourage free flow of electrons and promote the precious metal winning from solution. To keep the electrical resistance of the solution low during desorption and the electrowinning cycle, make-up caustic soda may sometimes be added to the strip solution tank. Barren eluant solution leaving the electrolytic cells will discharge to the barren eluate tank from which it will be pumped back to the strip solution tank for recycle through the elution column.

Periodically, all or part of the barren eluant will be dumped to the barren solution tank. Typically, about one-third of the barren eluant will be discarded after each elution or strip cycle. Sodium hydroxide and sodium cyanide will be added as required from the reagent handling systems to the barren eluant tank during fresh strip solution make-up.

The precious metal-laden cathodes in the electrolytic cells will be removed about once per week and processed to produce the final doré product. Loaded cathodes will be transferred to a cathode wash box where precipitated precious metals will be removed from the cathodes with a pressure washer. The resulting sludge will be pumped to a plate-and-frame filter press to remove water, and the filter cake will be loaded into pans for retorting.

17.8.5 Carbon Handling & Thermal Regeneration

The carbon preparation and storage system will include a 1-ton agitated carbon attrition tank, a 5-ton carbon storage tank, carbon dewatering screen, carbon fines storage tank, carbon fines filter press, and carbon transfer pumps. New and acid washed/regenerated carbon will be stored in the carbon storage tank to be returned to the CIC circuit as makeup carbon. Carbon being transferred to the carbon storage tank will pass to a carbon fines/dewatering screen in order to remove any carbon fines from the system. Carbon fines will be stored in a carbon fines storage tank, which will be periodically pumped through the carbon fines filter press; carbon fines from the filter press will be stored in bulk bags for removal from the system.

Fresh carbon being added to the system will first be attritioned in the carbon attrition tank before being pumped to the carbon dewatering screen to remove carbon fines and is then transferred to the carbon storage tank.

Thermal regeneration will consist of drying the carbon thoroughly and heating it to approximately 1300ºF for ten minutes in order to maintain carbon activity levels. The carbon regeneration circuit has been designed to regenerate 100% of the carbon.

Carbon from the elution circuit to be thermally reactivated will be dewatered on a vibrating circular screen, transferred to the regeneration kiln feed hopper and fed to the regeneration kiln by a screw feeder. Hot, regenerated carbon leaving the kiln will pass into a water-filled quench tank for cooling before being transferred to the carbon dewatering screen and carbon storage tank.

17.8.6 Refining & Smelting

Cathode sludge from the electrolytic sludge filter press will be dried and treated in a mercury retort to remove and recover any mercury that may be present. The sludge will be placed into pans and heated in the retort for a minimum of 6 hours at 1,100ºF to volatilize mercury. A vacuum system will remove mercury vapor from the retort and pass the vapor through a series of water-cooled condensers. Condensed mercury will be collected in a trap, and then transferred and stored in flasks. Cooled, mercury-depleted vapor leaving the trap will be passed through a sulfur-impregnated carbon scrubber to remove any residual mercury.

After mercury removal, fluxes will be mixed with the cathode sludge and then fed to an electric induction furnace. The furnace will be heated to approximately 2,200ºF. When the furnace charge is fully molten, it separates into two distinct layers: the slag (on the top) and metal (on the bottom). The slag layer, containing fused fluxes and impurities, will be poured first into conical pots. Once slag has been removed, the melted gold and silver (metal layer) will be poured into cascading molds to form Doré bars.

17.8.6.1 Mercury Abatement System

In addition to the mercury retort, the ADR facility will be fitted with an exhaust gas handling system to treat mercury emissions from the various pieces of equipment. The exhaust system will be designed to combine mercury-containing exhaust streams and treat them in two separate sulfur-impregnated carbon beds prior to discharge to the atmosphere.

The first carbon bed will be dedicated to treating fumes from the smelting furnace. The smelting furnace will be fitted with a hood which will collect fumes and direct them to a scrubber, which will remove suspended particles from the gas and cool the gas before passing through the carbon bed. The carbon bed will collect traces of mercury vapor before exhausting the gas into the atmosphere.

The second carbon bed will treat the combined exhaust gas streams from the electrowinning cells, eluant solution storage tank, elution vessel, and carbon regeneration kiln. The kiln exhaust gas will be first treated through a wet scrubber to remove particulates and cool the gas, which will then be combined with the remaining exhaust gas streams and pass through the carbon bed.

17.9 ADR Reagents and Utilities

Recovery plant reagents will include cyanide, caustic, nitric acid, antiscalant, activated carbon, and various furnace fluxes. Natural gas or propane may be used to fuel thermal equipment in the plant.

17.10 Laboratory Facilities

Analytical support, including fire assays and metallurgical testing required to support the Project operations, will be conducted on-site using a dedicated laboratory. It is assumed that approximately 100 samples per day will be delivered from the mine for assays. A small number of assays, solutions, and carbon assays will be required for metallurgical control for processing. A metallurgical lab area is also included for running bottle roll and column tests.

18 PROJECT INFRASTRUCTURE

The infrastructure for Black Pine has been developed to support mining and heap leaching operations. This includes the access road to the facility, power supply, communication, heap leach pad, process plant and ancillary buildings. Water supply to the site including tanks, pipelines, ponds, and diversions are described in more detail later in this section. Haul roads within the mining area as well as the mine waste storage facility are described in Section 16. The infrastructure envisioned is shown in Figure 18-1.

18.1 Access Road

The primary site access for the Black Pine Project will be from Salt Lake City, UT or Twin Falls, ID utilizing existing Highway-84. A new interchange consisting of new on and off ramps will be constructed approximately 5 miles north of the Idaho/Utah state line along Highway-84 at an existing overpass at BLM Road 601. From Highway-84 the route will continue due west along BLM Road 601 (Black Pine Road) for an additional 5 miles to the intersection of BLM Road 601 and BLM Road 587. The route will continue northwest along BLM Road 201 from this intersection for the last mile to the Black Pine security office.

The last mile of the route will be improved to a standard two-way road section consisting of a 12-foot-wide lane and 6-foot shoulder in each direction and have a rise of approximately 200 feet. The shoulders will provide area for any safety and drainage structures that will be needed along the route. All traffic will be required to check in at the security office before being allowed onto the site. Delivery of all personnel, operating equipment, consumables, and construction equipment will be along this primary access road.

18.2 Power Supply

Electrical utility service at the site is currently available. Electrical power will be supplied by Raft River Electric and transmitted to the Project via minor improvements to existing transmission and power lines capable of delivering up to 10MW. The improvements include substation and transmission line upgrades from the Curlew substation to the Black Pine site along the current access road. Preliminary provision has also been made within the design for an upgrade to the voltage regulators along the 25 KV line.

The costs associated with this recommendation have been captured in the Capital and Operating Cost Estimates.

18.3 Project Buildings

A truck shop is planned South of the heap-leach facility and West of the pits near the property access and administrative facilities. A fuel island will be constructed just south of the truck shop. Safety and training areas will be provided within the shop building. In addition, Mine Services offices are integral to the truck shop and a laydown yard is proposed directly northeast of the facility. The various pits are tied to their respective waste dumps and the leach pad by haul roads.

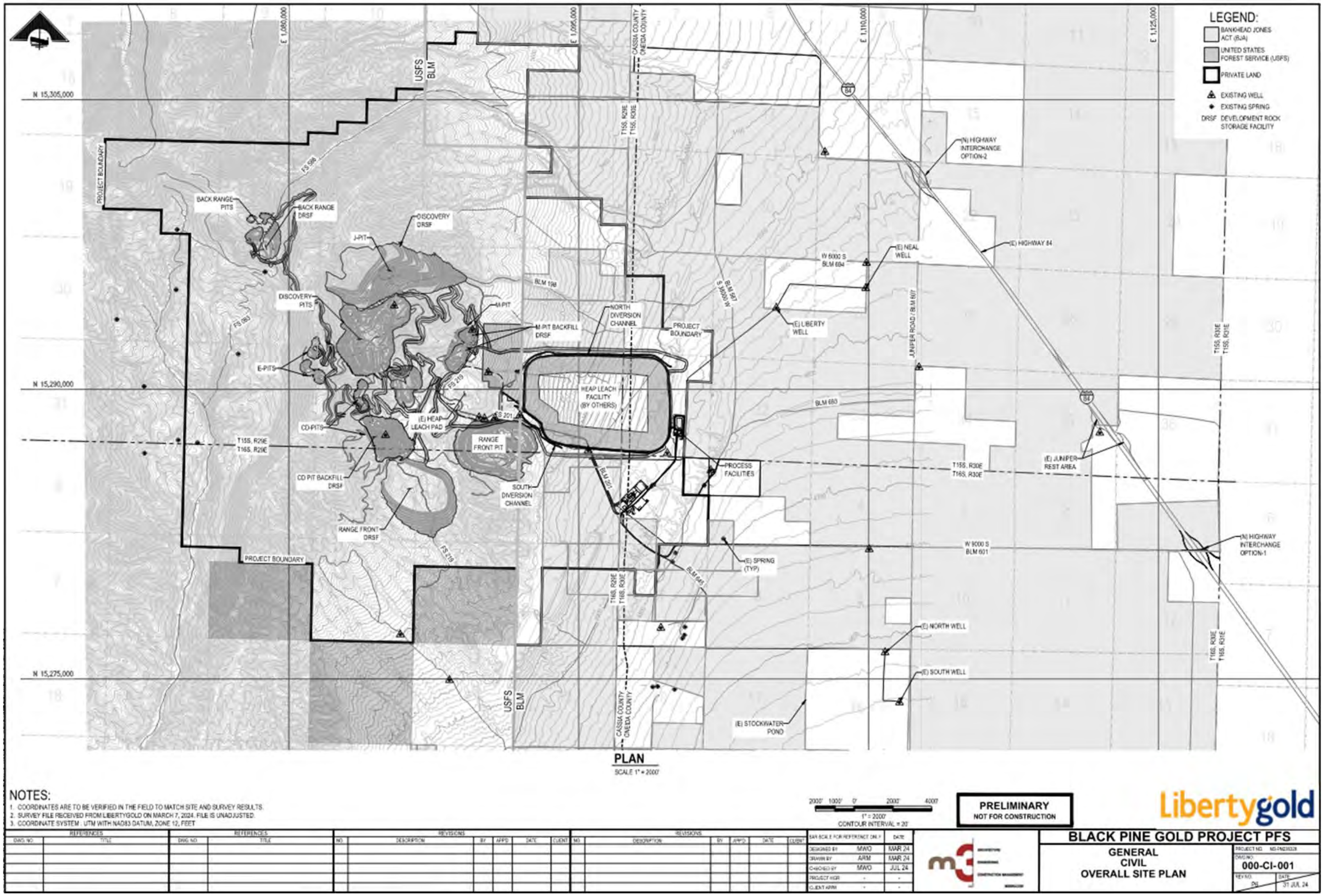


Figure 18-1: Site Plan Drawing

18.3.1 Security Building at Access Gate

The site Security Building is located up a hill for optimal visibility, approximately 1 mile along the main access road from the east property line. The Security Building includes an entry access gate that will control all site ingress egress. From the entry gate a continuous security fence surrounds the active facilities on site.

18.3.2 Administration Building

The site Administration Building is just past the Security Building also on the main road. The building will be comprised of (15) 12' x 60' mobile units that will be assembled into a single unit divided for the variety of use. The majority of these units will be used for the Administration Building, while also making provision within this facility for a large conference room and training room.

18.3.3 Truck Shop Building

As the access road continues from the Security Building to the northeast the Truck Shop is located just past the Primary Mine Substation and Fueling Station. The Truck shop is a 300' x 100' facility that has 6 bays with 2 of them design to have embedded rail to receive tracked vehicles or loaders with tire chains. The Mine Warehouse Facility is included within the footprint of the Truck Shop at the ground floor at the opposite of the bay side. The Mine Services Office and light vehicle maintenance bay is designed to be included adjacent to the warehouse space.

18.3.4 ADR Plant

The ADR Plant is located directly to the north and east of the Administrative Facilities. PLS from the Heap Leach Pad will be processed in an ADR (adsorption, desorption and recovery) plant where gold and silver will be adsorbed onto activated carbon and recovered by stripping the carbon and eventually recovering the precipitate by electrowinning. The ADR facility includes an open CIC circuit consisting of three carbon column trains operated in parallel as well as 17000 ft2 insulated, engineered steel walled building with an overall height of 45 feet. The building will contain the desorption, acid wash, and carbon handling and regeneration circuits, as well as an office, break/lunchroom, and men's and women's locker/bathroom facilities. The ADR facility also includes an attached refinery building which will be a 7500 ft2 insulated, engineered steel walled building with an overall height of 25 feet and will contain the electrowinning, mercury recovery, and smelting furnace. The ADR building includes two roll-up doors for forklift and maintenance vehicle access as well as man doors around building. The Refinery includes a secure man-door access as well as access for armored trucks via a roll-up door. The facility will include all necessary eyewash/safety shower water and fire protection systems.

18.3.5 Laboratory

The Laboratory building will be comprised of a series of mobile buildings that will be assembled into a single unit to allow for a more conventional layout. The layout will include (6) 12' x 60' buildings (60' x72' building footprint) and accommodate proper scrubbers, acid containment system, dust collection, and necessary sample processing equipment. Offices, restrooms, and change facilities for the Lab are incorporated into the layout.

18.4 Surface Water Management

The surface water management strategy entails minimizing contact water volumes by separating non-contact water surface flows from process solution sources. Potential process solution sources include the HLF, process plant, geomembrane-lined corridors, and other areas where process solution may be handled. Direct precipitation in these areas will remain within the primary and secondary containment systems. Direct precipitation from areas outside these process solution sources will be directed away from and downstream of contact areas after appropriate sediment controls. Figure 18-2 shows the conceptual surface water management plan at the end of the operating mine life. The surface water controls presented will be developed systematically throughout operations (as necessary) to minimize effects to mine property and the surrounding environment.

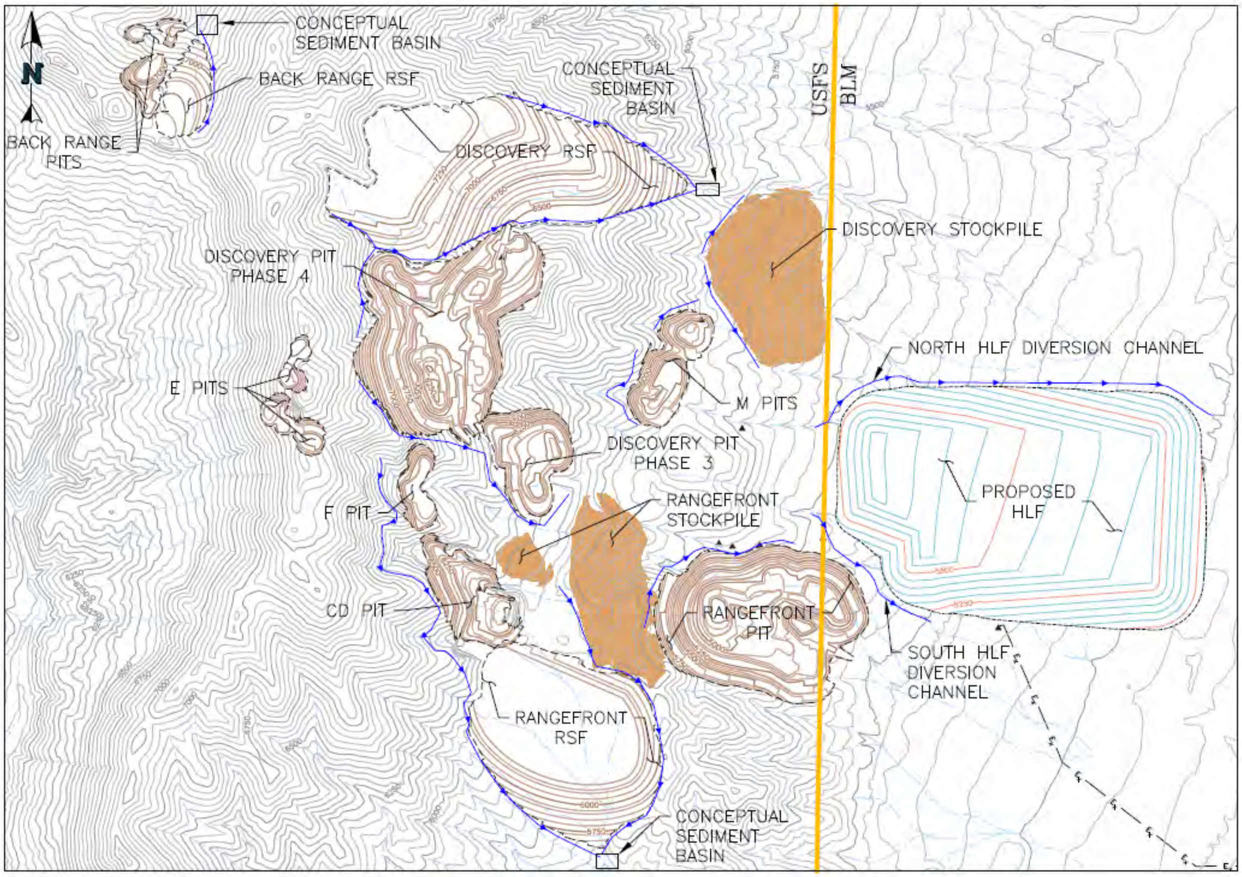


Figure 18-2: Conceptual Surface Water Management Plan

To inform the surface water management infrastructure designs, a site-wide hydrology model was developed. Culverts, sediment control structures, and diversions around the HLF were designed to contain estimated flows from the modeled 100-year, 24-hour design storm event. The National Oceanic and Atmospheric Administration ("NOAA") Atlas 2 Precipitation Frequency Estimates reported the 100-year, 24-hour design storm event at 3.06 inches for the Project. According to Idaho regulations, snow water equivalences ("SWE") were also applied to the 100-year, 24-hour design storm event modeled using the Temperature-Index Method. A constant melt rate (calculated so that the maximum SWE of 3.3 inches melts by the time of the design storm peak intensity) was applied to the hydrology model. Freeboard was included for surface water management to meet Idaho regulations, according to Table 18-1.

Table 18-1: Freeboard Requirements for Stormwater Controls

Infrastructure	Freeboard (feet)
Contact Water Channels and other Infrastructure	2
Non-Contact Water Channels	1
Sediment Ponds	2
HLF Pond (Process Component)	2

Non-contact surface water management infrastructure includes the following:

- Culverts where roads cross drainages;
- Sediment ponds downstream of development rock storage facilities to prevent discharge of sediment-laden water off-site;
- Surface water diversions around the upgradient side of the open pits and stockpiles to direct upland surface water runoff away from mining activities;
- Surface water diversions around the HLF to direct surface water runoff away from the containment area.

The diversions around the open pits and ore stockpiles were designed to protect stockpiles from erosion and to prevent nuisance flows into the pits during operations. The pit diversions were designed to convey flows from the 2-year, 24-hour design storm event, which is 1.4 inches according to the NOAA Atlas 2 Precipitation Frequency Estimates. During larger rainfall events, the diversions could overtop and flows would report directly into the pits. Water is expected to infiltrate into the pit rock or evaporate within a reasonable timeframe according to observations of the existing pits and should not cause appreciable delays in pit operations. The pit operations Ground Control Management Plan will define practices and equipment necessary to dewater the pits during operations, as necessary.

18.5 Heap Leach Facility Overview

The HLF will be constructed in four phases to contain up to 331 million dry tonnes (365 million dry tons) of leachable material: a Starter (Years 1 through 2), Phase 2 (Years 3 through 7), Phase 3 (Years 8 through 12), and Phase 4 (Year 13 through 18). The northwest portion of the HLF will receive Met 2 and Met 3 material types, located upgradient from the remainder of the HLF that will receive Met 1 material only. Met 2 and Met 3 materials are conceptually planned to be placed in the same portion of the pad; however, future studies will need to develop more detailed stacking plans to ensure the Met 2 and Met 3 materials are selectively handled to optimize recoveries and to properly account for the transition zone between material types. Within each phase, the base of the HLF will be divided into separate cells, both to manage solution draindown flows from Met 1, Met 2 and Met 3 materials, and to subdivide the HLF into smaller areas for solution collection pipe sizing.

The HLF achieves the design requirements outlined in the Idaho Rules for Processing by Cyanidation (IDAPA, 2024), including an 80-mil double-sided textured HDPE geomembrane liner underlain by 12 inches of compacted soil with a hydraulic conductivity equivalent to that of the prescribed 24-inch-thick layer of $1x10^6$ cm/s material. Additional investigations, technical studies, and evaluations will be required at future design stages, in collaboration with relevant regulatory authorities, to validate the PFS-level design concepts.

The solution collection system consists of a network of perforated pipes surrounded by free-draining gravel. The perforated pipes will transition to solid-wall HDPE pipes that will convey process solution from Met 2 and Met 3 material areas within the HLF to the process solution tanks. Process solution from Met 1 material will report to a tank located within the Event Pond, relying on the double-lined pond (with leak detection) for secondary containment.

Figure 18-3 shows the Starter HLF configuration and Figure 18-4 shows the Ultimate (Phase 4) HLF configuration. The heap was designed at 3H:1V overall side slopes to accommodate the closure footprint; actual operational slopes will likely include benching and steeper inter-bench face slopes. The maximum heap height will be 330 feet, as measured from the pad grading to the top of the heap. Leachable material will be truck-stacked in nominal 30-foot lifts. Existing infrastructure such as the land application area for the legacy HLF, powerlines, and Black Pine Mine Well, will need to be removed or relocated prior to constructing the HLF.

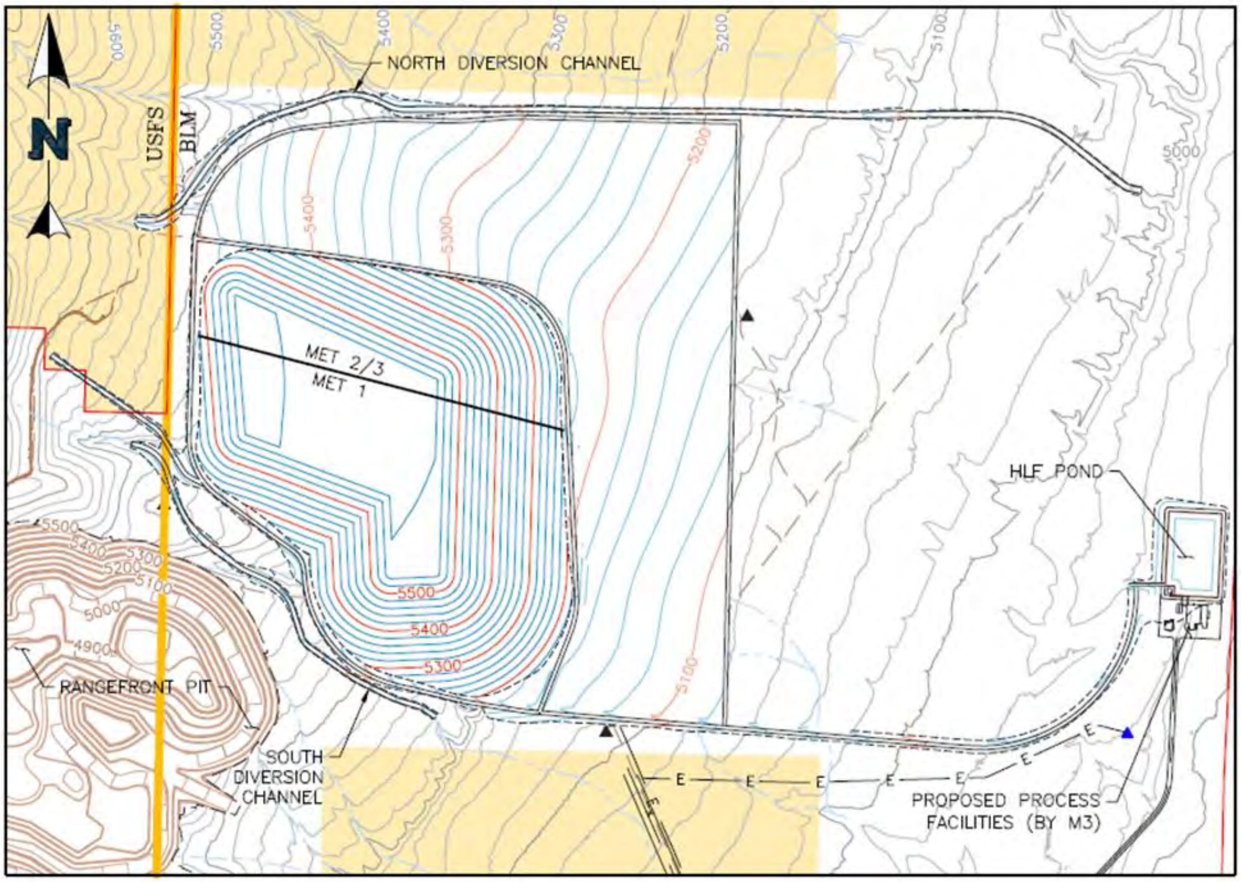


Figure 18-3: Starter HLF Configuration with Phase 2 Pad Grading

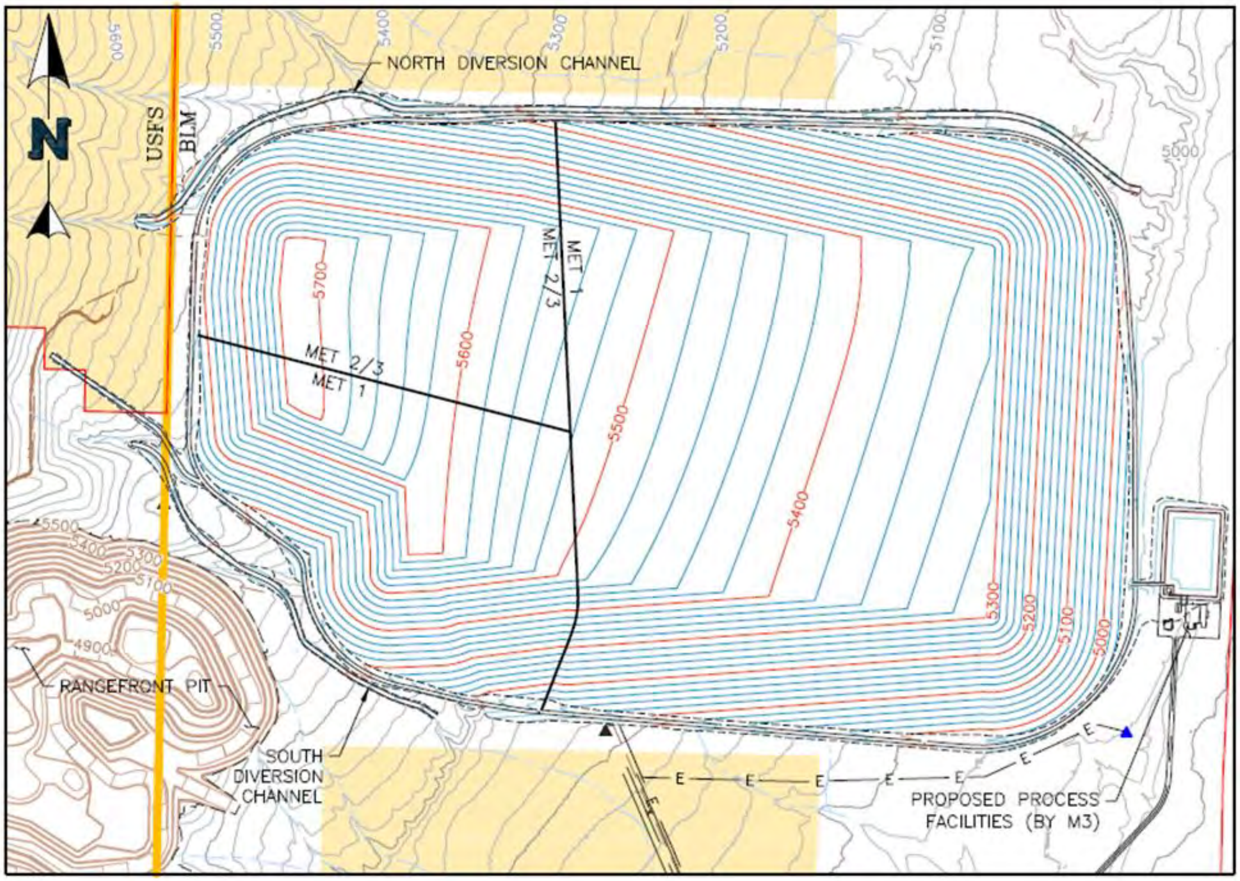


Figure 18-4: Ultimate HLF Configuration

18.6 HLF Phasing

Table 18-2 summarizes the HLF phasing and estimated capacities by phase.

Table 18-2: HLF Phasing

Phase	Estimated Construction Year	Years of Operation	Designed Capacity (million tonnes)
Starter	Year -1 / Year 0	Year 1 through Year 2	47.8
Phase 2	Year 2	Year 3 through Year 7	88.5
Phase 3	Year 7	Year 8 through Year 12	85.3
Phase 4	Year 12	Year 13 through Year 18	109.5
Total			331.1

As reported in Table 18-2, the HLF has a total capacity of approximately 331 million dry tonnes (365 million dry tons) at an assumed average placed dry density of 100 pounds per cubic foot (pcf); this exceeds the 299 million tonnes (330 million tons) estimated in the mine plan presented herein.

18.7 HLF Containment and Solution Collection/Conveyance Systems

The HLF was designed to meet regulatory requirements listed in the Idaho Rules for Processing by Cyanidation (IDAPA, 2024), including the following:

- HLF containment provided by 80 mil double-sided textured HDPE geomembrane liner underlain by 12-inches of low-permeability compacted soil beneath the geomembrane liner, equivalent in permeability to the prescribed 24-inches of $1x10^{-6}$ cm/s material.
- Solution collection and conveyance system designed to limit hydraulic head on the liner system to 12 inches:
  - A network of perforated 4-inch diameter dual-wall corrugated polyethylene ("CPe") secondary solution collection pipes at a spacing of 35 to 40 feet;
  - Primary perforated dual-wall CPe pipes, ranging from 12 inches to 24 inches in diameter depending on the total leach pad area draining to each engineered collection point;
  - A 2-foot-thick layer of clean drainage aggregate surrounding the primary solution collection pipes;
  - A 2-foot-thick layer of drainage material (likely sand and gravel) covering the surface of the geomembrane liner and solution collection pipe network; and
  - Solid-wall HDPE solution collection header pipes, ranging from 24 inches to 28 inches in diameter, to convey flows to the solution collection tank located within the Event Pond.
- Divider berms and solution collection channels within the HLF pad to direct solution draindown from Met 2 and 3 materials into a separate primary solid-wall solution collection header pipe from solution draindown from Met 1 materials.
- Solution draindown from the Met 2 and Met 3 materials will be directed to the barren tank so it can be recycled to the heap.
- Solution draindown from the Met 1 material will report to the Pregnant Solution Tank located on a tank shelf within the lined containment system of the Event Pond. From the Pregnant Solution Tank, solution will be pumped to the ADR plant for processing.
- Event or Emergency Pond sized for the maximum operational water inventory (based on the site-wide GoldSim water balance model) plus 24-hour solution draindown volume, with sufficient storage for the 100-year 24-hour design storm with two feet of freeboard. The Event Pond will consist of the following components:
  - Primary and secondary containment provided by 80 mil double-sided textured HDPE geomembrane liner;
  - Geomembrane layers separated by a 200 mil HDPE geonet layer;
  - 12-inches of low-permeability compacted soil beneath the geomembrane liner, equivalent in permeability to 24-inches of $1x10^{-6}$ cm/s material.
  - A leak detection sump and reclaim pump to return potential seepage from the primary liner back to the pond; and
  - Protective wear sheets, lean concrete protective layer, and structural concrete foundation where the tanks sit on a shelf within the geomembrane-lined pond.

Potential borrow sources with suitable engineering properties for earthworks construction were identified during the PFS geotechnical site investigation on nearby private land controlled by Pilot Gold. Extents of borrows were not determined for this study but future studies will include additional investigations to verify materials exist in sufficient quantities to support HLF development. Site investigation laboratory results suggest that borrow material with sufficiently low permeability exists to produce a 12-inch-thick secondary containment layer with a permeability equivalent to 24-inches of $1x10^{6}$ cm/s material. Additional studies will likely be required at further design stages to demonstrate chemical compatibility and other parameters necessary to satisfy Idaho regulatory requirements.

18.8 Legacy HLF Solution Draindown Management and Other Infrastructure

Solution draindown from the legacy HLF is currently collected in a series of sumps and pipes, treated, and applied to a solution application area that is within the footprint of the new HLF. During the Starter HLF construction, the solution draindown from the legacy HLF will be piped into a new 8-inch diameter HDPE pipe located in a channel lined with 80-mil double sided textured HDPE geomembrane liner for secondary containment. The pipe will terminate in the HLF Event Pond. Due to uncertainties in the flowrate from the legacy HLF draindown, the pipe is oversized; future studies will need to refine the flowrate estimate and evaluate the water chemistry to optimize the pipe sizing. There may also be a possibility that flows could be routed directly on to the new HLF, eliminating the separate pipeline and lined corridor.

The legacy HLF land application area will be regraded prior to placing the 12 inches of low-permeability soil and HDPE liner for the Starter HLF; no special treatment or removal for these soils that have been exposed to the legacy HLF draindown solution were included in the CAPEX. An abandoned powerline will also be decommissioned and removed during Starter HLF construction.

The Black Pine Mine Well and associated powerline will be relocated as a part of Phase 2 HLF construction. No additional infrastructure relocation is planned as a part of Phase 3 or 4 HLF construction.

18.9 HLF Geotechnical Evaluations

PFS-level geotechnical investigations and evaluations did not identify fatal flaws at the selected facility locations. Based on the current data, subsurface conditions within the HLF generally consist of a thin veneer of growth media, overlying alluvium consisting of loose to very dense, fine to coarse sand and gravel with varying amounts of clay and silt fines. Slope stability evaluations of the HLF slopes show the facility achieves target factors of safety for both the Starter and Ultimate HLF configurations.

18.10 HLF Design and Operational Safety

The HLF and associated structures have been designed to meet regulatory requirements and industry-accepted standards and practices, suitable for a PFS-level design. Additional investigations, evaluations, and analyses will be required at subsequent design phases to confirm assumptions and reduce the risk of encountering unforeseen conditions during construction.

During construction, a rigorous Construction Quality Assurance (CQA) program will be implemented to ensure the construction materials meet or exceed specified values that are key to HLF performance. Materials not meeting the specifications will either not be used in construction or approved after confirming the deviations will not negatively impact facility performance through modeling or other analyses, evaluations, and calculations.

A robust OMS manual will be a key component to ensure operations and monitoring controls are in place for the lifecycle of the structure. The OMS manual will include instrumentation and monitoring to provide early warning for potentially unstable conditions. These early warning systems will allow operators to monitor conditions at the HLF and provide recommendations if values trend toward thresholds for potentially unsuitable levels. The CAPEX includes preliminary instrumentation and monitoring systems.

In addition, final designs and significant design criteria or concept changes will be reviewed by a qualified third party at appropriate stages of the design process. A third-party review was completed for the PFS-level HLF design and any recommendations for the PFS have been incorporated into the designs presented herein.

18.11 Surface Water

There are no perennial streams within the Project area, with no Waters of the US, with all drainages considered ephemeral in the National Hydrology Dataset (NHD). Stantec 2020 classified a stretch of stream in Black Pine Canyon on the western side of the Project area as intermittent. Streams classified as ephemeral may occasionally convey water, especially during snowmelt, but Stantec 2020 determined that the majority do not meet the criteria for classification as a stream (Stantec 2020). Based on aerial photo imagery it appears that all intermittent streams infiltration into the ground prior to reaching the valley.

There are 34 identified seeps and springs found within and near the Project area (Donahoe 2020) (Figure 18-5). Maximum spring discharge rates, based on a review of satellite imagery, are assumed to be lower than 15 gpm (1 L/s). Some of the springs go dry seasonally or during drought periods. Surface water features are concentrated in the eastern part of the Black Pine Mountains, which suggest a geologic control on groundwater levels, with more permeable rocks in the west than in the east. There are only two surface water features in the immediate vicinity of the open pits, referred to as Silver Hills Spring and Mineral Gulch Spring. However, both “springs” are actually discharge points for old mine working adits. The discharge point elevations for the Mineral Gulch and Silver Hills Springs are approximately 6,119 and 7,054 ft (1,865 and 2,150 m) amsl, respectively. During recent sampling events the Mineral Gulch spring has been dry.

There are also six alluvial groundwater seep/spring complexes located southeast of the mine site (Figure 18-5). Discharge rates for these springs are mostly unknown, primarily because they are diffuse and are difficult to measure. The discharge at Anderson (Tree) Spring is estimated to have a combined flow of approximately 7 gpm (0.44 L/s). The discharge point elevations for these springs range between about 4,632 and 4,780 ft (1,410 and 1,430 m) amsl.

18.12 Geologic Conditions

Surface geology maps and cross sections are shown in Figure 18-6 and Figure 18-7. The Black Pine Mountains are situated in the northeast part of the Basin and Range Physiographic Province. Bedrock is composed of Devonian to Permian-age Paleozoic rocks (Mine Development Associates, 2018). Post-deposition, the rocks underwent a long period of low-angle thrust faulting and low-grade metamorphism, including the Robert Mountain Thrust and Elko / Laramide orogenies. These structural events occurred over a long period between the Late Devonian and late Cretaceous. Low-angle faulting was subsequently followed by the Cenozoic “Basin and Range” extension, resulting in substantial high-angle normal faulting that has exposed metamorphic rocks to surface. The normal faulting also contributed to formation of a horst-and-graben system of upthrown mountain blocks surrounded by valleys with significant thicknesses of Holocene- to Pleistocene- and Quaternary-age basin-fill material.

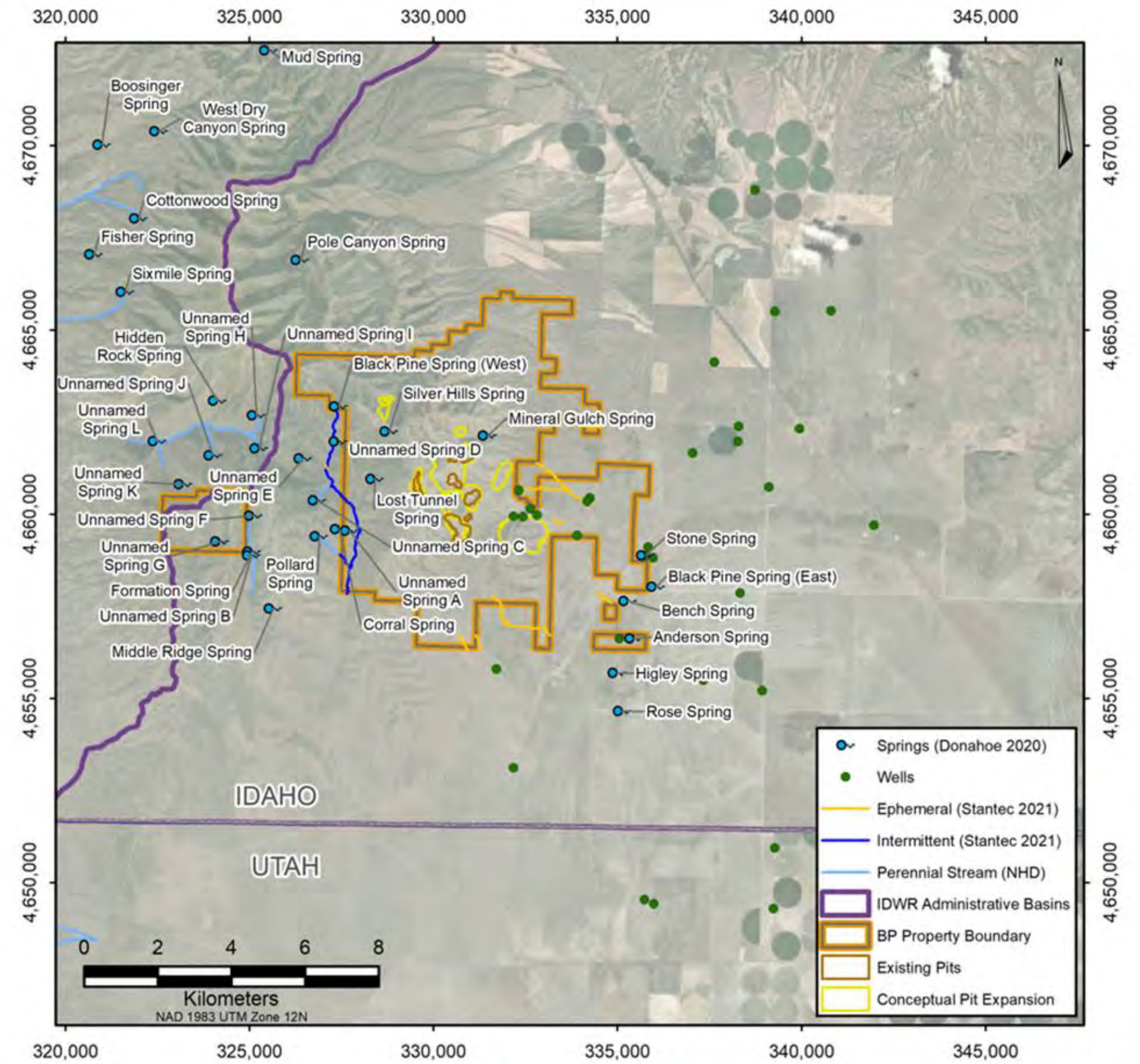


Figure 18-5: Black Pine Mountain Surface Water Features

The mine site geology includes three major structural plates:

- A lower structural plate that includes the Devonian Jefferson Formation and Late-Mississippian Manning Canyon Shale. The Manning Canyon Shale is up to 6,500 ft (2,000 m) thick.

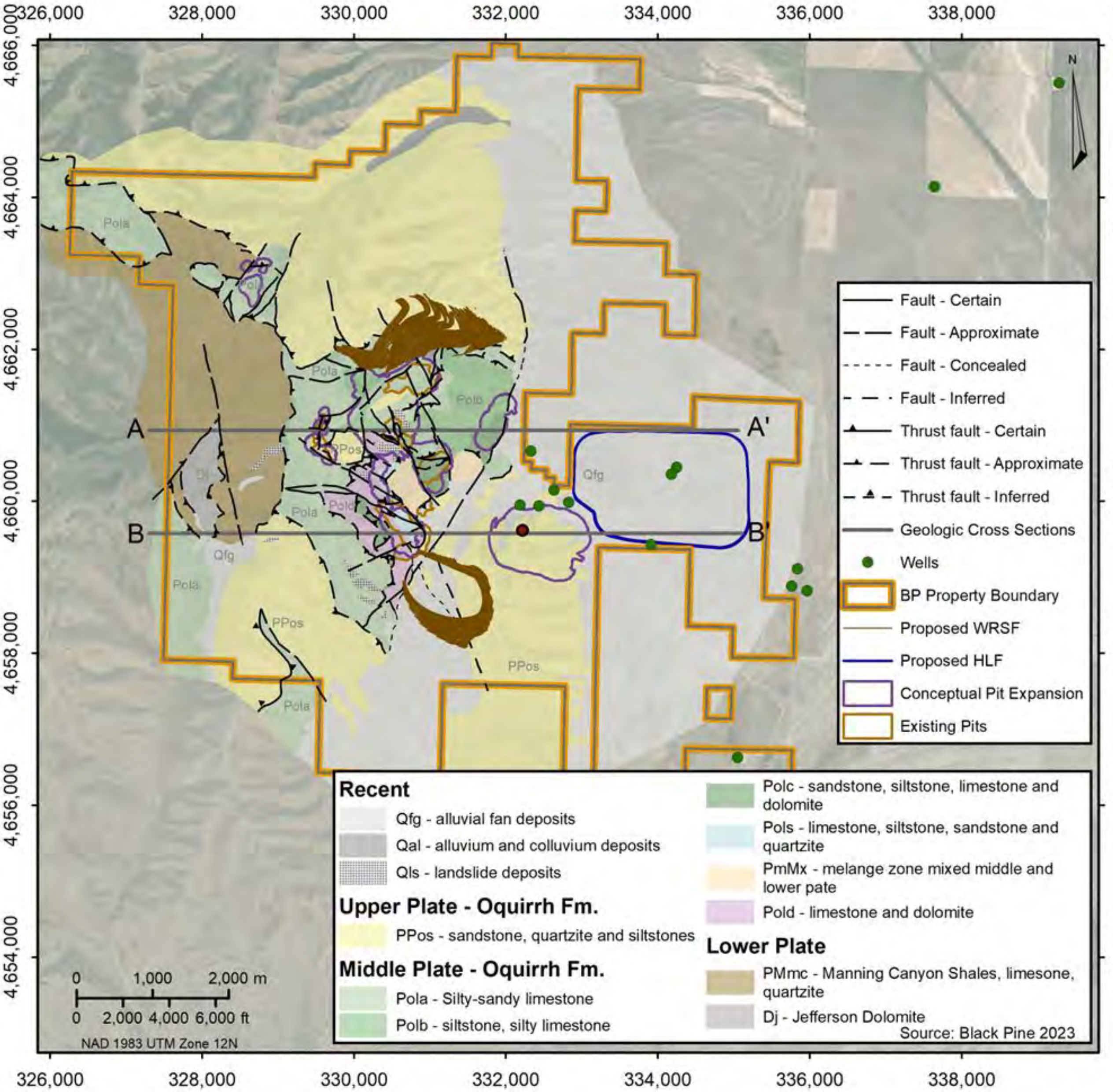


Figure 18-6: Surface Geology Map

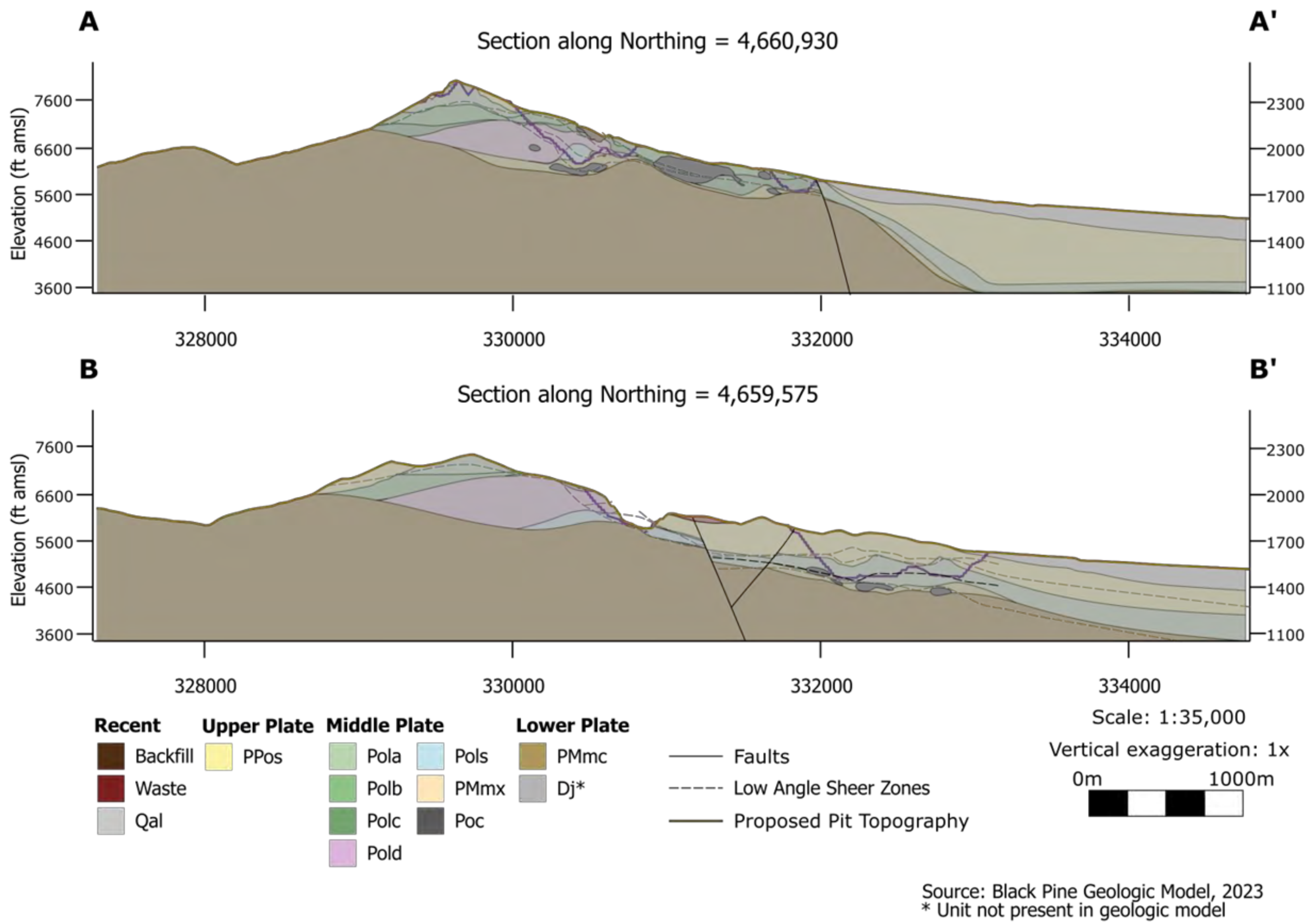


Figure 18-7: Black Pine Geology Sections

- A middle plate with limestone and dolomite units of the Oquirrh Group. In the mine site this unit is 650 to 1,300 ft (200 to 400 m) thick.
- An upper plate comprised primarily of siltstones and sandstones of the Oquirrh group.

There are narrow andesite dikes and sills intruding the Paleozoic rocks in the Project area (Mine Development Associates, 2018). They are typically narrow, less than a meter in width, with associated minor halos of contact metamorphism and alteration. The dykes are typically not mineralized.

There are significant zones of alteration and mineralization in the middle plate area (Mine Development Associates, 2018). The gold is finely-disseminated in calcareous shales and siltstone and in fault / dissolution breccias. Mineralization is concentrated at the intersection of stratigraphic units and along large low and high angle normal faults. Zones of quartz and calcite veining are also present.

These lower, middle, and upper plates are strongly folded and cut by faults. Major faults in the mine area include the low-angle structures that “stacked” the lower, middle and upper plates. There is shearing at the contact between the lower and middle plate and structural interleaving in the middle plate. Older thrust faulting is overprinted by younger normal faults. Normal faults tend to be brittle, with semi-ductile shears and milled breccia zones. There is also a significant set of range-front bounding structures at the eastern edge of the mine site that form the contact between bedrock and alluvium.

The older Holocene/Pleistocene Lake Bonneville basin-fill in Curlew Valley is comprised of alluvial and lacustrine deposits. Lake Bonneville deposits include shoreline sands and gravels formed on spits and terraces on the margins of Curlew Valley. Thick sequences of clays and silts formed at the lakebed floor. Younger and shallower Quaternary-age alluvium, colluvium and lake-shore deposits overly the Lake Bonneville deposits and are thickest along the higher elevation margins of the valley. In the northern part of the valley this includes well-drained surficial gravel deposits. In many places, the younger deposits are located above the deeper confined Lake Bonneville deposit groundwater system. However, the shallower Quaternary deposits likely play a key role in absorbing runoff and efficiently transmitting recharge along the mountain front (Bolke and Price, 1969).

The Holocene/Pleistocene basin-fill is underlain by Tertiary volcanic and continental sedimentary rocks. They often act as aquitards, but in some cases, they can provide groundwater to wells (Baker, 1974).

18.13 Hydrogeologic Conceptual Model

The Curlew Valley basin-fill groundwater system has been well-characterized by Bolke and Price (1969), Baker (1974) and Owsley (2009). The basin-fill groundwater system includes up to several thousand meters of Bonneville Lake deposits. Total recharge to this system is substantial, on the order of 24,800 gpm (1,565 L/s), as well as at least a million acre-ft (1.2 X 109 m^3) of groundwater held in storage (Bolke and Price, 1969). The estimated basin perennial yield is 6,820 gpm (430 L/s).

A preliminary study of the mine-area hydrogeology was completed by Donahoe Hydrogeology in 2020. This study developed an initial piezometric surface for the area using data from publicly available spring and groundwater levels. These data were augmented in 2024 with three multi-level Vibrating Wire Piezometer (VWP) strings targeting lower-plate saturated bedrock beneath the proposed Project Open Pits (Piteau, 2024). Figure 18-8 shows the latest-available groundwater levels for both the bedrock and basin-fill groundwater systems. Figure 18-9 shows hydrogeologic cross sections C C’ and D-D’ which intersect the mine area, the range-front faults, and the western edge of the alluvial groundwater system.

Key aspects of the Black Pine hydrogeologic conceptual model are as follows:

Bedrock Groundwater System

- The western Black Pine Mountains have relatively few springs and no perennial stream reaches, which indicates a geologic control on bedrock groundwater levels.
- Data from the three VWPs drilled in 2024 confirm the highly-fractured upper and middle plates in the western Black Pine Mountains are dry, with groundwater levels approximately coinciding with the top of the lower plate shales (Figure 18-9).
- Conceptually, this indicates surface infiltration passes freely through several hundred feet of the middle and upper plates, and subsequently reports to the saturated underlying shales.
- Bedrock groundwater levels range between about 4,745 ft (1,513 m) amsl in BPPZ-8 and 6,035 ft (1,840 m) amsl in BPPZ-4 (Figure 18-9 and Figure 18-10).
- The level in BPPZ-8 is about 225 ft (68.6 m) lower than the alluvial level in BPMW 1.5 miles (2.5 km) to the east.
- The groundwater level difference between these bedrock and alluvial measurements is attributed to a combination of (i) confining alluvial layers and (ii) the bedrock "high" beneath the Rangefront Pit (Figure 18-10) disconnecting the two systems hydraulically.
- Lateral groundwater gradients are very steep, up to about 9%, following the steep topography and lower plate contact elevation (Figure 18-7).
- Strong vertical gradients were also noted in all three VWP strings, ranging between 5 and 50%, indicating substantial vertical compartmentalization between fracture systems.
- The strong vertical gradients are probably at least in part influenced by low-angle faulting in the middle and lower plates.
- The range front faults on the eastern edge of the Black Pine Mountains are also likely isolate the bedrock and alluvial groundwater systems.
- The available data confirms at least 25 ft (7.6 m) of freeboard is anticipated below all planned Project open pits.

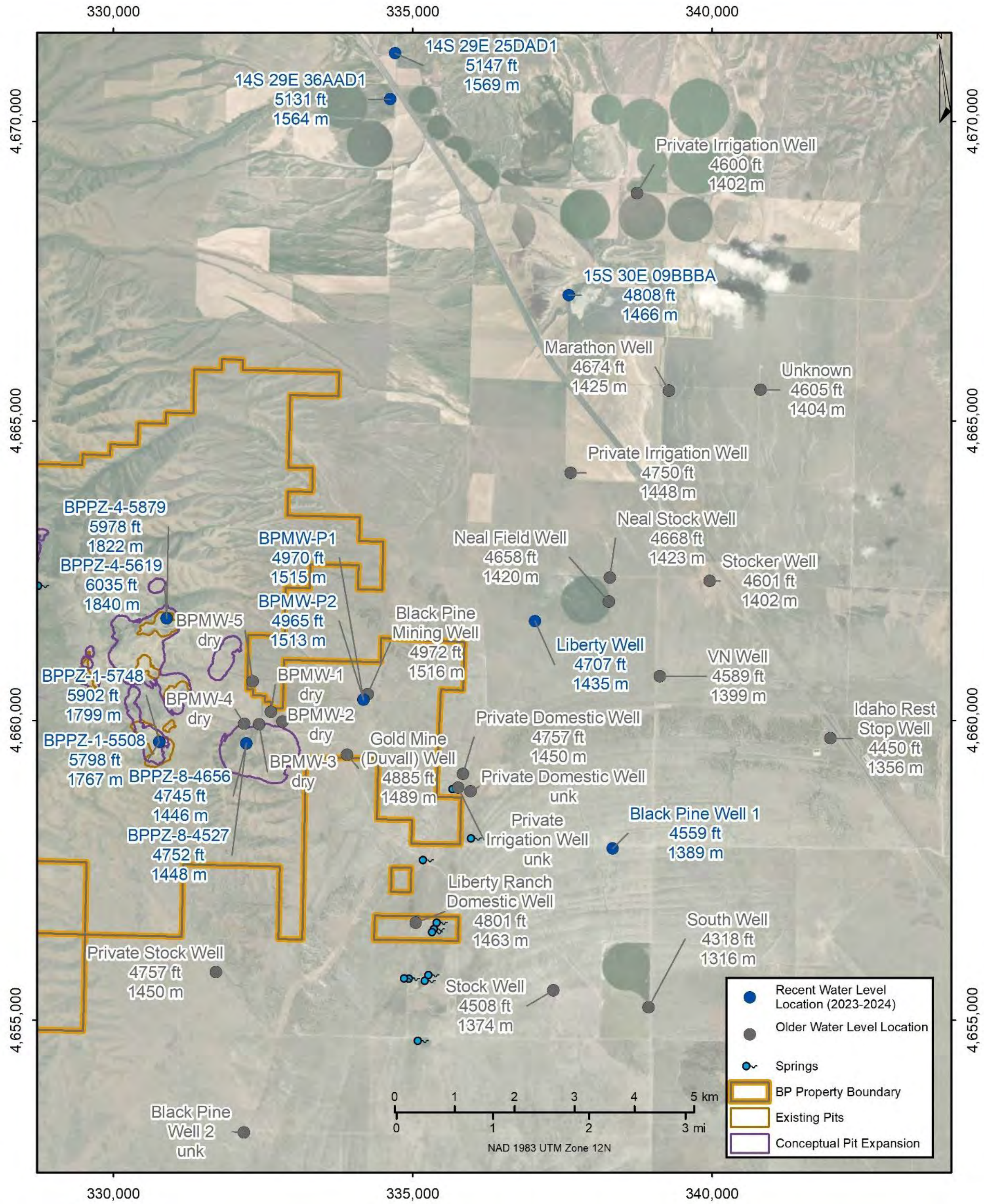


Figure 18-8: Northern Curlew Valley Groundwater Levels

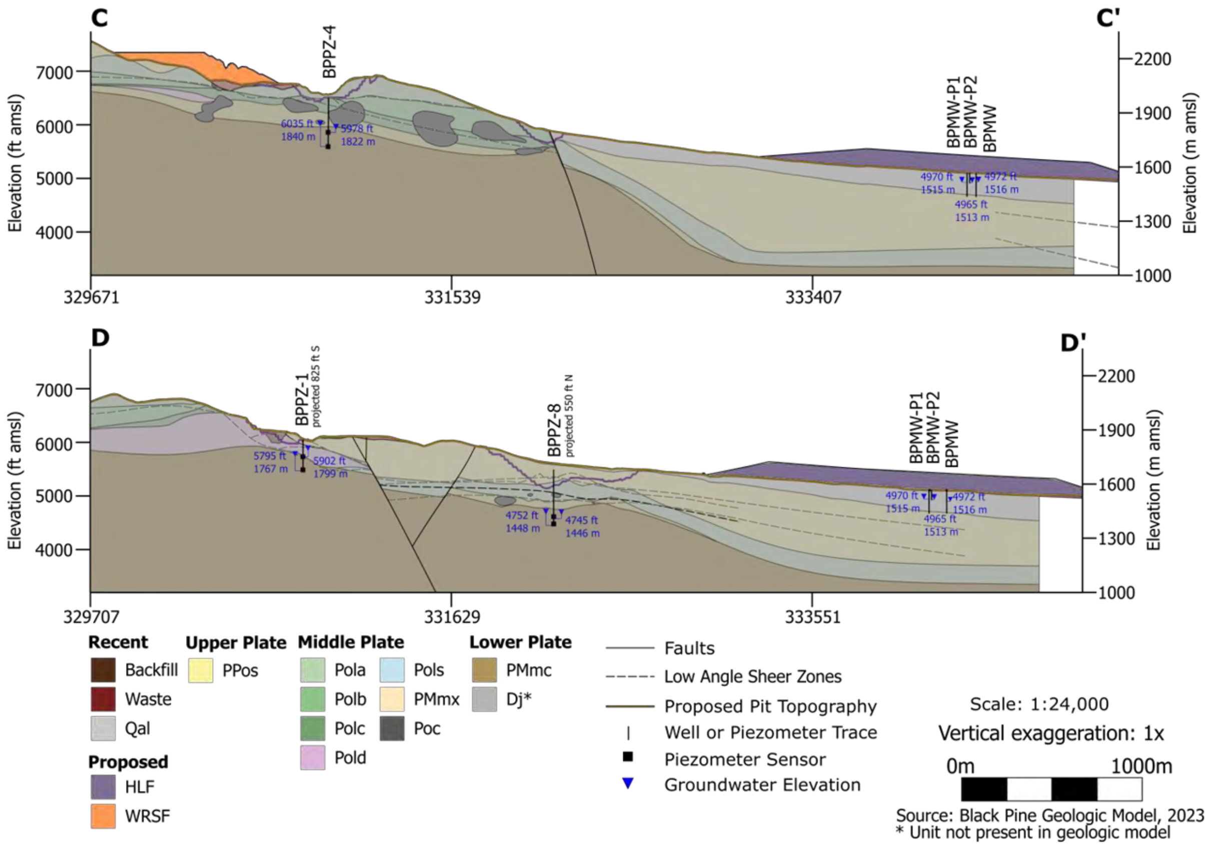


Figure 18-9: Hydrogeologic Cross Sections C-**C' and** D-**D'**

Alluvial Groundwater System

- The information from reconnaissance studies indicates the alluvial groundwater system east of the mine is hosted in (i) unconfined higher-permeability alluvial fan gravels at the range front margin and (ii) deeper confined Lake Bonneville sands and gravels, as well as Tertiary volcanics, towards the lower-elevation areas of the valley.
- The alluvial groundwater system is situated immediately east of the high-elevation Black Pine Mountains and is therefore in a significant and active groundwater recharge zone.
- There does not seem to be an indication that the wells near the Project area intercept the deeper higher-salinity groundwater system identified farther down-valley.
- Groundwater levels in the alluvial groundwater system range between 4,560 and 4,972 ft (1,390 and 1,515 m) amsl in the Black Pine Well 1 and Black Pine Mine Well, respectively (Figure 18-9).
- The six alluvial springs southeast of the mine site (Figure 18-9) have levels that range between 4,615 and 4,790 ft (1,407 and 1,460 m) amsl (Figure 18-9).
- The springs are located immediately downgradient of a large outcrop of middle-plate bedrock (Figure 18-4 and Figure 18-6).
- Given the relatively small alluvial catchment upgradient of the springs, and the difference between the spring discharge points and nearby alluvial well levels, the spring source may be deep bedrock, potentially upwelling on a range-front structure.
- Based on the sustained annual-average production rate of 530 gpm (35 L/s) from the Black Pine Mine Well in the 1990's (Mine Development Associates, 2018), and pumping rates reported by nearby agricultural center pivots that range between about 400 and 1,450 gpm (25 and 90 L/s), the alluvial groundwater system east of

the mine is (i) produces large sustainable quantities of groundwater to wells and (ii) is considered the most reliable, practical, and economic source for water supply.

- Water-supply pumping from mining will offset current agricultural pumping, so no net change in drawdowns from current conditions are expected.

18.14 Groundwater Quality

Groundwater quality sampling is completed in the spring and fall each year since 2017 at several wells and springs and at two abandoned mine adits (Silver Hills and Mineral Gulch “springs”), shown on Figure 18-10. The water quality results (Stantec, 2020) can be summarized as follows:

- The natural springs and wells have good water quality characteristics, with neutral to slightly alkaline pH, low to moderate TDS, moderate alkalinity, and low natural sulfate.
- There does not appear to be any influence of the higher-salinity deep groundwater system located to the south on the fringes of the Great Salt Lake, as would be expected given the alluvium near the Project area is actively recharged at the range front.
- Most samples have low levels of naturally-occurring barium, arsenic, and iron. Other detected trace metals in natural springs and wells include aluminum, copper, lead, and zinc. Low-level exceedances were noted for aluminum and iron. The source of these metals is not currently known but is probably related to natural mineralization.
- Anderson Tree Spring has had exceedance for iron and detectable concentrations of copper, iron, lead, manganese, selenium, and zinc.
- Higley Spring also has had an exceedance for iron and detectable concentrations of manganese, although only a single water sample is available due to access issues.

Trilinear plots for springs and mine adits are shown in Figure 18-11 and for wells in Figure 18-12. The trilinear plots show the basic cation/anion chemistry of the alluvial springs and wells are similar, with a calcium-magnesium bicarbonate, calcium-sulfate, and sodium-bicarbonate signature.

As expected, the adit discharges have distinctly different chemical profiles than the other wells and springs. Silver Hills Springs has elevated sulfate likely from water reacting in mineralized zones, but this is not indicative of a natural system. Mineral Gulch Spring is a calcium-carbonate dominated water with very low sulfate and higher alkalinity than the alluvial springs and wells. In recent sampling events it has been dry.

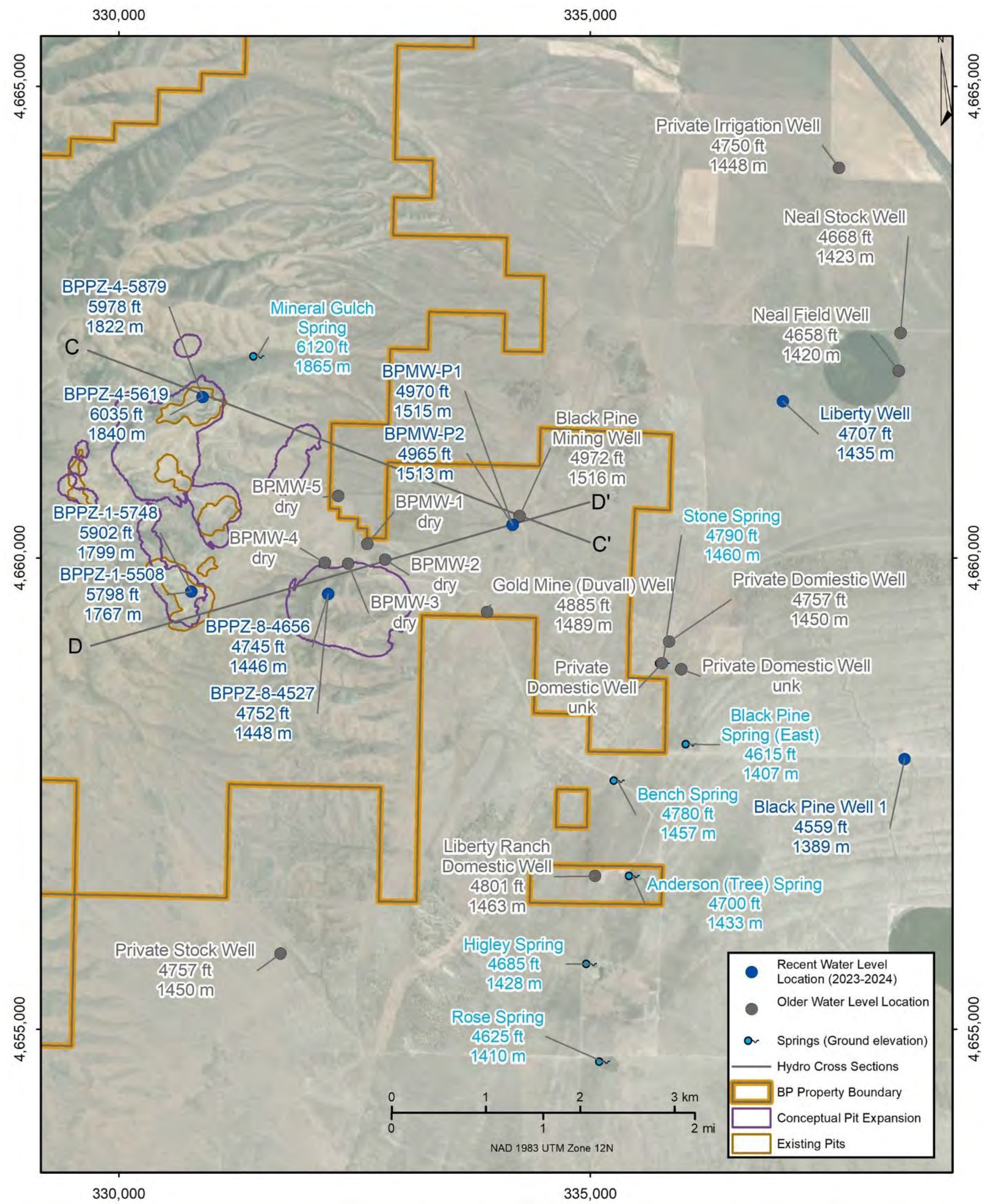


Figure 18-10: Groundwater Quality Sampling Locations

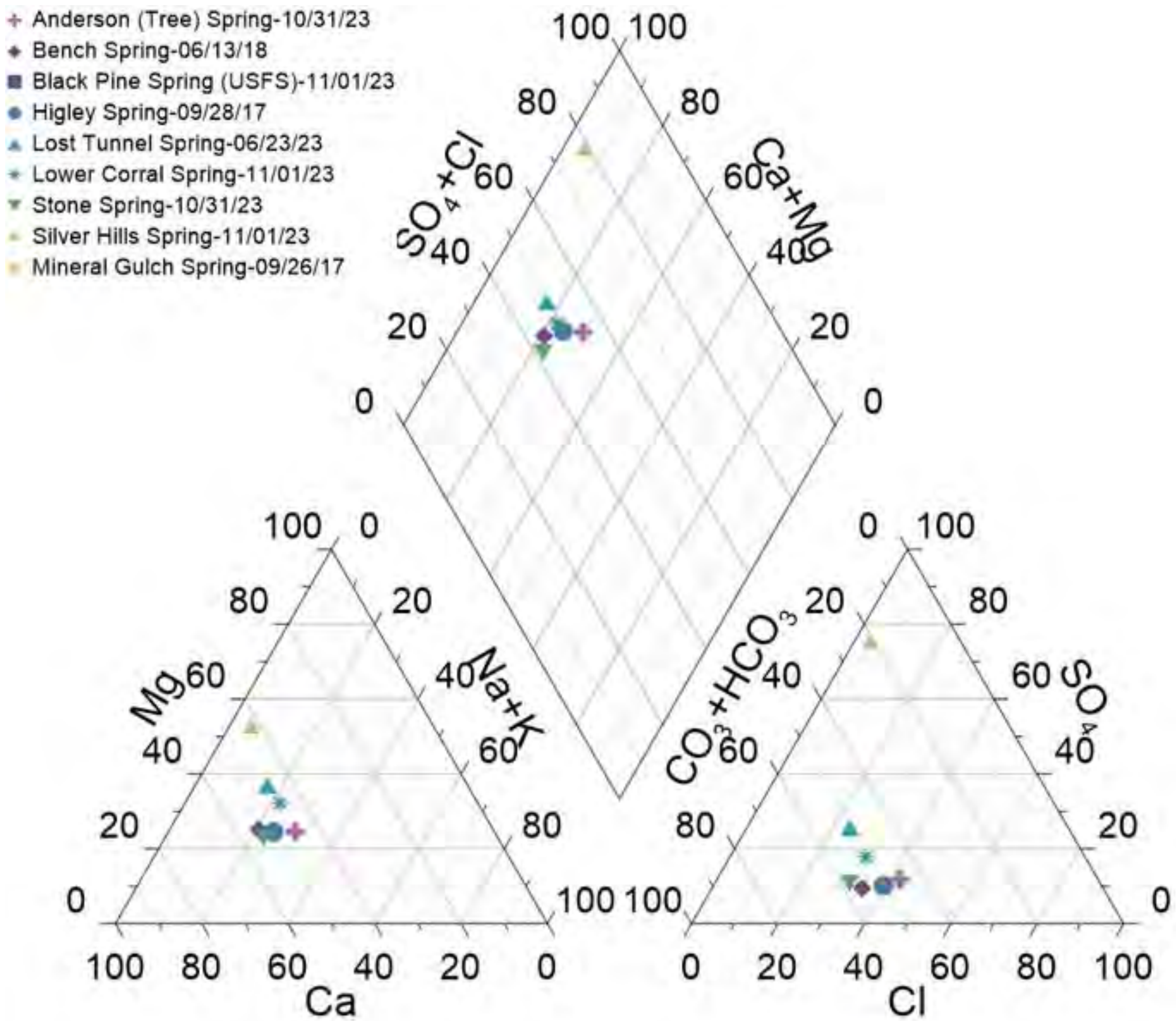


Figure 18-11: Spring Water Quality Trilinear Plot

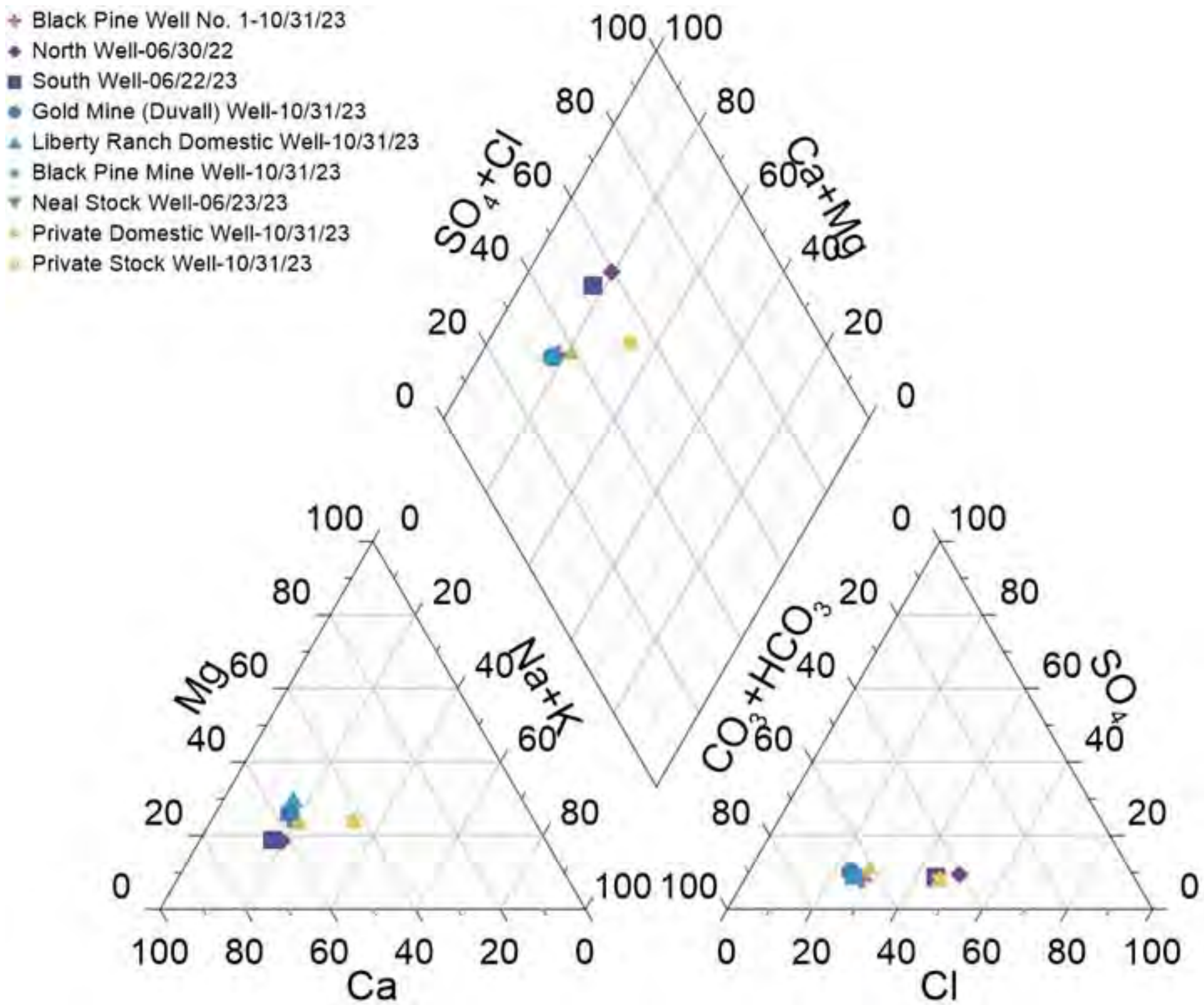


Figure 18-12: Groundwater Quality Trilinear Plot

18.15 Groundwater Supply Infrastructure Requirements

Pilot Gold has secured access to over 4,000 gpm (250 L/s) of groundwater supply for the Project. That is approximately double the peak projected rate of 2,000 gpm (126.2 L/s). Based on proximity to process facilities, the proposed water supply will be produced as follows (Figure 18-13):

- Sufficient water supply is available from the northernmost three wells (Black Pine Mine Well, Liberty Well, and Neal Field Well) (Figure 18-13).
- The existing Black Pine Mine Well is located within the footprint of the proposed HLF and will be abandoned prior to construction of the HLF. A replacement well will be needed for water supply.
- The proposed location for the replacement well BPMW-R is shown in Figure 18-13. It is assumed BPMW-R would be required to produce approximately 500 gpm (9.8 L/s).
- The Liberty Well was completed with a 16” casing and drilled to a total depth of 1,100 ft (335 m) bgs which is a reasonable diameter and depth for a production well in this setting.

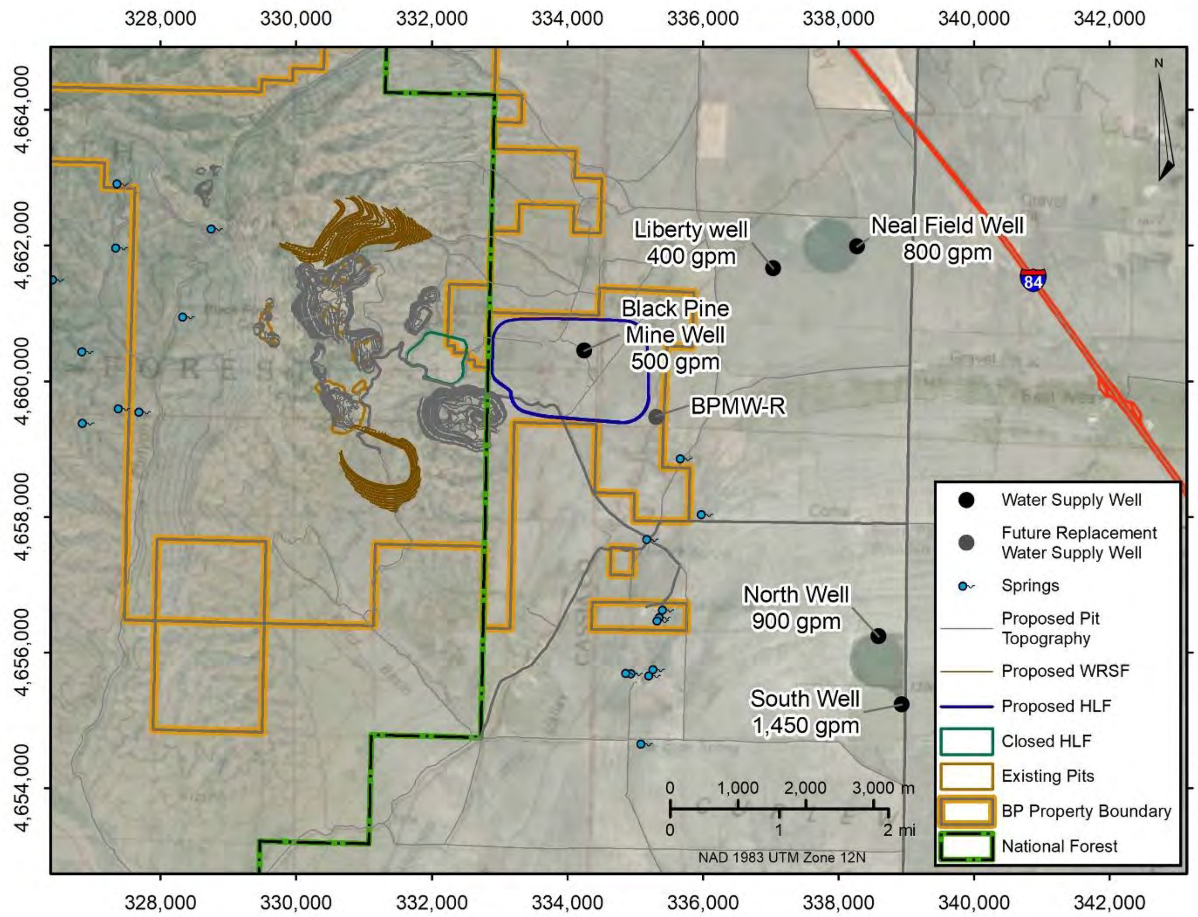


Figure 18-13: Black Pine Groundwater Supply Wells

A potable water-supply well with capacity for up to 50 employees and visitors will also be required. The cost estimate for this well is provided in Table 18-3.

Table 18-3: Potable Well Cost Estimate

Item	Cost (USD, 2023 Dollars)
Well drilling and completion	$175,000
Well development	$25,000
Power, pump, and controls	$50,000
Total	$250,000

18.16 Potential for Water Quantity and Quality Impact

The Black Pine Project proposed by Liberty Gold is considered to have low potential impacts for the following reasons:

Water Quantity:

- The 2024 field program determined the bedrock groundwater level coincides approximately with the top of the lower plate and that at least 25 ft (7.6 m) of freeboard will be maintained beneath the proposed pit. Maintaining an adequate freeboard will eliminate (i) the need for dewatering in the bedrock groundwater system and (ii) the formation of pit lakes in permanent closure.

- Water-supply pumping will be solely in alluvium. Water-supply pumping for the mine will offset pumping already being utilized for agriculture. Since the water-supply pumping for mine will either offset or be lower than agricultural pumping, the alluvial drawdowns will also be offset, thereby creating the possibility that alluvial groundwater levels could rebound.

Water Quality:

- A waste rock characterization program is underway to determine the potential for Acid Rock Drainage ("ARD"), which is currently considered to be low, and can be managed in closure, though waste rock management and partial pit backfill.

Maintaining dry open pits will substantially reduce the possibility of groundwater quality impacts to the underlying groundwater system.

19 MARKET STUDIES AND CONTRACTS

No market studies were completed, and no contracts are in place in support of this Technical Report. Gold production can generally be sold to any of a number of financial institutions or refining houses and therefore no market studies are required.

It is assumed that the doré produced at the Black Pine Project will be of a specification comparable with other regional gold producers and, as such, acceptable to all refineries.

Gold produced by the Black Pine Project would be sold to refineries or other financial institutions and the settlement price would be based on the then-current spot price for gold on public markets. There would be no direct marketing of the metal.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Existing Site Conditions

The Project and surrounding areas have been intermittently explored and developed (mined, milled and heap leached) since the late 1800's and the most recent mining operation was in the 1990's. There is evidence of these historical activities, which are in the form of roads, open pits, waste material (waste rock/overburden, spent/reclaimed heap leach pad) that are present within and adjacent to the Project. Modern day mining activities have been successfully reclaimed.

In 1943, the Virmyra Mining Company located claims near the Tolman Mine. The property was leased and mined by the Duvall Company from 1945 to 1955. The ore from the open pit mine was processed in a small mine erected near to the mine in 1950 (Bardy, 1984). The tailing from the mill appears to have been discharged into a drainage below the mill. In the early 1990s, Pegasus Gold Inc. commenced operation of the Black Pine Mine, an open pit, heap leach operation. As part of this operation, the Duvall open pit, mill area and tailing were covered with waste rock. In 1998, Pegasus Gold Inc. filed for bankruptcy. Eventually the Black Pine Mine site was partially reclaimed by the USFS, with efforts focused on the heap leach facility and waste rock dumps. The heap leach and waste rock dumps were regraded, covered with a growth media, and revegetated. The long-term drainage from the heap leach was routed to a treatment plant that uses iron as the treatment media. The treated water is piped to a leach field downgradient of the heap leach and the spent treatment media is placed in the heap leach.

20.2 Environmental Studies and Permitting

The Black Pine Project is located on National Forest System ("NFS") lands administered by the USFS, public lands administered by the BLM, State of Idaho mineral title, and private lands controlled by Pilot Gold (USA) Inc., a subsidiary of Liberty Gold in all or parts of Sections 13 through 16, 20 through 29, and 32 through 36, Township 15 South, Range 29 East (T15S, R29E), Section 31, T15S, R30E, Sections 1 through 5, 8 through 12, T16S, R29E, and Sections 7, T16S, R30E, Boise Base and Meridian. The mineral rights on the NFS lands, public lands, and state mineral title are controlled by Liberty Gold. For both the NFS lands and the public lands access to the minerals and the use of the surface estate are subject to different federal law. On the NFS lands, the vast majority of these lands within the Project area are subject to the General Law and the staking of mining claim provides access to the mineral estate. A small portion of the NFS lands in Sections 26 and 35, T15S, R29E are not open to the General Mining Law and are acquired lands and thus the mineral estate are covered by the Weeks Act of 1917, and subsequent mineral leasing laws. The BLM administers the minerals leasing program on NFS land with input from the USFS. A majority of the public lands within the Project area are open to the General Mining Law and acquired with the staking of mining claims. Section 36, T15S, R29E is acquired public lands where the State of Idaho holds the mineral estate. Portions of Sections 31, T15S, R30E and Section 6, T16S, R30E, are acquired public lands with the mineral estate in private ownership. To the east of the project area are acquired public lands that the BLM acquired under the Bankhead-Jones Act. The mineral estate on these lands is not available; however, the Project will need to utilize the BLM managed surface estate for certain project-related access activities.

Existing access to the Black Pine Project from US Interstate 84 (I84) is via the Utah State Route (SR) 30 interchange and then traveling west on SR 30 to North Black Pine Road. Travel north on North Black Pine Road into Idaho and then to the junction with Road 9000 South, then west on Road 9000 South to the Project area. In general, the proposed mine operations will consist of open pits, waste rock storage areas, and ore processing will use heap leaching. Liberty Gold plans the construction, operation, reclamation, and closing of this mining operation. Major components include:

- Seven areas of open pits;
- Two waste rock storage areas and in-pit backfilling;
- Crushing and conveying system;
- A heap-leach processing facility.

- Rerouting of the old leach field draindown into the processing facility;
- Reagent area;
- Laydown areas;
- A water delivery and distribution system;
- Power supply from the regional power grid;
- A power delivery and distribution system;
- Storm water diversion ditches and storm water sediment basins;
- Haul roads;
- Upgrade of the existing access road to the Project, including the development of an interchange with interstate 84 and Road 9000 South; and
- Truck shop, warehouse, refinery, fuel storage, and laboratory.

Liberty Gold proposes to mine approximately 50,000 tonnes per day of heap-leach-grade mineralized material. The LOM strip ratio will be 1.32 tonnes of waste per one tonne of ore. The life of the operation will be 18 years for mining, processing, reclamation and closure.

The mineralized material and waste rock would be extracted from the open pits using conventional surface mining methods of drilling, blasting, loading, and hauling. Liberty Gold would use hydraulic shovels or front-end loaders to load the blasted mineralized material and waste into the haul trucks. The haul trucks would transport the waste rock to the rock disposal area near the open pits. The haul trucks would also transport the mineralized material to the crushing system where the mineralized material would be crushed and delivered to the heap-leach pad for processing using a NaCN solution to leach the precious metals. A caron-in column process would be used to precipitate the precious metals. The precipitate would then be refined in a furnace to produce doré bars for shipment off site. The Project facilities would disturb approximately 2,729 acres.

The review and approval process for the Plan of Operations by the USFS and BLM constitutes a federal action under the National Environmental Policy Act ("NEPA"), and USFS and BLM regulations. Thus, for the USFS and BLM to process the Plan of Operations, they are required to comply with the NEPA and prepare either an Environmental Assessment ("EA"), or an Environmental Impact Statement ("EIS"). Based on the scale of the Project, Liberty Gold anticipates an EIS will be required to comply with NEPA. Currently, it is assumed that the USFS will be the Lead Federal Agency under NEPA.

The following sections provide information on historical and recent site characterization efforts, existing environmental conditions, status of project approval and permitting efforts, social and community considerations, proposed mitigation of stream and wetland disturbance, and reclamation and closure activities.

20.2.1 Environmental Baseline Studies

Liberty Gold has contracted qualified third parties to perform environmental baseline data collection and review the adequacy of existing environmental baseline reports and data. This baseline data collection is ongoing through 2024. Additionally, EAs were completed, and Plans of Operations approved in 1988, 1991, and 1993 for the Black Pine Mine by the previous operator for the site.

20.2.1.1 Air Quality and Meteorology

The general remote location of the Project affords the site relatively good air quality. Notable exceptions include fugitive dust emissions from vehicle travel on unimproved roads and from adjacent agricultural practices (planting) during dry conditions.

Climate has previously been characterized as arid in the lower project site elevation and sub humid on the slopes where the mine development will occur. Average precipitation (14.79 inches) has been extrapolated from nearby weather stations (Strevell, ID and Snowville, UT). Historical temperatures range from -10's (C) in the winter to 30's (C) in the summer. A Campbell Scientific weather station has been installed at the Project to correlate with regional data and establish site specific data (i.e., precipitation, evaporation, wind speed/direction, etc.). In addition, Liberty Gold has been collecting PM2.5 air quality data at the weather station.

20.2.1.2 Soils

Soil baseline is usually collected to determine quality and quantity that exists in undisturbed planned development area(s) and, what may be available for stockpiling/reclamation purposes. The general soil types have been characterized by past BLM surveys and includes the following: Argic Calciorthids, Lithic Calciorthids, Calcixerollic Duriargids, Pachic Argiborolls and Argic Pachic Cryoboroll Soils. Surface soils in the area consist mostly of loams and gravelly loams with moderate permeability and moderate resistance to erosion. In all cases, the agency(s) recommend collection, stockpiling and stabilization of available soils from disturbance areas for reclamation. In addition, a new soil survey will be completed on the NFS lands within the Project area in Q3 of 2024.

20.2.1.3 Hydrology

20.2.1.3.1 Surface Waters

Perennial and intermittent stream segments within the analysis area are only found in Black Pine Canyon, which is located in the Black Pine Range and west of the Project. Spring sources for these segments are very low flow springs, located on the west side of the drainage, geologically separated from the locations proposed for drilling high on the east side of the drainage. The perennial and intermittent reaches are short, extending downstream from springs for only a short distance (< ¼ mile) before becoming ephemeral. In addition, there are a series of at least six springs on the east edge of Project area.

Based on the 2021 jurisdictional delineation, there are not Clean Water Act Jurisdictional Waters of the United States within or adjacent to the Project.

A seep and spring survey within and surrounding the Project area will be completed in 2024.

20.2.1.3.2 Groundwater

There are two main types of aquifers present in the Project area; alluvial and bedrock. Alluvial aquifers in the Black Pine area occur in permeable zones consisting primarily of unconsolidated sediments (e.g. sand, gravel) within the alluvial deposits along the valley bottoms and flanking the Black Pine Range to the east of the Project. Bedrock aquifers are those that occur within fractures or structures within the Paleozoic bedrock that comprise the surrounding hillslopes and occur beneath the valley alluvium. Groundwater quality does not appear to be impacted by historical exploration and development at site.

Given the long history of mining and exploration, groundwater resources in the vicinity of the Black Pine Mine, including the proposed area of drilling, have been well documented. The main hydrostratigraphic units in the area of the proposed operations are most likely associated with the Middle and Lower Pennsylvanian Oquirrh Formation. However, given the structural complexity of these strata, the lateral and vertical extent of these units is limited.

To date, no measurable groundwater has been intersected in exploration drill holes, within the areas of the open pits, waste rock dumps or the area of the heap leach pad to depths ranging from 300 to 700 feet. A surface expression of groundwater is not present in the existing open pits that remain within the Project area in years since mining operations ceased at the Black Pine site. During drilling it is possible that there could be very minor pressure increases or decreases in the aquifer as a result of encountering lost circulation or discrete water entry zones.

A work plan to conduct groundwater data collection has been prepared by Liberty Gold's consultant. The work plan will be delivered to the USFS and BLM in July 2024 for their review and concurrence prior to commencing any activities. The work plan currently addresses data collection within the mining area.

20.2.1.4 Geochemical (Ore and Waste) Characterization

Geochemical analysis is currently ongoing and includes whole rock analyses, acid base accounting and static leaching tests on samples from historical and recent drilling. The initial work indicates that there is little risk of acid generation from the mined materials. However, the potential for neutral metal leaching does exist, particularly for aluminum, antimony, and arsenic, as well as barium and selenium. In addition, results from testing reveal that ore and waste rock material are highly alkaline and contain very little total sulfur. Thus, the acid generating potential has been characterized as negligible. The report on this initial geochemical work will be completed in June 2024 and presented to the USFS and BLM July 2024. Any additional geochemical work will commence once the agencies have provided input on the initial work.

20.2.1.5 Vegetation

The proposed Project area lies in a very sparsely populated portion of the sagebrush-grass region of Southern Idaho in a very northeastern portion of the Basin and Range Province. The area supports a semi-desert type of vegetation know as shrub-steppe. The location of the various vegetation communities found in any one area are influenced by soils, aspect, temperature, moisture, and elevation. Because both elevation and aspect change across the Project area, which results in a range of soils, temperature, and moisture, the vegetation found there is correspondingly diverse. It is possible to describe the site vegetation in terms of four major plant communities: Conifer/Mountain-mahogany, Sagebrush-Grass, Utah Juniper Subalpine, Sagebrush Grass Community Types. The majority of the proposed project would occur within the sagebrush grass community. The sagebrush-grass is the most extensive plant community in the study area, occurring in all habitats not otherwise supporting juniper, mountain mahogany/conifer or subalpine types. These habitats range from the warm, dry, gently sloping lower elevations extending to the valley floors, up the increasingly steeper slopes of the foothills to all but the steepest north-facing slopes and highest mountain ridges. As is true of any plant community found in such diverse environments, the sagebrush-grass community can be further broken down into several discrete plant associations, of habitat types based on the particular species of sagebrush and grasses dominating the site.

There is no documentation of the existence of a population of threatened, endangered, rare, or sensitive plants in the proposed Project area. Two sensitive plant species have been discovered in the Black Pine Range (Eriogonum desertorum and Pediocactus simpsonii), and they are found on War Eagle Peak, the highest elevation in the range, approximately 4.0 kilometers northwest of the proposed project.

Liberty Gold will be conducting additional vegetation surveys within the Project area in 2024.

20.2.1.6 Wildlife

There are no known occurrences of threatened, endangered, or candidate species or designated habitat for any ESA listed species within the Project area. The Project area does provide habitat for greater sage-grouse (Centrocercus urophasianus) ("GSG"), a USFS sensitive and Management Indicator Species ("MIS") species. On October 2, 2015, the U.S. Fish and Wildlife Service found the GSG does not warrant listing at this time.

The Sawtooth National Forest ("SNF") provides habitat for 20 terrestrial wildlife species on the Regional Forester's sensitive species list. The Minidoka Ranger District, Black Pine Division of the SNF provides potential habitat for 14 of these wildlife species. These species include:

Species	Project Likelihood
Bald Eagle (Haliaeetus leucocephalus)	low
California Bighorn Sheep (Ovis Canadensis californiana)	low
Columbian Sharp-tailed Grouse (Tympanuchus phasianellus columbianus)	moderate
Columbia Spotted Frog (Rana luteiventris)	low
Flammulated Owl (Psiloscops flammeolus)	low
Golden Eagle (Aquila chrysaetos)	high
Gray Wolf (Canis lupus)	low
Greater Sage Grouse (Centrocercus urophasianus)	high
Northern Goshawk (Accipter gentiles)	low
Peregrine Falcon (Falco peregrinus)	low
Pygmy Rabbit (Brachylagus idahoensis)	low
Townsend's Bigeared Bat (Corynorhinus townsendii)	moderate
Spotted Bat (Euderma maculatum)	moderate
North American Wolverine (Gulo gulo)	low
Yellow-billed Cuckoo (Coccyzus americanus)	low
Mule Deer (Odocoileus hemionus)	high
Moose (Alces alces shirasi)	high
Rocky Mountain Elk (Cervus elaphus canadensis)	high
Northern Leatherside Chub (Lepidomeda copei)	none
Yellowstone Cutthroat Trout (Oncorhynchus clarkii bouvieri)	none

Project development, operations and closure will need to determine if species/habitat exists and if so, mitigation of impact(s) may be required. GSG and its habitat in and around the Project area is considered to be the most sensitive species that may be impacted by the Project. The 2024 sage grouse survey identified two active leks within the vicinity of the Project. However, there appears to be a very limited number of birds utilizing these leks. Liberty Gold has also attempted, but with no success, to complete multiple tagging efforts. Given the habitat conditions in the vicinity of the Project area, any GSG mitigation will likely occur in the Greater Curlew Valley, away from the Project area.

Liberty Gold has funded with IDFG four telemetry surveys to assess the mule deer migration in the vicinity of the Project area. The final report was completed in June 2024. Within the Black Pine Range, most of the mule deer use is in Black Canyon to the west of the Project area. The mule deer migrate to the north and east to the Sublett Range.

Additional wildlife surveys will be conducted in 2024, including acoustical surveys for bats.

20.2.1.7 Archeological and Historical Resources

Cultural resource inventories for the Project area were conducted by ASM Affiliates, Inc. ("ASM") in the summer of 2017. Eligible and unevaluated cultural features, as identified in the cultural inventories would be avoided by all exploration activities. This avoidance strategy would be confirmed through the annual implementation plan for each phase. If previously undiscovered cultural resources are identified during exploration activities, the discoveries would be left intact and the agency(s) would be contacted and, if appropriate, tribal governments consulted for guidance. These activities will be coordinated through the State Historical Preservation Office ("SHPO"). The final surveys for the Project area will be completed in Q3 2024. Eligible sites within the Project area have been identified. These eligible sites are historic in character. Prehistoric sites have not been identified. Some of the eligible sites will be impacted by

Project activities. Treatment of those sites will have to be completed after project approval and prior to project activities in the vicinity of treated sites.

20.2.1.8 Tribal Relations

Because sovereign tribal governments can be affected by the policies and actions of the USFA and BLM in managing the lands and resources under their respective jurisdictions, the USFS and BLM each have a duty to consult with them on matters affecting their interests under their government-to-government responsibilities. Because of this government-to-government relationship, Liberty Gold expects the USFS and BLM to engage in efforts to involve sovereign tribal governments and to solicit their input regarding the Project.

At the request of the USFS, Liberty Gold has yet to initiate their own tribal engagement. The Shoshone Bannock are the tribal entity with interest in the area.

20.2.1.9 Visual Resources

The Project area is mapped with Visual Quality Objective's ("VQOs") of Partial Retention, Modification and a very small area of Maximum Modification. This site/area existed as a mining area before the Sawtooth Forest mapped VQOs in 1981. As a result, the USFS mapping for appropriate VQO does not reflect the preexisting disturbance associated with mining and reasonably foreseeable management action at the time VQOs were established. The appropriate VQO for an area with the type of minerals exploration and extraction activities would be Maximum Modification. This VQO, while acceptable to the levels of disturbance typical of this type of minerals extraction, also has very specific scenery guidelines protecting the resource, such that the Project area management activities must appear as natural from the background viewing distance, which would be reasonably considered achievable in a landscape such as this with appropriate management elements and design features and/or mitigations. This will need to be incorporated into reclamation and closure design(s). The USFS is requiring Liberty Gold to complete visual simulations of the planned Project features, which will be completed in 2024.

20.2.1.10 Land Use

The Project area is used primarily as undeveloped land that is managed for recreation and wildlife habitat purposes. The area surrounding the Project is predominantly a rural agricultural area (cropland and grazing). Livestock production is another key element with beef, sheep, swine and dairy production. Minerals that have been mined in the area include gold, silver, copper, mercury, and zinc.

20.2.2 Federal Permitting

As discussed above, the Project occurs on both NFS lands and public lands with use of these lands being subject to multiple USFS and BLM regulatory programs. The USFS will require a Plan of Operations under 36 Code of Federal Regulations ("CFR") 228, and the BLM will require a Plan of Operations under 43 CFR 3809. In addition, the BLM will require a Mining Plan under 43 CFR 3500 for those activities on the acquired lands at that NFS lands. For activities on the BLM managed acquired lands and those lands with private or State of Idaho mineral estate, the BLM will require a Plan of Development under 43 CFR 2800. It is anticipated that all these federal permitting requirements can be addressed in a single Mine Plan of Operations ("MPO") permit application to the USFS and the BLM.

20.2.2.1 United States Forest Service/Bureau of Land Management Plan of Operations

The MPO is submitted to the USFS and the BLM for any mining operation on NFS lands and public lands. The MPO describes the operational procedures for the construction, operation, and closure of the Project. As required by the USFS and the BLM, the MPO includes a waste rock management plan, quality assurance plan, a storm water plan, a spill prevention plan, reclamation plan, a monitoring plan, and an interim management plan. In addition, a reclamation report with a Reclamation Cost Estimate ("RCE") for the closure of the Project is required. The content of the MPO is based on the mine plan design and the data gathered as part of the environmental baseline studies. The MPO includes all mine and processing design information and mining methods. The BLM determines the completeness of the MPO and, when the completeness letter is submitted to the proponent, the NEPA process begins. The RCE is reviewed by the USFS, and the BLM and the bond is determined prior to the BLM issuing a decision on the MPO.

The MPO will be submitted for the Project when operational and baseline surveys are complete and operations and design for the Project are at a level where an MPO can be developed to the necessary level of detail. Submittal of the MPO is likely to occur in the fourth quarter of 2024.

20.2.2.2 Environmental Impact Statement

Approval of any MPO and Reclamation Plan by the federal agencies for the Project as well as accordance with Section 404 requires an environmental analysis under the NEPA. NEPA requires federal agencies study and consider the likely environmental impacts of the proposed action before taking whatever federal action is necessary for the Project to proceed.

The purpose and need for the Project would be to conduct open pit mining and ore processing, which would disturb approximately 2,729 acres of unpatented mining claims and private land within the Project area and complete reclamation and closure activities, as well as long-term water treatment, to produce silver and gold from mineralized material of the estimated Mineral Resources. As a result, Liberty Gold anticipates that an EIS will be required to meet agency NEPA requirements.

The USFS is likely to be the lead federal agency for the preparation of the EIS given the scope of activities planned on NFS lands, and other agencies will be cooperating agencies. However, that has not been determined and the BLM may be the lead. The EIS and associated Record of Decisions ("RODs") effectively drive the entire permitting process timeline.

20.2.3 Idaho Permitting

20.2.3.1 Idaho Stormwater Permit

Storm water discharges associated with this industrial activity require a related permit. Storm water is defined as "storm water runoff, snowmelt runoff, and surface runoff and drainage." Where flows are from conveyances that are not contaminated by contact with overburden or other mine waste, a permit is not required. Hence, the water management scheme developed for the Project endeavors to collect and convey clean water around the mining operation and discharge downstream. Active storm water would be managed via a storm-water pollution prevention plan, which must be submitted to the IDEQ at least 180 days before commencing discharge.

20.2.3.2 Air Quality Operating Permits

The Project will require an Air Quality Permit to Construct. This will likely be a Tier II permit to establish a facility emission cap, establish mercury best available control technology limits, or establish synthetic minor emission rate limits. Incorporated into this permit or in a separate Title V permit, the Project will have to complete with the mercury emission limits establish in the US Environmental Protection Agency's Maximum Achievable Control Technology ("MACT") standard.

20.2.3.3 Cyanidation Permit

The cyanidation permit is required by the IDEQ and is applicable for a facility that processes mineralized material using cyanide as the primary reagent. Liberty Gold is proposing to process the gold and silver mineralized material at a heap-leach facility. The regulations apply to both operations and closure and reclamation of that Facility.

Prior to submitting an application, Liberty Gold should contact IDEQ during the initial stages of site characterization to schedule a pre-application conference and it is recommended to begin meeting with IDEQ at least one (1) year in advance of preliminary design submittal to discuss the following: environmental baseline data requirements; waste characterization requirements; siting requirements; operation and maintenance plans; emergency and spill response plans; quality assurance/quality control plans; and, required contents for permit applications. In addition, the proposed water quality monitoring and reporting required and monitoring well siting and construction plans should be discussed with IDEQ. The pre-application conference may trigger a period of collaborative efforts between Liberty Gold, IDEQ, and the Idaho Department of Lands to develop an application that complies with rule requirements and ensures the facility will not interfere with the beneficial uses of waters and will not endanger public safety or the environment.

As part of IDEQ's groundwater rule, there are minimum requirements for groundwater protection through standards and a set of aquifer protection categories. To implement the rule, Liberty Gold would need to establish points of compliance outside and down-gradient from the mine area(s). Liberty Gold would also establish reasonable upper-tolerance limits for all compliance wells, working directly with IDEQ. These upper-tolerance limits would need to take into account the high naturally occurring background levels for several parameters.

20.2.3.4 Other Major State Authorizations, Licenses, and Permits

The key authorizations, licenses, and permits required by the State of Idaho are summarized in this section. The federal and state application processes would be integrated and processed concurrent with the EIS as follows:

- Water Rights – Liberty Gold has secured approximately 3,200 acre-feet of groundwater rights with a combination of agricultural and mining use. The points of diversion, manner of use, and place of use will need some modifications for use by the Project. The modification from agricultural use to mining use will result in a decrease in the available acre-feet due to the loss return flows from the agricultural use. The Project will require approximately 1,300 acre-feet annually of water for the operations;
- Stream Channel Alteration Permit – Required by the IDWR for a modification, alteration, or relocation of any stream channel within or below the mean high-water mark. The Project contemplates relocating one or more unnamed drainages as part of the overall mine plan;
- Water and Wastewater Systems – The drinking water system(s) design for the contemplated facilities must be approved prior to use to ensure compliance with the Safe Drinking Water Act;
- Fuel Storage Facilities – Any proposed fuel storage must also comply with IDEQ design and operating standards, as well as EPA Standards under 40 CFR 112 (may require a Spill Prevention and Countermeasure Plan depending on the size of the facility), Idaho State Fire Marshall, and Owyhee County requirements;
- Reclamation Plan – All surface mines must submit and obtain approval of a comprehensive reclamation plan (Title 47) from the Idaho Department of Lands ("IDL"). The Reclamation Plan includes detailed operating plans showing pits, mineral stockpiles, overburdened piles, ponds, haul roads, and all related facilities. The Reclamation Plan must also address appropriate best management practices ("BMPs") and provide for financial assurance in the amount necessary to reclaim those mining activities. The plan must be approved prior to any surface disturbance;
- Encroachment Permit – A Right-of-Way Encroachment Permit from the Idaho Transportation Department will be necessary to develop the interchange with Interstate 84 and Road 9000 South;

- Sewer and water systems approval by South Central District of the Idaho Health Department for facilities in Cassia County, and from the Southeastern District of the Idaho Health Department for facilities in Oneida County; and
- Others – State requirements would also involve compliance with the Idaho Solid Waste Management Regulations and Standards, transportation safety requirements enforced by the Idaho Public Utilities Commission, and various other authorizations.

20.2.4 Local County Requirements

There are several other permits and approvals that would apply to the Project, if it proceeded to a full-scale mining proposal, including:

- Conformance with the Cassia and Oneida counties Comprehensive Plans; and
- Issuance of building permits by the county(ies);

Additionally, an annual authorization by the Cassia and Oneida counties road departments road use permits for any mining operation is essential. The permit addresses standard operating procedures for the road route to be used, seasonal limits, spill prevention and response planning, HAZWOPER or hazardous materials handling training, convoying, and other requirements.

Liberty Gold has not entered into any agreements with local communities; however, there have been discussions with the local communities.

20.2.5 Idaho Joint Review Process

The IDL is responsible for implementation of the Idaho Joint Review Process, which was established to coordinate and facilitate the overall mine permitting process in the state. It involves an interagency Memorandum of Understanding ("MOU") between involved state and federal agencies and addresses a process to achieve pre-analysis coordination in approving/administering exploration permits, interagency agreement on plan completeness, alternatives considered, draft and final permits, bonding during mine plan analysis, and interagency coordination related to compliance, permit changes and reclamation/closure for major mining projects. In Idaho, the Joint Review Process was established as the basis for interagency agreement (state, federal, and local) on all permit review requirements. The focus of the process is concurrent analysis timelines. This would include, for example, in the case of this project the NEPA process, air quality permit, reclamation permit, and the cyanidation permit. The Idaho Joint Review Process would play a key role in achieving two primary permitting goals: 1) increased communication and cooperation between the various involved governmental agencies, and 2) reduced conflict, delay, and costs in the permitting process.

20.3 EIS/Permitting Timelines and Costs

20.3.1 Permitting Timelines

The discussion below assumes that the USFS will be the lead federal agency for NEPA, and that the BLM will be a cooperating agency. With regard to the likely scope of the Project, the following conceptual description was developed as the basis for this permitting analysis:

- Regulatory – EIS required; USFS Lead Agency; BLM, IDEQ, and IDL are cooperating agencies;
- Mining– Estimated at 50,000 tonnes per day of mineralized material with a 1.32:1 waste to mineralized material strip ratio;
- Processing – Heap leaching of ore;
- Power – It is anticipated that the existing line power would be utilized to meet power needs;

- Waste Rock – Some selective placement would be required likely due to potential geochemical reactivity; large volumes would be stored and managed;
- Water Supply – Available from existing water rights;
- Project Access – Utilizing existing county-maintained roads and the development of an interchange with Interstate 84 and Road 9000 South;
- Manpower – Up to 250 direct and indirect jobs during construction; estimated 420 during operations;
- Operating Schedule – Mining and processing year-round;
- Total Land Disturbance – Approximately 2,729 acres of disturbance on NFS lands, public lands, and private lands; and
- The Plan of Operations will be submitted to the USFS and BLM by the end of 2024.

An EIS/permitting sequencing is summarized below in five primary permitting windows.

- Commence preparation of the Initial Plan of Operations. Concurrently, develop all other permit applications for submittal. Liberty Gold would file its Plan of Operations with the USFS/BLM, which would trigger the EIS. This is currently anticipated for the end of 2024;
- Selection of third-party EIS Contractor, Start baseline confirmatory studies for surface and ground water, geochemistry, aquatic resources, wildlife, cultural resources, vegetation and soils as well as air quality and wetlands work. This work should be completed in 2024. Negotiate an MOU with the USFS for preparing the EIS. Conduct initial internal scoping with "high up" agency and political contacts;
- The third-party EIS contractor would finalize the EIS work plan and initiate early environmental baseline adequacy determination write-ups for the various resource categories (air, water, socio-economics, etc.);
    - The contractor would also write the alternatives section of the EIS. This is a significant section that must present only reasonable and potentially feasible alternatives as required under NEPA;
    - The USFS/BLM would publish the Notice of Intent to prepare an EIS during this timeframe;
- A preliminary draft EIS would be completed by the USFS (Third-Party Contractor). This document would be for the lead and cooperating agencies and Liberty Gold review only;
- A Draft EIS would be produced for public review and the public comment period;
- The USFS/EIS Contractor would review and develop responses to the public comments. In addition, the preliminary Final EIS is prepared for review by the USFS, cooperating agencies, and Liberty Gold.
- A Final EIS would be produced for public review during a 30-day period; and
- Ideally, the USFS and BLM would issue their RODs at the end of the 30-day period.

20.4 Social and community

The Project is located in rural Cassia and Oneida counties, close to Interstate 84 and near the Utah border. The closest substantial community is Juniper, a very small agricultural community. However, when the mine previously operated in the 1990s the employees most likely lived in Snowville, Utah or in Burley, Idaho and the surrounding communities. To date, Liberty Gold has not completed the development of a community engagement plan. Liberty Gold has initiated discussion with the federal state regulatory agencies.

20.5 Waste Characterization

Liberty Gold is conducting a mine waste characterization program as part of the planning and impact assessment for the Project. Geochemical testing of mine waste materials provides a basis for assessment of the potential for metal leaching ("ML") or ARD, prediction of contact water quality, and evaluation of options for design, construction, and closure of the mine facilities. This work also supports the next phase of the Project's potential advancement, including environmental assessment and permitting. The characterization effort focuses on the assessment of waste rock geochemistry, evaluation of tailing material from mineral benefaction processes, and determination of final pit wall geochemistry.

Geochemical characterization is an iterative process and sample collection for the Project is being completed in phases. The first phase is complete and involved the collection of samples from core generated during the recent exploration drilling activities and conducting static geochemical testing. Subsequent phases of the characterization program would focus on improving the spatial representation of the dataset as drill core from the ongoing exploration and geotechnical drilling becomes available, as well as completed additional static testing and completed kinetic testing.

20.6 Closure and Reclamation Strategy

A comprehensive reclamation and closure plan would be developed for all disturbances and infrastructure associated with the Project. Reclamation objective standards established by industry best practices and regulatory requirements for reclamation would be fulfilled. Liberty Gold would seek to develop an economic mine plan and closure/reclamation strategy that integrates habitats and restoration components. It is anticipated that the reclamation and closure of the heap-leach facility would consist of fluid management through first active and then passive evaporation and then discharge of any long-term draindown in a leach field, either with or without treatment. The reclaimed facilities will be covered with growth media, and then revegetating. The estimated reclamation costs for the Project, using the Nevada Standardized Reclamation Cost Estimator is $54 million.

The goals of this reclamation and closure plan are expected to evolve based on cooperative discussions, public and regulatory input; however, the initial goals include:

- Protecting water quality;
- Restricting or eliminating the migration of potential contaminants of concern from all sources based on the proposed mine plan;
- Restricting or eliminating potential public safety risks associated with the potential decommissioned and reclaimed mine site;
- Restoring the property, to the extent possible, to the current pre-mining conditions; and
- Improving the property by incorporating environmental mitigation projects as identified through the NEPA process.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated for the pre-feasibility study by AGP (mine development) and M3 (process plant, site development, power transmission and distribution, and ancillaries), and NewFields (heap leach facility). Table 21-1 shows the estimated capital costs for the Project. This includes $326.6 million in Year -1 and $219.8 million for sustaining capital. Total capital costs are estimated at $546.3 million.

Table 21-1: Capital Cost Summary

Category	Units	Initial	Sustaining	Total
Site General (Earthworks)	K USD	$11,785	-	$11,785
Process Plant (ADR, Refinery, Reagents)	K USD	$47,741	$9,474	$57,215
Power Systems	K USD	$4,253	-	$4,253
ADR Bldg. & Ancil. (Warehouse, Maint, Admin, Fuel)	K USD	$22,924	-	$22,924
Freight (Process Plant)	K USD	$3,408	-	$3,408
Sub-Total Direct Cost (Process Plant & Support)	K USD	$90,111	$9,474	$99,585
Construction Support (inc. Mobilization)	K USD	$2,703	-	$2,703
Engineering, Procurement, & Const. Mgmt.	K USD	$15,500	-	$15,500
Vendor Support	K USD	$1,618	-	$1,618
Spare Parts (Capital, Commissioning)	K USD	$1,578	-	$1,578
First Fills (Process Plant)	K USD	$480	-	$480
Contingency (Process Plant)	K USD	$22,398	-	$22,398
Owner's Cost	K USD	$9,200	$10,625	$19,825
Sub-Total Indirect Cost (Process Plant & Support)	K USD	$53,477	$10,625	$64,102
Heap Leach Facility Direct Cost (NewFields)	K USD	$47,515	$107,791	$155,306
Heap Leach Facility Indirect Cost (NewFields)	K USD	$14,755	$35,467	$50,222
Sub-Total Heap Leach Facility	K USD	$62,269	$143,258	$205,528
Mine Capital Equipment (AGP)	K USD	$31,411	$56,398	$87,809
Mine Preproduction Costs (AGP)	K USD	$89,291	-	$89,291
Sub-Total Mine Capital	K USD	$120,702	$56,398	$177,100
TOTAL CAPITAL COST	K USD	$326,560	$219,755	$546,315

Table 21-2 shows the estimated operating costs for the LOM Project. Operating costs were estimated at $2.726 billion for the LOM. This is $9.11 per tonne processed or $1,245 per ounce of gold produced.

Table 21-2: Operating Cost Summary

		Production Cost	
Category	K USD	$ / tonne	$ / Au oz
Mining Costs	$1,945,536	$6.50	$888.96
Process Plant	$538,322	$1.80	$245.97
G&A	$219,950	$0.73	$100.50
Refining	$21,908	$0.07	$10.01
TOTAL OPERATING COST	$2,725,716	$ 9.11	$1,245.43

21.1 Process Capital

21.1.1 Process Capital Cost Summary

The process plant costs are comprised of costs for the process facilities, infrastructure development, power systems and ancillaries. The direct costs are developed from labor, materials, plant equipment, sub-contracts, construction equipment, and freight. Indirect costs are applied to the direct costs to account for items such as: construction support; engineering, procurement, and construction management ("EPCM"); vendor support during specialty construction and commissioning; spare parts; contingency; and owner's costs. Together, the direct and indirect costs form the capital costs.

The process plant includes the adsorption, desorption and recovery plant, as well as the refinery and reagents. The ancillaries include components such as the truck shop and warehouse, truck wash, laboratory, mine dry, and the fuel station.

Indirect costs were then calculated following industry accepted methodologies, including application of contingency based on the completed level of design on a scope or individual work type basis. The agglomerate contingency for the process plant is estimated at 20% of total contracted cost. Total contracted costs include all process plant direct costs, plus construction support costs, EPCM costs, vendor support costs, and spare parts costs. First fills were calculated by M3. Owner's Costs were developed in collaboration with Liberty Gold.

Process plant capital costs were independently developed, and all capital cost estimates are based on the purchase of new equipment.

The total evaluated project cost is projected to be in the accuracy range of -20%/+25%.

Table 21-3: Initial Capital Process Plant Cost Summary

Category (all costs are in USD 1,000)	Labor	Plant Equip.	Material	Sub Contract	Const. Equip.	Total
Site General	6,978	535	1,590	561	2,122	11,785
Solution Transfer	3,348	3,092	3,079	157	593	10,269
ADR	7,217	13,614	4,741	556	992	27,120
Refinery	906	294	682	125	132	2,139
Water Systems	1,776	3,444	921	67	241	6,449
Power Systems	1,590	1,320	1,260	83	0	4,253
Reagents	658	616	359	36	95	1,764
Ancillary Facilities	4,071	3,649	3,401	11,129	674	22,924
Freight		2,125	1,283			3,408
Sub-Total Direct Cost (Process Plant)	26,544	28,688	17,315	12,714	4,849	90,111
Construction Support (inc. Mobilization)						2,703
Engineering, Procurement, & Const. Mgmt.						15,500
Vendor Support						1,618
Spare Parts (Capital, Commissioning)						1,578
First Fills (Process Plant)						480
Contingency (Process Plant)						22,398
Owner's Cost						9,200
Sub-Total Indirect Cost (Process Plant)						53,477
TOTAL CAPITAL COST (Process Plant)						143,588

21.1.2 Freight

Estimates for equipment and material freight costs are based on bulk freight loads and have been estimated at 8% of the equipment and material cost.

21.1.3 Construction Support

Mobilization is included as an indirect cost at 3% of total direct field costs for process plant direct costs.

21.1.4 EPCM

Engineering is included at 6.5% of total constructed cost ("TCC") for the process plant scope. Project management and administration is included at 0.75% of TCC. Project services are included at 1.0% of TCC. Project controls are included at 0.75% of TCC. Construction Management is included at 6.0% of TCC.

An EPCM Fee is included at 1% of total direct field cost.

Temporary construction facilities are included at 0.5% of TCC. Temporary construction power is included at 0.1% of TCC.

21.1.5 Vendor Support

Vendor supervision of specialty construction is included at 1.5% of plant equipment supply costs. Vendor pre-commissioning is included at 0.5% of plant equipment supply costs. Vendor commissioning is included at 0.5% of plant equipment supply costs.

Start-up Assistance by the contractor is included at 1% of Total Direct Field Costs.

21.1.6 Process Operating Cost Exclusions

The following operating costs are excluded from the process plant operating cost estimate:

- G&A costs (see section below)
- Access road and internal roads maintenance
- Operating cost contingency
- Escalation costs
- Currency exchange fluctuations

21.1.7 Spare Parts

Capital spare parts are included at 5.0% of plant equipment supply costs. Commissioning spare parts are included at 0.5% of plant equipment supply costs. Two-year operating spare parts are excluded.

21.2 **Owner's Costs**

Owner's Costs were developed in collaboration with Liberty Gold. The Owner's Costs include items such as salaries and wages for the Project personnel, housing, and accommodations for owner's team during project development, transportation for owner's team during project development, owner's team vehicles, office services, and travel during project development. There is also an allowance for external services, such as geotechnical investigation and permit support.

The Owner's Costs are estimated at 5% of Total Contracted Cost. Additionally, there is an allowance of $3.6 million for a royalty percentage buydown that is envisioned to be executed by Liberty Gold.

21.3 HLF and Associated Infrastructure, Water Management, and Closure Capital Costs

21.3.1 Unit Rates

Unit rates were estimated using a combination of the following:

- Supply vendor quotes for specific materials such as piping and geosynthetics.
- Contractor labor rates built up using Idaho Prevailing Wages, with additional factors to account for per diem rates (given the remote project site), estimated contractor markup, a small hand tool allowance, and a separate tax for the contractor's labor.
- Contractor equipment rates are estimated using the available equipment rates in the Nevada Standardized Reclamation Cost Estimator (SRCE, 2023).
- Production rates for each unit rate were calculated based on similar project experience and estimates of contractors' unit rates. The Caterpillar Performance Handbook Edition 49 was referenced for equipment specifications and capabilities.

21.3.2 Quantities/Material Take-Offs

The CAPEX is quantity-based using engineering-derived material take-off ("MTO") quantities from the PFS-level designs. The MTO was generated from AutoCAD Civil 3D design surfaces and 2023 survey data as well as typical layouts, sections, and details developed for this Project.

This estimate includes site preparation, earthworks, geosynthetics, piping, and associated surface water management infrastructure such as diversions, sediment ponds, and applicable erosion protection measures. Additional allowances have been made for indirect costs such as engineering, construction management, quality control, and surveying, as well as contractor costs to mobilize, demobilize, and provide temporary stormwater controls during construction. Variable contingency and growth rates were applied based on the level of confidence in the MTOs and unit rates, as well as for construction timing (lower contingency and growth applied to the Starter and Phase 2, and higher rates applied to Phases 3 and 4).

A detailed SRCE cost model was not developed to inform the CAPEX for this PFS; the CAPEX reflects estimated construction costs required to close and reclaim the Project, rather than SRCE which is intended to aid in estimating reclamation bonds. A high-level SRCE model was developed for comparison purposes only; the resultant closure costs were similar to those reported in the CAPEX. The general assumptions used in the closure CAPEX includes regrading, capping, and revegetating the HLF and roads, converting the HLF emergency pond to an evaporation or evapotranspiration cell, demolishing structures, and capping/revegetating concrete foundations.

21.4 Mining Capital Costs

21.4.1 Pre-Production Mining

Mining activity starts in Year -1 at Black Pine and is scheduled to move 13.9 Mt of waste material and place 13.1 Mt of mineralized material into stockpiles. The $89.3 million in operating costs associated mining in this period are included in the capital cost estimate. The pre-production operating cost covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology labor, grade control costs, and mobilization costs. It also includes any finance costs that have been added to the operating cost for that period.

A portion of the waste material moved will be used to develop mine haul roads, infrastructure platforms and heap leach facility construction needs. The earthwork costs for placing this material is not included as a mining cost and covered in other areas.

21.4.2 Mine Equipment

The mine fleet will be financed to reduce initial capital requirements. The terms are based on standard terms of a 20% down payment applied to capital with the remainder applied to operating costs with a provided interest rate of 6.0%. Equipment pricing was based on quotations from vendors with some equipment information from AGP's database of recent projects. The base costs provided by the vendors are included for each unit cost calculation and include added options. The base cost, initial downpayment and full finance cost for the major mining equipment is shown in Table 21-4.

Table 21-4: Major Mine Equipment – Capital Cost and Full Finance Cost

Equipment	Capital Cost $	Initial Payment $	Total Finance Cost $
Primary Drill (165 mm)	2,688,000	537,600	2,961,706
Secondary Drill (165 mm)	1,171,000	234,200	1,290,237
Hydraulic Shovel (22 m^3)	7,560,000	1,512,000	8,527,487
Hydraulic Excavator (6.7 m^3)	1,924,000	384,800	2,170,223
Haul Truck (144 t)	2,641,000	528,200	2,978,981
Haul Truck (64 t)	2,445,000	489,000	2,757,898
Production Loader (11.5 m^3)	2,130,000	426,000	2,402,586
Track Dozer	1,644,000	328,800	1,854,390
Grader	401,000	80,200	441,832

Various pieces of mining equipment were not financed including items such as spare truck boxes, and shovel buckets. Other equipment not financed includes the blasting truck, and pump truck, as well as the ambulance, fire truck, 100 tonne lowboy and tractor, and the crushing and screening plant for road surface material and blasthole stemming.

The cost of spare truck trays, loader buckets and shovel clams are included in the capital cost for the major equipment cost estimate. Options around rebuilds and recertification of equipment like track dozers is not considered, nor is used equipment. Mobile mining equipment is diesel, pit electrification is not planned for Black Pine in this study.

The blasthole drills purchased for Black Pine are diesel powered. The diesel drills have more mobility than electric drills and can be moved to the different pits and to areas in the steep terrain where access is limited. The purchase of a 100-tonne lowboy and tractor will assist in moving drills and dozers between the various pit areas.

Table 21-5: Major Equipment Purchases – Initial and Sustaining

Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
Primary Drill (165 mm)	3	2	1				2	2	2									
Secondary Drill (165 mm)	1	1						1	1									
Hydraulic Shovel (22 m^3)	1	2					2											
Hydraulic Excavator (6.7 m^3)	2	1																
Production Loader (11.5 m^3)	2	1																
Haul Truck (144 t)	12	11		5	2		1					3	8	9				
Haul Truck (64 t)	11	2		2							4							
Track Dozer	6	2							6	1								
Grader	3	1								2	1							

Table 21-6: Major Equipment Fleet Size

Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
Primary Drill (165 mm)	3	5	6	6	6	6	6	6	5	6	6	6	6	6	5	4	3	
Secondary Drill (165 mm)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Hydraulic Shovel (22 m^3)	1	3	3	3	3	3	3	4	4	4	4	4	4	4	4	3	3	3
Hydraulic Excavator (6.5 m^3)	2	3	3	3	3	3	3	2	2									
Production Loader (11.5 m^3)	2	3	3	3	3	3	3	3	3	3	3	6	3	3	3	3	2	1
Haul Truck (144 t)	12	23	23	28	30	30	31	31	31	31	31	34	34	32	27	27	22	22
Haul Truck (64 t)	11	13	13	15	15	15	15	15	15									
Track Dozer	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	7	3
Grader	3	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3	3	1

The number of equipment units is determined by the mine schedule and the estimate of required operating hours. There are periods where the maximum hours per fleet required are below the maximum fleet hours available. In this case, the required hours are distributed evenly across the available equipment fleet on site. The smaller support equipment was based on the number of units required for various locations around the mine and is replaced after a certain number of years of usage. For instance, light plants are replaced every 4 years. Replacement times for the equipment are average values from AGP's experience

A rock crusher and screening plant is purchased and will be operational in Year-1. The crusher will be operated by the road crew and generate material for road construction, road maintenance and stemming for blastholes and pre-production construction needs. This plant is estimated to cost $2.3 million.

The timing of equipment purchases by mining area, initial and sustaining, are shown in Table 21-5. Fleet size for the major equipment is in Table 21-6.

21.4.3 Miscellaneous Mine Capital

The cost associated with the engineering office includes in miscellaneous mine capital cost. This includes such items as computers, mining and geology software, survey equipment (GPS, drones and total stations), and associated peripherals.

Pit dewatering pumps and piping is included. The pits are considered dry and costs associated with pit dewatering are relatively small and used to control water accumulation in the deeper pits from storm events.

Construction of the access roads to the leach pad, stockpiles and waste rock dumps is $12.4 million. Based on the mine schedule, the access haul road costs are applied one year prior to when required. The explosive bulk storage pad is estimated to be $2 million.

Total miscellaneous mine capital cost including dewatering infrastructure is estimated at $18.4 million, most of which is related to surface haul road construction.

Table 21-7: Mine Capital Cost Estimate

Mine Capital Category	Initial Cost $	Sustaining Cost $	Total Cost $
Pre-Production Mining	89,291,291	-	89,291,291
Mining Equipment	24,010,600	45,356,400	69,367,000
Miscellaneous Mine Capital	7,400,000	10,409,567	17,809,567
Dewatering Infrastructure	-	632,000	632,000
Mine Electrification	-	-	-
Total Mine Capital	120,701,891	56,397,967	177,099,858

21.5 Process Operating Cost Summary

Process operating costs have been estimated by M3 from the first principles. Labor costs were estimated using project specific staffing, salary and wage, and benefit requirements. Unit consumptions of materials, supplies, power, and delivered supply costs were also estimated. LOM overall average processing costs are estimated at an average cost of $1.80 per ton.

Operating costs were estimated based on 3rd quarter 2024 US dollars and are presented with no added contingency based upon the design and operating criteria present in this Technical Report. Operating costs are considered to have an accuracy of +/- 20%.

The process operating costs presented are based upon the ownership of all process production equipment and site facilities. The owner will employ and direct all operating, maintenance, and support personnel for all site activities.

Operating costs estimates have been based upon information obtained from the following sources:

- Project metallurgical test work and process engineering
- Development of a detailed equipment list and demand calculations
- M3 In-house data for reagent pricing
- Experience with other similar operations

Where specific data do not exist, cost allowances have been based upon consumption and operating requirements from other similar properties for which reliable data exist.

21.5.1 Personnel and Staffing

Staffing requirements for process personnel have been estimated by M3 based on experience with similar-sized operations in the region. Total process personnel requirements are estimated at 70 persons for the ROM operation. Personnel requirements and costs are estimated at $7.9 million per year for the ROM operation.

21.5.2 Power

Power usage for the process and process-facilities was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage and cost. Power requirements for the Project are presented in Table 21-8.

Table 21-8: Power Requirements Summary

	ROM Process		
Area Description	Connecter Power (kW)	Demand (kW)	Annual (kWh)
Area 310 - Heap Leach Pad & Ponds	17	15	133,145
Area 350 - Solution Transfer	6,201	3,614	31,660,744
Area 400 - Adr	233	122	1,068,500
Area 500 - Refinery	213	91	793,104
Area 650 - Water Systems	525	350	3,062,325
Area 800 - Reagents	26	23	200,293
Area 900 - Ancillary Facilities	57	26	224,790
Area 960 - Fuel Station	5	4	38,214
Total	7,278	4,244	37,181,114

Power is available via transmission line connected to the Idaho Power grid; the estimated power cost of $0.04/kWh, based on input from Liberty Gold.

21.5.3 Consumable Items

Operating supplies have been estimated based upon unit costs and consumption rates projected by metallurgical tests. Freight costs are included in all operating supply and reagent estimates. Reagent consumptions have been derived from test work and from design criteria considerations. Other consumable items have been estimated by M3 based on experience with other similar operations. Table 21-5 presents average consumptions for major consumables.

Table 21-9: Process Consumables Average Annual Consumptions

Item	Form	Average Annual Consumption
Sodium Cyanide	Liquid at 30% NaCN by Weight	3,285 tonnes
Lime	Bulk Delivery (20 tonnes)	17,703 tonnes
Antiscalant	Liquid Tote (IBC)	183 tonnes
Carbon	1000 lb Supersacks	91 tonnes
Nitric Acid	Liquid at 57% Acid by weight	456 tonnes
Caustic	Liquid at 50% NaOH by Weight	274 tonnes
Refinery Fluxes	Dry Solid Bags	182 tonnes

Operating costs for consumable items have been distributed based on tonnage and gold production or smelting batches, as appropriate.

21.5.4 Maintenance

Annual maintenance costs have been included for the process facilities. The maintenance costs are estimated from the capital cost of the plant equipment at an allowance of 5% for parts repair or replacement. Maintenance labor is also included. The maintenance labor includes one maintenance supervisor, twelve mechanics, and four electricians. These personnel are included as part of the overall process personnel quantity. An allowance for outside repairs is also included at 10% of the maintenance parts allowance. The total annual maintenance is estimated at $4.864 million.

21.5.5 Supplies and Services

Estimates for supplies and services have been included for items such as lubricants, third-party services for the process plant, safety items, and minor supplies and tools outside of maintenance. The total annual supplies and services is estimated at $765 thousand.

21.6 G&A Costs

G&A costs were included based on benchmarks for similar-sized facilities in the surrounding region.

G&A costs are included at $13.25 million per year for the first sixteen years of operation, which are the full years of active mining and ore stacking on the pad. A G&A cost of $7.95 million is included for Year 17.

21.7 Mine Operating Costs

Mine operating costs are estimated from base principles. Key inputs to the mine costs are fuel and labor. The fuel cost is estimated using current pricing and future oil price projections. A fuel cost of $3.26 /gal ($0.86 /L) is used in this estimate.

Staff cost estimates were based on queries to other operations and recent salary surveys. Shift schedules are 12-hour shifts with a 4 days on/ 4 days off schedule. Some management positions will be on a 4-days on and 3-days off basis. A burden rate of 35% was applied to all rates. The mine staff labor remains consistent for the mine life. Staff positions and salaries are shown in Table 21-10.

Table 21-10: Open Pit Mine Staffing Requirements and Annual Salaries

Staff Position	Employees	Full Load Annual Salary $/a
Mine Maintenance		
Maintenance Superintendent	1	161,585
Maintenance General Foreman	1	137,596
Maintenance Shift Foremen	4	122,746
Maintenance Planner/Contract Admin	2	97,615
Clerk/Secretary	1	70,408
Subtotal	9	
Mine Operations		
Mine Ops/Technical Superintendent	1	180,796
Mine General Foreman	1	189,000
Mine Shift Foreman	4	122,746
Junior Shift Foreman	4	98,654
Trainers	1	98,654
Road Crew/Services Foreman	1	109,038
Clerk/Secretary	1	70,408
Subtotal	13	
Mine Engineering		
Chief Engineer	1	189,000
Senior Engineer	1	133,546
Open Pit Planning Engineer	2	103,327
Geotech Engineer	1	103,327
Blasting Engineer	1	103,327
Blasting/Geotech Technician	1	91,385
Dispatch Technician	2	91,385
Surveyor/Mining Technician	2	91,385
Surveyor/Mine Tech helper	2	75,808
Engineering/ Geology Clerk/Secretary	1	70,408
Subtotal	14	
Geology		
Chief Geologist	1	168,127
Senior Geologist	1	121,396
Grade Control Geologist/Modeler	1	99,485
Sampling/Geology Technician	4	78,923
Subtotal	7	
Total Mine Staff	41	

The number of loader, truck, and support equipment operators is estimated using the projected equipment operating hours. The maximum number of employees is four per unit to match the mine crews. Hourly employee labor force levels in the mine operations and maintenance departments fluctuate with production requirements. The drilling labor force is based on one operator per drill, per crew while operating. A snapshot of the labor makeup for Year 6 is shown in Table 21-11.

Shovel and loader operators are at 28 in year 6 and start to tail off after that. Haul truck drivers peak at 132 in Year 10 and then taper off to the end of the mine life.

Labor costs are based on an owner operated scenario. The mine is responsible for the maintenance of the equipment with its own employees.

Table 21-11: Hourly Manpower Requirements and Annual Salary (Year 6)

Hourly Position	Employees	Full Load Annual Salary $/a
Mine General		
General Equipment Operator	8	83,749
Road/Pump Crew	4	78,019
General Mine Laborer	8	73,143
Trainee	4	58,514
Subtotal	24	
Mine Operations		
Driller	32	83,749
Blaster	2	93,623
Blasters Helper	4	75,581
Loader Operator	8	93,257
Hydraulic Shovel/Excavator Operator	20	93,257
Haul Truck Driver	120	76,800
Dozer Operator	19	83,749
Grader Operator	6	83,749
Transfer Loader Operator	3	83,749
Snowplow/Water Truck	8	75,654
Subtotal	222	
Mine Maintenance		
Heavy Duty Mechanic	59	106,360
Light Duty Mechanic	2	111,958
Welder	33	106,360
Electrician	2	106,360
Apprentice	9	80,799
Tire Maintenance	5	85,334
Lube Truck Driver	4	85,334
Subtotal	114	
Total Mine Hourly	360	

Overseeing all mine operations, engineering, and geology functions is a Mine Operations Superintendent. This person will have the Mine Maintenance Superintendent, Mine General Foremen, and Chief Engineer reporting to them. The Mine Operations Superintendent reports to the Mine General Manager.

The mine has four mine operations crews, each with a Senior Shift Foremen who has one Junior Shift Foreman reporting to him. For the mine life, there is also a Road Crew/Services Foreman responsible for roads, drainage, and pumping around the mine. This person would also be a backup Senior Mine Shift Foreman. The Mine Operations department has its own administrator.

The Mine Maintenance Superintendent has the Maintenance General Foreman reporting to him. Four Mine Maintenance Shift Foremen will report to the Maintenance General Foreman. Also, there are two maintenance planners/mechanical engineers and an administrative assistant.

Maintenance factors are used to determine the number of heavy-duty mechanics, welders and electricians are required and are based on the number of drill operators, shovel operators, truck drivers, etc. Heavy duty mechanics work out to 0.25 mechanics required for each drill operator. Welders are 0.25 per drill operator and electricians are 0.05 per drill operator. This method of estimating maintenance requirements is used for each category of the mine operating cost and is summarized in Table 21-12.

Table 21-12: Maintenance Labor Factors (Maintenance per Operator)

Maintenance Job Class	Drilling	Loading	Hauling	Support
Heavy Duty Mechanic	0.25	0.25	0.25	0.25
Welder	0.25	0.25	0.25	0.25
Electrician	0.05	0.10	-	0.25
Apprentice	-	-	-	0.25

The Chief Engineer reports to the Mine Operations Superintendent and has two Senior Engineer, two Planning Engineers and Geotechnical Engineer as direct reports. The planning engineers will take the short-range planning role with the Senior Engineers coordinating the long range mine plans and assisting with short range planning. The Geotechnical engineer would cover all aspects of the wall slopes and waste dumps and share the surveyors/mine technicians with the short-range team. The surveyors/mine technicians will assist in the field with staking, surveying, and sample collection with the geology group.

In the Geology department, there is one Senior Geologist reporting to the Chief Geologist. The grade control geologist works in short range and grade control drilling and the Senior Geologist will be responsible for long range/reserves. Four grade control field technicians will mark off ore in the field and assist as required with sampling.

Tire costs were also collected from various vendors for the sizes expected to be used. The estimates of the tire life are based on AGP's experience and conversations with mine operators. The operating cost of the tires is expressed in a $/h form. The life of the 144 tonne haulage truck tires is estimated at 6,500 hours per tire with proper rotation from front to back. Each 144 tonne truck tire costs $31,000 so the cost per hour for tires is $28.62/h for the truck using six tires in the calculation.

Table 21-13: Major Equipment Operating Costs – No Labor ($/h)

Maintenance Job Class	Fuel/ Power	Lube/ Oil	Tires/ Under-Carriage	Repair & Maintenance	GET/ Consumables	Total
Primary Drill (165 mm)	97.18	9.72	4.62	73.59	94.12	279.23
Secondary Drill (165 mm)	55.90	5.59	2.31	101.49	73.95	239.24
Hydraulic Shovel (22 m^3)	215.00	21.50	incl	195.28	25.00	456.78
Hydraulic Excav. (6.5 m^3)	68.80	10.32	5.00	58.50	8.00	150.62
Loader (11.5 m^3)	81.70	8.17	33.36	98.25	10.00	231.48
Haul Truck (144 t)	77.40	7.74	26.29	99.00	4.00	214.43
Haul Truck (64 t)	38.70	3.87	10.79	30.68	2.00	86.04
Track Dozer	60.20	6.02	8.00	74.00	4.00	152.22
Grader	18.92	2.70	1.89	14.00	3.00	40.51

Ground Engaging Tool ("GET") costing is estimated from other projects and conversations with personnel at other operations. This is an area of cost that is expected to be fine-tuned during mine operations.

Drilling in the open pit will be performed using conventional down the hole (DTH) blasthole rigs with 165-mm bits. The initial drill platforms and pioneering benches will be developed by the smaller drill. The pattern size for leach material is slightly smaller waste. A fragmentation study was completed to model the drill and blast design parameters for various rock types. The parameters shown in Table 21- fall within the guidance of the fragmentation study.

Table 21-14: Drill Pattern Specification

Specification	Unit	Production/Presplit Drill		Production Drill	
		Ore	Waste	Ore	Waste
Bench Height	m	10	10	10	10
Sub-Drill	m	1.2	1.2	1.2	1.2
Blasthole Diameter	mm	165	165	165	165
Pattern Spacing – Staggered	m	5.50	5.70	5.50	5.70
Pattern Burden – Staggered	m	5.00	5.20	5.00	5.20
Hole Depth	m	11.2	11.2	11.2	11.2

Table 21-15: Drill Productivity Criteria

Drill Activity	Unit	Production/Presplit Drill		Production Drill	
		Ore	Waste	Ore	Waste
Pure Penetration Rate	m/min	0.55	0.55	0.60	0.60
Hole Depth	m	11.2	11.2	11.2	11.2
Drill Time	min	20.4	20.4	18.7	18.7
Move, Spot and Collar Hole	min	3.0	3.0	3.0	3.0
Level Drill	min	0.5	0.5	0.5	0.5
Add Steel	min	0.5	0.5	0.0	0.0
Pull Drill Rods	min	1.5	1.5	1.0	1.0
Total Setup/Breakdown Time	min	5.5	5.5	4.5	4.5
Total Drill Time per Hole	min	25.9	25.9	23.2	23.2
Drill Productivity	m/h	26.0	26.0	29.0	29.0

Drill consumables were estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity for the larger production drill is estimated at 29.0 m/h and the smaller pre-split/production drill is 26.0 m/h. The parameters used to estimate drill productivity are shown in Table 21-15.

Quotations from local explosive vendors were obtained which included delivery to the blasthole. The explosives cost includes monthly fees from the explosive vendor for magazine rental and all costs associated with delivering the product to the open pit and down the hole.

An emulsion product will be used for blasting to provide water protection when required. With the dry conditions expected, it is believed that ANFO will be the primarily explosive which is applied to 95% of all blastholes. The powder factors used in the explosives calculation are shown in Table 21-16.

Table 21-16: Design Powder Factors

Description	Unit	Leach	Waste
Powder Factor	kg/m^3	0.514	0.468
Powder Factor	kg/t	0.197	0.179

Total monthly cost in the service of delivering the explosives down the hole is $40,040 which includes $27,792 for the vendor's ANFO trucks, prill silos, emulsion silo, accessories magazines, shop and office Connex plus $12,248 in labor costs.

Loading costs for both leach feed and waste are based on the use of hydraulic shovels and front-end loaders. The large shovels will be the primary digger with the front-end loaders and smaller shovels used in the small pits, tight mining areas and as a backup/support unit. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-17. This highlights the focus of the larger more efficient shovels over the smaller loading units.

Table 21-17: Loading Parameters

Description	Unit	Hydraulic Shovel (22 m³)	Hydraulic Shovel (6.7 m³)	Loader (11.5 m³)
Bucket Capacity	m³	22	6.7	11.5
Waste Tonnage Loaded	%	75	18	7
Leach Ore Tonnage Mined	%	76	17	7
Bucket Fill Factor	%	97	93	95
Cycle Time	sec	38	38	42
Trucks Present at the Loading Unit	%	90	90	90

Haulage profiles were determined for each pit phase to the heap leach pad, stockpile or waste rock destinations. Cycle times were generated for each period by tonnage from each phase and stockpile to the destination and used to estimate the haulage costs. Maximum speed on trucks is limited to 50 km/h for tire life and safety reasons. Haul speed is reduced to 20 km/h when approaching the loading and dump areas. Haulage speeds for the haul trucks over various segments are shown in Table 21-18.

Table 21-18: Haulage Cycle Speeds

Truck Size	Down (10%)	Down (6%)	Down (3%)	Flat (0%)	Up (3%)	Up (6%)	Up (10%)
Haul Truck (144 t)							
Loaded (km/h)	16	25	40	50	26	15	11
Empty (km/h)	25	35	45	50	38	32	22
Haul Truck (64t)							
Loaded (km/h)	25	38	45	50	33	21	12
Empty (km/h)	30	40	45	50	45	35	28

Support equipment hours and costs are determined using the percentages shown in Table 21-19. These percentages resulted in the need for seven track dozers plus one dozer for the leach pad. Their tasks include cleanup of the loader faces, roads, dumps, and blast patterns.

Four graders, and three water trucks are required due to the spread-out nature of the various pit areas and maintenance of the large surface road network. The graders will maintain the leach feed and waste haul routes. In addition, water trucks have the responsibility for patrolling the haul roads and controlling fugitive dust for safety and environmental reasons. A small backhoe will be responsible for cleaning out sedimentation ponds and water ditch repairs together with the two small dump trucks.

These hours are applied to the individual operating costs for each piece of equipment. Many of these units are support equipment so no direct labor force is allocated to them due to their function.

Table 21-19: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track Dozer	25%	Of haulage hours to a maximum of 6 dozers
Grader	15%	Of haulage hours to a maximum of 3 graders
Auxiliary Loader	5%	Of loading hours to maximum of 1 loader
Support Backhoe	10%	Of loading hours to maximum of 2 backhoe
Water Truck	10%	Of haulage hours to a maximum of 2 trucks
Lube/Fuel Truck	6	h/d
Mechanic's Truck	14	h/d
Welding Truck	8	h/d
Blasting Loader	8	h/d
Blaster's Truck	8	h/d
Integrated Tool Carrier	4	h/d
Compactor	2	h/d
Lighting Plants	12	h/d
Pickup Trucks	10	h/d
Dump Truck – 20 ton	6	h/d

Financing of the mine fleet is used to reduce initial capital with the downpayment allocated to capital and the remainder of the financing cost allocated as an operating cost.

The major mine equipment, and a large majority of the support equipment where it was considered reasonable, are financed. If the equipment has a life greater than the finance period, then the following years onwards of the equipment does not have a finance payment applied. In the case of the mine trucks, with an approximate 10 year working life, the finance period would be complete in 5 years and the trucks would simply incur operating costs after that time. For this reason, the operating cost can vary annually depending on the equipment replacement schedule and timing of the financing.

Utilizing the financing option adds $ 0.40/t to the mine operating cost over the life of the mine. On a cost per tonne of leach ore basis, it is $ 0.93/t.

The total life of mine operating costs per tonne of material moved and per tonne of leach ore processed are shown in Table 21-20 and Table 21-21.

Table 21-20: Open Pit Mine Operating Costs – with Finance Cost ($/t Total Material)

Open Pit Operating Category		Year1	Year 5	Year 10	LOM
General Mine and Engineering	$/t moved	0.18	0.17	0.25	0.22
Drilling	$/t moved	0.20	0.20	0.20	0.20
Blasting	$/t moved	0.24	0.25	0.23	0.24
Loading	$/t moved	0.32	0.27	0.36	0.30
Hauling	$/t moved	1.07	1.15	1.49	1.21
Support	$/t moved	0.29	0.26	0.39	0.33
Grade Control	$/t moved	0.02	0.03	0.01	0.02
Finance Costs	$/t moved	0.69	0.29	0.25	0.40
Dewatering	$/t moved				
Total	$/t moved	3.01	2.62	3.19	2.93

Table 21-21: Open Pit Mine Operating Costs – with Finance Cost ($/t Leach Ore)

Open Pit Operating Category		Year1	Year 5	Year 10	LOM
General Mine and Engineering	$/t leach ore	0.50	0.50	0.48	0.50
Drilling	$/t leach ore	0.60	0.60	0.39	0.46
Blasting	$/t leach ore	0.75	0.75	0.44	0.56
Loading	$/t leach ore	0.80	0.80	0.69	0.70
Hauling	$/t leach ore	3.41	3.41	2.86	2.81
Support	$/t leach ore	0.77	0.77	0.75	0.77
Grade Control	$/t leach ore	0.07	0.07	0.03	0.05
Finance Costs	$/t leach ore	0.86	0.90	0.47	0.93
Dewatering	$/t leach ore				
Total	$/t leach ore	7.96	7.80	6.12	6.80

22 ECONOMIC ANALYSIS

The economic analysis in this study includes a pre-feasibility study-compliant modeling of the annual cash flows based on projected production volume, sales revenue, initial capital, operating cost, and sustaining capital with resulting evaluation of key economic indicators such as internal rate of return ("IRR"), net present value ("NPV"), and payback period (time in years to recapture the initial capital investment) for the Project. The sales revenue is based on the production of gold in doré bullion. The estimates of the capital expenditures and site production costs have been developed specifically for this Project and have been presented in Section 21 of this Technical Report.

22.1 Mining Physicals

The cash-flow model uses the mining and production schedules as discussed in Section 16 and summarized in Table 22-1. Results from the heap leach metal production model are included with this table to facilitate direct comparison between placed ounces, recoverable ounces, and recovered ounces. Placed ounces are per the mine plan and stacking plan. Recoverable ounces are placed ounces that are amenable to a positive response to leaching after cyanide-bearing solution has started being applied. Recovered ounces incorporate time-based constraints for the time it takes leached ounces to reach the pad liner and report to the metal recovery plant. Ore is placed on the pad during a sixteen-year period. Solution application continues for an additional 1 year to allow recovery of the solubilized ounces.

22.2 Process Plant Production Statistics

Ore will be processed by cyanide heap leaching as ROM and recovered via an ADR facility as described in Section 17 of this Technical Report. Overall production over the life-of-mine is summarized in Table 22-2.

.

Table 22-1: Yearly Mine & Process Physicals

Material Mined	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Total
Total Ore	K Tonnes	-	18,198	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	7,416	299,363
	Au gpt	-	0.367	0.552	0.331	0.454	0.547	0.346	0.345	0.324	0.199	0.185	0.190	0.247	0.341	0.193	0.476	0.148	0.140	0.323
	K oz Au	-	215	324	194	266	321	203	202	190	117	109	112	145	200	113	279	87	33	3,110
Total Waste	K Tonnes	13,862	28,233	29,122	33,581	32,557	28,339	32,546	33,425	26,451	28,727	25,585	29,107	15,033	12,917	15,928	8,906	-	-	394,320
Total Mined	K Tonnes	13,862	46,431	47,372	51,831	50,807	46,589	50,796	51,675	44,701	46,977	43,835	47,357	33,283	31,167	34,178	27,156	18,250	7,416	693,684
Strip Ratio	W : O		1.55	1.60	1.84	1.78	1.55	1.78	1.83	1.45	1.57	1.40	1.59	0.82	0.71	0.87	0.49	-	-	1.32
Total Ore Processed	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr` 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Total
Total Ore Processed	K Tonnes	-	18,198	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	299,363
	Au gpt	-	0.367	0.552	0.331	0.454	0.547	0.346	0.345	0.324	0.199	0.185	0.190	0.247	0.341	0.193	0.476	0.148	0.140	0.323
Total Placed	K oz Au	-	215	324	194	266	321	203	202	190	117	109	112	145	200	113	279	87	33	3,110
Total Recovered	K oz Au	-	140	226	130	187	231	131	131	133	66	75	80	109	158	82	228	61	23	2,191
Recovery	% Au	-	65.3%	69.6%	67.2%	70.3%	71.9%	64.8%	64.8%	70.1%	56.6%	68.7%	71.8%	75.1%	78.9%	72.2%	81.7%	69.9%	68.7%	70.4%

Table 22-2: Life of Mine Process Statistics

Total Ore (kt)	299,363
Gold Grade (gpt)	0.323
Contained Gold (kozs)	3,110
Gold Recovery %	70.4%
Recovered Gold (kozs)	2,191

22.3 Smelter Return Factors

No contractual payable metal rates have yet been negotiated with smelters. M3 used typical rates based on industry experience or published guidelines. Payable rates for metals used were 99.9% for gold. A bullion refining, transportation and insurance charge of $10 per troy ounce of gold was applied.

22.4 Capital Expenditure

The capital expenditure schedule for the life-of-mine is shown in Table 22-3 below.

Table 22-3: Capital Expenditure Schedule

Capital Expenditure, $000	Initial	Sustaining																
	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Mine Pre-Production	$89,291																	
Mine Capital	$31,411	$20,965	$2,211	$3,843	$3,212	$2,757	$3,208	$2,081	$4,243	$1,004	$660	$1,697	$5,316	$5,202				
Process	$196,658		$70,039					$43,453					$39,240					
Owner's Cost	$9,200	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625
Total	$326,560	$21,590	$72,875	$4,468	$3,837	$3,382	$3,833	$46,159	$4,868	$1,629	$1,285	$2,322	$45,181	$5,827	$625	$625	$625	$625

22.5 Revenue

Annual revenue is determined by applying metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life-of-mine production without escalation or hedging. Gold bullion revenue is based on the gross value of the payable metals sold before refining and transportation charges. Gold metal pricing is based on the long-term consensus price estimates of several metals price analysts; the base case gold price utilized in the economic assessment is $2,000 per troy ounce.

22.6 Total Production Cost

The total production cost includes mine operations, process plant operations, general administration, reclamation and closure, and government fees. Table 22-4 shows the estimated operating costs by area based on payable metals for the life of mine.

Table 22-4: LOM Operating Costs

LOM Operating Cost ($000)	
Mining	$1,945,536
Process Plant	$538,322
G&A	$219,950
Refining	$21,908
Total Operating Cost	$2,725,716
Royalty	$10,888
Salvage Value	$0
Reclamation/Closure	$54,439
Total Production Cost	$2,791,043

22.7 Depreciation

The depreciation cost was calculated using a 5-year straight line depreciation for project initial and sustaining capital and a 10-year straight line depreciation for development costs, which consist primarily of the heap leach expansion costs.

22.8 Royalties

Liberty Gold intends to buy down certain existing royalties prior to production. The royalty value in Table 22-4 reflects the expected net royalty amounts.

22.9 Government Fees

No government fees have been applied to the financial model.

22.10 Taxation

An Idaho Mine License Tax of 1% is applied to the taxable amount of total revenue minus total operating costs minus depreciation.

An Idaho Corporate Income Tax of 5.695% is applied to the taxable amount of net income minus loss carryforward.

Federal corporate tax of 21% is applied to taxable corporation income after adjustments for state tax, if any, and Idaho Mine License Tax.

22.11 Net Income After-Tax

The net income after-taxes is projected to be $871 million.

22.12 Project Financing

It is assumed that the Project will be all equity financed.

22.13 Economic Indicators

The economic analyses for the Project are summarized in Table 22-5 below. The NPV calculations have been conducted per the Year-End discounting method.

Table 22-5: Key Economic Results

Indicators	Before-Tax	After-Tax
LOM Cash Flow ($000)	$1,039,763	$871,018
NPV @ 5% ($000)	$655,883	$550,207
NPV @ 10% ($000)	$423,897	$352,440
IRR	34.5%	31.8%
Payback (years)	3.1	3.3

22.14 Sensitivity Analysis

Table 22-6 below shows the sensitivity analysis of the key economic indicators (cash flow, NPV, IRR, and payback) to changes in gold prices.

Table 22-6: Sensitivity Analysis

Financial Indicators	Base Case +$1500	Base Case +$1350	Base Case +$1200	Base Case +$1050	Base Case +$900	Base Case +$750	Spot Case (Base Case +$600)	Base Case +$450	Base Case +$300	Base Case +$150	Base Case	Base Case -$150	Base Case -$300
Gold Price (per troy oz)	$3,500	$3,350	$3,200	$3,050	$2,900	$2,750	$2,600	$2,450	$2,300	$2,150	$2,000	$1,850	$1,700
Pre-tax Cash Flow, $M	$4,314.4	$3,986.9	$3,659.5	$3,332.0	$3,004.5	$2,677.1	$2,349.6	$2,022.2	$1,694.7	$1,367.2	$1,039.8	$712.3	$384.8
Pre-tax Net Present Value (5%) in $M	$2,947.8	$2,718.6	$2,489.4	$2,260.2	$2,031.0	$1,801.8	$1,572.6	$1,343.4	$1,114.3	$885.1	$655.9	$426.7	$197.5
Pre-tax Internal Rate of Return (IRR)	112.5%	105.0%	97.5%	90.0%	82.4%	74.7%	67.0%	59.2%	51.2%	43.0%	34.5%	25.5%	15.5%
Pre-tax Payback (Years)	1.1	1.1	1.2	1.3	1.3	1.4	1.5	1.7	1.8	2.2	3.1	3.7	4.3
After-tax Cash Flow, $M	$3,469.7	$3,214.1	$2,957.5	$2,698.9	$2,439.4	$2,179.9	$1,919.0	$1,656.2	$1,394.2	$1,133.9	$871.0	$605.4	$342.1
After-tax Net Present Value (5%) in $M	$2,392.4	$2,211.3	$2,029.6	$1,846.5	$1,662.7	$1,479.0	$1,294.3	$1,108.1	$922.2	$737.2	$550.2	$360.6	$172.0
After-tax Internal Rate of Return (IRR)	104.3%	97.3%	90.3%	83.3%	76.2%	69.1%	62.0%	54.6%	47.2%	39.6%	31.8%	23.6%	14.6%
After-tax Payback (Years)	1.1	1.1	1.2	1.3	1.3	1.4	1.5	1.7	1.9	2.5	3.3	3.8	4.3

22.15 Detailed Financial Model

The detailed financial model, shown in Table 22-7 below, was developed in compliance with the PFS requirement. This model has captured all the parameters of the mine production volume, annual sales revenue, and all the associated costs. This model was used to calculate the economics of the Project, as well as for the sensitivity analysis.

Table 22-7: Detailed Financial Model

Liberty Gold - Black Pine Project - Financial Model	ROM																					
M3-PN230328	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18
Mine																						
Ore (tonnes)	299,363,496			-	18,197,694	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	7,415,803	-
Gold (gpt)	0.323			-	0.367	0.552	0.331	0.454	0.547	0.346	0.345	0.324	0.199	0.185	0.190	0.247	0.341	0.193	0.476	0.148	0.140	-
Silver (gpt)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	3,110			-	215	324	194	166	321	203	202	190	117	109	112	145	200	113	279	87	33	-
Contained Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste (tonnes)	394,320,449			13,862,211	28,232,871	29,122,422	33,581,369	32,556,932	28,339,111	32,546,488	33,424,730	26,451,116	28,726,848	25,585,274	29,106,698	15,033,208	12,917,080	15,928,339	8,905,751	-	-	-
Total Material Mined (tonnes)	693,683,946			13,862,211	46,430,564	47,372,422	51,831,369	50,806,932	46,589,111	50,796,488	51,674,730	44,701,116	46,976,848	43,835,274	47,356,698	33,283,208	31,167,080	34,178,339	27,155,751	18,250,000	7,415,803	-
Process Plant																						
ROM Processing																						
Met_1 (tonnes)	247,898,673			-	12,600,796	10,828,091	15,277,628	13,937,086	15,543,516	11,191,219	12,367,132	17,154,479	8,448,411	15,817,069	17,948,552	17,898,488	18,046,204	17,977,900	18,097,481	17,885,654	6,878,966	-
Gold (gpt)	0.325			-	0.400	0.666	0.333	0.464	0.555	0.372	0.346	0.306	0.208	0.184	0.191	0.248	0.341	0.193	0.476	0.148	0.141	-
Silver (gpt)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	2,588			-	162	232	163	208	277	134	138	169	57	94	110	143	198	112	277	85	31	-
Contained Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Recovery %	74.2%				70.1%	76.8%	69.9%	74.7%	74.8%	71.0%	71.0%	72.1%	64.7%	71.8%	72.1%	75.5%	79.2%	72.5%	81.9%	70.3%	69.9%	0.0%
Silver Recovery %	0.0%				0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recoverable Gold (kozs)	1,920			-	114	178	114	155	207	95	98	122	37	67	79	108	157	81	227	60	22	-
Recoverable Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Met_2&3 (tonnes)	51,464,824			-	5,596,897	7,421,909	2,972,372	4,312,914	2,706,484	7,058,781	5,882,868	1,095,521	9,801,589	2,432,931	301,448	351,512	203,796	272,100	152,519	364,346	536,837	-
Gold (gpt)	0.315			-	0.294	0.386	0.321	0.419	0.499	0.305	0.342	0.604	0.192	0.192	0.160	0.189	0.306	0.205	0.478	0.154	0.130	-
Silver (gpt)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	522			-	53	92	31	58	43	69	65	21	60	15	2	2	2	2	2	2	2	-
Contained Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Recovery %	51.9%				50.5%	51.6%	53.2%	54.3%	53.4%	52.8%	51.4%	53.9%	49.1%	48.9%	53.1%	50.1%	50.1%	52.8%	54.7%	52.6%	51.1%	0.0%
Silver Recovery %	0.0%				0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recoverable Gold (kozs)	271			-	27	48	16	32	23	37	33	11	30	7	1	1	1	1	1	1	1	-
Recoverable Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total ROM (tonnes)	299,363,496			-	18,197,694	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	7,415,803	-
Gold (gpt)	0.323			-	0.367	0.552	0.331	0.454	0.547	0.346	0.345	0.324	0.199	0.185	0.190	0.247	0.341	0.193	0.476	0.148	0.140	-
Silver (gpt)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	3,110			-	215	324	194	266	321	203	202	190	117	109	112	145	200	113	279	87	33	-
Contained Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Recovery %	70.4%																					
Silver Recovery %	0.0%																					
Recoverable Gold (kozs)	2,191			-	140	226	130	187	231	131	131	133	66	75	80	109	158	82	228	61	23	-
Recoverable Silver (kozs)	-			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Processing																						
Total Ore (tonnes)	299,363,496			-	18,197,694	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	7,415,803	-
Gold (gpt)	0.323				0.367	0.552	0.331	0.454	0.547	0.346	0.345	0.324	0.199	0.185	0.190	0.247	0.341	0.193	0.476	0.148	0.140	-
Silver (gpt)	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Liberty Gold - Black Pine Project - Financial Model	ROM																					
M3-PN230328	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18
Contained Gold (kozs)	3,110				215	324	194	266	321	203	202	190	117	109	112	145	200	113	279	87	33	-
Contained Silver (kozs)	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Recovery %	70.4%				65.3%	69.6%	67.2%	70.3%	71.9%	64.8%	64.8%	70.1%	56.6%	68.7%	71.8%	75.1%	78.9%	72.2%	81.7%	69.9%	68.7%	
Silver Recovery %	0.0%																					
Recovered Gold (kozs)	2,191				140	226	130	187	231	131	131	133	66	75	80	109	158	82	228	61	23	-
Recovered Silver (kozs)	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Metals																						
Gold (kozs)	2,189				140	225	130	187	230	131	131	133	66	75	80	109	158	82	228	61	23	-
Silver (kozs)	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Metal Prices																						
Gold ($/oz)	$2,000.00				$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$0.00
Silver ($/oz)	-				$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$25.00	$0.00
Revenues ($000)																						
Gold	$4,377,121				$280,201	$450,728	$260,675	$373,576	$460,896	$262,658	$261,578	$266,301	$132,420	$149,009	$160,006	$217,573	$315,215	$163,541	$455,612	$121,243	$45,889	$0
Silver	$0				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Revenues	$4,377,121				$280,201	$450,728	$260,675	$373,576	$460,896	$262,658	$261,578	$266,301	$132,420	$149,009	$160,006	$217,573	$315,215	$163,541	$455,612	$121,243	$45,889	$0
Operating Cost ($000)																						
Mining	$1,943,403				$144,898	$145,348	$160,306	$154,377	$141,815	$128,818	$130,084	$121,192	$101,893	$111,635	$116,030	$104,154	$112,240	$97,053	$103,577	$49,921	$20,063	$0
Process Plant	$538,322				$32,606	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$15,770	$0
G&A	$219,950				$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$7,950	$0
Refining	$21,908				$1,402	$2,256	$1,305	$1,870	$2,307	$1,315	$1,309	$1,333	$663	$746	$801	$1,089	$1,578	$819	$2,280	$607	$230	$0
Total Operating Cost	$2,723,583				$192,157	$193,517	$207,524	$202,160	$190,034	$176,046	$177,306	$168,438	$148,469	$158,294	$162,743	$151,156	$159,731	$143,784	$151,771	$96,441	$44,013	$0
Royalty	$10,888				$697	$1,121	$648	$929	$1,146	$653	$651	$662	$329	$371	$398	$541	$784	$407	$1,133	$302	$114	$0
Salvage Value	$0				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation/Closure	$54,439				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$54,439
Total Production Cost	$2,788,911				$192,854	$194,638	$208,172	$203,089	$191,181	$176,699	$177,957	$169,100	$148,799	$158,665	$163,141	$151,697	$160,515	$144,191	$152,904	$96,742	$44,127	$54,439
Operating Income	$1,588,210				$87,347	$256,091	$52,502	$170,487	$269,715	$85,959	$83,621	$97,201	-$16,378	-$9,656	-$3,136	$65,877	$154,700	$19,350	$302,708	$24,501	$1,762	-$54,439
Depreciation ($000)																						
Total Capital	$545,777				$65,312	$69,630	$77,094	$77,987	$78,754	$14,119	$10,567	$14,994	$15,074	$14,633	$14,213	$13,911	$11,478	$11,670	$11,469	$11,337	$10,998	$5,589
Total Depreciation	$545,777				$65,312	$69,630	$77,094	$77,987	$78,754	$14,119	$10,567	$14,994	$15,074	$14,633	$14,213	$13,911	$11,478	$11,670	$11,469	$11,337	$10,998	$5,589
Net Income after Depreciation	$1,042,433				$22,035	$186,461	-$24,591	$92,500	$190,961	$71,840	$73,053	$82,207	-$31,452	-$24,289	-$17,349	$51,965	$143,222	$7,680	$291,238	$13,163	-$9,236	-$60,029
Taxes ($000)																						
Idaho Mine License Tax	$12,354				$227	$1,876	$0	$934	$1,921	$725	$737	$829	$0	$0	$0	$525	$1,440	$81	$2,924	$135	$0	$0
Idaho Corporate Income Tax	$35,653				$128	$4,836	$0	$1,270	$6,894	$2,044	$2,078	$2,397	$0	$0	$0	$296	$2,511	$228	$12,592	$380	$0	$0
Federal Corporate Tax	$121,388				$458	$16,364	$0	$4,221	$23,570	$6,954	$7,070	$8,163	$0	$0	$0	$1,057	$8,290	$776	$43,174	$1,292	$0	$0
Total Taxes	$169,395				$813	$23,076	$0	$6,426	$32,385	$9,723	$9,885	$11,389	$0	$0	$0	$1,878	$12,241	$1,085	$58,690	$1,806	$0	$0
Net Income after Taxes ($000)	$873,038				$21,222	$163,385	-$24,591	$86,074	$158,576	$62,117	$63,168	$70,818	-$31,452	-$24,289	-$17,349	$50,087	$130,981	$6,595	$232,549	$11,357	-$9,236	-$60,029
Cash Flow ($000)																						
Net Income before Taxes	$1,042,433			$0	$22,035	$186,461	-$24,591	$92,500	$190,961	$71,840	$73,053	$82,207	-$31,452	-$24,289	-$17,349	$51,965	$143,222	$7,680	$291,238	$13,163	-$9,236	-$60,029
Add back Depreciation	$545,777			$0	$65,312	$69,630	$77,094	$77,987	$78,754	$14,119	$10,567	$14,994	$15,074	$14,633	$14,213	$13,911	$11,478	$11,670	$11,469	$11,337	$10,998	$5,589

Liberty Gold - Black Pine Project - Financial Model	ROM																					
M3-PN230328	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18
Operating Cash Flow	$1,588,210			$0	$87,347	$256,091	$52,502	$170,487	$269,715	$85,959	$83,621	$97,201	-$16,378	-$9,656	-$3,136	$65,877	$154,700	$19,350	$302,708	$24,501	$1,762	-$54,439
Working Capital ($000)																						
Accounts Receivable	$0			$0	-$7,677	-$4,672	$5,207	-$3,093	-$2,392	$5,431	$30	-$129	$3,668	-$454	-$301	-$1,577	-$2,675	$4,155	-$8,002	$9,161	$2,065	$1,257
Accounts Payable	$0			$26,841	-$9,272	$4,283	-$4,427	-$493	-$1,034	-$1,113	$3,582	-$4,123	-$1,908	$779	$451	$2,570	-$2,530	-$1,738	$656	-$4,548	-$4,309	-$3,669
Inventory (parts)	$0			$0																		
Total Working Capital	$0			$26,841	-$16,949	-$389	$780	-$3,586	-$3,426	$4,319	$3,612	-$4,252	$1,760	$325	$150	$993	-$5,205	$2,417	-$7,346	$4,613	-$2,245	-$2,412
Initial Capital Expenditures ($000)																						
Pre-stripping	$89,291			$89,291	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mining	$31,411			$31,411	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process	$196,658			$196,658	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$9,200			$9,200	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Expansion Capital Expenditures ($000)																						
Mining	$0				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process	$0				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$0				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital Expenditures ($000)																						
Mining	$55,860				$20,965	$1,673	$3,843	$3,212	$2,757	$3,208	$2,081	$4,243	$1,004	$660	$1,697	$5,316	$5,202	$0	$0	$0	$0	$0
Process	$152,732				$0	$70,039	$0	$0	$0	$0	$43,453	$0	$0	$0	$0	$39,240	$0	$0	$0	$0	$0	$0
Owner's Cost	$10,625				$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$0
Total Capital	$545,777			$326,560	$21,590	$72,337	$4,468	$3,837	$3,382	$3,833	$46,159	$4,868	$1,629	$1,285	$2,322	$45,181	$5,827	$625	$625	$625	$625	$0
Cash Flow before Taxes ($000)	$1,042,433			-$299,719	$48,808	$183,364	$48,814	$163,064	$262,907	$86,445	$41,074	$88,081	-$16,246	-$10,616	-$5,308	$21,688	$143,668	$21,142	$294,737	$28,489	-$1,108	-$56,851
Cumulative Cash Flow before Taxes ($000)				-$299,719	-$250,911	-$67,547	-$18,733	$144,331	$407,238	$493,683	$534,757	$622,838	$606,591	$595,975	$590,667	$612,355	$756,024	$777,166	$1,071,903	$1,100,392	$1,099,284	$1,042,433
Cash Taxes Payable	$169,395			$0	$0	$813	$23,076	$0	$6,426	$32,385	$9,723	$9,885	$11,389	$0	$0	$0	$1,878	$12,241	$1,085	$58,690	$1,806	$0
Cash Flow after Taxes ($000)	$873,038			-$299,719	$48,808	$182,551	$25,739	$163,064	$256,482	$54,060	$31,351	$78,196	-$27,635	-$10,616	-$5,308	$21,688	$141,790	$8,902	$293,652	-$30,201	-$2,914	-$56,851
Cumulative Cash Flow after Taxes ($000)				-$299,719	-$250,911	-$68,360	-$42,622	$120,442	$376,924	$430,984	$462,335	$540,531	$512,896	$502,279	$496,971	$518,660	$660,450	$669,351	$963,004	$932,803	$929,889	$873,038
Discounted Cash Flow Parameters																						
Year (Year-End Discounting)				0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Factor @ 0%	0.0%			1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Factor @ 5%	5.0%			1.000	0.952	0.907	0.864	0.823	0.784	0.746	0.711	0.677	0.645	0.614	0.585	0.557	0.530	0.505	0.481	0.458	0.436	0.416
Factor @ 10%	10.0%			1.000	0.909	0.826	0.751	0.683	0.621	0.564	0.513	0.467	0.424	0.386	0.350	0.319	0.290	0.263	0.239	0.218	0.198	0.180
Discounted Cash Flow before Taxes ($000)																						
DCF @ 0%	$1,042,433			-$299,719	-$250,911	-$67,547	-$18,733	$144,331	$407,238	$493,683	$534,757	$622,838	$606,591	$595,975	$590,667	$612,355	$756,024	$777,166	$1,071,903	$1,100,392	$1,099,284	$1,042,433
DCF @ 5%	$658,280			-$299,719	-$253,236	-$86,919	-$44,751	$89,402	$295,397	$359,903	$389,094	$448,710	$438,238	$431,720	$428,616	$440,693	$516,884	$527,562	$669,336	$682,387	$681,903	$658,280
DCF @ 10%	$426,014			-$299,719	-$255,348	-$103,808	-$67,133	$44,242	$207,487	$256,283	$277,360	$318,450	$311,560	$307,467	$305,607	$312,517	$354,133	$359,700	$430,258	$436,458	$436,239	$426,014
Discounted Cash Flow after Taxes ($000)																						
DCF @ 0%	$873,038			-$299,719	-$250,911	-$68,360	-$42,622	$120,442	$376,924	$430,984	$462,335	$540,531	$512,896	$502,279	$496,971	$518,660	$660,450	$669,351	$963,004	$932,803	$929,889	$873,038
DCF @ 5%	$552,093			-$299,719	-$253,236	-$87,656	-$65,422	$68,731	$269,691	$310,031	$332,312	$385,238	$367,424	$360,907	$357,803	$369,880	$445,074	$449,570	$590,822	$576,987	$575,715	$552,093
DCF @ 10%	$354,147			-$299,719	-$255,348	-$104,480	-$85,142	$26,233	$185,488	$216,003	$232,091	$268,570	$256,850	$252,757	$250,897	$257,807	$298,879	$301,223	$371,521	$364,948	$364,372	$354,147
Financial Indicators before Taxes ($000)																						
NPV @ 0%	$1,042,433																					
NPV @ 5%	$658,280																					
NPV @ 10%	$426,014																					
IRR	34.6%																					

Liberty Gold - Black Pine Project - Financial Model	ROM																					
M3-PN230328	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18
Payback (years)	3.1				1.0	1.0	1.0	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Financial Indicators after Taxes ($000)																						
NPV @ 0%	$873,038																					
NPV @ 5%	$552,093																					
NPV @ 10%	$354,147																					
IRR	31.9%																					
Payback (years)	3.3				1.0	1.0	1.0	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Au (kozs)	2,189			-	140	225	130	187	230	131	131	133	66	75	80	109	158	82	228	61	23	-
Mining	$1,943,403			$0	$144,898	$145,348	$160,306	$154,377	$141,815	$128,818	$130,084	$121,192	$101,893	$111,635	$116,030	$104,154	$112,240	$97,053	$103,577	$49,921	$20,063	$0
Process Plant	$538,322			$0	$32,606	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$32,663	$15,770	$0
G&A	$219,950			$0	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$13,250	$7,950	$0
Refining	$21,908			$0	$1,402	$2,256	$1,305	$1,870	$2,307	$1,315	$1,309	$1,333	$663	$746	$801	$1,089	$1,578	$819	$2,280	$607	$230	$0
Royalty	$10,888			$0	$697	$1,121	$648	$929	$1,146	$653	$651	$662	$329	$371	$398	$541	$784	$407	$1,133	$302	$114	$0
Cash Cost before By-Product Credit	$2,734,471			$0	$192,854	$194,638	$208,172	$203,089	$191,181	$176,699	$177,957	$169,100	$148,799	$158,665	$163,141	$151,697	$160,515	$144,191	$152,904	$96,742	$44,127	$0
$/Au oz	$1,249			$0	$1,377	$864	$1,597	$1,087	$830	$1,345	$1,361	$1,270	$2,247	$2,130	$2,039	$1,394	$1,018	$1,763	$671	$1,596	$1,923	$0
Silver Credit	$0			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Cash Cost after By-Product Credit	$2,734,471			$0	$192,854	$194,638	$208,172	$203,089	$191,181	$176,699	$177,957	$169,100	$148,799	$158,665	$163,141	$151,697	$160,515	$144,191	$152,904	$96,742	$44,127	$0
$/Au oz	$1,249			$0	$1,377	$864	$1,597	$1,087	$830	$1,345	$1,361	$1,270	$2,247	$2,130	$2,039	$1,394	$1,018	$1,763	$671	$1,596	$1,923	$0
Sustaining Capital Expenditures																						
Mining	$55,860			$0	$20,965	$1,673	$3,843	$3,212	$2,757	$3,208	$2,081	$4,243	$1,004	$660	$1,697	$5,316	$5,202	$0	$0	$0	$0	$0
Process	$152,732			$0	$0	$70,039	$0	$0	$0	$0	$43,453	$0	$0	$0	$0	$39,240	$0	$0	$0	$0	$0	$0
Owner's Cost	$10,625			$0	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$625	$0
Salvage Value	$0			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation/Closure	$54,439			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$54,439
Idaho Mine License Tax	$12,354			$0	$227	$1,876	$0	$934	$1,921	$725	$737	$829	$0	$0	$0	$525	$1,440	$81	$2,924	$135	$0	$0
AISC	$3,020,482			$0	$214,672	$268,851	$212,640	$207,860	$196,484	$181,257	$224,853	$174,797	$150,427	$159,950	$165,463	$197,403	$167,781	$144,897	$156,453	$97,502	$44,752	$54,439
$/Au oz	$1,380			$0	$1,532	$1,193	$1,631	$1,113	$853	$1,380	$1,719	$1,313	$2,272	$2,147	$2,068	$1,815	$1,065	$1,772	$687	$1,608	$1,950	$0
Total Capital Expenditures ($000)																						
Pre-stripping	$89,291																					
Mining	$87,271																					
Process	$349,390																					
Owner's Cost	$19,825																					
Total	$545,777																					
Pre-Strip & Mining	$176,562																					
Process & Owner's Cost	$369,215																					

23 ADJACENT PROPERTIES

There are no adjacent properties to report in this section.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Geometallurgy Zones

At Black Pine a large effort has been put into geologic modelling, taking into consideration the large effort put into geo-metallurgical characterization of all ore types, and their corresponding metallurgical response to oxide cyanide solubility ranges, high clay and blocky ore zones, variability bottle roll and column leach testing response and subsequent development of gold recovery modelling. This ongoing effort has revealed significant differences in gold recoverability throughout Black Pine, necessitating the development of 3-D geo-metallurgical models that have currently been incorporated into the mine block model.

The key 3-D geo-metallurgical models are defined as:

1. Gold cyanide solubility zones:
   - MET-1 (AuCN >65%)
   - MET-2 (AuCN >50%, <65%)
   - MET-3 (AuCN >25%, <50%)
2. High clay ore zones that require blending and
3. Blocky ore zones that require blending.

These 3-D geologic models have been developed and incorporated into the mine block model to allow for planning and optimization of mine sequencing, blending, heap stacking and the separation of oxide ore types with differing metallurgical characteristics throughout the mine life.

At the Pre-Feasibility Study stage, there is insufficient time to develop optimal short to medium term mine/process plans, in sufficient detail, to define realistic mine/heap blending and heap stacking plans which will be required for the final Feasibility Study.

The heap leach pad and process plant have been designed to incorporate separation of met ore types, on the pad, and the use of solution collection and stacking options during operations. It is recommended that Black Pine Project planning and engineering teams incorporate these recommendations into the Feasibility Study mine/process planning to insure optimal performance of the Black Pine Heap Leach and gold and silver recovery operations. If not done properly, as recommended, a measurable degradation in gold and silver recoveries might be expected.

24.2 Silver

Historical Pegasus reports estimate 174,200 ounces of silver were produced in the historical mining operations, with approximately 0.48 ounces of silver recovered for every 1.0 ounce of gold (G.Simmons internal Memo, June 19, 2023). A silver resource was not included in the current gold Mineral Resource estimation due to several deficiencies in the data.

The Black Pine database contains over 38,800 historic silver assays analyzed using a 4-acid digestion (AAS) with a Fire assay finish and over 157,000 modern silver assays analyzed by 2-acid aqua-regia digestion (ME-ICP41). Aqua regia is not considered a total digestion for silver, furthermore recovery from metallurgical test work uses a silver head grade from a 4-acid digestion. For these reasons confidence in the actual silver grades across Black Pine is low.

As part of the metallurgical test work for gold, silver was also tested. Currently the Black Pine database has 180 Liberty Gold metallurgical tests. Each of the metallurgical composites is assayed at KCA labs in Reno for silver by 4-acid digestion and additionally a silver head grade is calculated from the tails screen assay.

In order to quantify the difference between the two acid and four acid assay methods, Liberty Gold re-submitted 117 sample pulps previously assayed by two acid ICP to be re-assayed by ALS labs for silver using their method ME-ICP61 (4-acid digestion). These same intervals make up 117 of the 180 metallurgical composites. As expected, Figure 24-1 shows the samples assayed by 4-acid digestion have much better duplication than those assayed by 2-acid aqua regia when compared to the KCA calculated silver head grade.

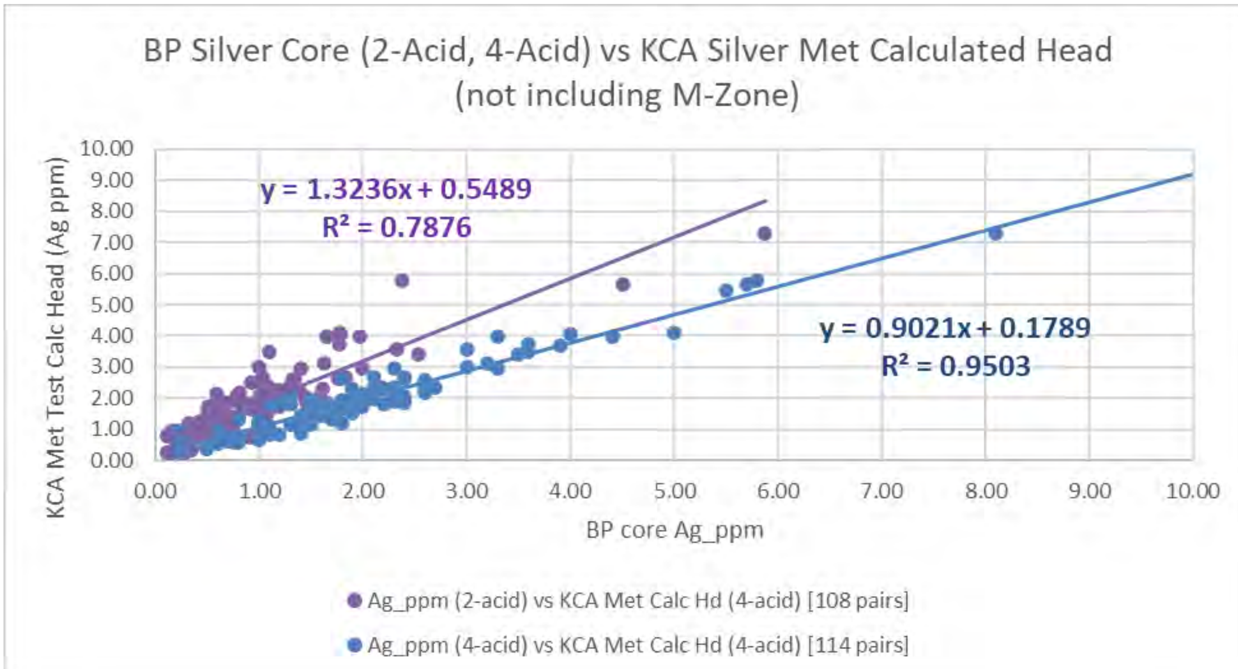


Figure 24-1: Comparison of 2-acid and 4-acid Silver digestions to Calculated Silver Head grades

Figure 24-2 generally shows the underestimation in the 2-acid digestion compared to the 4-acid digestion particularly in the lower grades (<1.0 g/t Ag_ppm).

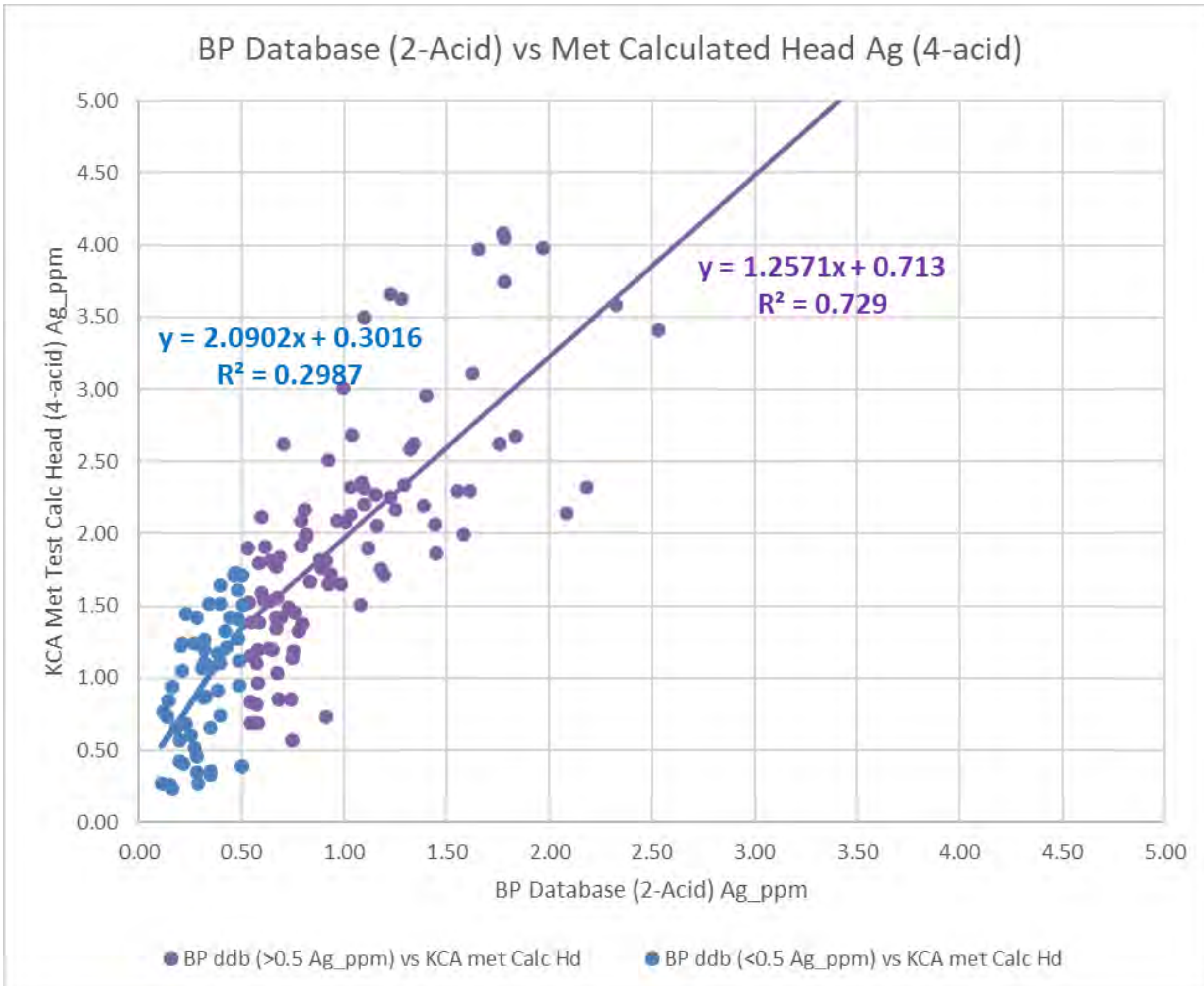


Figure 24-2: Comparison of 2-acid digestion to calculated Silver Head grades

M- Zone and Backrange are currently the only metallurgical composites that have tested elevated silver (>3 ppm Ag). Figure 24-3 would indicate that the 2-acid digestion is more in-line with the 4-acid digestion at higher silver grades. Currently there is insufficient data to determine at what threshold 2-acid and 4-acid are aligned or if the M-zone and Backrange assays were an anomaly.

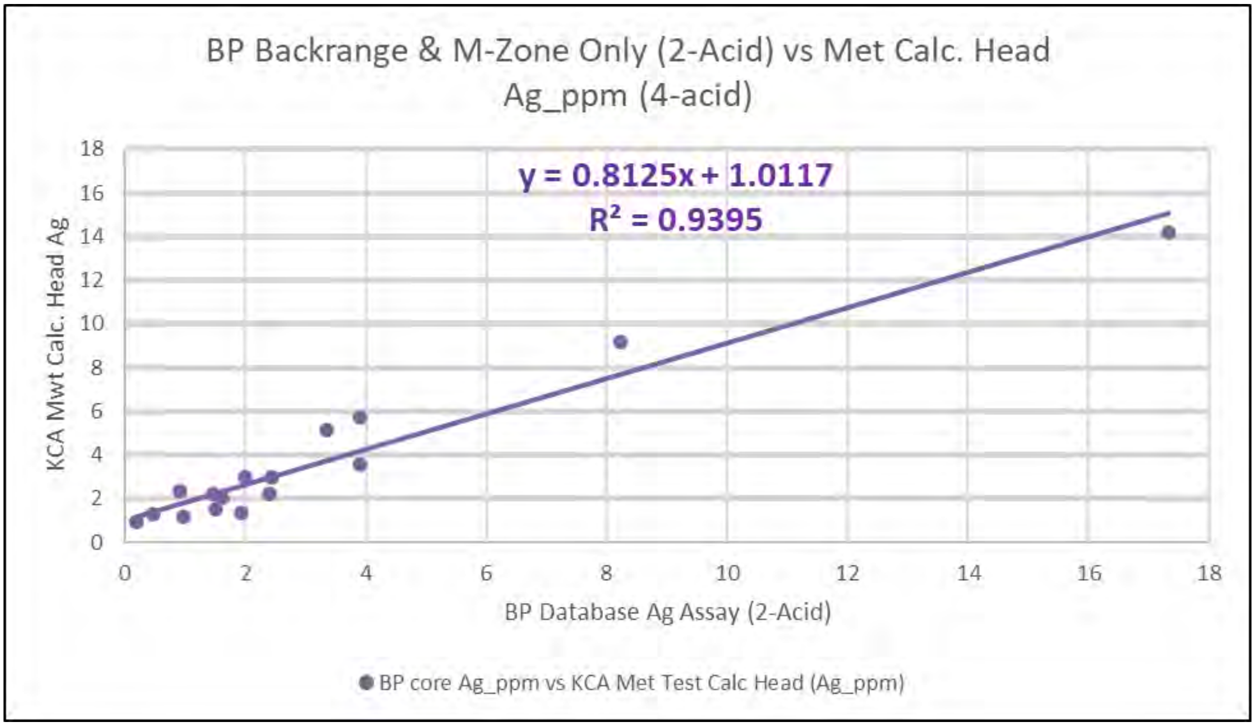


Figure 24-3: Comparison of 2-acid digestion to Calculated Silver Head grades at M Zone and Backrange

Table 24-1 shows the recovery equations for silver from 177 metallurgical composites testing all zones across the Project with M Zone and Backrange having their own recovery equations based on higher silver grades and better recoveries.

Table 24-1: Black Pine – ROM Silver Recovery Equations (source Section 13)

Geo-met Recovery Zone	P_{80}	Silver Recovery, %	Range
Met-1,2,3: All Zones, Except M-Zone and BR	All P_{80}'s	=5.503*ln(HG) + 6.698	Au HG < 3.0 g/t
		=3.696*ln(HG) + 8.536	Au HG > 3.0 g/t
Met-1,2,3: M-Zone	All P_{80}'s	=5.733*ln(HG) + 11.371	Au HG < 3.0 g/t
		=3.863*ln(HG) + 13.283	Au HG > 3.0 g/t
Met-1,2,3: Backrange	All P_{80}'s	=2.574*ln(HG) + 35.714	Au HG < 3.0 g/t
		=1.076*ln(HG) + 37.193	Au HG > 3.0 g/t

Historic mining in the 1990s did recover significant silver as a byproduct to the gold operation. The silver grades were generally higher in those historic pits than what is currently in the ground with the exception of M-Zone and Backrange. Adding a similar estimation class for silver to the current gold resource estimation will require extensive additional work.

24.3 Potential Impact of Inferred Mineral Resources

Inferred Mineral Resources do not have sufficient geological confidence to apply economic considerations to them and there is no guarantee that Inferred Mineral Resources will convert to Indicated or Measured Mineral Resources with additional exploration work, thus all Inferred Mineral Resources have been treated as waste material and excluded from the economic analysis presented herein. However, there is sufficient geological confidence to suggest that much of the Inferred Mineral Resource should be able to be converted to Indicated or Inferred with additional infill drilling.

Table 24-2 below detailed the quantities of Inferred Mineral Resources which are mined and treated as waste in the production schedule presented in Section 15. A total of 41.1M tonnes of Inferred Mineral Resources at 0.154 g/t Au (diluted grade) and containing 136,000 potentially recoverable ounces of gold are mined over 15 years.

Table 24-2: Inferred Mineral Resources Mined by Year

Period	Indicated			Inferred		
	Tonnes	Diluted Au Grade	Recovered Au Grade	Tonnes	Diluted Au Grade	Recovered Au Grade
pp-1	13,137,789	0.272	0.170	488,484	0.184	0.112
y1	19,830,873	0.313	0.200	790,001	0.200	0.113
y2	20,877,578	0.499	0.344	1,018,015	0.235	0.139
y3	21,239,920	0.308	0.205	2,477,405	0.182	0.111
y4	22,081,087	0.408	0.284	2,502,262	0.173	0.110
y5	25,974,336	0.457	0.321	2,555,507	0.231	0.142
y6	19,628,494	0.327	0.212	1,028,593	0.207	0.133
y7	17,629,306	0.341	0.221	2,048,971	0.153	0.099
y8	17,637,078	0.317	0.229	2,201,699	0.134	0.091
y9	1,382,668	0.191	0.138	1,537,060	0.122	0.086
y10	9,414,726	0.170	0.121	2,809,466	0.113	0.078
y11	18,845,886	0.184	0.132	3,747,939	0.120	0.083
y12	24,966,792	0.217	0.161	5,002,227	0.142	0.100
y13	27,082,920	0.279	0.215	5,366,943	0.124	0.085
y14	14,071,661	0.202	0.147	3,270,939	0.150	0.106
y15	25,562,383	0.389	0.312	4,255,296	0.172	0.124
Total	299,363,496	0.323	0.228	41,100,805	0.154	0.103

25 INTERPRETATION AND CONCLUSIONS

The authors of this Technical Report believe that Black Pine is a project of merit and warrants advancing the study to detailed engineering and ultimately project construction.

The authors have reviewed the Project data, including the drill-hole database and available metallurgical information, and have visited the Project site. The authors believe that the data provided by Liberty Gold, as well as the geological interpretations that have been derived from the data, are generally an accurate and reasonable representation of the Black Pine property. Based on the positive results of this PFS, the Project should continue on a path to a production decision.

Results of historical metallurgical tests and those commissioned by Liberty Gold indicate there are multiple metallurgical material types within the various gold deposits. Due to the multiple material types and the dependence of gold recoveries on head grades, numerous different gold ROM recovery equations are used to project the processing and gold production estimates presented in this Technical Report.

The process selected for recovery of gold and silver from the Black Pine mineralized material is a conventional heap-leach recovery circuit. The material will be mined by standard open-pit mining methods and trucked from each deposit to a centralized area of heap-leach pads and processing facilities.

This study indicates an average gold production over the estimated 17-year LOM of about 129,000 ounces per year, with peak production in Year 5 of 231,000 ounces of gold. Cash costs are estimated to be $1,250 per ounce of gold, and AISC are estimated to be $1,381 per ounce of gold. The resulting after-tax cash flow is $871.0 million, for an after-tax NPV (5%) of $550.2 million and an estimated payback period of 3.3 years.

25.1 Geology and Mineral Resources

The Black Pine Project is a sedimentary rock-hosted, Carlin-type gold deposit with a history of mining and exploration activities, including the extraction of 30 million tonnes of ore, production of 434,000 ounces of gold, and the delineation of mineralization through 1,877 drill holes. The project database has undergone rigorous auditing and verification methods, and in the opinion of the Qualified Person (QP), it is adequate for Mineral Resource estimation. Liberty Gold's QA/QC protocols and results meet industry standards, providing confidence in the assays used in the database. Mineral Resource estimates for the project have been prepared using acceptable estimation methodologies, with classifications of Indicated and Inferred Mineral Resources conforming to CIM Standard definitions. The geologic and resource interpretation models for the project are reliable representations of its geology and mineralization, and the Mineral Resource estimation approach, including interpolation design and grade restriction, is considered reasonable.

Total Mineral Resources at the Black Pine Project, above a gold cut-off grade of 0.1 g/t Au, are estimated to include:

- Indicated - 403 Mt grading 0.32 g/t Au, containing 4.16 Moz Au
- Inferred - 98 Mt grading 0.23 g/t Au, containing 0.71 Moz Au

Additionally, there is potential to outline further Mineral Resources with additional exploration drilling programs.

25.2 Metallurgical Test Work

The Black Pine Project is predominantly an oxide deposit with minimal sulfide content and some organic carbon. All carbonaceous and sulfide materials have been geologically identified through 3D modeling and excluded from the run-of-mine (ROM) heap leach metallurgical recovery domains. The oxide resources are well-suited for low-cost ROM conventional heap leaching, demonstrating low sensitivity to feed particle size. Cyanide and lime consumption levels are low, contributing to the cost efficiency of the processing.

The deposits are characterized by low silica and high carbonate content, making them non-acid generating and environmentally favorable for permitting and closure practices. Some blending of clay-rich materials from specific resource areas, such as Pola and Polb zones in the Discovery Zone, is planned at the mining face and heap-leach benches. To ensure optimal gold recovery, Met Types 1, 2, and 3 will be stacked separately on distinct solution collection cells to avoid intermixing, which could degrade recovery performance.

The Black Pine deposits also exhibit low levels of potential cyanide-consuming elements (S=, Cu, Ni, and Zn) and very low levels of other potentially toxic elements such as arsenic, mercury, and selenium, further enhancing the metallurgical and environmental advantages of the project

25.3 Mineral Reserve Estimates

The Mineral Reserve estimation for the Project conforms to industry-accepted practices and is reported using the CIM Standards.

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes to recovery assumptions based on further metallurgical test work and determination of material separation on the heap leach due to different metallurgical properties; effective execution of surface haul road construction to allow access to the pits located at the top of the Black Pine mountains; effective excavation and control of open pit slopes, and maintaining bench advance rate by efficiently dealing with the steep slopes and small operating benches when opening new areas to mining; and management of snow and rain conditions in a mountainous setting. Further factors that may affect the estimate include the ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

25.4 Mining Methods

A PFS level open pit slope design study has been completed and the current geotechnical dataset is considered adequate for the pre-feasibility study level designs.

All pits are considered dry, and dewatering is not required. The ultimate pit bottoms in the later phases of the Rangefront Pit are near the expected groundwater table and will require monitoring for groundwater fluctuations.

Each pit phase was designed to accommodate the proposed mining fleet for that pit. Mining will occur on 10 m benches with catch benches spaced 20 m vertically. Berm widths will vary depending on lithology type and kinematic sector. The haul roads will be 30.2 m or 22.8 m in width depending on the fleet selected and have a road grade of 10%.

The mine schedule for open pit mining consists of 299 Mt of leach feed at a 0.323 g/t mine diluted gold grade over a processing life of 17 years. The open pit waste tonnage totals 394 Mt and will be placed into waste storage areas. The current mine life includes one-year of pre-preproduction followed by 15 years of production mining plus two additional years of stockpile reclaim.

The leach production rate is 18.25 Mt/a starting in Year 1. Ore stockpiling begins during the pre-production year and continues throughout the mine life as low grade and lower recovery leach material is deferred when better material is available. A total of 92 Mt of ore is sent to the stockpiles.

There are three Waste Rock Facilities and two backfill pits for waste rock storage. Four stockpiles are available for lower grade ore and will separately store the leach ore based on gold cyanide solubility.

The mine equipment fleet is anticipated to be leased to lower capital requirements.

25.5 Opportunities

There are numerous opportunities within the Black Pine Project as follows:

- Silver as a by-product
    - In the economic model, silver is not accounted for and as such is not an economic contributor.
    - Although comparably lower value than gold, silver could have a minor positive impact in the economic model as a by-product and would not require additional infrastructure to process.
- Investigating and quantifying the potential gold ounces contained and recoverable from the historical HLF. If exploration work on the historical HLF lead to the estimation of a mineral resource within that facility, that material would potentially have a positive impact on the existing mine plan by providing an readily accessible ore source at the start of the mine life.
- Access to the property is generally in good condition on existing roads. The Project currently carries provision for an additional access and off ramps to be developed from the interstate, however; through further evaluation this additional access may be determined superfluous and as such removed from the Project scope.
- Conversion of mineral resources from inferred to indicated. Section 24 details the Inferred Resources which are included in the current pit designs which total 41.1M tonnes at 0.15gpt Au and 136,000 recoverable Au ounces.
- Discovery of additional resources through exploration and expansion drilling. There are numerous exploration targets within the Project boundaries that Liberty has initiated drill programs on in 2024.
- Evaluation and incorporation of alternative material handling technologies to reduce the haulage operating cost of the project (i.e. electric trucks or conveyor technologies)
- Evaluate the feasibility and economic impacts of adding an additional carbon column line in the ADR plant and increasing the HLF feed rate to 24M tpa. This would reduce the tonnes being placed in stockpile, requiring rehandling and additional operating costs as well as bring potential revenue forward in time with potential benefits to the Project's discounted cash flow.
- Optimize the haulage fleet for downhill loaded hauling. There is a long downhill haul from the open pits to the HLF and selecting the haulage fleet with the best dynamic braking performance and highest downhill speed could improve haulage productivity and improve safety for the operation.
- Evaluate alternate technologies to reduce carbon emissions from the operation such as wind or solar renewable energy, supply of renewable diesel, electrification of the mining fleet, use of electric conveyor systems.
- Investigate the relationship between blast fragmentation and gold recovery and redesign drill and blast patterns to optimize HLF feed particle size distribution.

25.6 Risks

Details surrounding a project of this nature also present risks that need to be taken into consideration. Risks for this Project include the following:

- Social, permitting and geopolitical risks may impact the project's ability to advance to construction and into operations.
- Metal and commodity prices over time may affect the economics of the project. Diesel and electricity price changes may impact the cost basis of the project while global gold prices could impact the project's revenue.
- Future market conditions may impact the ability of the Company to acquire the required financing to make a construction decision.
- Grade control and mining near the ore contacts present a risk of potentially mining too much dilution material or losing high-grade material. Performance of grade control methods and mining techniques should be continually evaluated to manage this risk.

- Delays in surface road construction to the different mining areas will impact the mine plan sequence and result in not being able to achieve production targets.
- Due to the extensive road network in difficult terrain the support fleet will be tasked maintaining a considerable amount of surface and access roads. This is considered an important, but manageable operating risk to meet production targets.
- Mining in the steep terrain when opening new mining areas will be inefficient and could result in reduced mining production, therefore impacting the mine plan sequencing and gold production.
- Failure to properly identify and manage any heap leach feed which contains carbonaceous material, high levels of clay or insufficient fragmentation may reduce gold recovery below the formula stated herein.
- Significant downhill loaded hauls may present a risk of increased maintenance costs, reduced productivity and potential health and safety risk that need to be managed closely.

26 RECOMMENDATIONS

The authors of this technical report believe that Black Pine is a project of merit and warrants significant additional investment with the following recommendations.

26.1 Geology and Mineral Resources

- Complete a drill program consisting of infill Core and RC drilling focused on converting Inferred Mineral Resources to Indicated within and adjacent to the PFS pit shells.
- Complete a sonic drill program on the historic heap leach pad to determine the extent and metallurgical characteristics of the material and to classify any remaining recoverable ounces in a future resource update
- Continue exploration drilling focused on identifying additional resource areas to expand the overall resource base.
- Complete a mid-year 2025 resource update.
- Future exploration drilling is also envisioned across project development, however, not included within the provision of this technical report.

26.2 Mining Methods

Additional study is required in the following areas above the normal mine planning activities for a feasibility study.

- Blast Optimization
    - Fine tune explosives and blast patterns with detailed rock information to reduce consumable costs and achieve the optimum particle size to the heap leach pad
- Ore Blending
    - Establish further definition regarding blending of clay and block ore types. This is to minimize the possibility of solution plugging issues in the pad with too many clays or solution channeling issues in the pad with blocky ores
- Equipment Selection
    - Review equipment selection and fine tune drill size, loading unit size and haul truck selection to optimize maximize carrying capacity and downhill haul speeds to /reduce unit costs.
    - Includes electrification opportunities and regenerative braking.
- Alternative material handling
    - Evaluate alternative material handling technologies to reduce haulage costs including reduction in the surface road construction/ reclamation costs.
- Mine infrastructure improvements
    - Review and study waste storage and stockpile location and access to reduce haulage hours and operating costs. May include additional condemnation drilling and permit modification not included in the cost estimate.
    - Evaluate investment in the haulage road network to improve safety and haulage hours with capital cost trade off against the reduced operating costs.

26.3 Metallurgical Test Work

The following work is needed to progress Black Pine metallurgical development to feasibility level.

1. Complete Phase 5A and 5B variability composite testing at KCA.
2. Determine if there are any gaps in metallurgical core drilling and variability composite testing to a feasibility level.

3. Continue with select metallurgical domain environmental characterization of composite heads and residues to assist project engineering/design and permitting.
4. Update the Met 1, 2 & 3 gold and silver recovery models and equations as new testing data is received from Phase 5A and 5B test programs.

26.4 Legacy HLF Draindown Solution

- Define the lateral extents and the full depth of the existing HLF land application area. This includes confirming the soil that may have been exposed to constituents of potential concern (COPCs) from the legacy HLF draindown solution can be regraded within the new HLF footprint area prior to covering with the new HLF composite liner system.
- Further define draindown flowrates with time, as little data is available on the flowrates. This includes installing flow monitoring devices on the existing system, the using the data collected over a sufficient recording period to resize pipes and incorporate the flows into the water balance model.
- Further define the water chemistry and potential interaction with the Met 1 and Met 2/3 draindown solution. There is a possibility that flows could be more efficiently routed directly onto the new HLF, eliminating the need to transmit flows a much more significant distance to the process facilities area.

26.5 Geotechnical Investigations and Testing

- Further define the quality and quantity of the potential construction material borrow sources (low-permeability soil liner, overliner, drainage aggregate, road wearing course) during the feasibility level field investigation campaign. Laboratory testing to define the quality of the material would also include geochemical sampling on potential construction material borrows.
- Further explore the concept of using a low-permeability soil layer equivalent to that specified in regulatory documents, including evaluating the use of alternate materials such as geosynthetic clay liner (“GCL”) and permeability variability within natural low permeability soil borrow sources. This should also include discussions with regulatory agencies.
- Complete additional investigations such as geophysics, drilling, sampling, and test pitting within the footprint of the HLF and associated ponds, surface water management infrastructure, truck shop, and plant area. The investigations should be developed to further define foundation/subsurface conditions, determine if excavated soils will be a suitable fill source during cut-to-fill pad construction, and define depth to groundwater. This may also include geophysical investigations.
- Complete liner interface testing and liner puncture testing with proposed design materials, including soil borrows (low permeability soil and overliner), geosynthetics, and spent ore.

26.6 HLF

- Refine the assumed material properties for the various metallurgical material types. The HLF is currently designed at a capacity of approximately 331 million dry tonnes (365 million dry tons) at an assumed average in-place dry density of 100 pcf. Additional future work should include laboratory testing to refine the average in-place dry density after leaching and hydraulic conductivity for Met 1, Met 2, and Met 3 materials under the anticipated loading due to the heap.
- Refine the stacking plan to accommodate the distinct metallurgical ore types, including evaluating the transition zone and potential impacts on the solution drainage and recoveries.
- Complete a risk assessment for the HLF construction sequencing (Starter heap constructed at the farthest uphill point instead of a more traditional configuration with the Starter constructed at the farthest downhill point, immediately adjacent to the tank/pond).
- Further evaluate expanding the HLF to the west, including reprocessing the spent ore from the legacy HLF on the new HLF or using the spent ore as overliner (after processing).

- Evaluate the possibility of re-routing the legacy HLF draindown solution directly on the new HLF.

26.7 Climatology

- Design storms were only available in NOAA Atlas 2; NOAA Atlas 14 data for a similar elevation just to the south has significantly lower values for the design storm event. Site-specific evaluations should be completed to optimize the design storm event determination.

26.8 Hydrology and Hydraulics

- Complete site reconnaissance and/or investigations at the diversions proposed around the perimeter of the pits. If rock is present, the diversions may need to be redesigned or replaced with alternate surface water control measures such as in-pit sumps (depending on the extents of the rock).
- Refine the designs for diversions around the RSF and stockpiles, including better defining the stockpile extents throughout the life of mine and surface conditions. Additional erosion protection may be required, and diversions may need to be re-routed to avoid potential rock outcrops or other difficult excavation. The diversions may be constructed in a phased approach to minimize initial CAPEX.
- Refine the designs for the sediment control ponds at the base of RSFs, including sediment cleanout interval, outlet structures, erosion protection, etc.
- Develop specific basin models for sections of the haul road and refine designs for the haul road culverts for each road segment.

26.9 Hydrogeology

- Piezometric Levels: Additional characterization of groundwater levels in the bedrock groundwater system will be needed as part of the permitting and operational monitoring program to improve confidence in the adequate freeboard analysis. Two to three additional vibrating wire piezometer strings are envisioned prior to commencing operations. Additional VWP strings and monitoring wells may also be installed during mine development.
- Bedrock Groundwater Quality: Between three and five bedrock groundwater monitoring wells are required to (i) further reduce uncertainty in bedrock groundwater levels and (ii) define the baseline groundwater quality beneath the existing and future open pits for environmental permitting.
- Hydraulic Testing: Hydraulic testing is required to characterize the BPMW well performance under increased pumping and the effect this pumping will have on the surrounding alluvial and bedrock groundwater systems. These data will be important to confirm (i) the capacity and viability of the water supply (ii) confirm there will be no drawdowns to other water users and nearby springs (iii) hydraulic connection between the bedrock and alluvial groundwater systems (if any).
- Potential impacts predictions: Once the Project is defined at the feasibility level, a groundwater modeling study should be undertaken for environmental permitting, to accurately quantify the potential changes to groundwater levels during operations and 100 years into permanent closure. The simulated groundwater flow results should also be combined with the geochemical studies to confirm there is no potential for adverse/ unmitigable groundwater quality impacts to the underlying bedrock groundwater system.

26.10 Permitting

It is recommended to advance the Project into the permitting process. The first step of which is to use this PFS as the basis to apply for a Mine Plan of Operations.

26.11 Recommendations Cost Estimate

Table 26-1 is a summary of the costs of the recommended work to advance the project to a construction decision.

Table 26-1: Cost Estimate for the Recommended Study Program

Activity	Cost
Exploration	$9,900,000
Testwork	
Metallurgy	$800,000
Hydrogeology & Water	$850,000
Permitting	$550,000
Geotechnical (incl. hydrology, hydrogeology, climate and HLF)	$1,000,000
Engineering and Feasibility Study	$3,500,000
Total	$16,600,000

27 REFERENCES

AMTEL 2020. Gold Deportment in Oxide Ore Composites from Black Pine, prepared for Liberty gold by AMTEL in August 2020.

Anderson, A.L., 1931, Geology and Mineral Resources of Eastern Cassia County, Idaho, Idaho Bureau of Mines and Geology Bulletin 14, 169 p.

Baker, C.J., 1974. Water Resources of the Curlew Valley Drainage Basin, Utah and Idaho. State of Idaho Department of Natural Resources Technical Publication No. 45.

Bolke, E. L., and D. Price, 1969. Hydrologic Reconnaissance of Curlew Valley, Utah and Idaho. US Geological Survey in cooperation with the Utah Department of Natural Resources. Technical Publication No. 25.

Brady, Bruce T., 1984, Gold in the Black Pine Mining District, Southeast Cassia County, Idaho, U.S. Geological Survey Bulletin 1382-E, 15 p.

Brown, G.A., 2016, Stantec Consulting Services, Inc. Review Report, Mineral Gulch WPR Project, prepared for Pilot Gold USA Inc., Stantec Project Number 203706052, 4 p.

Clemson, J.E., 1988, Delimiting heap leachable zones in the Black Pine Gold Deposit, Black Pine Gold District, Idaho, Process Mineralogy VIII, edited by D.J.T. Cason and A.H. Vassiliou, The Minerals, Metals and Materials Society, p. 57-79.

Cline, J., 2018, Nevada's Carlin-type gold deposits: What we've learned during the past 10 to 15 years: Society of Economic Geologists, Inc., Reviews in Economic Geology, v. 20, p. 7-37.

Cline, J., Hofstra, A., Muntean, J., Tosdal, R. and Hickey, K., 2005, Carlin-type gold deposits in Nevada: Critical geologic characteristics and viable models: Economic Geology, 100th Anniversary Volume, p. 451-484.

Cook, H.E., 2015, The evolution and relationship of the western North America Paleozoic carbonate platform and basinal depositional environments to Carlin-type gold deposits in the context of the carbonate sequence stratigraphy, in Pennell, W.M., and Garside, L.J., eds., New concepts and discoveries: Geological Society of Nevada Symposium proceedings: Reno, Nevada, Geological Society of Nevada, p. 1–80.

Dawson, H.A., 1980, Results of preliminary cyanide leach testing on composite of samples submitted by Pioneer Nuclear: Dawson Metallurgical Laboratories Inc., File No. P-345, 3 p.

DeCelles, P.G., 2004. Late Jurassic to Eocene evolution of the Cordilleran thrust belt and foreland basin system, western U.S.A.: American Journal of Science, v. 304, p. 105–168.

Defilippi, C., 1988, Black Pine carbonaceous lab testwork final report, report by Kappes, Cassiday & Associates for Noranda Exploration, Inc., 9 p.

Dix, R.B., 1984, Tallman Mine ore samples cyanide leach tests, report by Kappes, Cassiday & Associates for Coeur d'Alene Mines Corporation, 6 p.

Dix, R.B., 1990, Black Pine drill rejects agitated cyanide leach tests, report by Kappes, Cassiday & Associates for Noranda Exploration, Inc., 7 p.

Donahoe Hydrogeology, 2020. Preliminary Hydrogeologic Characterization and Groundwater Resources Impact Assessment, Black Pine Mineral Exploration Project, Cassia and Oneida Counties, Idaho. August.

Ennis, K., undated (1975?), Gold Resources Inc. Black Pine Property Crush-Leach Test, internal report for Gold Resources Inc. by Newport Minerals Inc., 3 p.

Fritz, F.P., 2012, Mineral Gulch Project, Cassia County, Idaho, Aeromagnetic Survey Interpretation, internal Report for Western Pacific Resources, 9 p.

Gustin, M.M., Simmons, G.L. and Smith, M.T., 2021, Updated Technical Report and Resource Estimate for the Black Pine Gold Project, Cassia County, Idaho, USA, NI 43-101 Technical Report for Liberty Gold Corp., 193 p.

Hardie, Byron S., 1964, Black Pine Gold Prospect, Cassia County, Idaho, Black Pine Mining District, internal Newmont Exploration Ltd. Memorandum, 3 p.

Hefner, M.L., Loptien, G.D., Ohlin, H.N., 1991, Geology and Mineralization of the Black Pine Gold Deposit, Cassia County, Idaho, in Geology and Ore Deposits of the Northeastern Great Basin, Field Trip No. 5, edited by Shaddrick, D.R., Kizis, J.A., Hunsaker, E.L., Geological Society of Nevada, Reno, Nevada, 8 p.

Hintze, L.H., 1991, Stratigraphic Evolution of the Northeastern Great Basin, in Geology and Ore Deposits of the Northeastern Great Basin, Field Trip No. 5, edited by Shaddrick, D.R., Kizis, J.A., Hunsaker, E.L., Geological Society of Nevada, Reno, Nevada, 32 p.

IDAPA, 2024 (Idaho Administrative Procedure Act) - Department of Environmental Quality, Waste Management and Remediation Division, 58.01.13 - Rules for Ore Processing by Cyanidation. 2025 for Rulemaking and 2026 Legislative Review

Kappes, Cassiday & Associates (KCA-2020a), Black Pine Project Bulk Pit Samples, Report of Metallurgical Test Work, June 2020, by Kappes, Cassiday & Associates, Project No. 9315C, Report I.D.: KCA0190072_BP02_02, File: 9315, 175 p.

Kappes, Cassiday & Associates (KCA-2020b), Black Pine Project Phase I Metallurgical Composites, Report of Metallurgical Test Work, October 2020, by Kappes, Cassiday & Associates, Project No. 9315C, Report I.D.: KCA0200012_BP03_06, File: 9315, 1233 p.

Kappes, Cassiday & Associates (KCA-2021), Black Pine Project GEOMET Samples Report of Metallurgical Test Work, October 2021, by Kappes, Cassiday & Associates, Project No 9305C, Report I.D.: KCA0210025_BP04_06, File: 9315, 1655 p.

Kappes, Cassiday & Associates (KCA-2024b), Black Pine Project Phase 5A Core Composites, Bottle Roll Test Work, Report of Metallurgical Test Work, August 2024, by Kappes, Cassiday & Associates, Project No. 9315C, Report I.D.: KCA0240010_BP12_01, File: 9315, 439 p.

Kappes, Cassiday & Associates (KCA-2023a), Black Pine Project, Rangefront Deposit, Report of Metallurgical Test Work, May 2023, by Kappes, Cassiday & Associates, 7590 Security Circle, Reno, NV 89506. Project No. 9315C, Report I.D.: KCA0220049_BP06_07. File 9315.

Kappes, Cassiday & Associates (KCA-2023b), Black Pine Project, Phase 4B, Report of Metallurgical Test Work, October 2023, by Kappes, Cassiday & Associates, 7950 Security Circle, Reno, NV 89506. Project No. 9315 C. Report I.D.: KCA0220102_BP08_05. File: 9315.

Kappes, Cassiday & Associates (KCA-2024a), Black Pine Project, 4C Metallurgical Composites, Report of Metallurgical Test Work, January 2024, by Kappes, Cassiday & Associates, 7950 Security Circle, Reno, Nevada 89506. Project No. 9315 C, Report I.D.: KCA023001_BP09_04. File: 9315.

Kappes, Cassiday & Associates (KCA 2024b), Black Pine Project, Phase 5A Core Composites, Report of Metallurgical Test Work, June 2024, by Kappes, Cassiday & Associates, 7950 Security Circle, Reno, Nevada 89506. Project No. 9315C, Report I.D.:KCA0240010_BP12_01A.

Knight Piésold Ltd, Preliminary Economic Assessment Open Pit Slope Design, October 2022, by Knight Piésold Ltd..

Konstantinou, A., Strickland, A., Miller, E and Wooden, J.P., 2012, Multistage Cenozoic extension of the Albion –Raft River-Grouse Creek metamorphic core complex: Geochronologic and stratigraphic constraints: Geosphere, DOI:10.1130/GES00778.1, 38 p.

Loptien, G.D., 1986, Paleozoic Rocks in the Black Pine Mountains, Cassia County, Idaho, informal sketch in Noranda Exploration, Inc. files.

Mine Development Associates, 2018. Technical Report on the Black Pine Gold Project, Cassia County, Idaho, USA. September.

Mueller, K. J., Cerveny, P. K., Perkins, M. E, and Snee, L. W., 1999, Chronology of polyphase extension in the Windermere Hills, northeast Nevada: Geological Society of America Bulletin, v. 111, no. 1, p. 11 – 27.

Noranda, 1989, Black Pine Project proposed offer to Kiewit to acquire Black Pine Mine, text and cash flow analysis, internal Noranda Exploration, Inc. communication, 66 p.

Novak, E., and Pegnam, M., 2022, Geotechnical data collection factual report for PFS slope design study Black Pine project, Oneida County Idaho

Ohlin, H.N., 1988, Geological Map of the Black Pine Mining District, Cassia County, Idaho, 1:500 scale.

Ohlin, H.N., 1989, Geology of the Black Pine Mining District, internal consultant report for Noranda Exploration, Inc., 49 p.

Owsley, D., 2009. Summary of Ground Water Levels in the Curlew Valley Ground Water Monitoring Network – 2009 Update. Idaho Department of Water Resources Open-File Report. November.

Paterson, N.R., 1979, Black Pine Project, Cassia Co., Idaho, geophysical report for Asarco from Paterson, Grant and Watson Limited/Consulting Geophysicists, 4 p.

Piteau, 2024. Black Pine Mine Hydrogeologic Field Characterization Program. DRAFT. October.

Potter, G.M., 1974, Letter to Donald Curry re percolation cyanidation tests carried out by the US Bureau of Mines, 1 p.

Powell, Randy, 2012a, Report on Heap Leach Production and Recovery, Black Pine, internal memorandum for Western Pacific Resource, 4 p.

Prochnau, J., 1985, Memo to R. Willden re review of the Black Pine property, Noranda Exploration, Inc. internal files, 21 p.

Robert, F., Brommecker, R., Bourne, B.T., Doback, P.J., McEwan, C.J., Rowe, R.R. and Zhou, X., 2007, Models and exploration methods for major gold deposit types, in Ore Deposits and Exploration Technology, Paper 48, edited by Milkereit, B., Proceedings of Exploration 07, Fifth Decennial International conference on Mineral Exploration, p. 691-711.

Sawyer, V., Scott, C.C., Comba, P., 1997, Black Pine Mine Heap Leach Closure Pre-Closure Planning and First Year Closure Activities, internal report for Pegasus Mining Corp., 20 p.

"Standardized Reclamation Cost Estimator (SRCE)." *Nevada Department of Environmental Protection*, ndep.nv.gov/land/mining/reclamation/reclamation-cost-estimator. Accessed 1 Sept. 2023.

Shaddrick, D.R., 2013, Technical report on the Mineral Gulch project, Cassia County, Idaho, USA, NI 43-101 Technical Report for Western Pacific Resources Corp., 67 p.

Shaddrick, D.R.; Kizis, J.A., Jr.; Hunsaker, E.L., III, 1991, The Geology and Ore Deposits of the Northeastern Great Basin, an Overview, in Geology and Ore Deposits of the Northeastern Great Basin, Field Trip No. 5, edited by Shaddrick, D.R., Kizis, J.A., Hunsaker, E.L., Geological Society of Nevada, Reno, Nevada, 13 p.

Simmons, G.L., 2023, Unpublished internal reports and spreadsheets used for metallurgical studies, provided to Liberty Gold by G.L. Simmons Consulting LLC.

Smith, J.F., Jr., 1982, Geologic Map of the Strevell 15-Minute Quadrangle, Cassia County, Idaho, U.S. Geological Survey Miscellaneous Investigations Series Map 1-1403, scale. 1:62,500.

Smith, J.F., Jr., 1983, Paleozoic Rocks in the Black Pine Mountains, Cassia County, Idaho, US Geological Survey Bulletin 1536, 36 p.

Smith et al. 2020, The black Pine Oxide Gold Project Southeastern Idaho: A New Look at the Historic Mining District. GSN symposium 2020.

Stantec, 2020. Black Pine Mine Exploration – Fall 2020 Groundwater Monitoring. November.

Stepperud, Erik C., (Stepperud 2020), Letter Report to Kappes, Cassiday & Associates, Comminution Testing, Hazen Project 12752, Report and Appendices A and B, May 22, 2020, 67 p.

Stepperud 2021, Phase 2 Report - Comminution Testing, Revision 1, Hazen Project 12752, Report and Appendices A and B, April 14, 2021.

Theodore, T.G., Armstrong, A.K., Harris, A.G., Stevens, C.H. and Tosdal, R.M., 1998, Geology of the Northern Terminus of the Carlin Trend, Nevada: Links Between Crustal Shortening During the Late Paleozoic Humboldt Orogeny and Northeast Striking Faults, in Contributions to the Gold Metallogeny of Northern Nevada, USGS Open File Report 98-338, p. 69-105.

Thorman, C.H, and Peterson, Fred, 2004, The Middle Jurassic Elko Orogeny: A Major Tectonic Event in Nevada-Utah, Search and Discovery Article #3022, adapted from "extended abstract" for presentation at the AAPG Annual Meeting, Salt Lake City, Utah, May 11-14, 2003, 7 p.

Threlkeld, R.W., 1983, Black Pine Project, Cassia County, Idaho, internal report for Coeur Explorations Inc, 6 p.

USDA Forest Service - Sawtooth National Forest, 1993, Draft Environmental Impact Statement, Black Pine Mine Expansion Project.

Willis, G.F., 2004, Pit Geologic Mapping for Mine Mappers, internal geologic maps for Western Pacific Resources, 5 maps.

Willis, G.F., 2011, Pit Geologic Mapping for Mine Mappers, internal geologic maps for Western Pacific Resources, 5 maps.

Wright, J.L., 2022, Black Pine Property Gravity Survey – 2022 Phase 2 GIS Database

Yernberg, 1988, Cyanide leach tests – final report, partial internal report by Kappes, Cassiday & Associates for Noranda Exploration, Inc. (missing the first 18 pages), 88 p.

Appendix A

Pre-Feasibility Study Contributors and Professional Qualifications

Certificate of Qualified Person (“QP”)

CERTIFICATE OF QUALIFIED PERSON

Matthew Sletten

I, Matthew Sletten, P.E., do hereby certify that:

1. I am employed by M3 Engineering and Technology Corp., 2175 W. Pecos Rd. Suite 3, Chandler, AZ 85224;

2. I graduated with a Master of Science in Structural Engineering and a Bachelor's in Civil Engineering from the South Dakota School of Mines and Technology in 2004 and 2006, respectively;

3. I am a registered Professional Engineer in good standing in the state of Arizona in the area of Civil Engineering, License #51936;

4. I have worked as an engineer and project manager in the base metals and precious metals industry for a total of 19 years;

5. My work experience includes detailed engineering, engineering management, project management, corporate management, capital and operating cost development and report development for major mining projects throughout the world;

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

7. I am independent of the Company as described in Section 1.5 of NI 43-101;

8. I am a contributing author for the preparation of the technical report titled Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study", Cassia County, Idaho, USA (the "Technical Report"), dated November 2, 2024, with an effective date of June 1, 2024 (the "Technical Report"), prepared for Liberty Gold;

9. I am responsible for Sections 1.1, 1.2, 1.20, 1.21, 2, 3, 4, 5, 18.1-18.4, 18.11-18.16, 19, 23, 25.5, 25.6, 26.11, and 27 of the Technical Report;

10. I have visited the Black Pine project site on October 31, 2023, and May 8, 2024, and reviewed the plant location site;

11. I have no prior involvement with the property that is the subject of the Technical Report.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

13. Neither I, nor any affiliated entity of mine, is at present, under any agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Liberty Gold, or any associated or affiliated companies;

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

15. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 day of November, 2024

(Signed) "Matthew Sletten"______
Signature of Qualified Person

Matthew Sletten_______________
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Benjamin Bermudez

I, Benjamin Bermudez, P.E., do hereby certify that:

1. I am currently employed as a Chemical/Process Engineer at M3 Engineering & Technology Corporation, 2051 W Sunset Rd, Suite 101, Tucson, AZ 85704, USA.

2. I am a graduate of Arizona State University and received a Bachelor of Science degree in Chemical Engineering in 2009.

3. I am a Registered Professional Engineer in good standing in the State of Arizona in the area of Chemical Engineering (No. 54919).

4. I have worked as an engineer for a total of 16 years. My experience includes mineral process plant engineering, support of new and on-going process plant operations, financial modeling of mineral properties, and project management.

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

6. I am contributing author for the preparation of the technical report titled “Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study”, Cassia County, Idaho, USA” dated November 21, 2024, with an effective date of June 1, 2024 (the “Technical Report”), prepared for Liberty Gold.

7. I am responsible for the preparation of Sections 1.14, 1.18, 1.19, 17, 21.1, 21.2, 21.5, 21.6, 22, and 26.10. I have not visited the project site.

8. I have no prior involvement with the project or property that is the subject of the Technical Report.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 day of November 2024.

(Signed) *“Benjamin Bermudez”*
Signature of Qualified Person

Benjamin Bermudez, PE

CERTIFICATE OF QUALIFIED PERSON

Gary L. Simmons

I, Gary L Simmons, Qualified Professional (QP), do hereby certify that:

1. I am the Principal Owner of:
GL Simmons Consulting, LLC
15293 Shadow Mountain Ranch Road
Larkspur, CO 80118

2. I graduated with a Bachelor of Science Degree in Metallurgical Engineering from the Colorado School of Mines, Golden, Colorado, USA, in 1973.

3. I am a Professional Metallurgical Engineer, registered with the Mining and Metallurgical Society of America, Qualified Professional (QP) Member in Metallurgy, Member Number – 01013QP, in good standing in the USA.

4. I have practiced in my profession since 1973. My relevant experience includes mine site and corporate level process development, project engineering, operations supervision and as a mineral processing project development consultant, in the base metals and gold/silver mining business, for a total of 51 years.

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

6. I am a contributing author for the preparation of the technical report titled ""Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study", Cassia County, Idaho, USA" dated November 21, 2024, with an effective date of June 1, 2024 (the "Technical Report"), prepared for Liberty Gold.

7. I am a contributing author for Sections 1.10, 13, 25.2, and 26.3. I have visited the project site on the following dates: June 3, 2019 (1 day), October 18-19, 2019 (2 days), June 22-23, 2020 (2 days), May 3, 2023 (1 day), and August 1-2, 2023 (2 days).

8. I have been involved in the Black Pine Project since it was acquired by Liberty Gold and have directed all metallurgical testing and participated in Black Pine technical reports requiring a metallurgical QP signature: 1. NI 43-101 Technical Report entitled "Updated Technical Report and Resource Estimate for the Black Pine Gold Project, Cassia County, Idaho, USA", effective date June 20, 2021, and 2. NI-43-101 Technical Reprot titled "Technical Reprot on the Updated Mineral resource Estimate at the Black Pin e Gold Project, Cassia and Oneida Counties, Idaho, USA", effective January 21, 2023.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 of November 2024.

(Signed) *"Gary L. Simmons*"
Signature of Qualified Person

Gary L. Simmons
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Richard DeLong

I, Richard DeLong, M.S., P.G., MMSA QP, do hereby certify that:

1. I am Senior Technical Advisor of:
   Trinity Consultants, a WestLand Resources, Inc. Company
   5401 Longley Lane, Suite 5, Reno, NV 89511

2. I graduated with a Master's Degree in Geology and a Master's Degree in Resource Management from the University of Idaho.

3. I am a Professional Geologist in good standing in the State of Idaho in the area of Geology (No. 727). I am also recognized as a Qualified Person Member with special expertise in Environmental Permitting and Compliance with the Mining and Metallurgical Society of America (No. 01471QP).

4. I have worked as an environmental permitting and compliance specialist for a total of 34 years. My experience includes permit acquisition of state and federal permits and baseline data acquisition programs for mining and exploration operations.

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

6. I am contributing author for the preparation of the technical report titled "Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study", Cassia County, Idaho, USA" dated November 21, 2024, with an effective date of June 1, 2024 (the "Technical Report"), prepared for Liberty Gold.

7. I am responsible for the preparation of Sections 1.17, 20, 26.10. I have not visited the project site.

8. I do not have prior involvement with the project or property that is the subject of the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 of November 2024.

(Signed) "*Richard DeLong*"
Signature of Qualified Person

Richard DeLong
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Todd Carstensen

I, Todd Carstensen, do hereby certify that:

1. I am employed as a Principal Mine Engineer with AGP Mining Consultants Inc., with an office address 246-132 Unit K, Commerce Park Drive, Barrie, Ontario L4N 0Z7, Canada.

2. I graduated with a Bachelor of Science, Mining Engineering in 1984 from the University of Wisconsin-Platteville.

3. I am a Registered Member of Society of Mining, Metallurgy & Exploration SME, RM #04063866. I am also recognized as a Qualified Person Member with special expertise in Mine Planning.

4. I have worked as a mining professional for a total of 38 years. My experience includes resource estimation, mine planning, scheduling, and financial evaluation for precious and base metal deposits including mine operations and project evaluations.

5. I have read the definition of "qualified person" set out in National Instrument 43101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am contributing author for the preparation of the technical report titled "Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study", Cassia County, Idaho, USA" dated November 21, 2024, with an effective date of June 1, 2024 (the "Technical Report"), prepared for Liberty Gold.

7. I am responsible for the preparation of Sections 1.15, 1.16, 18.5-18.10, 21.3, and 26.4-26.9. I visited the Black Pine project site on October 31, 2023, for the duration of 1 day.

8. I have been previously involved with the Black Pine Project. I participated in the mine planning activities prior to the start of the Pre-Feasibility Study.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 of November 2024.

(Signed) "*Todd Carstensen*"
Signature of Qualified Person

Todd Carstensen
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Valerie Wilson

I, I, Valerie Wilson, M.Sc., P.Geo., do hereby certify that:

1. I am a Principal Resource Geologist with SLR Consulting Australia Pty Ltd., of Level 1, 500 Hay Street, Subiaco, WA, Australia, 6008.

2. I am a graduate of the Camborne School of Mines, University of Exeter, UK in 2010 with a Master of Science degree in Mining Geology and a graduate of the University of Victoria, BC in 2006 with a Bachelor of Science degree in Geoscience.

3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #2113).

4. I have worked as a geologist for a total of 18 years since graduation from my bachelor's degree. My relevant experience for the purpose of the Technical Report is:
    a. Mineral Resource modelling, estimation, and reporting work on numerous Carlin-style gold projects in the Mountain West region of the USA.
    b. Mineral Resource estimation work on a variety of Projects from around the world.

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

6. I am contributing author for the preparation of the technical report titled "Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study", Cassia County, Idaho, USA" dated November 21, 2024, with an effective date of June 1, 2024 (the "Technical Report"), prepared for Liberty Gold.

7. I am responsible for the preparation of Sections 1.5-1.9, 1.11, 6, 7, 8, 9, 10, 11, 12, 14, 24, 25.1, and 26.1.

8. I visited the site on April 11, 2024

9. I have previous involvement with the project that is the subject of the Technical Report, acting in technical support and project management roles for technical assignments undertaken for Liberty Gold from 2021 to 2023, and acting as qualified person for the previous Mineral Resource estimate (effective February 15, 2024).

10. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 of November 2024.

(Signed) "*Valerie Wilson*"
Signature of Qualified Person

Valerie Wilson
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Nicholas Rocco

I, Nicholas Rocco, do hereby certify that:

1. I am Principal Engineer of:

   NewFields MDTS
   9540 Maroon Cricle, Suite 300
   Englewood, Colorado 80112

2. I graduated with a Ph.D. in Civil Engineering from the Missouri University of Science & Technology.

3. I am a Professional Civil Engineer in good standing in the State of Idaho (No. 15525). I am also recognized as a Qualified Person Member with special expertise in civil and geotechnical design and evaluation of heap leach facilities.

4. I have worked as a Geotechnical Engineer for a total of 20 years. My experience includes civil and geotechnical design and evaluation of heap leach facilities.

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

6. I am contributing author for the preparation of the technical report titled “Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study”, Cassia County, Idaho, USA” dated November 21, 2024, with an effective date of June 1, 2024 (the “Technical Report”), prepared for Liberty Gold.

~~7.~~ I am responsible for the preparation of Sections 1.15, 1.16, 18.5-18.10, 21.3, and 26.4-26.9. I personally visited the site on April 17, 2024.

8. I do not have prior involvement with the project or property that is the subject of the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 21 of November 2024.

(Signed) “*Nicholas Rocco*”
Signature of Qualified Person

Nicholas Rocco
Print Name of Qualified Person