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Black Pine Project



NI 43-101 Feasibility Study

Southern Idaho, USA

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LIBERTY GOLD

DATE AND SIGNATURES PAGE

The effective date of this report is August 1, 2026. The issue date of this report is September 18, 2026. See Appendix A, 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 PROJECT
FORM 43-101F1 TECHNICAL REPORT

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LIST OF APPENDICES

APPENDIX	DESCRIPTION
A	Feasibility Study Contributors and Professional Qualifications • Certificate of Qualified Person (QP)

1 SUMMARY

This Technical Report (Technical Report) has been prepared by M3 Engineering and Technology Corporation (M3) with the assistance of 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 a revised Reserve estimate and the Black Pine feasibility study (FS), which incorporates conceptual new design-work, scheduling, and projected costs, in support of the disclosure of the Mineral Reserve estimates in the Black Pine gold deposits.

Liberty Gold is listed on the Toronto Stock Exchange (TSX:LGD) and holds its interest in the Black Pine Project through its wholly owned subsidiary, Liberty Gold USA Inc., a Delaware, USA Corporation. Liberty Gold holds a 100% ownership interest in the federal lode claims that host the Mineral Reserve at the Black Pine property.

This Technical Report has been prepared as a feasibility study in accordance with National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* and Form 43-101F1 for the purpose of evaluating the technical and economic viability of the selected development configuration for the Black Pine Project as of the Effective Date. This Technical Report is not, and is not intended to constitute, a permit application, a modification or supplement to the current Mine Plan of Operations submitted to the applicable regulatory agencies (the "MPO"), or an authorization to construct or operate any Project component. For greater certainty, the MPO and the supporting environmental, engineering and technical information submitted to the applicable regulatory agencies remain the basis for the ongoing environmental review and permitting process, and nothing in this Technical Report is intended to amend, supersede, or expand the proposed action currently under federal and state agency review.

The development configuration evaluated in this Feasibility Study is fundamentally consistent with the Project described in the MPO; however, it expands upon the current MPO by incorporating feasibility-level optimizations and refinements not contemplated therein or under existing permitting. The Feasibility Study does not contemplate a change to the fundamental mining or mineral-processing methods described in the MPO. The Project remains a conventional open-pit mining operation in which run-of-mine oxide ore is placed directly on an engineered heap leach facility, and gold and silver are recovered through conventional cyanide leaching and an adsorption, desorption and recovery ADR plant and refinery. The Feasibility Study expands on the current MPO and envisions changes to scheduled mining rates, process flow rates, equipment capacities and the sequencing or configuration of certain facilities. The outcomes of the Feasibility Study will need to be further advanced through additional studies and future approval by Liberty Gold. Where these finalized studies indicate differences with the current MPO, Liberty will engage with applicable regulatory agencies and, when required, address differences through supplemental permitting processes and approvals.

The engineering designs, production schedules, process criteria, equipment selections, quantities and capital and operating cost estimates presented in this Technical Report have been developed to a level of detail considered by the Qualified Persons to be appropriate for a feasibility study and to support the mining rates, process flow rates, production forecasts, Mineral Reserve estimates and economic analyses presented herein. Certain designs contained herein may be described as conceptual or preliminary solely because they have not yet been advanced to final, permit-specific, issued-for-construction or detailed engineering status; such terminology is not intended to characterize the designs as speculative, unsupported or equivalent to the conceptual information customarily associated with a preliminary economic assessment. The designs and estimates remain subject to normal refinement through continued site exploration, advanced mine planning, and detailed engineering provided that such refinement is not presently expected to alter the fundamental development concept evaluated in this Technical Report.

1.1 PRINCIPAL FINDINGS

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

Table 1-1: Key Project Parameters & Findings

Mine Life	16 Years (including 1 year of stockpile reclaim)
Mine Type	Open Pit
Process Description	Run of Mine (ROM) heap leach Gold recovery by Adsorption, Desorption, and Recovery (ADR) plant & Refinery, three carbon column trains
Total Mineral Reserve Estimate[1]	433.3 M tonnes
Average Grade (5 year / 10 year / LOM)	0.35 g/t Au / 0.30 g/t Au / 0.29 g/t Au
Contained Gold (5 year / 10 year / LOM)	1,463 k oz Au / 2,339 k oz Au / 4,047 k oz Au
Produced Gold (5 year / 10 year / LOM)	1,011 k oz Au / 1,625 k oz Au / 2,841 k oz Au
Average Recovery (based on produced)	70.2% Au
Payable Gold	2,838 k oz Au
Average Annual Tonnes Mined	64 M tonnes (16 years)
Strip Ratio	1.34:1
Process (ROM) Throughput (tonnes/day)	74,200 (16 years)
Capital Expenditures	
Initial Capital Expenditures	$411.37M
Sustaining Capital Expenditures	$254.02M
Closure Capital Expenditures	$160.10M
Total Capital Expenditures (All Phases)	$825.49M
Unit Operating Costs	
Average Life of Mine (LOM) Mining Costs	$6.08 / ore tonne
Average LOM Process Costs	$2.07 / ore tonne
Average LOM G & A Costs	$0.80 / ore tonne
Average LOM Refining Costs	$0.02 / ore tonne
Average LOM Total Operating Costs	$8.98 / ore tonne
Cash Costs	$1,388 / oz Au
All in Sustaining Costs (AISC) – See Note	$1,566 / oz Au

[1]See Table 1-3 and the notes hereto.
Note: AISC includes; Sustaining Capital (Mining, Process, and Owner's Costs), Reclamation & Closure Costs, Idaho Mine License Tax, and Idaho Property Tax.
Note: Totals may not sum precisely due to rounding.

Financial Indicators	Alternate High Case	Alternate High Case	**Base Case**	Alternate Low Case	Alternate Low Case
Gold Price (per troy oz)	$4,500	$4,000	**$3,250**	$2,500	$2,000
Pre-tax Cash Flow in $M	$7,990	$6,578	**$4,460**	$2,342	$930
Pre-tax NPV (5%) in $M	$5,371	$4,394	**$2,929**	$1,463	$486
Pre-tax Internal Rate of Return (IRR)	109.4%	91.3%	**63.7%**	35.3%	15.7%
Pre-tax Payback (Years)	1.2	1.4	**2.0**	3.3	4.7
After-tax Cash Flow in $M	$6,343	$5,239	**$3,579**	$1,905	$775
After-tax NPV (5%) in $M	$4,336	$3,562	**$2,397**	$1,215	$397
After-tax IRR	104.3%	87.0%	**60.5%**	33.4%	14.3%
After-tax Payback (Years)	1.2	1.4	**2.0**	3.4	4.7

1.2 PROPERTY DESCRIPTION

The Black Pine Project (the 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, at the intersection of Utah State Highways 30 and 42. The approximate geographic center of the Black Pine property is 42.082°North latitude and 113.047°West 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 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

As of the Effective Date of this report the Black Pine property consists of a largely contiguous block of 670 unpatented federal lode and placer mining claims covering 11,904 acres (4,817 ha), one lease of State of Idaho mineral rights over a one square mile (642 acres) section (260 ha), a majority interest in 911 acres (369 ha) of private mineral rights, 139 acres (56 ha) of Company-owned private property, and a federal Hardrock Prospector Permit covering 1,762 acres (713 ha), all located within either Cassia or Oneida counties. The combined properties occupy an area of 15,359 acres (6,215 ha). The portion of the Black Pine property on which the Mineral Resource lies consists of a largely contiguous block of unpatented Federal mining claims and a single state minerals lease within Cassia County, Idaho (Figure 4-2).

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 through an agreement dated June 15, 2016. Liberty Gold staked the remaining 325 claims between 2016 and 2025.

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, 2027. County recording fees are also required annually, as well as a lease payment to the State of Idaho, Hardrock Prospector Permit rent, and private property taxes. Liberty Gold's total annual property holding costs were US$139,960 in 2026.

In September 2023 Liberty Gold bought back the aforementioned Western Pacific/Deer Trail Mining Company, LLC royalty, and assigned a new 0.5% NSR royalty to Wheaton Precious Metals (Cayman) Co. (WPM). The WPM royalty covers all claims, leases, private properties, and mineral rights. This royalty is subject to a 50% buyback for $3.6M, the cost of which has been included in the cash flow analysis in this FS in Year 1 of production. Production of metallic minerals from the private mineral rights lands described above will be subject to a 0.25% NSR. 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. In addition, production is subject to a 5% NSR royalty payable to the State of Idaho. 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 regionally from areas such as 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 kilovolt (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 9.9 megawatts (MW) of electrical power along the distribution line, which is managed by Raft River Rural Electric Co-op Inc.

Access to the site is via unpaved roads maintained by Oneida County, Bureau of Land Management, and United States Forest Service.

Liberty Gold offices and equipment are staged on private property located approximately 3.5 miles (5.6 km) southeast of the central Project area.

Water for exploration drilling is sourced 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. A small open pit was operated at the Tallman (Virmyra) mine 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 66,731 m (218,934 ft) 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 117,601 m (385,830 ft).

Approximately 24.0 million tonnes (26.5 million tons) of waste rock and 30.8 million tonnes (34.0 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 7,217 m (23,678 ft) prior to vending the property to Liberty Gold in 2016.

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 most of the gold mineralization of interest, has a structural thickness ranging from approximately 656 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 as disseminated fine-grained mineralization and little evidence of descrete vein hosted mineralization. Within the broader siliciclastic-carbonate sedimentary sequence of the middle plate higher-grade zones are hosted within favorable stratigraphic units, such as calcareous siltstones, 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 modeling 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 the overthickened zone of massive limestone and dolostone.

1.7 DRILLING

Liberty Gold carried out drilling activities in 2017 and 2019 through 2025, totaling 300,909 m in 1,287 holes, including 77 core holes (11,253 m), 1,210 RC holes (289,655 m), and 26 sonic holes (1,469 m). This drilling defined gold mineralization in several discrete bodies including the Discovery, Rangefront, Backrange, CD, M, J, F, and E zones. In the 2024 and 2025 drill programs, drilling was designed to convert Inferred resources to Indicated, and to expand the Mineral Resource at the Rangefront zone. Approximately 50,000 m of drilling is planned for 2026 with a on focused on upgrading resource confidences levels to Measured for the first 5 years of mining, geotechnical drilling for detailed engineering efforts around infrastructure, compliance drilling in areas for planned rock storage facilities and the leach pad, and exploration drilling to expand the mineral resource beyond the current pit margins.

1.8 SAMPLING, SAMPLE PREPARATION, ANALYSIS AND SECURITY

While documentation with respect to drilling and assaying methods and protocols is not available for all historical holes, all of the historical operators were reputable, well-known mining/exploration companies and include the likes of Newmont, Noranda, and Pegasus. The independent laboratories used to analyze the drill samples of the historical operators 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 manual checking of the database values using historical records, and statistical analyses.

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 Black Pine Mineral Resource.

The QP is of the opinion that the Black Pine data is 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 mining operations that began in 1991 were reviewed. These include bulk samples (~2.4" P_{80}) from Noranda (1988) which have been integrated into the FS recovery models. 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 (2019-2027)

Liberty Gold initiated metallurgical testing in 2019. Eight metallurgical test programs were subsequently completed, and two more programs are ongoing; ROM Column Leach test program (Phase 6) and Legacy Heap Leach Facility (LHLF) sonic drill sample testing (Phase 8). All leach testwork has been carried out by Kappes, Cassiday, & Associates (KCA) labs in Reno, Nevada. A summary of all test work completed for use in this feasibility study can be seen in Table 1-2.

Table 1-2: Summary of Metallurgical Test Work Completed – Black Pine Project

Sample Type	Count
Bottle-Rolls	
75 µm (DL)	220
75 µm (CIL)	212
1,700 µm (DL)	212
1,700 µm (CIL)	133
12.5 mm (DL)	25
Total Bottle Rolls	802
Columns	
12.5 mm (1/2")	82
25 mm (1")	136
>25 mm (>1")	9
Total Columns	227
Total Extraction Tests:	1,029
Total Samples:	216
Total Extraction Tests Completed:	1,029

A summary of the metallurgical test programs completed by Liberty Gold is included below.

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 across a range of grades and lithologic types 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 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 similar to those described above.
6. 2022 Discovery, E Zone, CD, F and M Zone (Phase 4B, 4C) - In early 2023 Liberty Gold selected 36 metallurgical composites for a testing program similar to those described above.
7. 2023 Large-diameter PQ Core Program (Phase 5A and 5B) - In 2023 Liberty Gold drilled 11 PQ core holes in the Backrange and J zone areas and infilled metallurgical gaps in M Zone, Discovery Zone and F Zone.

8. 2026 Run-of-Mine (ROM) Bulk Sample Test Program (Phase 6). In 2026 a ROM Bulk Sample 4 ft-diameter by 20 ft-tall column leach test program was initiated and is ongoing at KCA. Results will be published once the program is finished and a final report is issued.
9. 2026 large-diameter PQ Core Program (Phase 7). In April 2026 Liberty Gold delivered forty-four (44) 5-gal buckets containing broken PQ core material representing nine (9) individual low-grade variability composites to KCA for geo-metallurgical characterization, bottle-roll and column leach testing. Results will be published once the program is finished and a final report is issued.
10. 2026 LHLF sonic drill samples (Phase 8B). In April 2026 Liberty Gold delivered material for seven (7) composites from sonic drilling of the Legacy Heap Leach Facility. These are being tested at KCA to evaluate the potential for re-leaching at least a portion of this material. Results will be published once the program is finished and a final report is issued.

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 composites that have been tested.

- PPos – sandstone, quartzite, and siltstones (16) - upper plate
- Pola – limestone and sandy limestone (51) - middle plate
- Polb – siltstone, sandy limestone, and dolomite (36) - middle plate
- Polc – siltstone, limestone, sandstone, and dolomite (55) - middle plate
- Pold – limestone, dolomite, sandstone, and quartzite (27) - middle plate
- Pols – limestone, sandstone, and quartzite (27) - middle plate
- PMmx – limestone, siltstone, quartzite (7) - middle plate
- PMmc – shale, limestone, and quartzite (0) – lower plate, underlies most oxide gold mineralization. It was not tested for leach extraction as it is generally strongly carbonaceous

Recovery equations have been assigned to each of the lithologic units, with separate equations for each domain (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 lithologic types and styles and grade of mineralization.

1.11 MINERAL RESOURCES ESTIMATE

The Mineral Resource is based on 1,853 historical RC holes and 24 historical core holes, as well as 1,122 RC, 44 core holes, and 26 sonic 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.

The Mineral Resources at the Black Pine Project were modeled and estimated by SLR Consulting as QP and Liberty using the following process:

- Developing a geological model in Leapfrog Geo 3D modeling software, 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, internal low-grade material (below 0.1 g/t Au) was captured using indicator shells. A low-grade domain was generated using a 30 m buffer around the medium-grade domains.

- Reviewing the silver mineralization trends and defining medium-grade silver domains based on the 0.1 g/t gold domains.
- Compositing data to 10 ft (3.048 m) within the gold domains.
- Coding a block model comprised of 10 x 10 x 5-m (x, y, z) blocks and sub-blocked to 2.5 x 2.5 x 1.25-m blocks within the domains.
- Analyzing the modeled 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 and silver grade estimations.
- Evaluating, statistically and visually, the resulting model in detail prior to finalizing the mineral resource estimation.

The Mineral Resource was constrained by optimized pit shells created using a gold price of $2,800/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 - $2.59/tonne mined; heap leaching - $2.10/tonne processed; and G&A cost of $0.80/tonne processed at an assumed 54.4 million tonnes (Mt) per year mining rate. These values were determined from early estimates for the optimization effort and may not match exactly with final financial evaluations. Gold recoveries are based on equations derived from metallurgical data and vary by grade and rock unit. A 0.25% NSR royalty was also applied.

The in-pit resources were further constrained by the application of a cut-off grade of 0.10 g/t Au to all model blocks lying within the optimized pit shells. The portions of blocks coded as carbonaceous or designated as Met 4 material were assigned a recovery of 0% and thus excluded from the resource estimate. Met 4 material is defined as material with a gold cyanide solubility value less than 25% and is further defined in Section 13.13.2.

The 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 currently no Measured Mineral Resources at the Black Pine Project; any future declaration is contingent on additional technical data from the 2026 Drill-to-Measure (DtM) program. Indicated Mineral Resources were defined where drill hole spacing of 164 ft to 197 ft (50 m to 60 m) was achieved. The drill hole 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 portions of the low-grade domain that contain mineralization above the cut-off grade were classified as Inferred Mineral Resources. Following a 2025 sonic drill program, Mineral Resources for the LHLF were estimated by Forte Dynamics and are reported in the Indicated category. Due to the requirement for the LHLF material to remain in a containment facility, any potential mining scenario must consider mining and stacking all of the material on the new leach pad. The LHLF is reported in its entirety at a 0.1 g/t gold cut-off grade.

The total estimated Mineral Resources at the Project are presented in Table 1-3. The Gold Mineral Resource comprises 529.7 million tonnes (Mt) at an average grade of 0.30 g/t Au containing 5.04 million ounces (Moz) in the Indicated category and 153.6 Mt at an average grade of 0.21 g/t Au containing 1.04 Moz Au in the Inferred category. The Mineral Resources for silver were only reported within the Discovery and M Zones, which are supported by analytical results from 4-acid digestion and at a drill spacing threshold for the gold resource classification. Silver Mineral Resources total 8.75 Moz in the Indicated category and 1.13 Moz in the Inferred category.

Table 1-3: Summary of Mineral Resources for the Black Pine Project - April 30, 2026

Material Type	Zone	Classification	Tonnage (kt)	Average Grade		Contained Metal	
				Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
In-Situ	CD	Indicated	37,614	0.28	-	336	-
		Inferred	12,223	0.20	-	80	-
	Discovery	Indicated	188,399	0.35	1.35	2,097	8,147
		Inferred	27,503	0.19	1.03	169	912
	E	Indicated	9,657	0.26	-	82	-
		Inferred	11,825	0.22	-	82	-
	F	Indicated	6,543	0.32	-	66	-
		Inferred	1,217	0.24	-	10	-
	J	Indicated	3,601	0.33	-	38	-
		Inferred	12,731	0.25	-	104	-
	M	Indicated	7,324	0.51	2.57	120	606
		Inferred	5,406	0.21	1.25	36	217
	Rangefront	Indicated	237,679	**0.26**	-	1,998	-
		Inferred	59,787	**0.19**	-	371	-
Total In-Situ		**Indicated**	**490,816**	**0.30**	-	**4,737**	**8,754**
		Inferred	**130,691**	**0.20**	-	**851**	**1,133**
Backfill and Waste Dumps		**Indicated**	**0**	-	-	**0**	-
		Inferred	**18,144**	**0.25**	-	**145**	-
LHLF		**Indicated**	**30,844**	**0.19**	-	**190**	-
		Inferred	-	-	-	-	-
Total		**Indicated**	**521,660**	**0.29**	-	**4,927**	**8,754**
		Inferred	**148,835**	**0.21**	-	**996**	**1,133**

Notes:

1. The Mineral Resource has an effective date of April 30, 2026.
2. The Mineral Resource was prepared by Ms. Valerie Wilson, P.Geo., Principal Resource Geologist at SLR Consulting (Canada) Inc. (SLR) but for the LHLF estimate, which was prepared by Mr. Don Hulse, RM-SME of Forte Dynamics. Ms. Wilson is the Qualified Person (QP) responsible as defined under NI 43-101 for the Mineral Resource estimate but not for the LHLF material; Ms. Wilson is independent of Liberty Gold. Mr. Don Hulse is the QP for the LHLF estimate and is independent of Liberty Gold. CIM (2014) definitions were followed for Mineral Resources.
3. Mineral Resources are estimated at a gold cut-off grade of 0.10 g/t using a long-term gold price of US$2,800 per ounce.
4. Gold is reported for all blocks of in-situ, backfill, waste dump, or LHLF material. The waste dumps and backfill are limited to the CD, Discovery, F, J, and Rangefront zones. The LHLF is within the Rangefront zone, with the northern portion crossing into M Zone.
5. Silver is reported only for those blocks that meet or exceed the gold cut off grade of 0.10 g/t Au. It is limited to the Discovery and M zones, areas which are supported by analytical results from 4-acid digestion at a drill spacing threshold for gold class criteria.
6. Mineral Resources are estimated using variable recoveries derived from metallurgical studies.
7. Bulk density is variable by rock type and ranges between 2.43 t/m^3 and 2.63 t/m^3 in the mineralized domains. Quaternary deposits, LHLF, Backfill, and Waste Dumps were assigned a value of 1.8 t/m^3.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are inclusive of Mineral Reserves.
9. In-situ, backfill, and waste dump material are reported within conceptual open pit shells. Silver grades were not used in the pit optimization.
10. LHLF material is reported within the LHLF wireframe above a 0.10 g/t Au cut-off grade. Silver is not considered to be recoverable within the LHLF and is not reported.
11. The QPs for this Mineral Resource estimate are not aware of any environmental, permitting, legal, title, taxation, socio economic, marketing, political, or other factors that could materially affect the Mineral Resource Estimates presented in this news release.
12. Numbers may not add or multiply correctly due to rounding

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Black Pine Mineral Resource.

1.12 MINERAL RESERVE ESTIMATE

The Black Pine Project is planned as a conventional open pit mining operation using large-scale mining equipment. The Mineral Reserve estimate is based on the mine designs and production schedule developed by AGP Mining Consultants Inc. (AGP) for the Feasibility Study.

The Mineral Reserve consists of Indicated Mineral Resource blocks contained within the ultimate pit designs and meeting the economic cut-off criteria, including a minimum recoverable gold grade of 0.05 g/t Au. No Measured Mineral Resources are currently declared for the Black Pine deposits and, consequently, there are no Proven Mineral Reserves. The Black Pine open pits comprise the larger Discovery and Rangefront pits and the smaller CD, E, F, J, and M pits.

Ultimate pit limits were developed using economic pit optimization incorporating gold price, mining and processing costs, metallurgical recovery, haulage costs, and geotechnical criteria. Detailed pit designs incorporate feasibility-level geotechnical recommendations developed by Knight Piésold Ltd., including bench configuration, inter-ramp slope angles, berm widths, and other geotechnical design parameters. Inferred Mineral Resources were treated as waste for Mineral Reserve estimation purposes, and no additional mining dilution or ore loss has been applied.

The Mineral Reserves for the Black Pine Deposit are listed in Table 1-4 with the gold grade estimates based on the gold grades in the block model.

Table 1-4: Proven and Probable Reserves – Black Pine Project

Reserve Class	Tonnes (Mt)	Au (g/t)	Contained Ounces (Mozs)
Proven	-	-	-
Probable	433.3	0.29	4.04
Proven and Probable	**433.3**	**0.29**	**4.04**

Notes:

1. The Mineral Reserve estimate was prepared by Todd Carstensen RM-SME, Principal Mining Engineer at AGP Mining Consultants Inc., Barrie ON, Canada (AGP) and has an effective date of August 1, 2026. Mr. Carstensen is the Qualified Person responsible as defined under NI 43-101 for the Mineral Reserve estimate, he is 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 0.05 g/t Au recoverable gold within the designed pit based on a gold price of US$2,250/oz.
4. Metallurgical recovery of gold is based on a variable gold leach recovery model derived from extensive metallurgical studies.
5. Modeled carbonaceous units and lower recovery metallurgical domains have been treated as waste. Overall gold leach recovery averages 70.2%.
6. Units are metric tonnes, metric grams & troy ounces.
7. The QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.
8. Reserve pit designs are based on a pit optimization run using $2,250/oz Au, variable mining costs, and $1.88/t processing costs. Other costs around royalties, payables, and refining costs are also included.

The mine plan includes 433.3 Mt of leach feed grading 0.35 g/t Au in the first 5-years, 0.30 g/t Au in the first 10 years, and 0.29 g/t Au over the life of mine and containing a total of 4.04 Moz of gold. The mine life includes one year of pre-production mining followed by 15 years of in-pit mining, with processing of stockpiled ore continuing for one additional year. Approximately 582 Mt of waste rock is planned to be placed in rock storage facilities or backfilled into mined-out pits, resulting in an overall life-of-mine strip ratio of approximately 1.3:1.

Leach ore placement is planned up to 30.0 Mtpa in Years 1 and 2 and up to 37.0 Mtpa from Year 3 onward, subject to annual mine material movement requirements. The resulting production schedule and associated cost estimates form the basis for the economic evaluation of the Project.

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 two large pits and several smaller pits that will 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. The majority of the material movement for the first five years will be downhill loaded from the Discovery Pit Phases with an elevation change from 2,070 m at the pit exit to 1,635 m at the leach pad access. All ore from the Discovery Phases and the smaller satellite phases will be downhill loaded haulage to the leach pad, this makes up 46% of the total ore tonnes with the remaining 54% coming from the Rangefront phases that will be generally uphill loaded haulage. Ore from the pits will be hauled to the leach facility and end-dumped on to the leach 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 an elevation limit in the Rangefront and CD pits. The CD limit is set at 1,760 masl and Rangefront is limited to 1,470 masl. The elevation limit provides a 50 ft (15 m) offset above the measured water table in both pits.

Two mining fleets are planned at Black Pine. The larger mining fleet includes three 34 m^3 diesel hydraulic shovels and twenty-two 240 tonne rigid body trucks. The smaller fleet is planned to be provided through a contract miner and has 6.7 m^3 diesel hydraulic excavators and 64 tonne rigid body trucks. Additional loading support will be provided by 29.5 m^3 and 8.6 m^3 front end wheel loaders. The smaller pits, pioneering cuts, and mining areas with narrow bench widths and limited mining space will be mined with the smaller mining fleet. The 64 tonne trucks will allow for decreased ramp width, which in turn will reduce the construction and maintenance cost on the surface roads that transverse the steep terrain leading to pits that are located at higher elevations. Production blasthole drilling will be completed with diesel down the hole hammer (DTH) drills with 171 mm bits, presplit drilling will be completed with diesel top hammer drills with 127 mm bits.

There are two main rock storage facilities (RSF). The Discovery RSF is located in a valley north of the Discovery pit and the Rangefront RSF is located southwest of the Rangefront pit near the CD pit. The Discovery Phase 3 (Tallman), CD, and M pits will be backfilled with waste rock. Based on geochemical characterization to date, no specific waste rock management procedures are anticipated.

The stockpile area is located northeast of M pit. Different stockpiles will be established for the Met 1, Met 2, and Met 3 material. The Met 1 and Met 2 stockpiles will be reclaimed in Year 1 and the Met 3 stockpile will be reclaimed after the completion of in-pit mining.

Grade control will utilize the blast hole sampling method. A high-density RC drill program is planned in 2026 and will assist in the evaluation of RC grade control as a potential alternative that will be evaluated as part of detailed engineering. All pits are considered dry and are not anticipated to require dewatering.

1.14 RECOVERY METHODS

The process selected for recovering 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 leach pad as ROM ore directly, without crushing, agglomeration or conveyor stacking. Lime will be added directly to the haul trucks for pH control.

The stacking rate will be in accordance with the mine plan. The ore placement rate will be equivalent to a Life of Mine (LOM) average of 74,200 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 15,000 to 18,000 gallons per minute. The leached gold will be recovered from solution using a carbon adsorption circuit as part of a fully conventional Adsorption, Desorption & Recovery (ADR) plant. The carbon adsorption circuit will consist of a conventional cascading-column configuration, with each column having an 8-tonne carbon capacity. The gold will be stripped from the 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 abatement, followed by refinement in a melting furnace to produce gold doré bars.

1.15 INFRASTRUCTURE – HEAP LEACH AND WATER MANAGEMENT

The new heap leach facility (HLF) is sited in the eastern portion of the Project area and is designed in three phases to contain the estimated 433 million dry metric tonnes of leachable material and material from the LHLF. The HLF phasing was developed to accommodate the mine plan schedule and defer initial capital where possible.

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

The heap was designed at overall 3H:1V side slopes to accommodate the closure footprint; actual operational slopes will include benching and steeper inter-bench face slopes with typical slopes being reflective of the angle of repose of the ore. The maximum heap height will be 500 ft (approximately 150 m). ROM leachable material will be truck-stacked in nominal 50-ft (15 m) lifts with the first lift of each phase being 20 ft (6 m) high.

Water from process solution sources will be routed to the tank system within a process pond located at the topographic low at the southeast corner of the HLF. Flows exceeding the capacity of the process pond will overflow via a lined containment berm into an event pond. The pond system will be double-lined with a HDPE geomembrane liner and will include a leak detection and pumpback system.

Storm water diversions will route non-contact water away from the HLF, conveying clean runoff water from native areas downstream to planned discharge points. Diversions are also planned around the open pits, development rock storage areas, and stockpiles. Runoff from the stockpiles and development rock storage areas will report to sediment control ponds prior to discharging downgradient 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 FS-level designs. Additional investigations, evaluations, and analyses will be required during detailed design to confirm assumptions and reduce the risk of encountering unforeseen conditions during construction or operation.

During construction, a 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.

An Operations, Maintenance, and Safety (OMS) manual will be a key component to ensure operations and monitoring controls are in place for the life of the structure. The OMS manual will include descriptions of instrumentation and instructions for monitoring to provide early warning systems which will allow Liberty Gold to assess conditions at the HLF and provide recommendations if parameters trend toward thresholds that would trigger corrective action. The Capital Cost Estimate includes preliminary instrumentation and monitoring systems.

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 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 (Idaho State Department of Lands), and private lands controlled by Liberty Gold USA Inc. The mineral rights on the NFS lands, public lands, and Idaho State minerals are controlled by Liberty Gold.

Liberty Gold prepared a 2026 Mine Plan of Operations (MPO) based on the Pre-Feasibility Study mine plan and submitted it under applicable USFS, BLM, and IDL mine plan and reclamation requirements. The MPO was deemed administratively complete by the USFS Sawtooth National Forest, Minidoka Ranger District, and the BLM Pocatello Field Office on November 24, 2025. The USFS then published a Notice of Intent (NOI) on April 3, 2026, initiating preparation of an Environmental Impact Statement (EIS) and formally starting the National Environmental Policy Act (NEPA) review process. Public scoping was initiated through NOI to collect input on the proposed action, potential alternatives, environmental effects, and relevant studies to inform the scope and content of the EIS.

In January 2026, the Project was accepted into the FAST-41 process as a Covered Project, providing a coordinated permitting timetable, a project advisor, public permitting dashboard, and greater transparency and accountability for federal and state permitting milestones. The Draft EIS is currently identified as in progress, with the FAST-41 timetable targeting publication of the Draft EIS Notice of Availability on February 5, 2027, the Final EIS Notice of Availability on November 12, 2027, and a Record of Decision on January 5, 2028.

In addition to the MPO and EIS process, Liberty Gold must obtain BLM authorizations for certain access, utility, and surface-use components of the Project. The BLM identified the need for Right-of-Way (ROW) applications under 43 CFR 2800 for access routes and pipelines, and a lease under 43 CFR 2920 for operations and facilities on BLM surface with non-federal mineral estate and certain BLM-Bankhead Jones lands. Final decisions on the ROW applications and lease are currently scheduled for March 15, 2028, according to the FAST-41 permitting timetable.

Subsequent to receiving approval for the current MPO, Liberty Gold would need to develop and formally submit a modified MPO and supporting Reclamation Plan based on any additional study outcomes for the project aspects outlined as part of this Feasibility Study. The modified MPO and Reclamation Plan must provide sufficient engineering, environmental, reclamation, and operational detail for the USFS and BLM to evaluate the changes relative to the 2026 MPO and determine whether the revisions would result in new or different environmental effects. In addition to submitting a modified MPO and Reclamation Plan, Liberty Gold would need to evaluate and, as necessary, update Idaho State cyanidation facility approvals for the Feasibility-level mine plan.

Liberty Gold initiated baseline environmental studies, including multiple field data collection surveys, to characterize existing environmental conditions within the Project area in support of what became 4 separate exploration EAs completed between 2017 and 2022. Since issuance of the Pre-Feasibility Study Technical Report (M3 Engineering,

2024), extensive additional baseline studies continue to be conducted to support ongoing permitting processes associated with the current MPO and to advance understanding of the potential for the deposit to sustain future growth opportunities.

Liberty Gold has implemented an active and extensive stakeholder and community outreach program since June 2019. Engagements have included regular project updates and introductory meetings with municipal, county, and state officials in Idaho and Utah, including the Malta and Snowville city councils; Cassia, Oneida, and Box Elder county commissioners; and members of the Idaho Legislature. The Project team has also met with regulatory agencies, industry representatives, utilities, emergency-response organizations, local businesses, and community service groups to provide information on Project planning and development.

At the time of this report, no material environmental or social issues have been identified that are expected to preclude securing the required permits and authorizations to develop the Project.

1.18 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated for the Feasibility Study by AGP (mine development), M3 (process plant, site development, power transmission and distribution, and ancillaries), and NewFields (heap leach facility and stormwater management). Table 1-5 shows the estimated capital costs for the Project. This includes $411.4 million in pre-production capital and $414.1 million for sustaining capital. Total LOM capital costs are estimated at $825.5 million, which are inclusive of reclamation and closure, presently estimated at $160.1 million and applied at the end of the operational life (starting in Year 20).

Table 1-5: Capital Cost Summary

Category	Units	Initial	Sustaining	Total
Site General (Earthworks)	K USD	$16,765	-	$16,765
Process Plant (ADR, Refinery, Reagents)	K USD	$76,123	$1,400	$77,523
Power Systems	K USD	$7,853	-	$7,853
ADR Bldg. & Ancil. (Warehouse, Maint, Admin, Fuel)	K USD	$33,062	$2,975	$36,037
Freight (Process Plant)	K USD	$5,421	-	$5,421
Sub-Total Direct Cost (Process Plant & Support)	K USD	**$139,225**	**$4,375**	**$143,600**
Construction Support (inc. Mobilization)	K USD	$4,177	-	$4,177
Engineering, Procurement, & Const. Mgmt.	K USD	$19,930	-	$19,930
Vendor Support	K USD	$2,578	-	$2,578
Spare Parts (Capital, Commissioning)	K USD	$2,609	-	$2,609
First Fills (Process Plant)	K USD	$595	-	$595
Contingency (Process Plant)	K USD	$25,367	-	$25,367
Owner's Cost	K USD	$24,367	$3,600	$27,967
Sub-Total Indirect Cost (Process Plant & Support)	K USD	**$79,624**	**$3,600**	**$83,224**
Heap Leach Facility Direct Cost (NewFields)	K USD	$69,257	$109,146	$178,403
Heap Leach Facility Indirect Cost (NewFields)	K USD	$3,905	$6,295	$10,200
Heap Leach Facility Contingency (NewFields)	K USD	$13,827	$24,795	$38,621
Sub-Total Heap Leach Facility	K USD	**$86,990**	**$140,235**	**$227,225**
Stormwater Controls Direct Cost (NewFields)	K USD	$2,036	$18,299	$20,335
Stormwater Controls Indirect Cost (NewFields)	K USD	$121	$1,131	$1,252
Stormwater Controls Contingency (NewFields)	K USD	$379	$4,326	$4,705
Sub-Total Stormwater Controls	K USD	**$2,535**	**$23,757**	**$26,292**
Mine Capital Equipment (AGP)	K USD	$37,004	$46,199	$83,203

Category	Units	Initial	Sustaining	Total
Mine Preproduction Costs (AGP)	K USD	$59,382	$35,857	$95,239
Mining Capital Contingency (AGP)	K USD	$6,612	-	$6,612
Sub-Total Mine Capital	K USD	**$102,998**	**$82,056**	**$185,054**
Sub-Total Closure	K USD	**-**	**$160,100**	**$160,100**
TOTAL CAPITAL COST	K USD	**$411,370**	**$414,123**	**$825,493**

Table 1-6 shows the estimated operating costs for the LOM project. Operating costs were estimated at $3.89 billion for the LOM. This is equivalent to $8.98 per tonne of ore or $1,371 per ounce of payable gold.

Table 1-6: Operating Cost Summary

		Operating Cost	
Category	**K USD**	**$ / tonne processed**	**$ / Au oz**
Mining Costs	$2,635,819	$6.08	$928.74
Process Plant	$898,622	$2.07	$316.63
G&A	$347,100	$0.80	$122.30
Refining	$10,653	$0.02	$3.75
TOTAL OPERATING COST	**$3,892,193**	**$8.98**	**$1,371.43**

1.19 ECONOMIC ANALYSIS

The economic analysis for the Black Pine Project is based on a feasibility study-level financial model incorporating the mine and production schedules, gold production and sales, initial and sustaining capital expenditures, operating costs, royalties, taxes, and reclamation and closure costs. The model evaluates the Project on both a before-tax and after-tax basis using net present value (NPV), internal rate of return (IRR), life-of-mine cash flow, and payback period.

Ore is planned to be processed by run-of-mine (ROM) cyanide heap leaching, with gold recovered through an ADR facility as described in Section 17. The financial model incorporates the timing of gold recovery from the heap leach, including the lag between placement of ore, recovery of soluble gold, and production through the ADR facility. Ore is placed on the heap leach facility over a 16-year period, with solution application continuing for an additional four years to recover remaining soluble gold.

The mine plan schedule delivers 433 Mt of ore to the leach pad grading 0.35 g/t Au in the first 5 years, 0.30 g/t Au in the first 10 years, and 0.29 g/t Au over the 16 year mine life, including one year of placing only stockpiled ore after in-pit mining has ended. Total placement is approximately 4.05 Moz of gold at an average life-of-mine recovery of 70.2%. Approximately 2.84 Moz of gold are projected to be produced and approximately 2.84 Moz are payable.

Table 1-7: Life of Mine Process Statistics

Total Ore (kt)	**433,338**
Gold Grade (g/t)	0.29
Contained Gold (kozs)	4,047
Gold Recovery %	70.2%
Produced Gold (kozs)	2,841
Payable Gold (kozs)	2,838

The economic analysis assumes a base-case gold price of $3,250 per troy ounce, applied to all life-of-mine production without escalation or hedging. Gold revenues are based on payable gold production, with a 99.90% payable factor and an allowance of $3.75 per ounce for bullion refining, transportation, and insurance.

Initial capital expenditures are estimated at approximately $411.4 million. Total life-of-mine capital expenditures are estimated at approximately $825.5 million, including sustaining capital and approximately $160.1 million of reclamation and closure expenditures. Life-of-mine operating costs are estimated at approximately $3.89 billion, comprising mining, process plant, G&A, and refining costs. Including royalties, total production costs are estimated at approximately $3.94 billion.

The financial model applies five-year straight-line depreciation to initial and sustaining capital and ten-year straight-line depreciation to development costs, which consist primarily of heap leach expansion costs. The model incorporates applicable Idaho and U.S. federal taxes and assumes that the Project is financed entirely with equity. The after-tax economic analysis incorporates applicable federal and Idaho taxes, including an Idaho Mine License Tax of 1%, Idaho Corporate Income Tax of 5.3%, Idaho personal property tax of 0.663%, and U.S. Federal Corporate Income Tax of 21%. The financial model also reflects Liberty Gold's intention to buy down certain existing royalties prior to the end of the first year of production, resulting in a 0.25% NSR royalty over the life of mine, in addition to a separate 5% Idaho State NSR royalty applicable to certain ounces associated with the Rangefront deposit.

At the base-case gold price of $3,250 per ounce, the Project generates an after-tax NPV at a 5% discount rate of approximately $2.40 billion, an after-tax IRR of 60.5%, and an after-tax payback period of approximately 2.0 years. The corresponding before-tax NPV at a 5% discount rate is approximately $2.93 billion, with an IRR of 63.7% and a payback period of approximately 2.0 years. NPV calculations use mid-year discounting. No silver production has been included in the financial model or key economic indicators; any future silver production therefore represents potential upside to the Project.

Table 1-8: Key Economic Indicators

Indicators	Before-Tax	After-Tax
LOM Cash Flow ($000)	$4,460,218	$3,578,943
NPV @ 5% ($000)	$2,928,582	$2,396,566
NPV @ 10% ($000)	$2,016,198	$1,673,650
IRR	63.7%	60.5%
Payback (years)	2.0	2.0

Gold-price sensitivity analysis was completed over a range of $2,000 to $5,500 per ounce. At a gold price of $2,000 per ounce, the Project generates an after-tax NPV at a 5% discount rate of approximately $397 million, an after-tax IRR of 14.3%, and a payback period of approximately 4.7 years. At $5,500 per ounce, the corresponding after-tax NPV increases to approximately $5.89 billion, with an after-tax IRR of 139.4%.

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 following all required permitting and related approvals.

The authors have reviewed the Project data, including the drill-hole database and available metallurgical information, and all QPs 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 FS, the Project should continue on a positive path to a construction decision.

Results of historical metallurgical tests and those commissioned by Liberty Gold indicate that 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. Based on the quality and representativeness of the metallurgical testwork completed, the QPs are of the opinion that the recovery model meets or exceeds standard industry practice for feasibility-level determination of heap leach gold recoveries and heap leach production forecasting.

The process selected for recovery of gold and silver from the Black Pine mineralized material is a conventional ROM heap-leach recovery circuit. The material will be mined by standard open-pit mining methods and trucked from each deposit to a centralized heap-leach pad and conventional ADR processing facility and on-site refinery to produce gold doré bars

This technical report indicates an average payable gold over the estimated 16-year LOM of approximately 176,700 oz per year, with a year 1-5 peak payable gold in Year 5 of 277,400 ounces of gold, and a LOM peak in Year 13 of 295,000 oz of payable gold. Payable gold anticipated in the first 5 years of operation averages 202,000 oz per year. Cash costs are estimated to be $1,388/oz of payable gold, and AISC are estimated to be $1,566/oz of payable gold. The resulting after-tax cash flow is $3.58 billion, for an after-tax NPV (5%) of $2.40 billion, and an estimated payback period of 2.0 years.

Based on the findings of this feasibility study, the project continues to demonstrate strong technical and economic potential, with the capacity to generate significant long-term returns. The results summarized above provide a sound basis for advancing the project from feasibility study into execution.

Subject to the ongoing management of identified risks, confirmation of key assumptions, and completion of required approvals, the project is well positioned to proceed with permitting, detailed engineering, procurement, and execution-readiness activities. These next phases will further develop the engineering, cost estimate, schedule, and operating strategy, ultimately supporting a disciplined progression toward construction, commissioning, and commercial production. In addition, the acceptance of Black Pine as a covered project under FAST-41 notably derisks the overall projected timeline to acquire necessary federal and state permits required for initial construction.

1.21 RECOMMENDATIONS

Based on the results of the Feasibility Study, the Qualified Persons recommend that Liberty Gold advances the Project into the engineering, procurement and construction management (EPCM) and project execution phases, with the objective of constructing, commissioning and placing the Project into commercial production.

The Feasibility Study demonstrates that the Project has sufficient technical and economic merit to support advancement, subject to the assumptions, qualifications, risks and opportunities described in this Technical Report. The recommended execution program includes completion of detailed engineering; advancement and maintenance of required permits and authorizations; completion of project financing and commercial arrangements; procurement of long-lead equipment; construction planning and contracting; site preparation; mine development; construction of the process plant and supporting infrastructure; operational-readiness activities; commissioning; and production ramp-up.

The recommended work program is proposed in two phases:

- Phase 1 - EPCM and Execution Readiness: Complete detailed engineering, advance permitting, finalize the execution schedule and control budget, undertake procurement of long-lead equipment, complete financing and material commercial arrangements, and prepare the site and contracting strategy for full construction. The estimated cost of Phase 1 is approximately $35M.
- Phase 2 - Construction, Commissioning and Ramp-Up: Proceed with full project construction, mine development, installation of processing and infrastructure facilities, recruitment and training, operational

readiness, commissioning, and ramp-up toward commercial production. The estimated cost of Phase 2 is approximately $400M.

Advancement from Phase 1 to Phase 2 would be subject to confirmation that the principal conditions required for construction have been satisfied. These conditions should include, as applicable, receipt of material permits and authorizations; availability of adequate project financing; completion of critical engineering, procurement and operational readiness activities; satisfactory resolution or management of material environmental, social, technical, commercial and implementation risks; and approval by the Company's Board of Directors.

The Qualified Persons recommend advancement of the Project into EPCM and project execution to be the appropriate next stage of development.

2 INTRODUCTION

M3 has prepared this technical report on the Black Pine Project ("Black Pine" or the "Project"), located in Cassia and Oneida counties, Idaho, for Liberty Gold. The purpose of this report is to disclose the Mineral Reserve estimate. This report, with an effective date of August 1, 2026 (the "Effective Date"), conforms to National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects. The Mineral Resource and Mineral Reserve estimates presented in this Technical Report supersede all prior estimates for the Project.

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

Liberty Gold holds a 100% ownership interest in the federal lode claims that host the Mineral Reserve at the Black Pine property. The property was 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 return (NSR) royalty and 300,000 common shares of Liberty Gold. Portions of the property staked subsequent to the purchase are not subject to the NSR.

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 (except 14.16), as well as subsections 1.3-1.9, 1.11, 24, 25.1, and 26.1. 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, Lorraine Tam P. Geo, SLR Principal Resource Geologist, April Barrios, P. Geo, SLR Senior Resource Geologist, Rosmery Delgado, SLR GIS Specialist, Leah Longley, SLR Project Geologist, Owen Nicholls, Director of Technical Services 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.

Consulting Metallurgist Gary L. Simmons, MMSA, a QP under NI 43-101 with respect to metallurgy, prepared and is responsible for Sections 1.10, 13 (except 13.14), 25.2, and 26.3. Mr. Simmons has no affiliation with Liberty Gold except that of independent consultant/client relationship. Mr. Simmons conducted a site visit to the Black Pine property on June 3, 2019, October 18-19, 2019, June 22-23, 2020, May 3, 2023, August 1-2, 2023, June 2-4, 2025, August 13-15, 2025, and January 14-16, 2026.

Sections 1.1, 1.2, 1.20, 1.21, 2, 3, 4, 5, 18.1-18.3, 18.8-18.15, 19, 23, 25.7, 25.8, 26.7, 26.9, 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 (except 16.5), 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 June 17, 2026.

Sections 1.14, 1.18, 1.19, 13.14, 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, 18.4, 18.5.1-18.5.3, 18.6, 18.7, 20.10.2, 21.3, 25.5, and 26.4-26.6 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, remotely or through email with the various Liberty Gold team members including Black Pine personnel responsible for geology, mining, environmental activities and other team members for processing and infrastructure. These individuals include: Moira Smith, Ph.D., P.Geo., Corporate Technical Advisor for Liberty Gold; Jon Gilligan, B.Sc., Ph.D., Chief Executive Officer for Liberty Gold; Peter Shabestari, B.Sc., C.P.G., Vice President of Exploration for Liberty Gold; Will Lepore, M.Sc., P.Geo., Principal Geologist for Liberty Gold; and Gerald Heston, Director, Land Management and Spatial Analytics.

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, 18.1-18.3, 18.8-18.15, 19, 23, 25.7, 25.8, 26.7, 26.9, and 27
Benjamin Bermudez, P.E.	Chemical/Process Engineer, M3 Engineering and Technology Corporation	No visit to site	1.14, 1.18, 1.19, 13.14, 17, 21.1, 21.2, 21.5, 21.6, 22, 26.10
Todd Carstensen, RM-SME	Principal Mine Engineer, AGP Mining Consultants Inc.	October 31, 2023 and June 17, 2026	1.12, 1.13, 15, 16 (except 16.5), 21.4, 21.7, 25.3, 25.4, and 26.2
Kyle Brangers, CPG	GSI Environmental Inc.	No visit to site	1.17, 20 (except 20.10.2), 25.6, 25.9, and 26.8
Nicholas Rocco, Ph.D., P.E.	Principal Engineer, NewFields MDTS	April 17, 2024	1.15, 1.16, 18.4, 18.5.1-18.5.3, 18.6, 18.7, 20.10.2, 21.3, 25.5, and 26.4-26.6
Gary L. Simmons, MMSA	Consultant, GL Simmons Consulting LLC	June 3, 2019, October 18-19, 2019, June 22-23, 2020, May 3, 2023, August 1-2, 2023, June 2-4, 2025, August 13-15, 2025, January 14-16, 2026	1.10, 13 (except 13.14), 25.2, and 26.3
Valerie Wilson M.Sc. P.Geo.	Principal Resource Geologist, SLR Consulting (Canada) Ltd.	April 11, 2024	1.3-1.9, 1.11, 6, 7, 8, 9, 10, 11, 12, 14 (except 14.16), 24, 25.1, and 26.1
Brendan Fetter, P.E.	Forte Dynamics, LLC (part of SLR)	No visit to site	16.5, 18.5.4
Don Hulse, RM-SME	Forte Dynamics, LLC (part of SLR)	June 6, 2025	14.16

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 27 References. This report incorporates information and descriptions drawn from reports for Liberty Gold by Gustin et al. (2021) entitled "Updated Technical Report and Resource Estimate for the Black Pine Project, Cassia County, Idaho, USA" with an effective date of June 20, 2021 and Sletten et al. (2024) entitled "Black Pine Project NI 43-101 Technical Report" with an Effective Date of June 1, 2024.

The Effective Date of this technical report is August 1, 2026.

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.

Table 2-2: Units of Measurement

a	annum
A	ampere
ac	acre
bbl	barrels
Btu	British thermal units
°C	degree Celsius
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimeter
cm^2	square centimeter
d	day
dia	diameter
dmt	dry metric tonne
dwt	dead-weight ton
°F	degree Fahrenheit
ft	foot
ft^2	square foot
ft^3	cubic foot
ft/s	foot per second
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per liter
Gpm	Imperial gallons per minute
g/t	gram per metric tonne
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic meter
ha	hectare
hp	horsepower
hr OR hrs	hour(s)
Hz	hertz
in.	inch
in^2	square inch
J	joule
k/K	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometer
km^2	square kilometer
km/h	kilometer per hour
kPa	kilopascal
kV	kilovolt
kVA	kilovolt-amperes
kW	kilowatt
kWh/m^3	kilowatt-hours per cubic meter
kWh/yr	kilowatt-hours per year
kWh	kilowatt-hour
l	Liter (L in metallurgical use)
lb(s)	pound(s)
L/s	liters per second
m	meter
M	mega (million); molar
m^2	square meter
m^3	cubic meter
masl	meters above sea level
mg	milligram
mi	mile(s)
min	minute
mm	millimeter
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
oz	Troy ounce (31.1035g)
oz/st, opt	ounce per short ton
pcf	per cubic foot
ppb	part per billion
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	Imperial short ton (2,000 lbs.)
stpa	short ton per year
stpd	short ton per day
t	metric tonne(s)
tpa	metric tonne per year
tpd	metric tonne per day
US$/USD	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
wt%	weight percent
yd^3	cubic yard
yr	Year
µm	micron or 10^{-6} meters

Units of measurement are primarily in SI (metric) convention. Exceptions have been made to preserve consistency with information reference, where the original source data was presented in U.S. customary (imperial) units, or where imperial units match the jurisdiction of the location where the project will be developed.

Table 2-3: Acronyms and Abbreviations

Abbreviation	Description
AA	atomic absorption
ABA	acid-base accounting
afa	acre-foot per annum
Ag	silver
AJD	Approved Jurisdictional Determination
ARD	Acid Rock Drainage
ARMPA	Approved Resource Management Plan Amendments
Au	gold
Au_{CN}	cyanide-soluble gold
Au_{FA}	gold analysis by fire assay, total gold content
BDL	below detection limit
BFA	Bench Face Angles
BLM	Bureau of Land Management
BMP	Best management practices
BPMW	Black Pine Mine Well
BOOT	Build Own Operate Transfer
°C	degrees Celsius
CAD	Computer-Aided Design
Calc, calc	calculated
Calfm	calcite-cemented breccia
CAPEX	Capital Expenditure
CDN	CDN Resource Laboratories Ltd
CFR	Code of Federal Regulations
CIL	Carbon in Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CNI	Call & Nicholas, Inc.
core	diamond core-drilling method
CPe	corrugated polyethylene
COI	Constituents of Interest
COPC	Constituents of Potential Concern
CQA	Construction Quality Assurance
CRMs	certified reference materials
CSTG	Columbian Sharp-Tailed Grouse
CTGDs	Carlin-type gold deposits
DDH	diamond drill hole
DEM	digital elevation model
DGI	DGI Geoscience Inc.
DL	direct leach
DO	dissolved oxygen
DTH	down the hole
DtM	Drill-to-Measure
Dwi	drop-weight index
DZ	Discovery Zone
EIS	Environmental Impact Statement

Abbreviation	Description
EPCM	engineering, procurement, and construction management
ESA	Endangered Species Act
EV	Expected Value
Ext	extracted
FA	fire assay
FAST	Fixing America's Surface Transportation
FCF	Free Cash Flow
FeOx	iron (oxyhydr)oxides
FMS	Fleet management system
FPISC	U.S. Federal Permitting Improvement Steering Council
FS	Feasibility Study
G&A	General and Administrative
GA	General Arrangement
GAC	granulated activated carbon
GET	Ground Engaging Tool
GHMA	General Habitat Management Area
GPS	Global Positioning System
GRSG	Greater Sage-Grouse
HCL	hot cyanide leach
hd	head
HLF	Heap Leach Facility
HQT	Habitat Quantification Tool
HPP	Hardrock Prospector Permit
ICP	inductively-coupled plasma-emission spectrometric method
ICP-AES	inductively-coupled plasma atomic emission
ICP-MS	inductively-coupled plasma-emission and mass spectrometry
IDAPA	Idaho Administrative Procedures Act
IDEQ	Idaho Department of Environmental Quality
IDL	Idaho Department Land
IDS	International Directional Services
IPDES	Idaho Pollution Discharge Elimination System
IDWR	Idaho Department of Water Resources
IGES	Intermountain GeoEnvironmental Services, Inc.
IHMA	Important Habitat Management Area
IP	induced potential
IRA	inter-ramp angles
IRR	Internal Rate of Return
IT	information technology
LHLF	Legacy Heap Leach Facility
LoD	Limit of Detection
LOM	Life of Mine
MARC	Maintenance and Repair Contract
MEG	Minerals Exploration and Environmental Geochemistry
MIS	Management Indicator Species
MPO	Mine Plan of Operation

Abbreviation	Description
MTO	Material take-off
ML	Metal leaching
MWMT	meteoric-water mobility tests
NEPA	National Environmental Policy Act
NFS	National Forest Services
NHD	National Hydrology Dataset
NN	Neural Network
NNP	net neutralization potential
NOA	Notice of Availability
NOI	Notice of Intent
NPV	Net Present Value
NSG	north-seeking gyro
NSR	net smelter return
OK	Ordinary Kriging
OMS	Operations, Maintenance, and Safety
org	Organic
OSA	overall slope angle
OT	operational technology
P_{80}	the theoretical square screen-opening, through which 80 weight percent of the particles will pass.
PFS	Pre-Feasibility Study
PHMA	Priority Habitat Management Area
POC	Point of Compliance
PoO	Plan of Operations
QA	quality assurance
QA/QC	quality assurance and quality control
QAL	Quaternary
QC	quality control
QP	Qualified Person
QQ	quantile-quantile
QXRD	Quantitative X-Ray Diffraction
RC	reverse-circulation drilling method
RF	Random Forest
RMA	Roberts Mountains allochthon
ROD	Record of Decision
ROM	Run of Mine
RQD	rock-quality designation
RSF	rock storage facilities
SAG	semi-autogenous grinding
SCSE	SAG circuit specific energy
SD	standard deviations
SMC	mill comminution
SO_4	Sulfate
SPLP	Synthetic Precipitation Leaching Procedure
SRCE	Standardized Reclamation Cost Estimator
SRG	surface-recording gyro

Abbreviation	Description
T	Township
TCC	total constructed cost
TOC	total organic carbon
tot	Total
TSX	Toronto Stock Exchange
USACE	U.S. Army Corps of Engineers
roUSFS	United States Forest Service
USGS	United States Geological Services
WETLAB	Western Environmental Testing Laboratories
VCIC	vertical carbon in column
VO	Variable Orientation
VQO	Visual quality objective
VWP	Vibrating Wire Piezometer
WOTUS	Waters of the United States
XGB	eXtreme Gradient Boosting
XRF	X-ray fluorescence

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by M3 Engineering and Technology (M3) 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 information 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 certain private properties (Liberty Ranch), an opinion was provided by Parsons, Behle and Latimer dated April 22, 2022. For Split Estate Properties (Idaho state mineral lease and private mineral rights), an opinion was provided by Parsons, Behle and Latimer dated February 3, 2026. 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). The authors have also relied upon hydrogeologic evaluations that were completed by Piteau Associates (Rupp 2026) and climatology and water balance studies that were completed by WSP (Schaper 2026).

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, at the intersection of Utah State Highways 30 and 42 (Figure 4-1). The approximate geographic center of the Black Pine property is 42.082°North latitude and 113.047°West longitude.

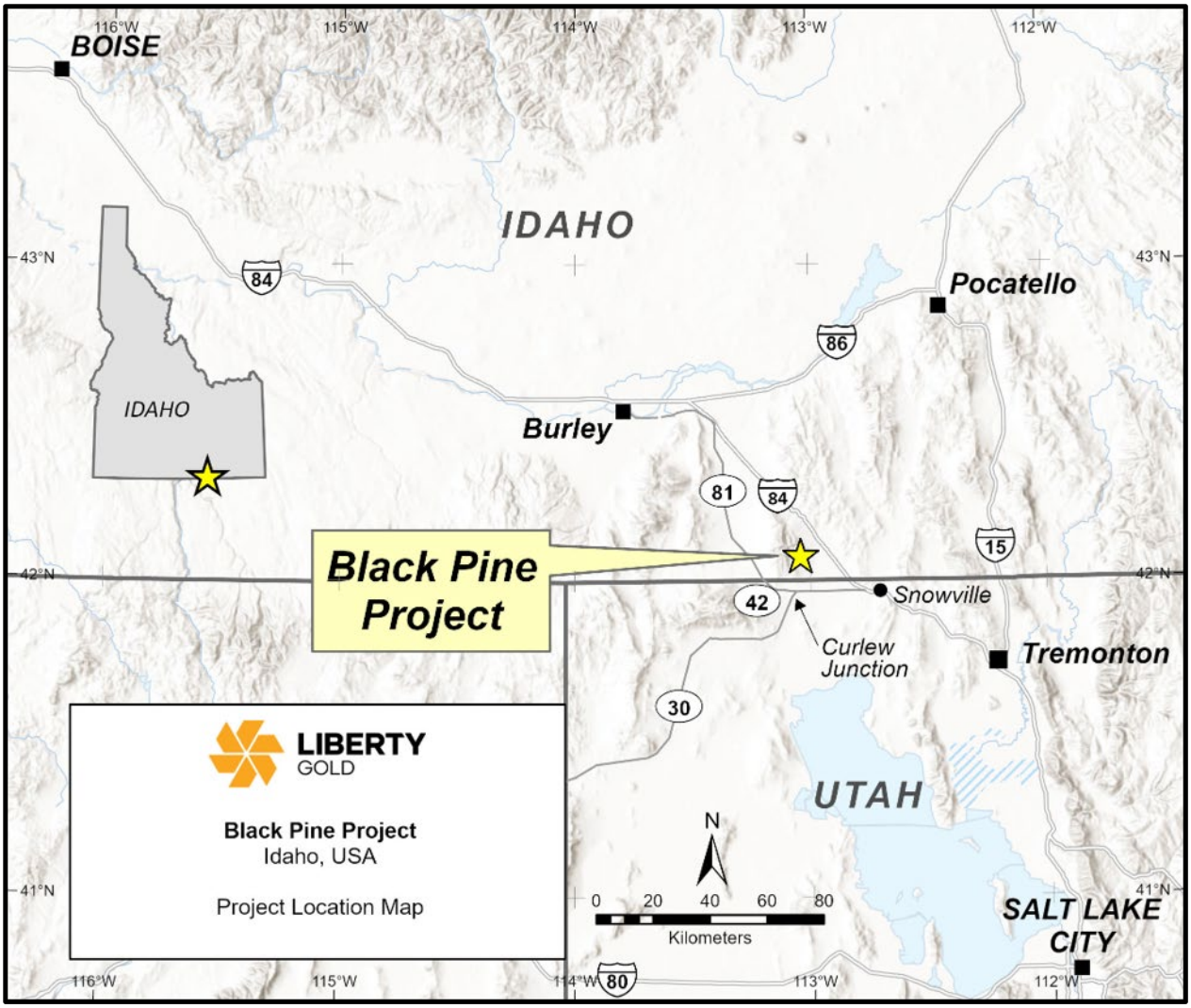


Figure 4-1: Location Map

4.2 LAND TENURE

The combined area of the Black Pine property occupies an area of 15,359 acres (6,215 ha), 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

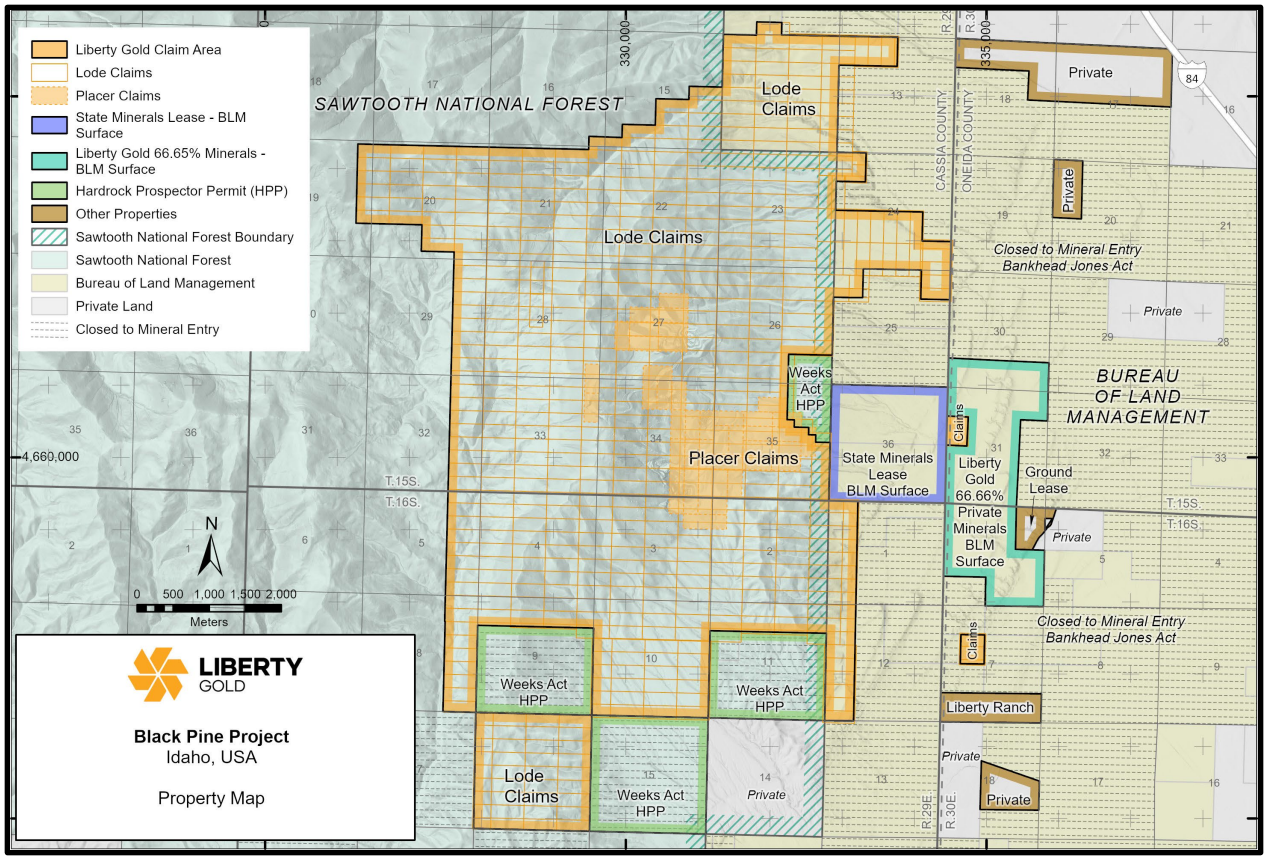
The portion of the Black Pine property on which the Mineral Resource lies consists of a largely contiguous block of 670 unpatented Federal mining claims within Cassia and Oneida counties, Idaho (Figure 4-2). The claims occupy a combined area of 11,904 acres (4,817 ha) as of the Effective Date of this 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 return (NSR) royalty 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. Subsequently, the NSR assignment was subject to a Liberty Gold buyback and assignment of a 0.5% NSR to Wheaton Precious Metals in 2023. Liberty Gold expanded the property by staking 325 unpatented claims between 2016 and 2025, including 45 placer claims over the LHLF, for a total of 670 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 107 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. 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, 2027. County recording fees are also required annually (Table 4-1). The unpatented claims do not expire as long as the Federal and county fees are paid.



Source: Liberty Gold, 2026

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	2025 (US$)	2026 (US$)	2027 Estimated (US$)
Unpatented BLM Claim Fees	$127,200	$134,000	$134,000
County Recording Fees	$83	$83	$83
Total Annual Claim Fees	**$127,283**	**$134,083**	**$134,083**
Section 36 Mineral Lease	$4,661	$4,726	$5,700
Private Property Taxes	$271	$271	$271
Hardrock Prospector Permit	**$880**	**$880**	**$880**
Total Land Holding Costs	**$133,095**	**$139,960**	**$140,934**

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). 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, remains 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 received 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, Liberty 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, Liberty 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, Liberty Gold purchased a 66.65% controlling interest in 911 acres (386.7 ha) of privately held 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 (358 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. In January 2026 Liberty Gold purchased an additional 108.1 acres (43.8 ha) of private land south of the Liberty Ranch. The property is located in Section 18 of T16S R30E. These properties are not included in the Black Pine Property combined area or holding costs.

4.2.7 Ground Lease Agreement

In March 2026 Liberty Gold entered into a Ground Lease Agreement with a private property owner near the project area. The Agreement allows Liberty Gold to place mine-related facilities on the surface. The leased property contains 57.8 acres (23.4 ha) in Sections 5 and 6 of T16S R30E. This property is not included in the Black Pine Property combined area or holding costs.

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 this portion 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

16,415 acres (6,643 ha). Liberty Gold has the option to buy back 0.25% of this royalty for $3,600,000 until January 1, 2030.

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 LHLF or other designated areas.

The LHLF, which lies 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 draindown water containing low-level CN and trace metals at the lower section of the lined heap leach facility, treating it with zero-valent iron in buried concrete vaults, and delivering the treated water to a 100.1-acre (40.5-hectare) land-application area downhill from the leach pad. This land application process following treatment relies on attenuation of residual low-level metals in natural alluvium and was analyzed and ultimately authorized as part of an Environmental Assessment and Finding of No Significant Impact completed by the USFS and BLM, in cooperation with IDEQ. The volume of water produced from draindown is estimated to be 7.5 million gallons annually based on USFS records. The draindown water is sampled two to four times during the land-application period each year, and the USFS provides an annual operational and water-quality monitoring report to the Idaho Department of Environmental Quality (IDEQ, (http://www.deq.idaho.gov/). The heap leach and land-application area are fenced off to preclude cattle access. The USFS and IDEQ hold the remainder of a nominal $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 do not disturb the repository.



Source: Liberty Gold, 2017

Figure 4-3: View of Reclaimed Black Pine 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 allowed for construction of 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 granted access to the historic Black Pine Mine Well (BPMW) and use of public roads on BLM-administered land (case number IDI-039132). Total permitted access included 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 mi^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 (26.9 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 included 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 was received 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, which was completed in 2025.

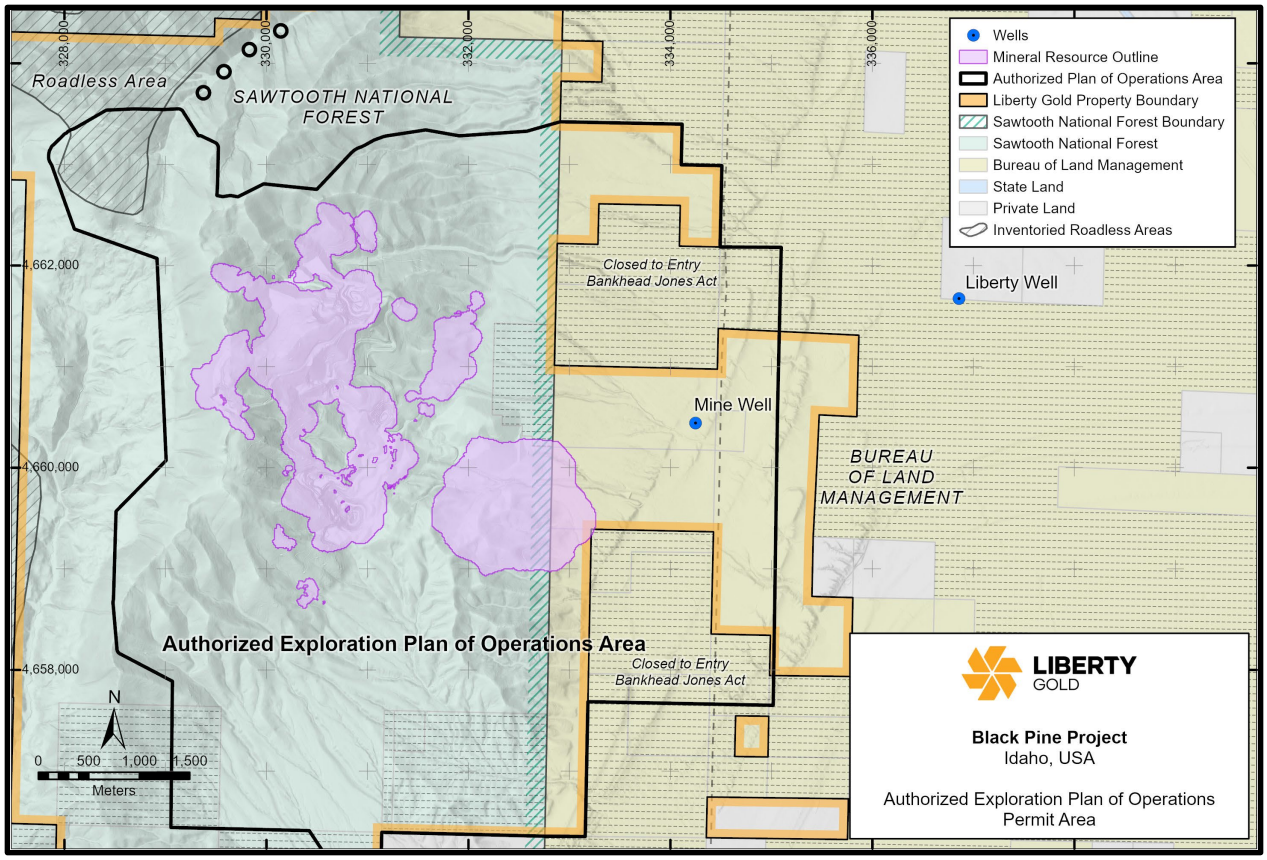
With the approval of the May 2023 modification and Hardrock Prospector Permit in the second half of 2024, combined authorized exploration activity under the USFS and BLM PoO's include 977 drill sites, 100.7 mi (162 km) of drill roads, and 405.7 acres (164.2 ha) of disturbance, over an area of 15.7 mi^2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

Permit areas are summarized in Figure 4-4.

As of June, 2026, there were 341 open drill sites and 319 reclaimed drill sites, 40.6 mi (65.4 km) of open drill roads, 16.6 mi (26.7 km) of reclaimed drill roads, and 149.5 acres (60.5 ha) of open disturbance and 66.8 acres (27.0 ha) of reclaimed disturbance for a total 216.2 acres (87.5 ha) of reclaimed and unreclaimed disturbance.

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 exploration PoO. At present, drilling on USFS land is restricted to the months between July 1 to February 28 within 2 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 $636,687 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 $5,127,087.



Source: Liberty Gold, 2026

Figure 4-4: Authorized Exploration Plan of Operations 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 right 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 BPMW. 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 BPMW, 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, Liberty Gold USA successfully purchased these two rights out of the historic Pegasus (Black Pine Mining Company) 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 statutory surface water rights through its acquisition of the private ranch described in Section 4.2.3.

In January of 2026, through the purchase of the Higley Ranch, an additional statutory agricultural surface water right of 5,067 afa was obtained.

Also, in January of 2026 a surface water lease was signed with the owners of the Anderson Farm, immediately east of the project area, for a total of 287 AFA of agricultural groundwater water rights.

In April of 2026 an adjudication claim (ownership claim) was made for the historic Duvall mining water rights (94.1 afa) the claim also transferred the point of diversion from the historic Duvall well to the BPMW.

In total, Liberty Gold has secured, through purchase and lease agreements, access to 8,655.9 afa of water rights. While the total amount of agricultural water rights may be reduced when transferred to mining use, the resulting water volumes available will be sufficient for the current MPO and 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 on 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 mi (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.

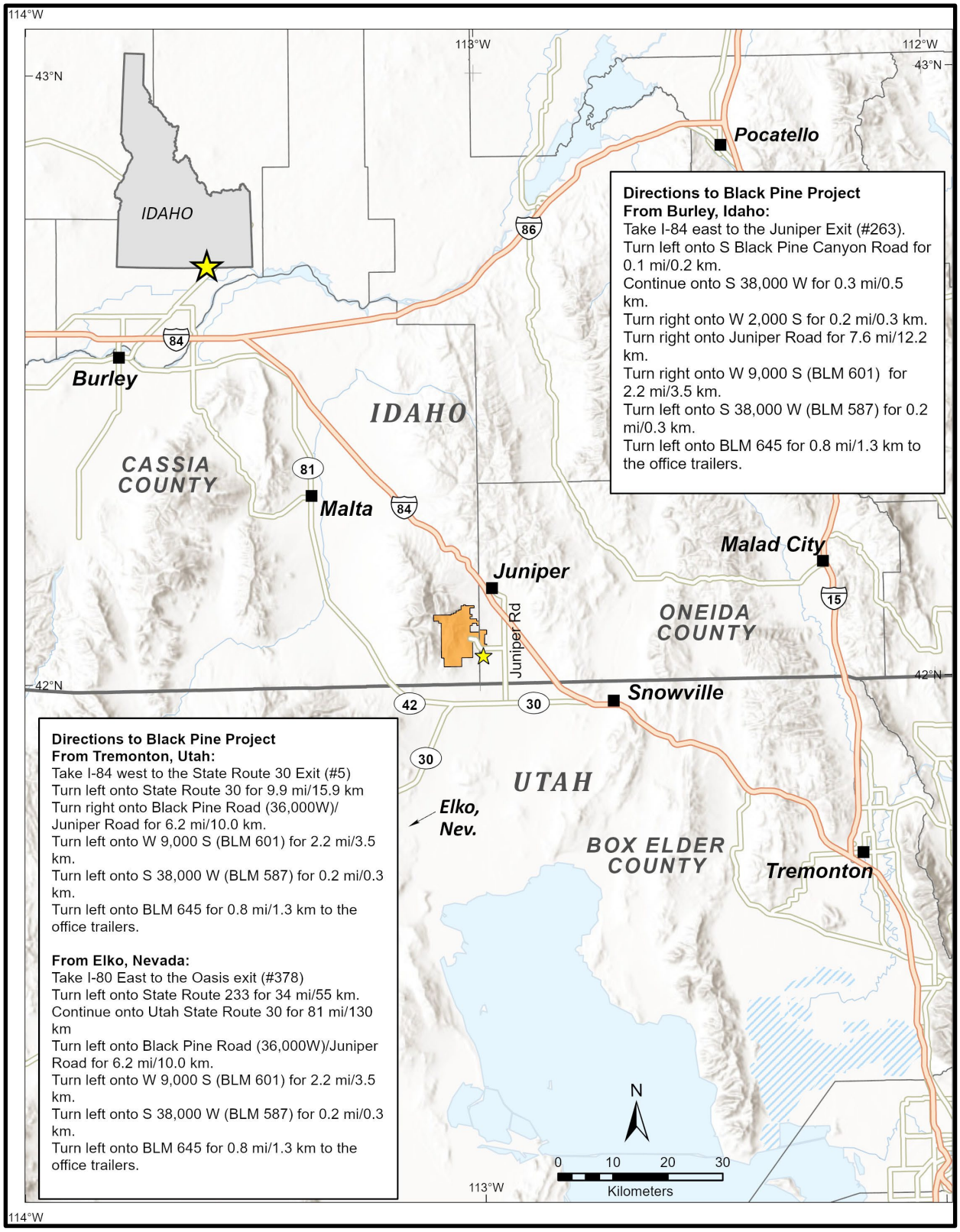
5.2 CLIMATE

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, forbs, and sagebrush. In the higher elevations, increased moisture allows for the growth of juniper, piñon, mountain mahogany, and (locally on steep, north-facing slopes), spruce. Wildfires in 2006-07 denuded much of the range and only a few scattered patches of trees remain on the property.



Source: Liberty Gold, 2026

Figure 5-1: Access Map

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The small agricultural community of Snowville, Utah, located about 18 mi (28 km) to the southeast is the nearest town to the Project. Basic lodging, fuel, and some supplies are available (Figure 5-1). Burley, Idaho, the seat of Cassia County, 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 BPMW, used during the previous mining operations, was drilled in the SE1/4 of the NE1/4 of Section 36, T15S R29E, with water encountered in alluvium just below 5,000 ft (1,524 m) in elevation, or approximately 185 ft 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). All Liberty Gold activities are described in subsequent sections of this report.

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 the Tallman open pit from 1949 to 1955 (Prochnau, 1985). Total production was reported to be 98,884 tonnes (120,000 tons) with an average gold grade of 5.14 g/t Au (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 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/t Au. 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. Following the bankruptcy in early 1998 of Black Pine Mining Inc, the reclamation bond was voluntarily surrendered and the USFS finished the remaining reclamation on 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 modeling.

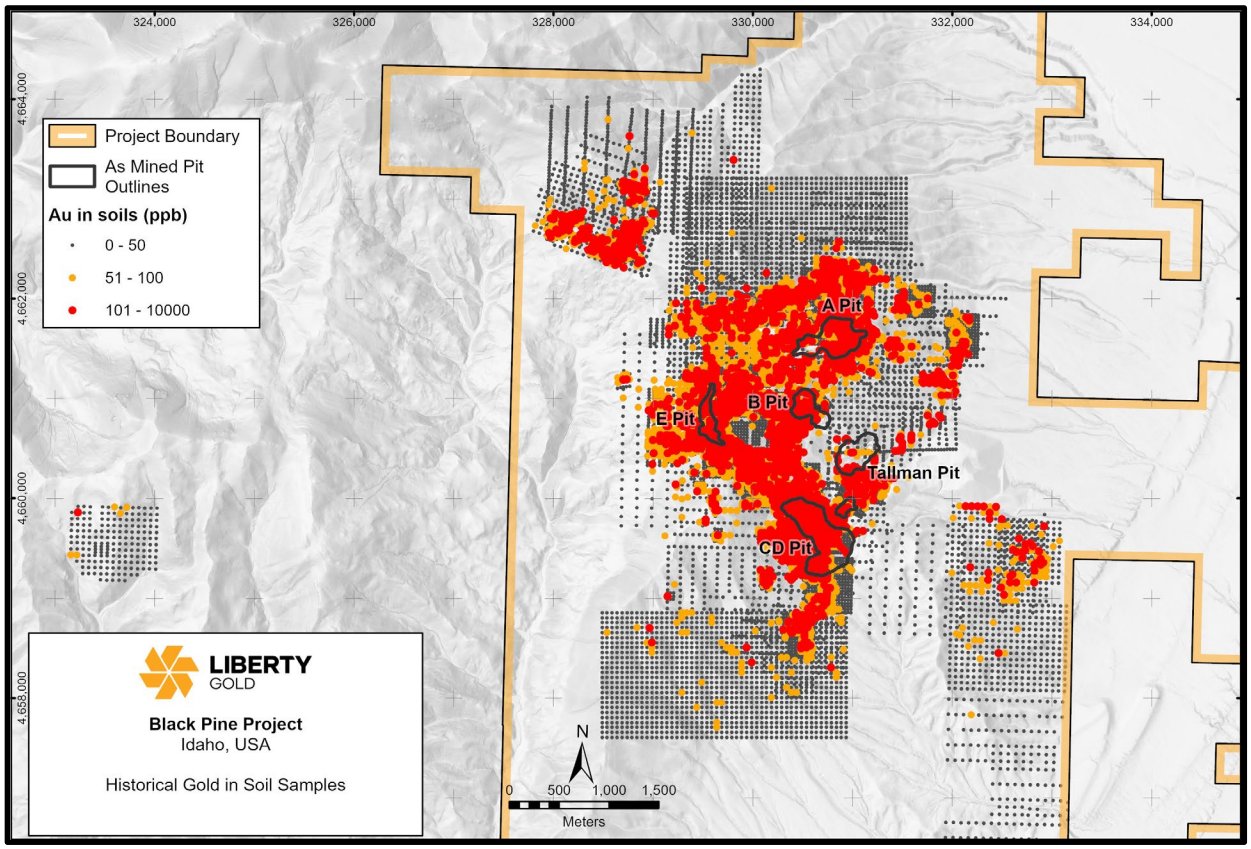
6.3 HISTORICAL SOIL SAMPLE AND STREAM SEDIMENT DATA

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/t Au and 3,205 samples that assayed greater than 0.100 g/t Au. These samples principally form a broad, diamond-shaped anomaly over the historical mine area, of approximately four km north-south by about 3 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.



Source: Liberty Gold, 2026

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

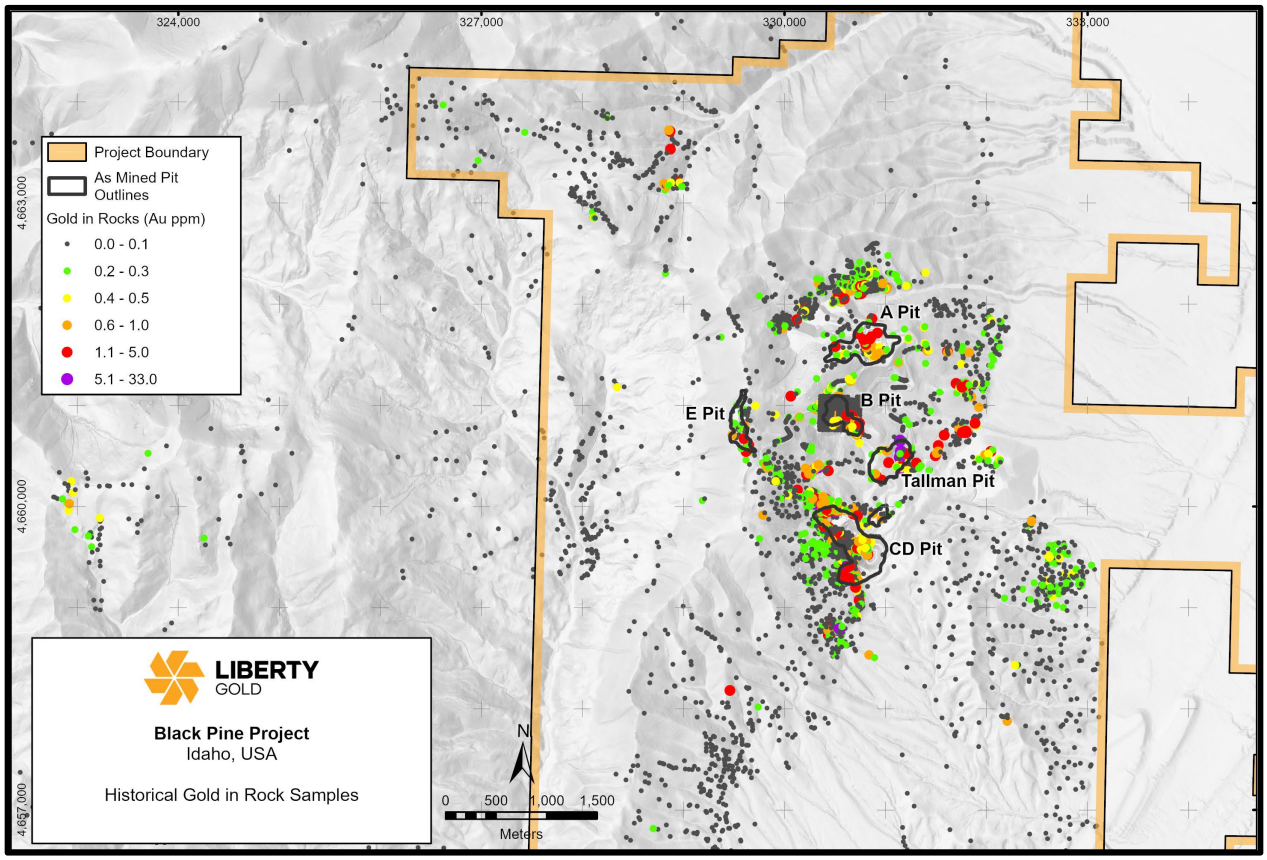
Stream-sediment surveys were carried out by previous operators across the broader Black Pine Mountains as part of a regional exploration effort. This data is not presently being used by Liberty Gold, as it has largely been superseded by soil and rock data.

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/t Au, 168 were in excess of 1 g/t Au, and 19 samples assayed greater than 5 g/t Au. The presently compiled historical gold results from rock samples are shown in Figure 6-2.



Source: Liberty Gold, 2026

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 1,877 holes at Black Pine, not including 34 conventional rotary or RC holes drilled by ASARCO in 1977, for which no data was found (Table 6-1). Approximately 99% of the holes and meters were drilled using conventional rotary and RC methods, and 24 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.

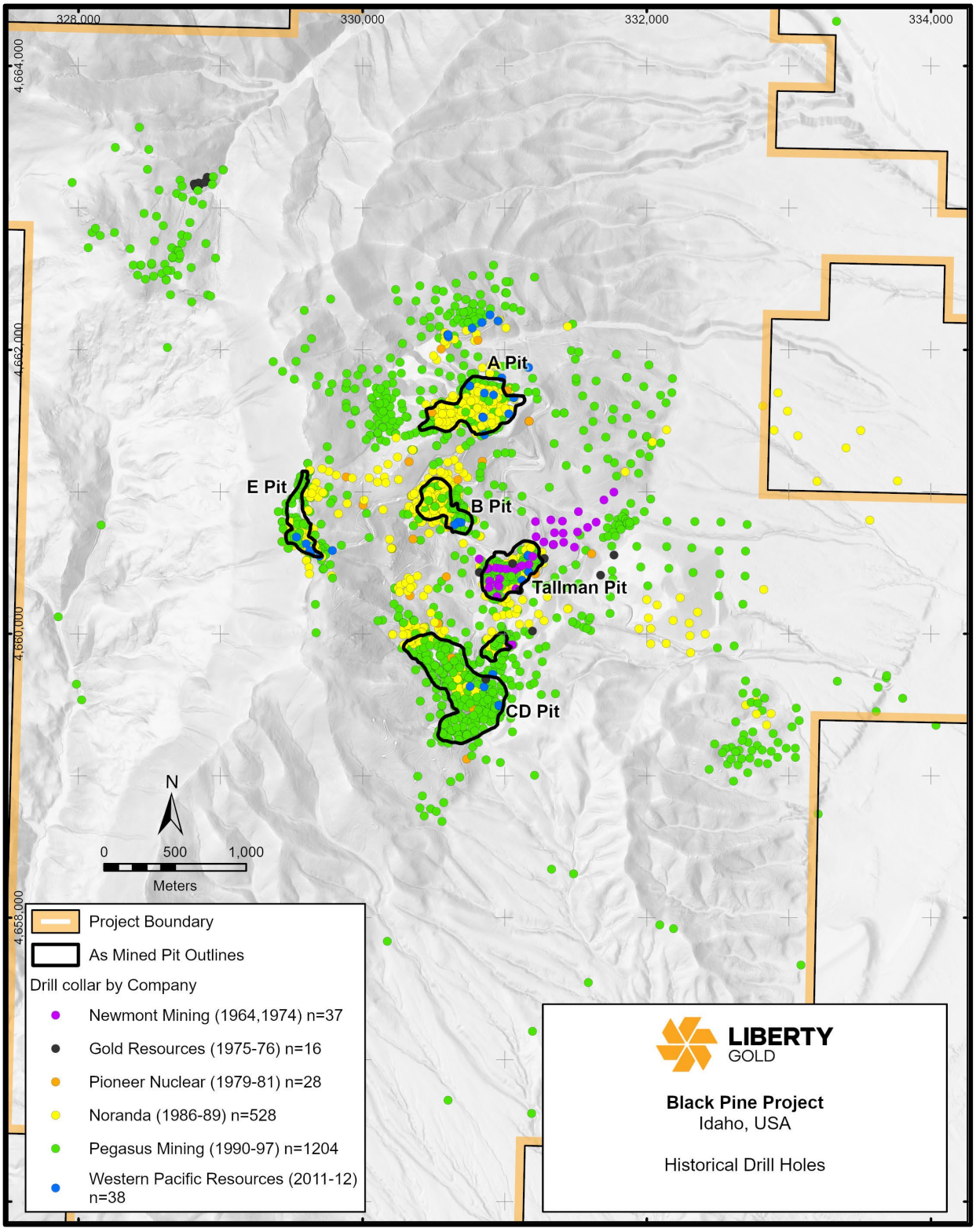
Table 6-1: Summary of Black Pine Project Historical Drilling

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)
Note:
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.



Source: Liberty Gold, 2026

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.

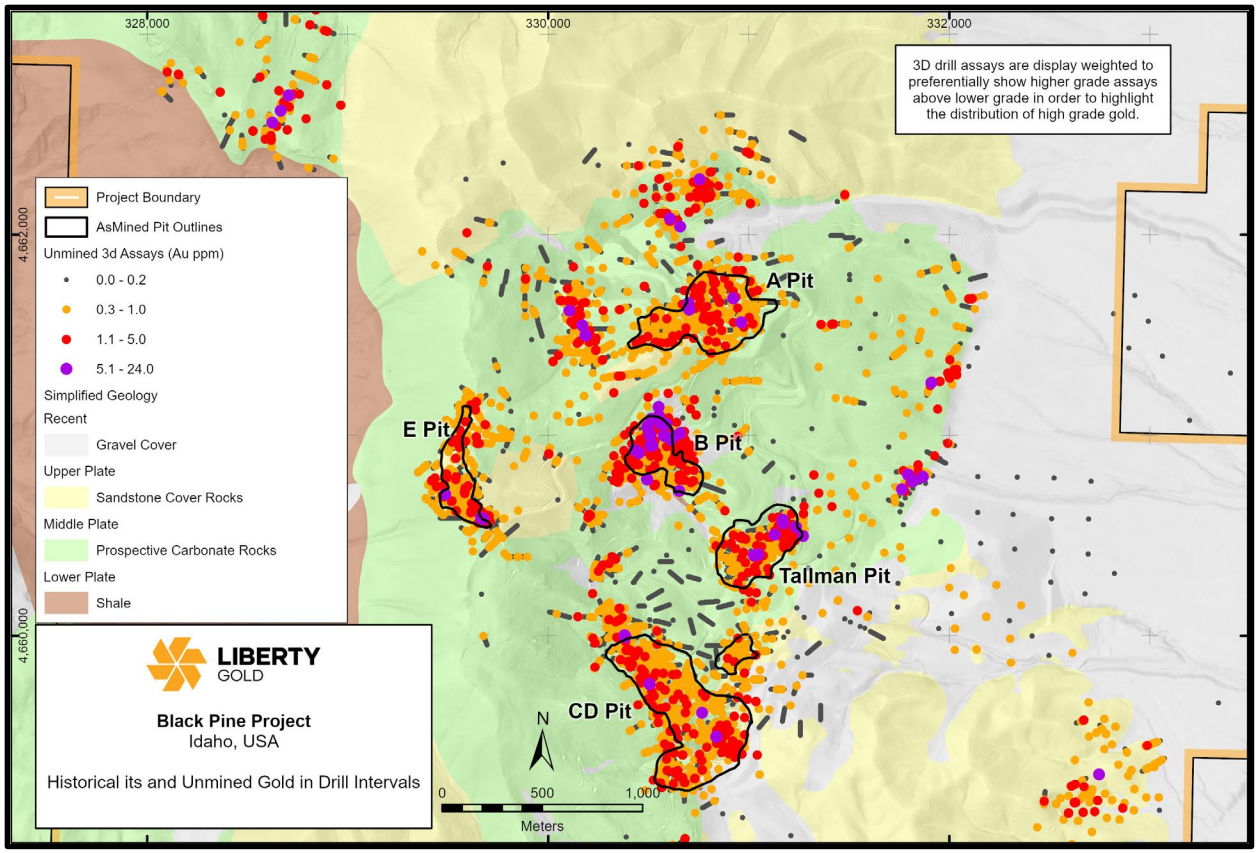
The historical drilling discovered and defined gold mineralization that was eventually mined from seven historical open pits. These pits produced approximately 435,000 oz of recovered gold from more than 30 Mt of ore between 1991 and 1997. The pits lie within mineralized zones of various sizes, and only 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/t Au 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 zone, Backrange, and Rangefront areas (Figure 6-4).



Source: Liberty Gold, 2026

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-m (5-ft) intervals. Drilling was carried out by Sprague and Henwood, Inc. with 14.3 cm tricone and 12.1 cm 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-m 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 area, 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 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 rotary 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 ft), 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 ft) or 6.1 m (20 ft) (<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.

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.

These Mineral Resource estimates are historical in nature, however, they are relevant as they indicate the past presence of 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.

6.6.1 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/ton Au, 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/ton Au. 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. In 1989 Noranda updated an 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"
Grade reported in ounces per short ton.

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).
3. Gold grades reported as ounces per short ton.

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/t Au, with "additional mineralized material" that totaled 1.7 Mt at a grade of 0.46 g/t Au (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 LHLF

Powell (2012a) of Tetra Tech, prepared an internal report for Western Pacific Resources, with an estimate of the remaining gold in the LHLF, 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 LHLF.

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 estimate should not be relied upon.

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, 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 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.

6.8 SUMMARY STATEMENT

The QP has not analyzed the historical 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.

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 kilometers 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.

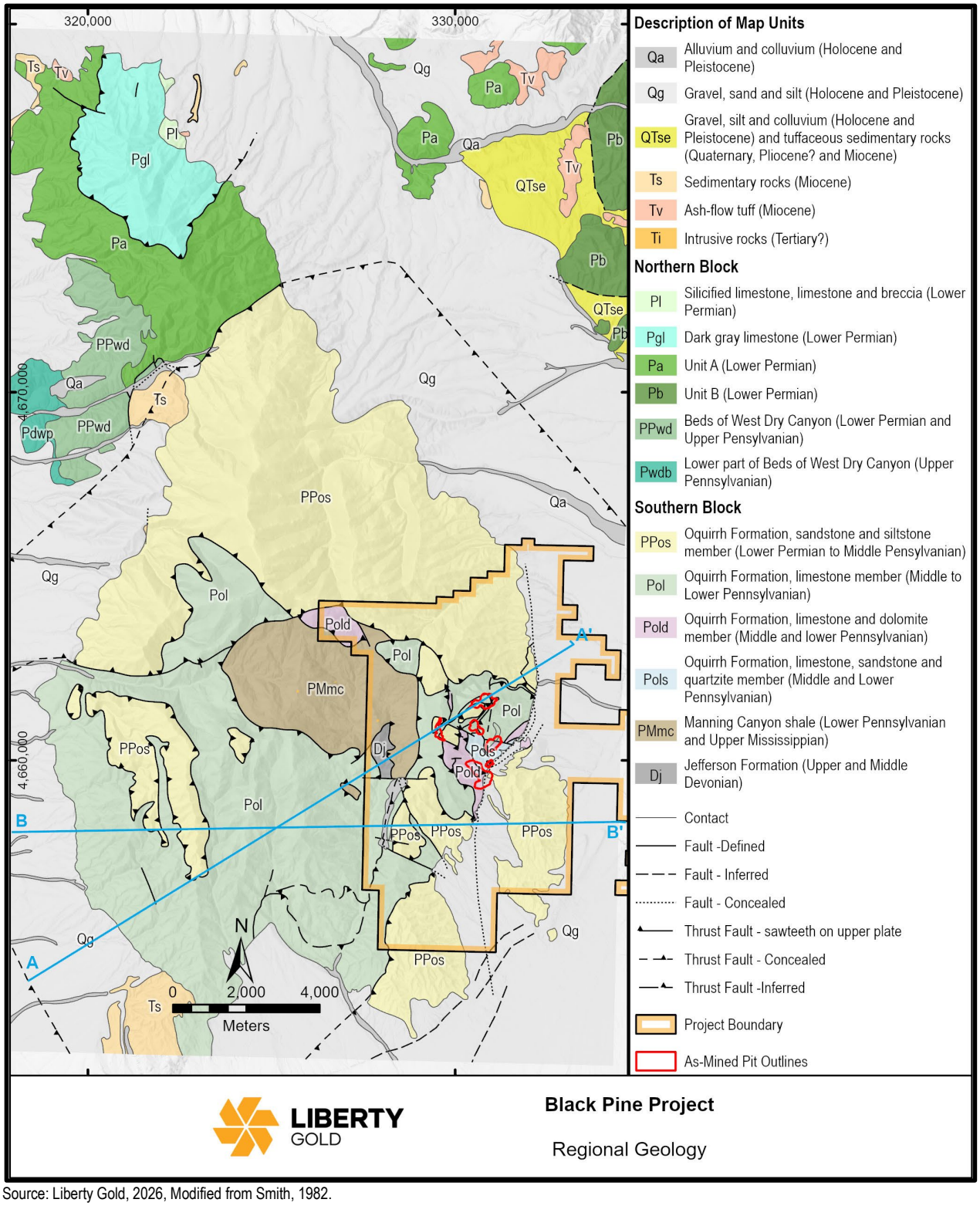
The latest manifestations of extension are “Basin and Range” style block faults that divide the Great Basin into its characteristic horsts and grabens. Some of these faults are still active today.

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 lower plate is believed to structurally overlie a basement of weakly metamorphosed rocks of suspected Cambro-Ordovician age (Smith, 1982; Figure 7-2).

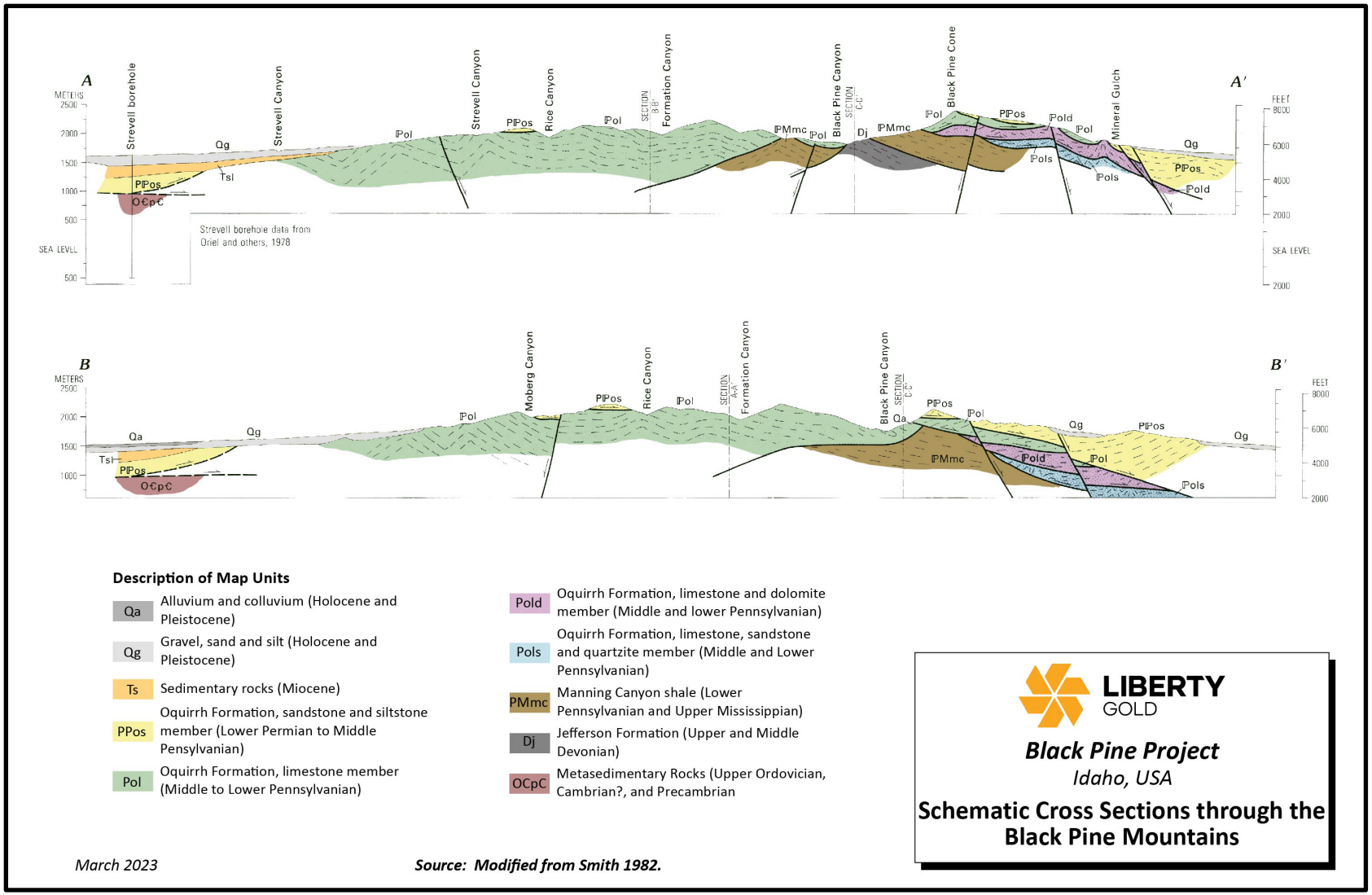
The northern block is comprised of two thrust plates. The lower thrust plate consists of four informally-named stratigraphic units, ranging from Late Pennsylvanian to Early Permian in age, probably corresponding to the upper portion of the Oquirrh Formation. The upper plate consists of limestone and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



Source: Liberty Gold, 2026, Modified from Smith, 1982.

Figure 7-1: Regional Geology



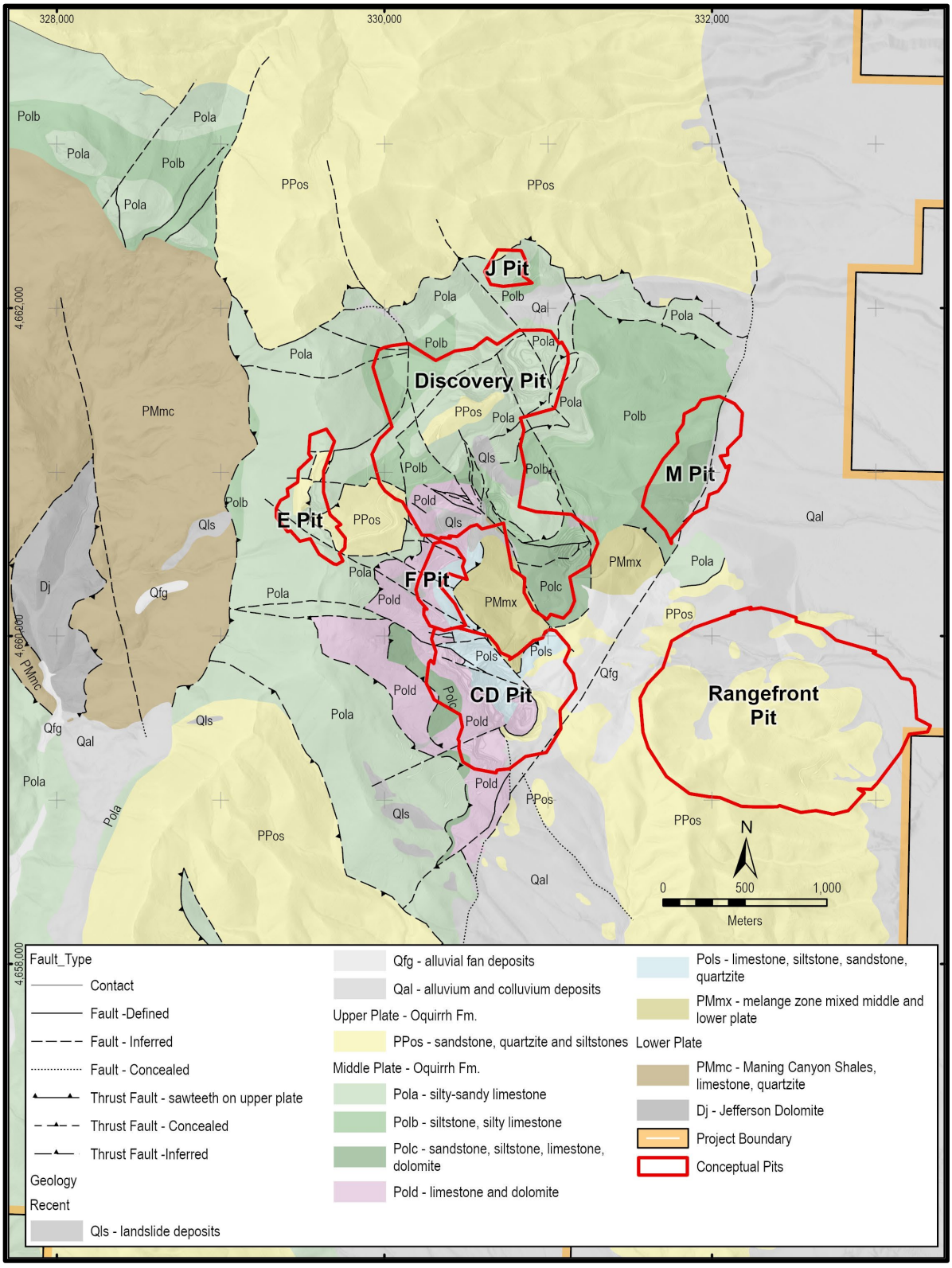
Source: Liberty Gold, 2026, Modified from Smith, 1982.

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).



Source: Liberty Gold, 2026

Figure 7-3: Geologic Map

7.2.1 Stratigraphy

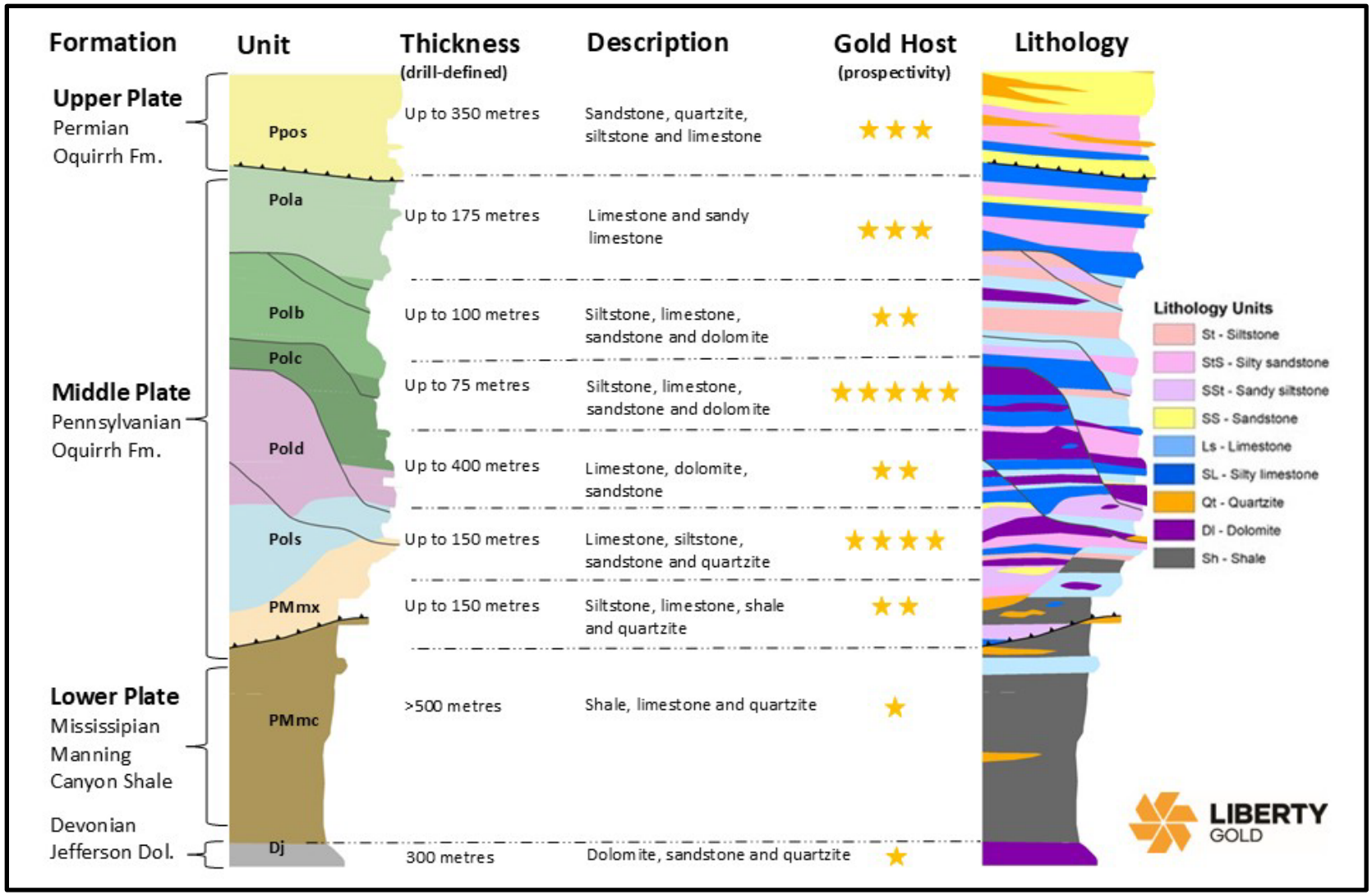
The stratigraphy in the Project area records the transition from the top of the Devonian shelf and platform, through foreland-basin sedimentation associated with the mid-Paleozoic Antler orogeny, to basin and platform conditions that persisted throughout much of the late Paleozoic era. Figure 7-4 illustrates 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 Mountains, although interleaving and attenuation of the section by low-angle faults makes stratigraphic analysis extremely difficult. In the southern structural, 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).



Source: Liberty Gold, 2026

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 Project 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. Polb can also contain locally strong zones of carbonaceous material.

- 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 at 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 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. Ppos in Rangefront can also host mineralization and appears to be in a stratigraphic contact with the underlying Pola.

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 an alteration product. 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 Project 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 clast/calcite cemented 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. Pervasive 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 dike 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 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 episodes.

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.

This same fine-grained, intrusive rock is also noted at the lower plate contact where it is typically unoxidized. These dikes 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 ICP-MS. 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)

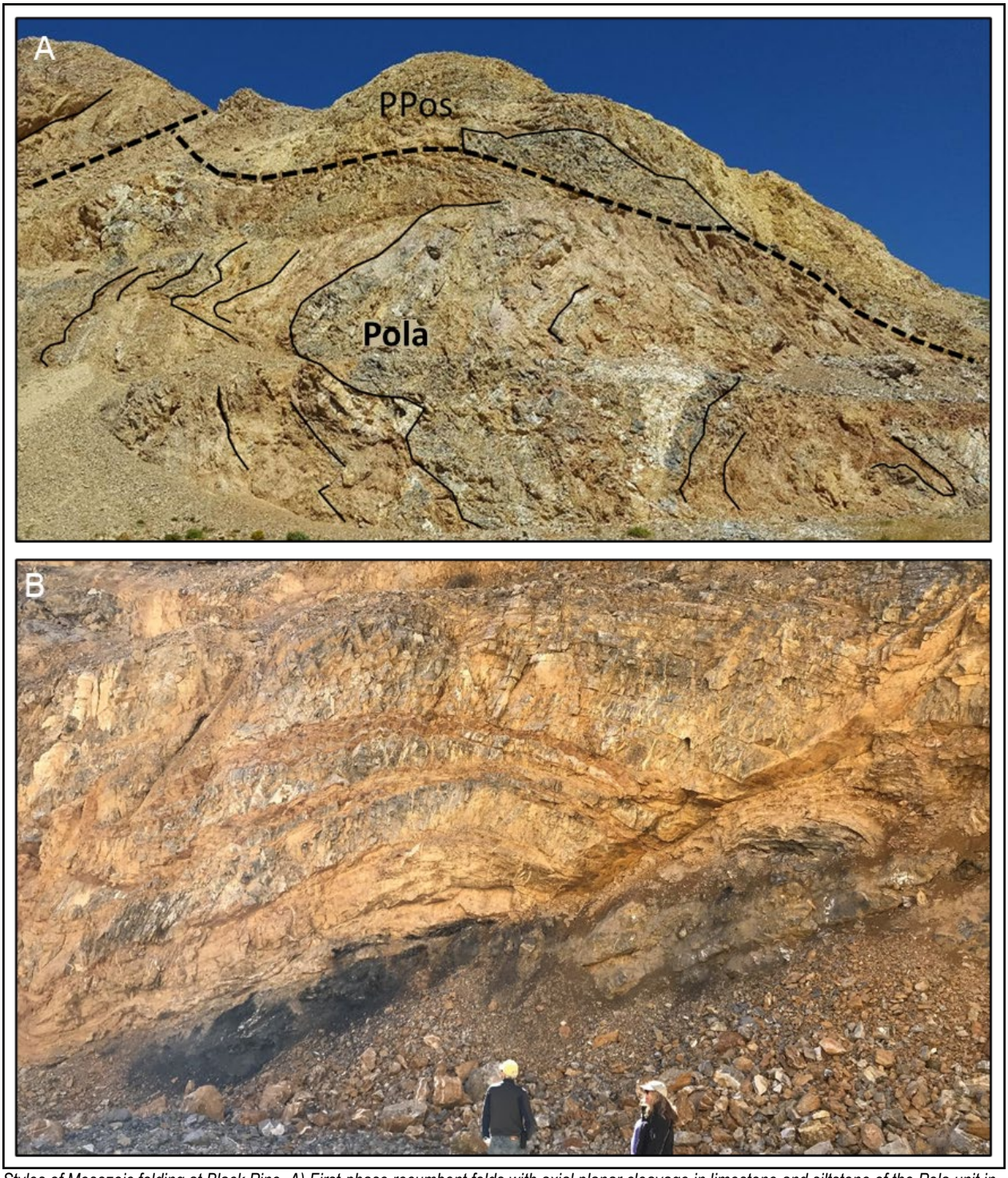
A polyphase, Mesozoic contractional event or events strongly affected rocks in the Black Pine Mountains.

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. (Source: Liberty Gold, 2020)

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 Mountains, including:

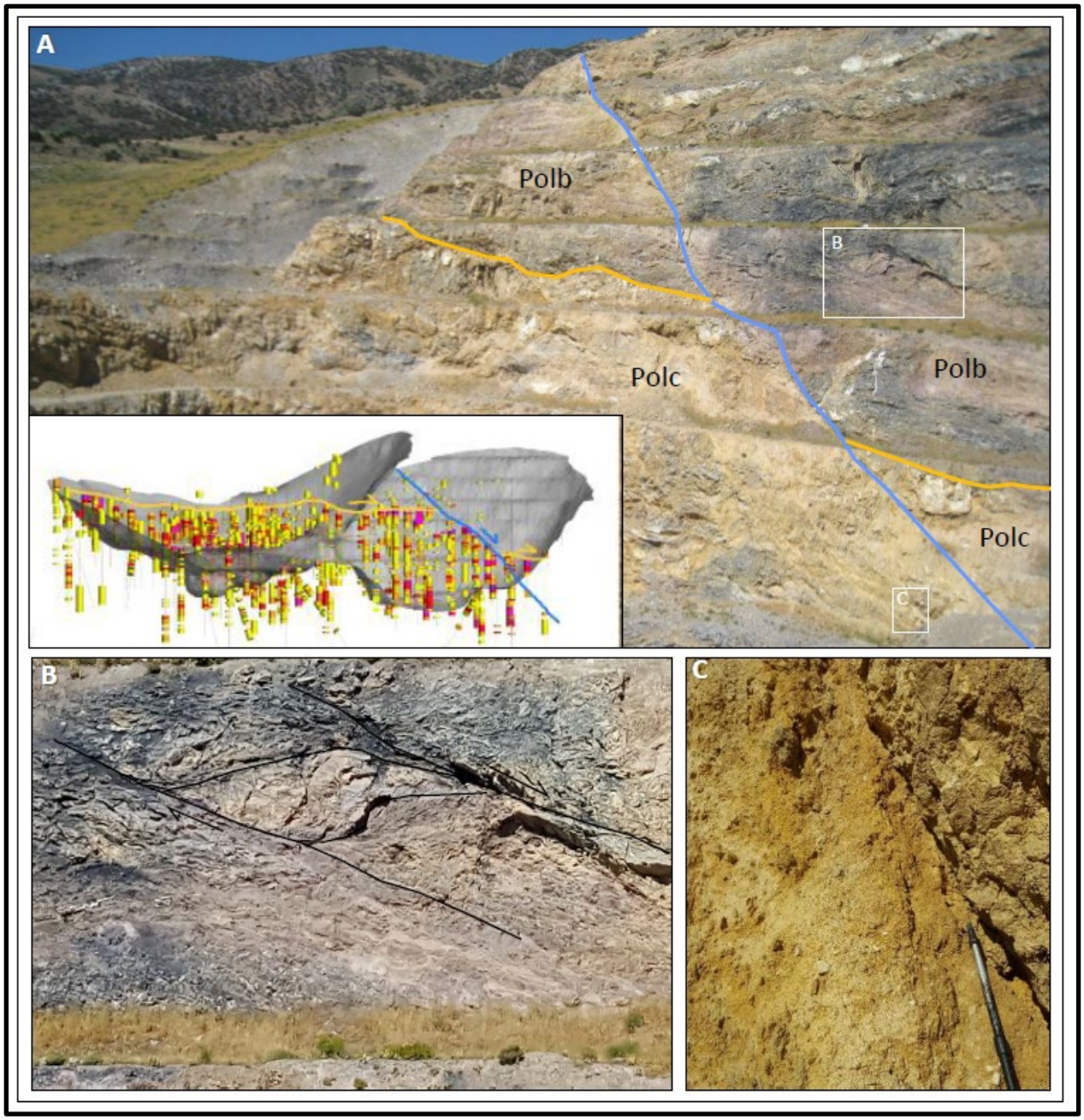
- Eocene listric normal faulting and volcanism (likely timing of gold mineralization)
- Oligocene extension, deep metamorphism and plutonism
- Miocene unroofing of the Albion-Raft River metamorphic core complex along low angle detachment faults
- Basin and Range block faulting

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 several 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 Discovery 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. A second, north-south striking listric normal fault is interpreted to underlie the Discovery Zone and may extend as far south as the CD Pit. It is interpreted to join the first listric fault to the north. The fault has a relatively steep orientation along the west side of the Discovery 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 Project 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. (Source: Liberty Gold, 2020)

Figure 7-6: Examples of Normal Faults in the Tallman 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. (Source: Liberty Gold, 2020)

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 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 Project area is weakly to strongly hydrothermally altered and can 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, also known as “sanding” 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:* Weak 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. Several large areas of barren jasperoid are present peripherally to the mineralized zones in Rangefront and Discovery zones.

- *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, in proximity 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 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 Project area as veins, veinlets, breccia cement and breccia clasts. In some cases, calcite forms both the clasts and cement in 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 also present in the upper plate in the Rangefront area, where the PPos unit is more calcareous than elsewhere. 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.

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. Unlike most Carlin style mineralization there is high silver in the deposit (~4:1 Ag:Au ratio). 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 assays (Figure 7-8) shows gold correlating most strongly with mercury and moderately with antimony, thallium, tellurium and arsenic, typical strong Carlin-style pathfinder elements. There are no recognized alterations or spatial patterns to these positive or negative correlations between gold and the trace elements. Silver anchors an independent polymetallic suite with strong correlations between silver and copper, zinc, lead, selenium and bismuth. Areas of high silver, lead, copper, bismuth and 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.

Correlation matrix - Pearson r on log10 (ppm), Au > 0.1 g/t intervals

n = 63,329 intervals

Element	Au	Ag	As	Ba	Bi	Cu	Hg	Pb	Sb	Se	Te	Tl	Zn
Au	1.00	0.30	0.28	0.16	0.18	0.21	0.54	0.21	0.22	0.21	0.45	0.14	0.22
Ag	0.30	1.00	0.58	0.15	0.60	0.71	0.54	0.73	0.58	0.57	0.53	0.18	0.59
As	0.28	0.58	1.00	0.10	0.63	0.67	0.40	0.52	0.65	0.48	0.48	0.45	0.51
Ba	0.16	0.15	0.10	1.00	0.02	0.11	0.30	0.20	0.24	0.08	0.06	0.27	0.20
Bi	0.18	0.60	0.63	0.02	1.00	0.73	0.23	0.57	0.44	0.57	0.55	0.27	0.37
Cu	0.21	0.71	0.67	0.11	0.73	1.00	0.38	0.64	0.56	0.52	0.51	0.28	0.56
Hg	0.54	0.54	0.40	0.30	0.23	0.38	1.00	0.50	0.47	0.26	0.44	0.22	0.49
Pb	0.21	0.73	0.52	0.20	0.57	0.64	0.50	1.00	0.59	0.35	0.39	0.28	0.60
Sb	0.22	0.58	0.65	0.24	0.44	0.56	0.47	0.59	1.00	0.31	0.46	0.45	0.56
Se	0.21	0.57	0.48	0.08	0.57	0.52	0.26	0.35	0.31	1.00	0.49	0.07	0.37
Te	0.45	0.53	0.48	0.06	0.55	0.51	0.44	0.39	0.46	0.49	1.00	0.16	0.34
Tl	0.14	0.18	0.45	0.27	0.27	0.28	0.22	0.28	0.45	0.07	0.16	1.00	0.24
Zn	0.22	0.59	0.51	0.20	0.37	0.56	0.49	0.60	0.56	0.37	0.34	0.24	1.00

Correlation matrix - Pearson r on log10 (ppm), complete cases

n = 194,355 intervals

Element	Au	Ag	As	Ba	Bi	Cu	Hg	Pb	Sb	Se	Te	Tl	Zn
Au	1.00	0.38	0.45	0.34	0.08	0.28	0.72	0.23	0.51	0.16	0.46	0.49	0.34
Ag	0.38	1.00	0.61	0.33	0.51	0.73	0.56	0.66	0.64	0.65	0.60	0.29	0.68
As	0.45	0.61	1.00	0.23	0.54	0.67	0.49	0.50	0.67	0.48	0.56	0.53	0.56
Ba	0.34	0.33	0.23	1.00	0.03	0.24	0.46	0.30	0.43	0.14	0.20	0.36	0.36
Bi	0.08	0.51	0.54	0.03	1.00	0.68	0.10	0.55	0.24	0.53	0.55	0.21	0.32
Cu	0.28	0.73	0.67	0.24	0.68	1.00	0.41	0.63	0.53	0.56	0.57	0.32	0.59
Hg	0.72	0.56	0.49	0.46	0.10	0.41	1.00	0.47	0.66	0.22	0.45	0.47	0.54
Pb	0.23	0.66	0.50	0.30	0.55	0.63	0.47	1.00	0.55	0.33	0.44	0.31	0.53
Sb	0.51	0.64	0.67	0.43	0.24	0.53	0.66	0.55	1.00	0.32	0.50	0.55	0.62
Se	0.16	0.65	0.48	0.14	0.53	0.56	0.22	0.33	0.32	1.00	0.54	0.09	0.49
Te	0.46	0.60	0.56	0.20	0.55	0.57	0.45	0.44	0.50	0.54	1.00	0.30	0.46
Tl	0.49	0.29	0.53	0.36	0.21	0.32	0.47	0.31	0.55	0.09	0.30	1.00	0.35
Zn	0.34	0.68	0.56	0.36	0.32	0.59	0.54	0.53	0.62	0.49	0.46	0.35	1.00

Pathfinder association with Au > 0.1 g/t vs all intervals

Element	r vs Au (>0.1 g/t)	r vs Au (all)	Change
Hg	0.54	0.72	-0.18
Te	0.45	0.46	-0.01
Ag	0.30	0.38	-0.09
As	0.28	0.45	-0.16
Sb	0.22	0.51	-0.29
Zn	0.22	0.34	-0.12
Cu	0.21	0.28	-0.07
Pb	0.21	0.23	-0.02
Se	0.21	0.16	+0.05
Bi	0.18	0.08	+0.10
Ba	0.16	0.34	-0.19
Tl	0.14	0.49	-0.35

Polymetallic suite — association with Ag (Au > 0.1 g/t) vs all intervals

Element	r vs Ag (>0.1 g/t)	r vs Ag (all)	Change
Pb	0.73	0.66	+0.07
Cu	0.71	0.73	-0.02
Bi	0.60	0.51	+0.10
Zn	0.59	0.68	-0.09
As	0.58	0.61	-0.03
Sb	0.58	0.64	-0.07
Se	0.57	0.65	-0.08
Hg	0.54	0.56	-0.02
Te	0.53	0.60	-0.07
Au	0.30	0.38	-0.09
Tl	0.18	0.29	-0.10
Ba	0.15	0.33	-0.18

Source: Liberty Gold (2026)
Note: 63,329 Liberty Gold drill samples containing >0.1 g/t Au

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 (Figure 6-4) around the margins of the Black Pine gold system. This type of mineralization is typically associated with the lower plate as well as steep faults, with 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 with some samples returning >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 mineralizing 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/I 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. For example, historical “reserves” disclosed in Section 6.6 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 now recognized at Black Pine, as defined through historical and Liberty Gold drilling.

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, and M, 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, and Burnt Basin remain a priority.

8 DEPOSIT TYPES

Black Pine mineralization is best assigned to 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 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.

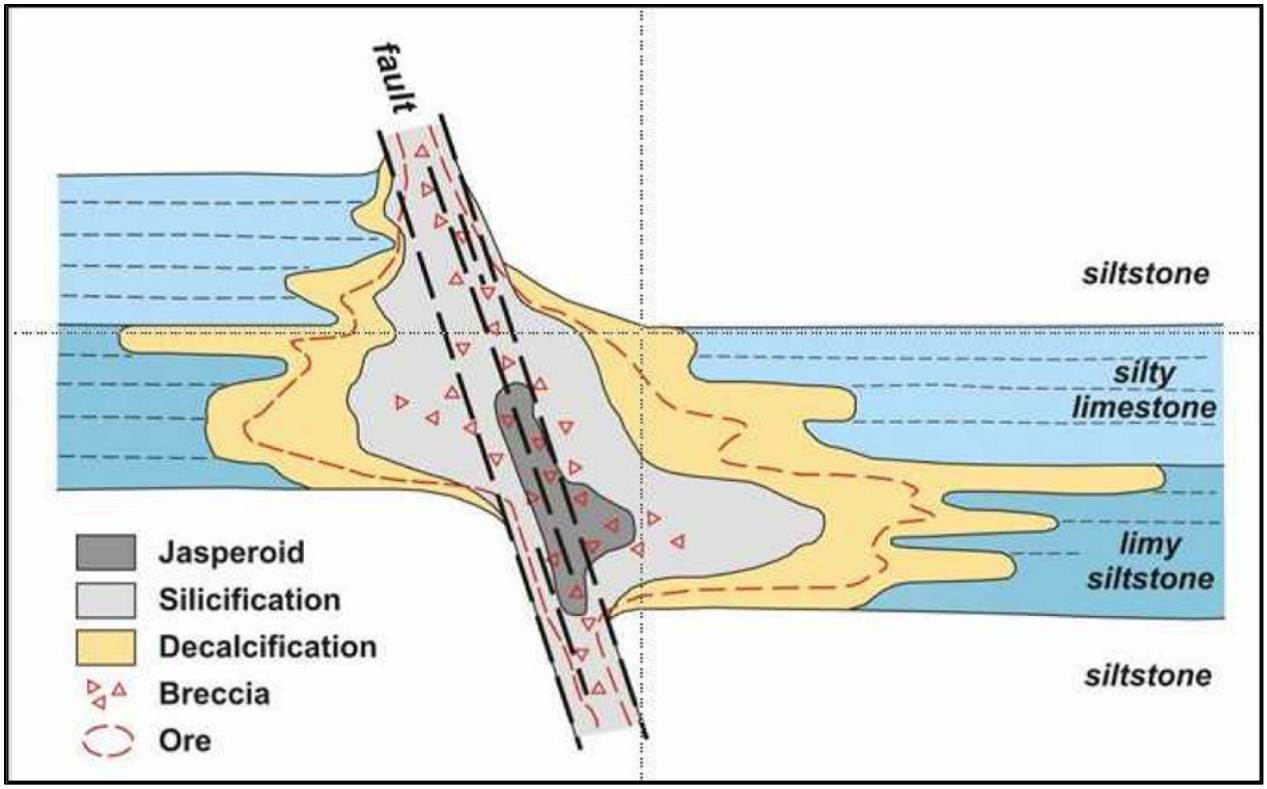
Deposit configurations and shapes are quite variable. CTGDs are typically somewhat stratiform in nature, with mineralization largely confined within specific favorable stratigraphic units. Fault and solution-collapse breccias can also be primary hosts to mineralization (Figure 8-1).

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. For example, the general location of the Project is outside the major gold deposit trends in Nevada and the younger age of host rocks.

Using the CTGD model as a guide, exploration at Black Pine has been focused on targeting zones of favorable stratigraphy primarily calcareous siltstone or silty limestone in Pola, Polb, Polc and sections of Ppos units where they intersect moderate to lower angle fault zones. Exploration has also been driven by surface geochemical anomalies in soils and rock chip sampling where elevated CTGD 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). Additional details and maps from Liberty Gold surface exploration work through 2021 can be found in Gustin et al. (2021).

9.1 COMPILATION OF HISTORICAL DATA 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 and no historic core or RC chips remain.

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

- Assembly and verification of raw data export files of drill-hole data into a coherent Access database. 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.

- Conversion of historical mine-grid coordinates into the UTM NAD 83, Zone 12 coordinate system. 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 m about one kilometer away, to 3 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.

- Verification of historical collar locations and surface samples after coordinating transformation. 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.

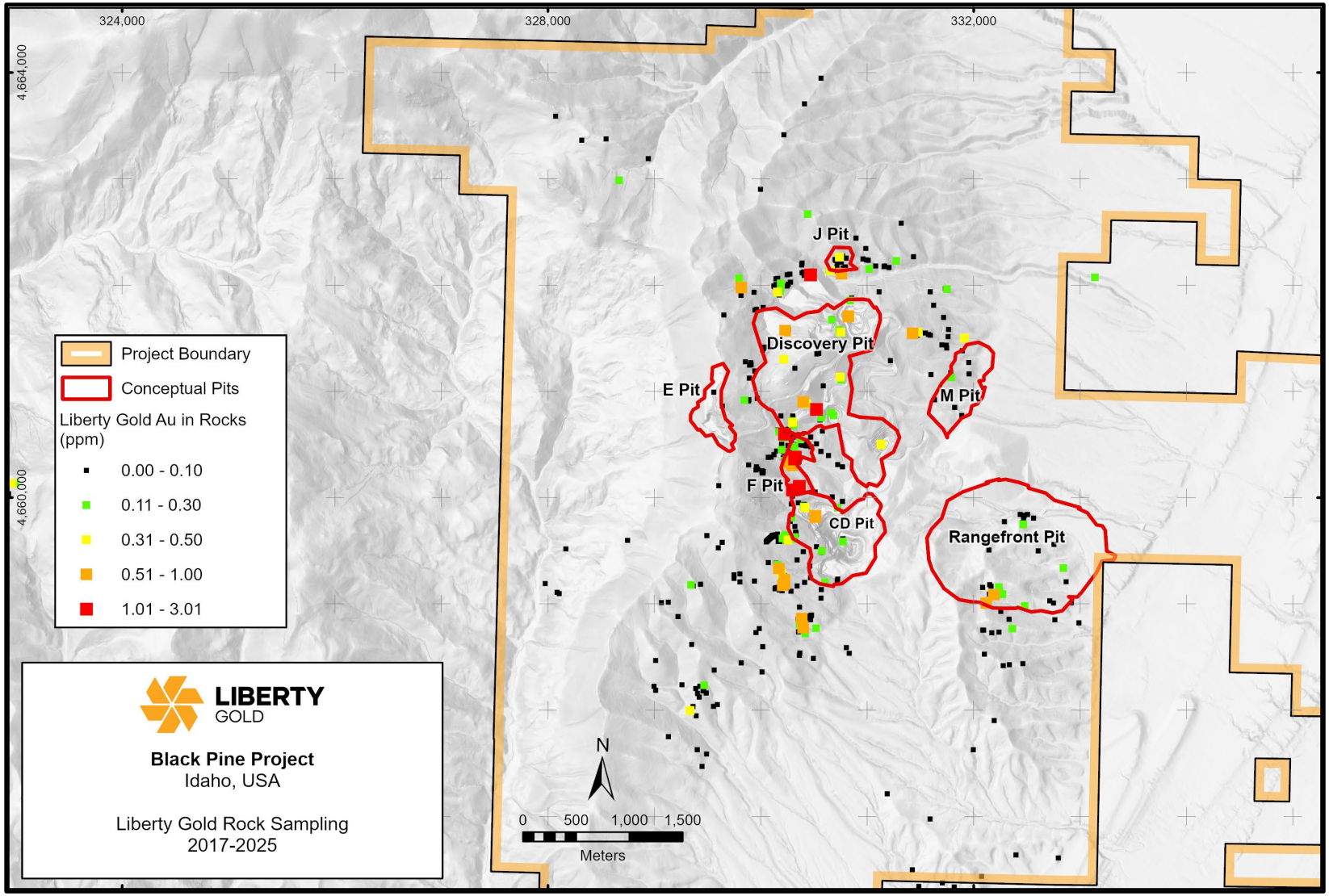
- Geologic map compilation. Surface geological maps created by Noranda were not updated significantly during the Pegasus operations. The Noranda map by Ohlin (1989) is still the best available historical property-scale geological map. Registration, digitization, and spot checking of Ohlin's map have 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 modeling 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, 771 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, Rangefront, and other various locations across the property (Figure 9-1). Gold assays range from below detection limit to a high of 3.01 g/t Au. 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.



Source: Liberty Gold, 2026

Figure 9-1: Liberty Gold Rock Sampling, 2017-2025

9.3 LIBERTY GOLD SOIL SAMPLING

Liberty Gold completed two north-south and east-west oriented soil samples lines over the Rangefront deposit area consisting of 117 samples. Samples were spaced at 50 m and the lines were designed to cross the deposit where it was outcropping as well as in areas of barren upper plate Ppos cover. The purpose of these soil samples was to define a geochemical signatures in soils that might indicate underlying mineralization.

9.4 LIBERTY GOLD GEOLOGIC MAPPING

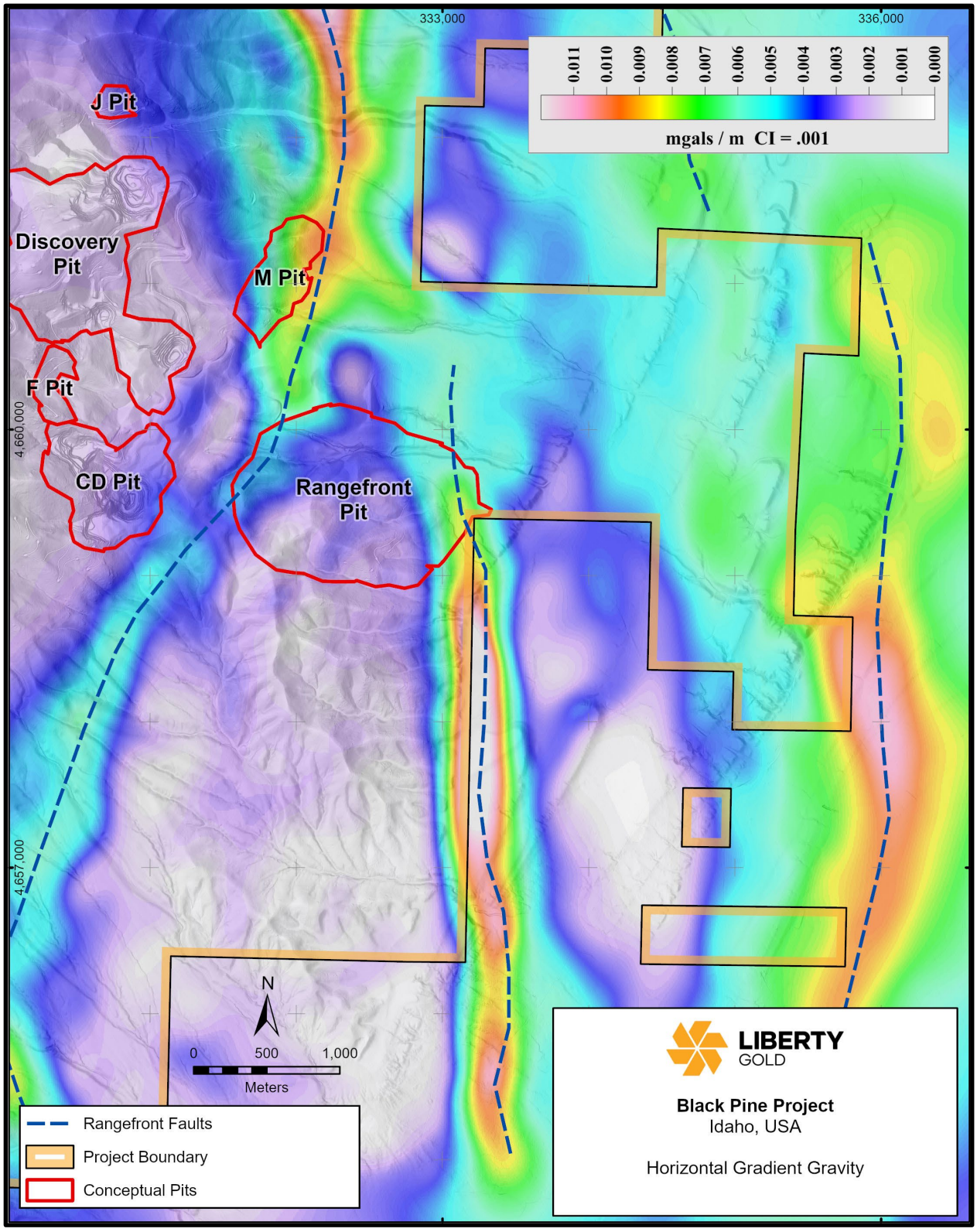
Liberty Gold has carried out geologic mapping at various times throughout the life of the Project, primarily by April Barrios, William Lepore, Randy Hannink, Moira Smith, and consulting geologist Tracy Dembrowski. Geologic mapping was facilitated using a combination of digital pads (ArcPad, Field Maps, FieldMove Clino etc.) and paper maps, and has been integrated by former 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.

9.5 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.

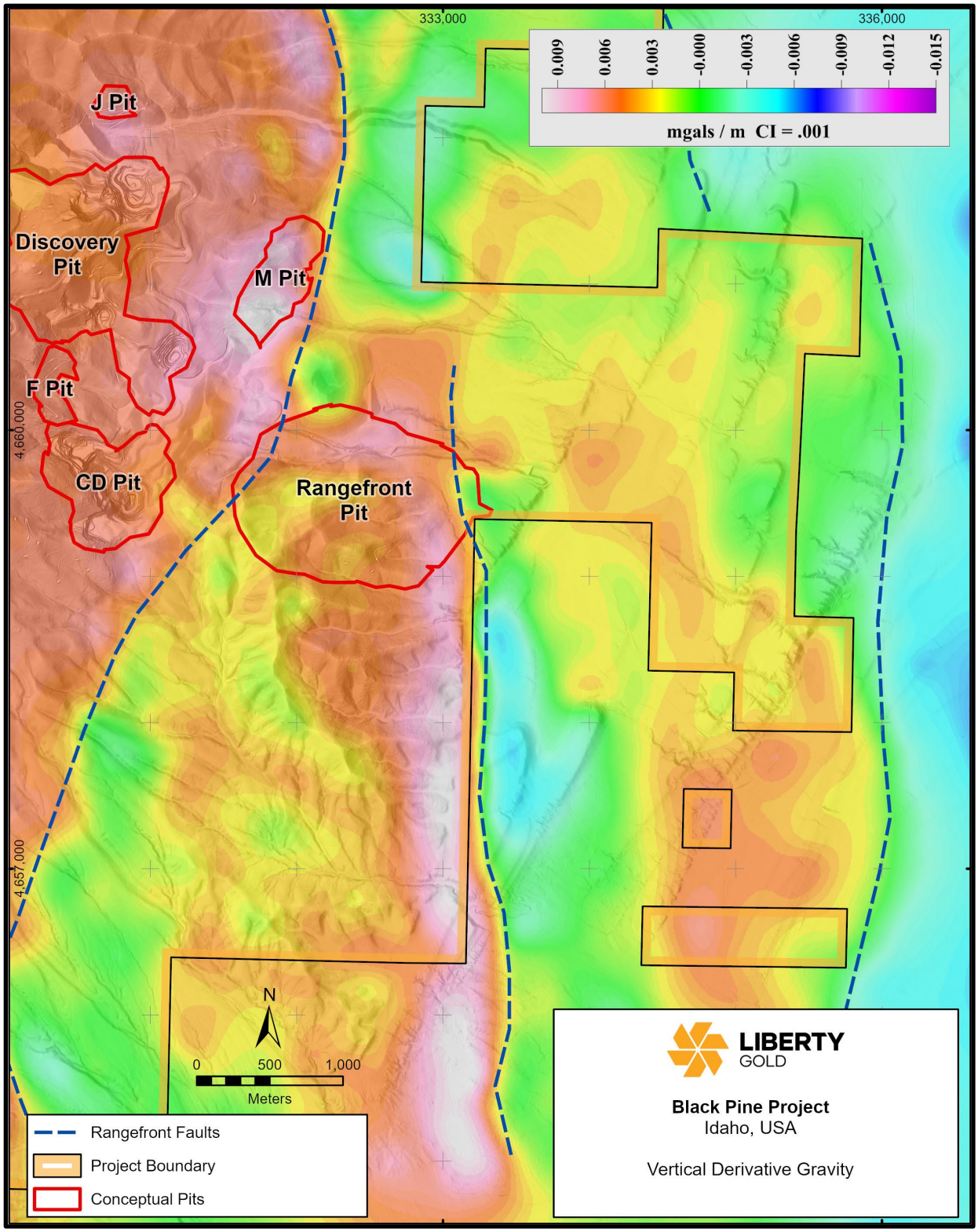
The horizontal gradient model illustrates the rate of change in gravity response over horizontal distances, and clearly delineates the Rangefront Fault, as well as other likely faults to the east that are concealed by gravel cover (Figure 9-2). Of note is a north-south linear immediately to the east 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.



Source: Liberty Gold, 2026, Wright, 2002.

Figure 9-2: Horizontal Gradient Gravity



Source: Liberty Gold, 2026, Wright, 2022.

Figure 9-3: Vertical Derivative Gravity

9.6 THREE-DIMENSIONAL MODELING

In order to integrate surface mapping, drilling, structural data and interpretations, Liberty Gold created a three-dimensional geological model for the Black Pine property in Leapfrog Geo.. The model is constantly 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, as well as surficial deposits including alluvium, pit backfill, and waste dumps.

9.7 MACHINE LEARNING TO PREDICT GOLD CYANIDE SOLUBILITIES

Geometallurgical domains at Black Pine were constructed based on gold cyanide solubility, defined as the ratio of drill interval gold concentration measured by agitated leach of pulp samples in dilute HCN to gold by fire assay (AuCN). These domains, used to inform gold recovery equations, are listed in Table 9-1 along with the corresponding AuCN value defining each category. Met 1-3 are all considered oxidized material. Met 2 and Met 3 have a lower AuCN response largely due to lower head grade, increased clays (preg borrowing), or presence of silica rich typically dolomitic rocks where gold may be encapsulated. Met 4, as well as all lithologic domains containing significant carbon (Poc) are not considered leachable and are given a 0% recovery in the resource model.

Table 9-1: Black Pine Project Geometallurgical Domains

Metallurgical Domain	Category	Cyanide Solubility (% AuCN)
Met 1	Oxide	65 > %Au_{CN}
Met 2	High Transition	50 > % Au_{CN} < 65
Met 3	Low Transition	25 > % Au_{CN} < 50
Met 4	Non-Leach	% Au_{CN} < 25

In 2021, Liberty Gold noticed that results for AuCN assaying of some pulps were considerably lower than expected based on previous assaying of similar material in adjacent holes. An internal investigation did not identify an explanation for the discrepancies. At the same time, assay results were received for a core twin of an RC hole in the Rangefront area. These samples had been prepared at a different prep laboratory and AuCN results were materially higher for the core hole. By mid-2022, a comprehensive investigation by Liberty Gold determined that one of the laboratories had been adding stearic acid (an anti-caking agent) well in excess of recommended amounts during the pulping step. Coarse rejects from most of the drill intervals from the lab that violated protocols were retrieved and submitted to the lab for prep and reassay, replacing the suspect data in the database. A follow-up investigation concluded that even small amounts of stearic acid may have a deleterious effect on recovery of gold by cyanidation. This handling error was found to impact approximately 23,600 cyanide leach assays across 465 drill holes at the Project, and these results were removed from the database.

This sampling preparation error, as well as earlier procedures that limited the grade threshold for cyanide leach testing to include only samples above 0.2 g/t Au, left large gaps in the database. This reduced the confidence in the modeling of the geometallugical domains, especially proximal to those gaps.

To improve the modeling around the data gaps, Liberty Gold worked with SLR on a machine learning model (ML) to predict the gold cyanide solubilities in the holes with contaminated or missing data. Three ML algorithms, Neural Network (NN), Random Forest (RF), and eXtreme Gradient Boosting (XGB) (LeCun, Bengio & Hinton, 2015; Chen and Guestrin, 2016; Pedregosa et al., 2011) were iteratively trained on selected drill hole data available entries. Prior to ML modeling, multiple pre-processing steps were performed to prepare the data for use in supervised learning tasks. The goal of the ML modeling was to better model metallurgical domains through classification, and actual cyanide soluble gold values through regression.

The best ML model was able to correctly classify the geometallurgical domains with a 91% weighted accuracy. The resulting infilled database allowed for metallurgical domain modeling to proceed across the deposit without the need for costly and time-prohibitive re-drilling of data gaps. The study also showed increased resolution and prediction of transitional categories compared to earlier models.

9.8 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.

10 DRILLING

10.1 SUMMARY

Liberty Gold carried out drilling programs in 2017 and 2019-2025 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 as well as holes drilled primarily for water-related investigations. Figure 10-2 shows all drill holes in the database relative to the outline of modeled gold mineralization.

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

Year	No. RC Holes[1]	RC[2] (m)	No. Core Holes[1]	Core[2] (m)	Total (Holes)[1]	Total[2] (m)
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	319	66,552	6	947	325	67,499
2023	149	27,461	11	1,401	160	28,862
2024	70	20,551	0	0	70	20,551
2025[3]	167	36,759	33	3,138	200	39,897
Total	**1210**	**289,655**	**77**	**11,253**	**1,287**	**300,909**

Notes:
1. The number of holes includes both holes that were abandoned/lost 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/lost holes.
3. Meterage includes meters drilled for geotechnical and metallurgical samples.

10.2 DRILLING CONTRACTORS AND EQUIPMENT

The 2017-2024 drilling contractor was Boart Longyear Company of Elko, Nevada. One or more track-mounted Foremost MPD 1500 drill rigs were utilized with 14 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 air drying.

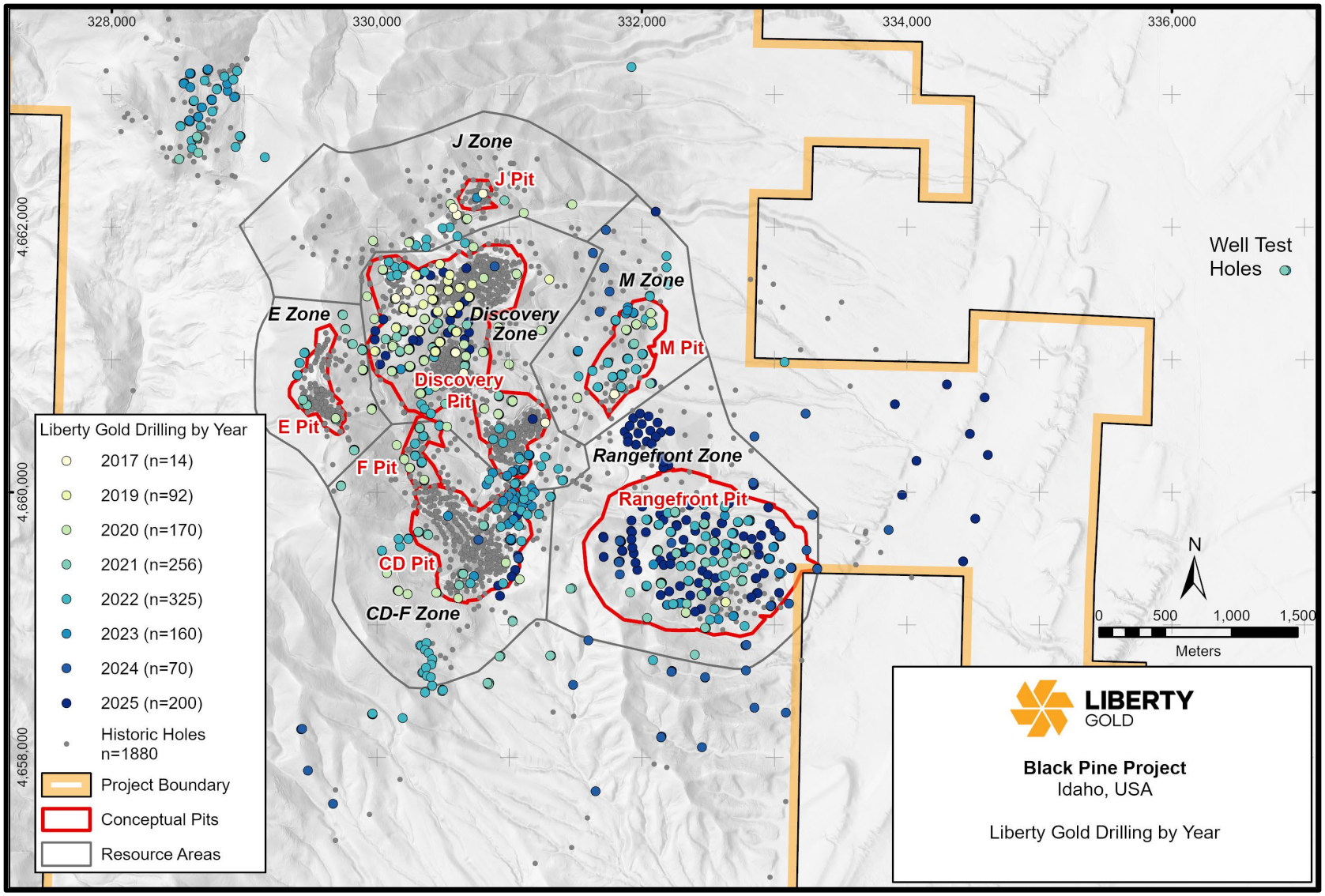
The 2019 core drilling contractor was Timberline Drilling Inc. of Elko Nevada. Drilling was carried out using a track-mounted Atlas Copco CS14 using primarily PQ (8.5 cm diameter core) tools. The 2020 - 2022 core drilling contractor was Major Drilling of Salt Lake City, Utah. Drilling was carried out using a track-mounted LF90 and a truck-mounted LF230 to recover primarily PQ and some HQ (6.35 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. In 2025, Major Drilling drilled PQ-sized metallurgical and geotechnical core holes with a skid mounted LF-90. Drilling was carried out using a track-mounted Atlas Copco CS14 using primarily PQ (85 cm diameter core) tools. PQ core was recovered in wax impregnated cardboard boxes with hole depths recorded by the drillers. These boxes were transported to Liberty Gold's secure core logging facility where a wet and dry photo was taken. Core was then logged for RQD, recovery, lithology and alteration. Sample intervals were taken based on lithology and alteration and most samples were on 1.52 m intervals. Core was then transported to Elko for cutting and sampling or (post 2023) cut and sampled on site.

10.2.1 Drill

The 2020 – 2022 core drilling contractor was Major Drilling out 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.

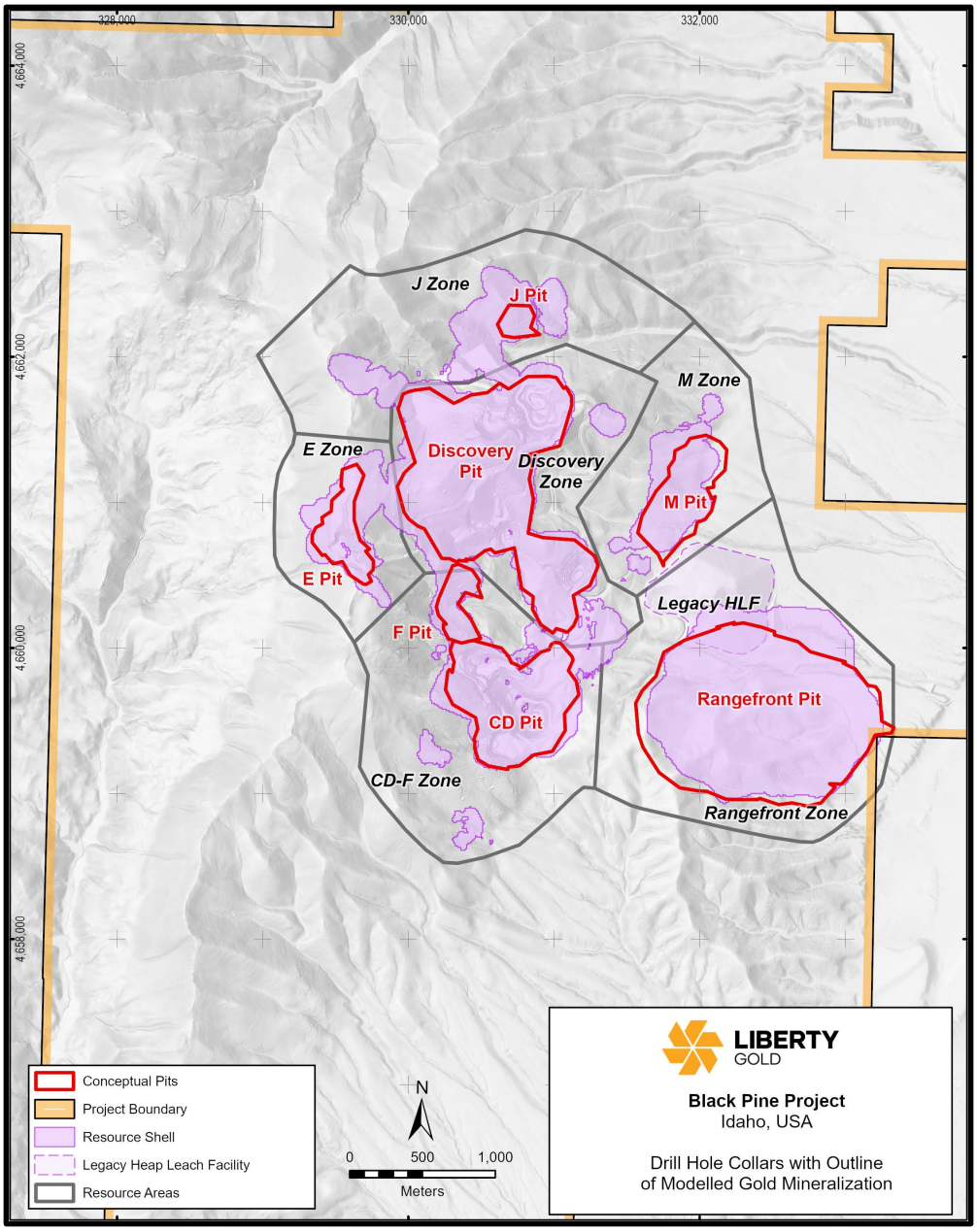
The 2025 drill contractors for RC drilling were Boart Longyear with two MPD 1500's. Boart's core division supplied a sonic rig for drilling in the legacy heap leach and waste dumps. Legacy Drilling out of Elko Nevada had an Apex truck mounted RC rig and a Schramm 685 RC rig. Major Drilling drilled additional PQ-sized metallurgical and geotechnical core holes with a skid mounted LF-90.

The 2026 drill program to date has included two MPD 1500 RC rigs from Boart Longyear. National Drilling out of Elko, Nevada had a track mounted EDM 95 Cawler and a track mounted Schramm T-455GT Crawler. Alford Drilling out of Winnemucca, Nevada provided a single buggy mounted drill rig.



Source: Liberty Gold (2026)

Figure 10-1: Liberty Gold Drilling by Year



Source: Liberty Gold (2026)

Figure 10-2: Drill Hole Collars with Outline of Modeled 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 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 placed in the cement surface plug, and their locations were surveyed by Liberty Gold personnel using a Trimble GeoXH GPS receiver.

From 2020 through 2026, 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 2025 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 continuously 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 down-the-hole and the up-the-hole data for 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 are installed on the tool to reduce rotation during the survey.

For Liberty Gold's 2026 drill program a north-seeking, solid state gyroscopic tool (Reflex EZ-Gyro) 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. Surveys were run by the drill crews and data downloaded by Liberty Gold personnel.

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 by 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 ft (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 (Section 7.2.1, Figure 7-4), was sustained groundwater intersected.

During the detailed explicit modeling of the gold mineral domains discussed in Section 14, 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 lack 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 for 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 in turn 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 2026 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 2026. Drilling for these years is briefly summarized below.

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 Liberty Gold's preferred 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. 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 modeling 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.

In 2021, RC drilling consisted primarily of 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 (derived from historic shallow drilling), 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).

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 and F zones.
- Amalgamation of zones consisting of two or more small resource pits defined by primarily shallow historical drilling. Target areas included the M and F 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 outlying target areas.

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

- Resource upgrade in Rangefront, M, and Discovery zones
- Step out and expansion drilling in Rangefront East, M zone and Backrange, and between the historic CD Pits and Tallman Pits, including testing of historic waste rock dumps.
- One hole also drilled in South Rangefront. (approximately 2 km south of the Rangefront Zone)
- Eleven PQ sized core holes in the J, M, F, Rangefront and Discovery zones to collect material for metallurgical testing.

In 2024, RC drilling commenced in early June and continued through the end of November. Drilling was focused on:

- Exploration/Compliance drilling in Section 36
- Resource expansion drilling in Rangefront and M Zone
- Exploration at the Lower Burnt Basin Target Area

- Piezometers drilled at Rangefront, CD Pit and A Pit

In 2025, drilling commenced in early May and continued through the end of December. Drilling was focused on:

- 24 sonic holes drilled into the LHLF
- RC drilling to upgrade and expand mineralization in Rangefront, Discovery and CD Areas
- Piezometers and monitoring wells in M Zone, Tallman, Rangefront, and CD Pit
- 5 Core holes in Rangefront and Disco for metallurgy and geotechnical studies

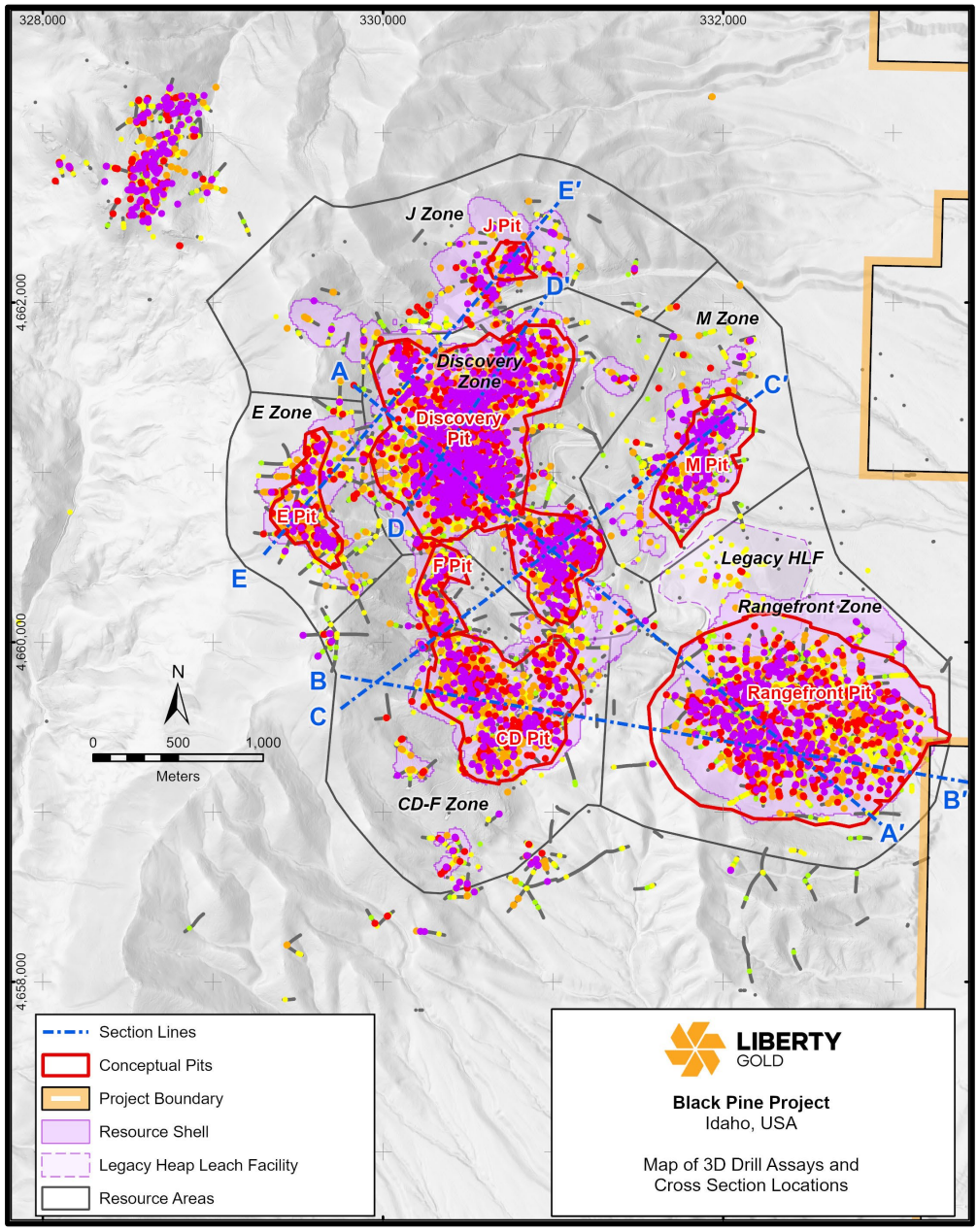
In 2026, drilling commenced in May and is planned to continue through the end of December. Drilling is focused on:

- Compliance drilling in the proposed heap leach pad area
- Drill to measured programs in the first 2 years in Discovery and phase 1 Rangefront areas

10.7 MINERALIZED ZONE DESCRIPTIONS

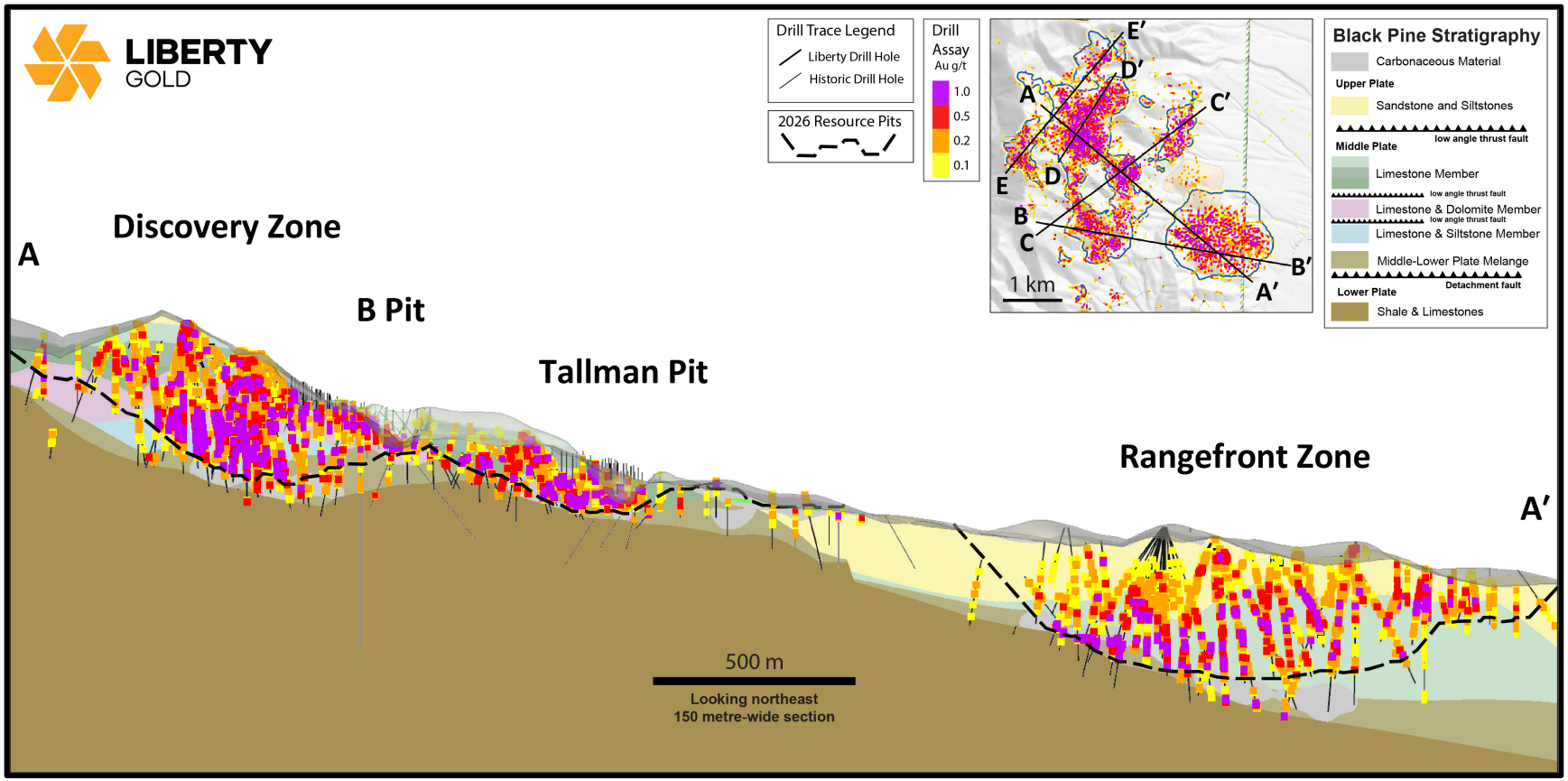
Zones subject to Liberty Gold drilling are described in more detail below and illustrated in Figure 10-3 through Figure 10-8. Zone descriptions reflect the most recent drilling and interpretations in these areas since the previous resource estimate (Sletten et al., 2024). Additional tables of drill results from 2017 - 2025, cross sections and zone descriptions can be found in Sletten et al. (2024).

Over time, continued drilling by Liberty Gold has 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 rendered modeling 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 10-8 illustrate the major mineralized zones in cross section.



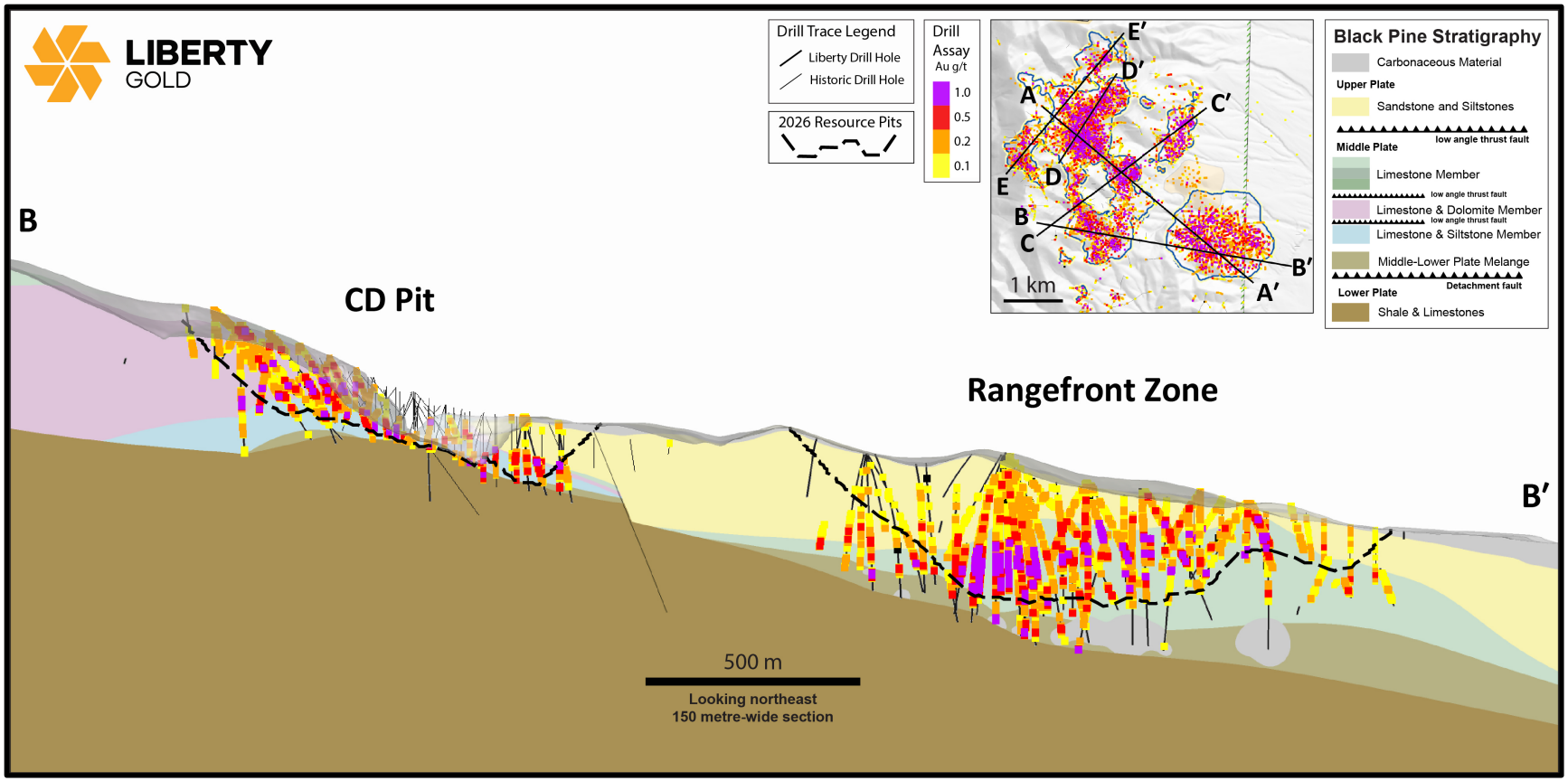
Source: Liberty Gold (2026)

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



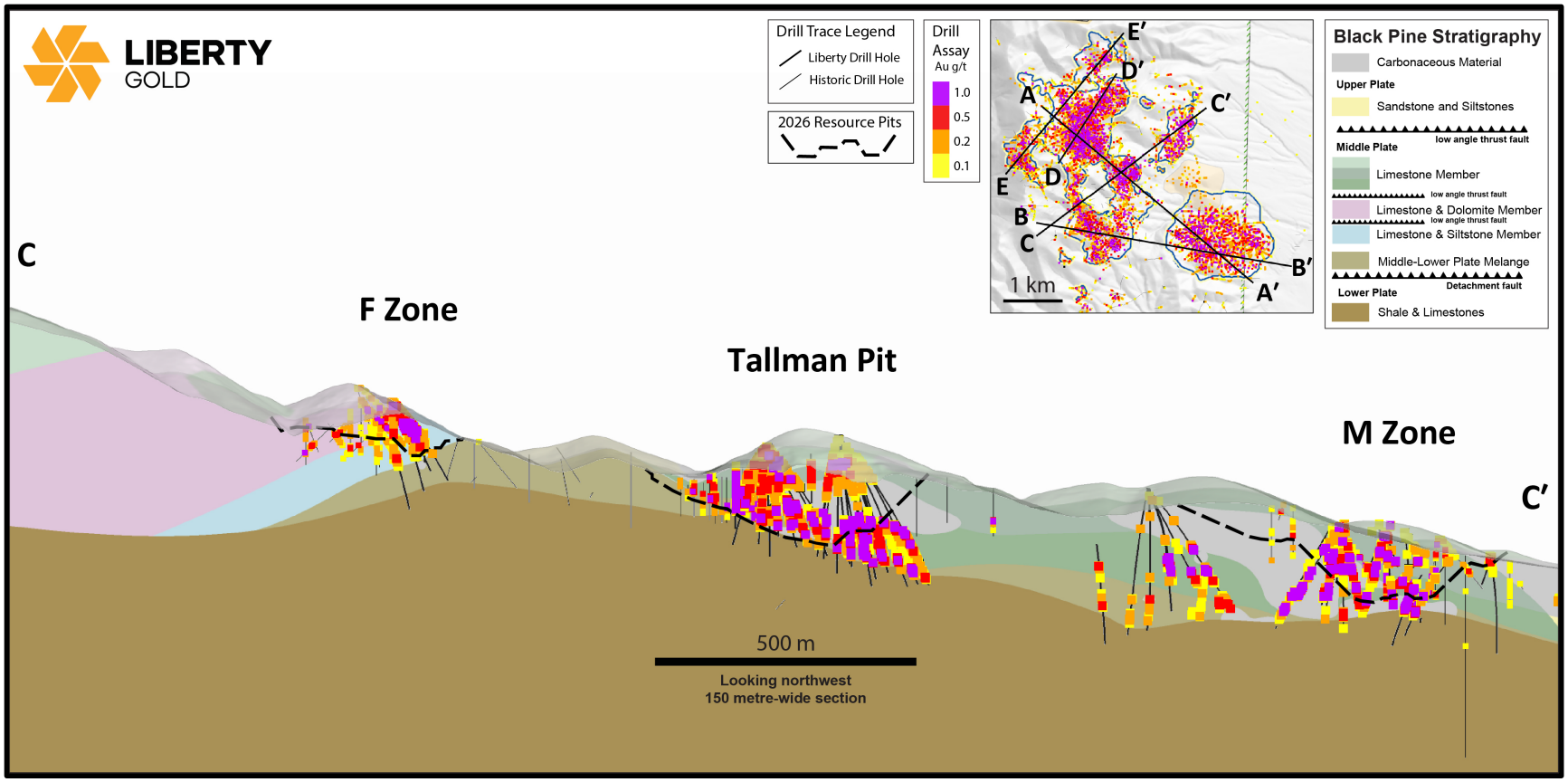
Source: Liberty Gold (2026)

Figure 10-4: Cross Section A-A'



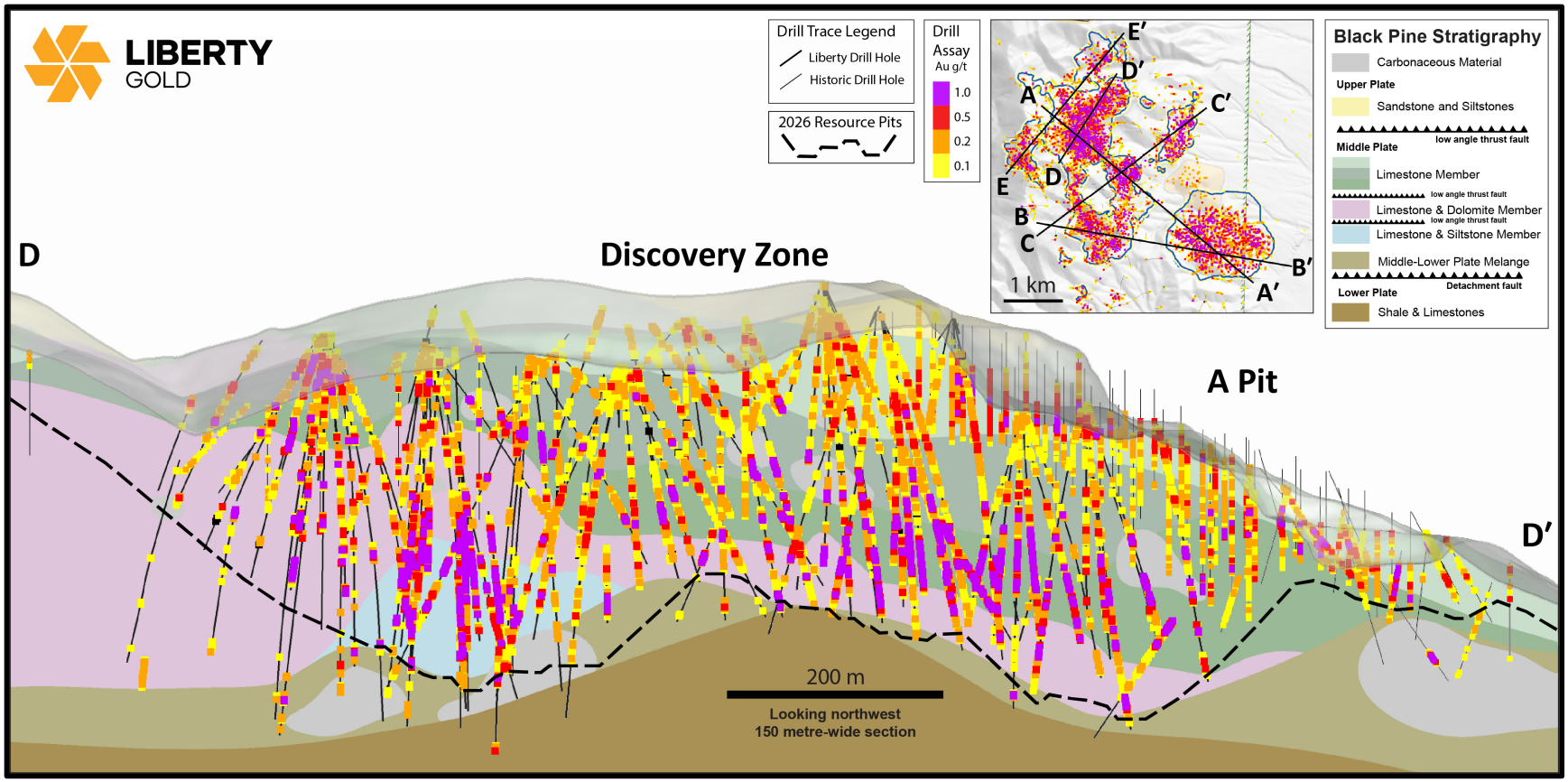
Source: Liberty Gold (2026)

Figure 10-5: Cross Section B-B'



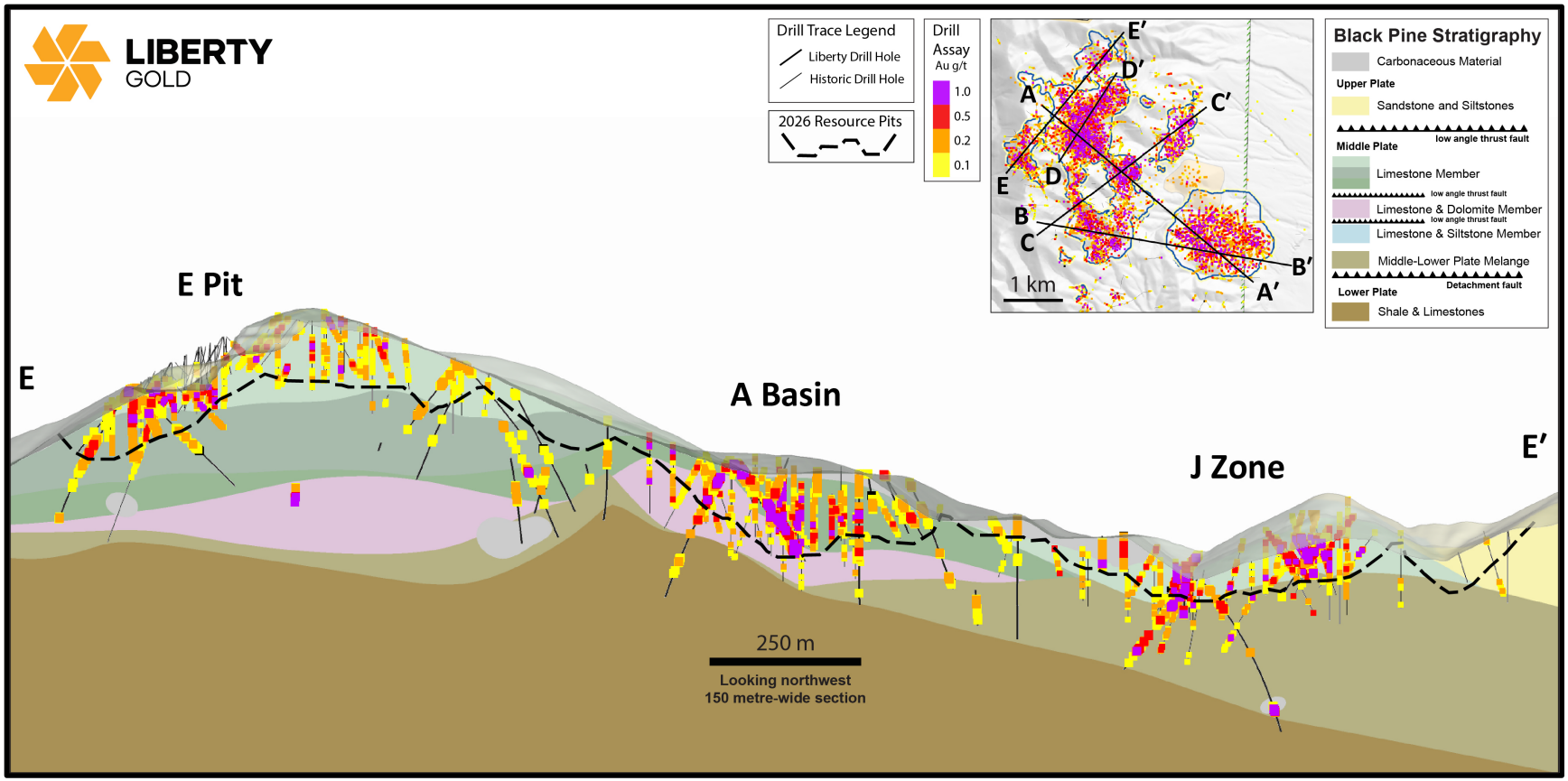
Source: Liberty Gold (2026)

Figure 10-6: Cross Section C-C'



Source: Liberty Gold (2026)

Figure 10-7: Cross Section D-D'



Source: Liberty Gold (2026)

Figure 10-8: Cross Section E-E'

10.7.1 Discovery Zone

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 remnant 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 original mineralized target 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 this 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-m-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 Discovery SE Extension 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 sheared 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 Pola 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 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 modeled 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. Drilling continued in late 2021 and throughout 2023 and identified a zone measuring approximately 1 km by 1.3 km in

223 RC holes and 8 core holes drilled as of October 2023. In 2024, 46 RC drill holes totaling 14,817 m were completed in Rangefront including one that was completed as a peizometer water monitoring hole. In 2025 there was a total of 118 holes totaling 27,004 m. Drilling included RC, metallurgical, and geotechnical core and peizometer and monitoring wells. Mineralization at Rangefront 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 and is locally carbonaceous with reduced cyanide solubility.

10.7.5 M Zone

The M Zone is an area of shallow gold mineralization lying along the range front north of the LHLF (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 99 RC and 4 core holes by Liberty Gold in 2017 through 2025, 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 of the Effective Date of this report, the M Zone is still open to the north and west and is relatively untested to the east.

10.7.6 E Zone

A number of historical drill holes with shallow gold intercepts extend south, west, and north of the historic E Pit. Between 2020 and 2025, 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 Silver Hills

A number of small historic workings north of the main deposit in an area called the Silver Hills (referred to as Back Range in Section 13), were mined for silver, gold and some base metals, similar to the Hazel Pine mine (now M-Zone). Historic 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 holes 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 N-NE trend is controlled by the distribution of highly deformed siltstone bodies are interpreted to be part of the Polb unit. 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. Currently, there are no plans to advance the Silver Hills area beyond what has been completed to date. The area is outside our mine plan area and is not contemplated for development at this time.

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 ft) are appropriate for the style of the Black Pine mineralization.

The mineralization at Black Pine is predominated by gently dipping zones that follow 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 modeling 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.

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 unchanged from Sletten et al. (2021).

11.1 SAMPLE 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 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. Liberty 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. Liberty 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 (g) 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 g aliquots. Samples from previously analyzed holes were sent to Assay Lab Inc. (Assay Lab) in West Jordan, Utah for 30 g fire assay of gold and silver. Cuttings for at least 12 holes were sent to GSI for 30 g fire assay with gravimetric finish. Samples from multiple holes were also sent to Chemex in North Vancouver, British Columbia, for 30 g 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 g 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 g 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 g 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 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/ton Au (0.17 g/t Au), 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/ton Au 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/ton Au 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/ton Au (0.034 or 0.068 g/t Au). 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 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 g fire-assay fusion with an AA finish (ALS method code Au-AA23). Separate 1 g 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.

A summary of the historical analytical methods and laboratory protocols is shown in Table 11-1 below.

Table 11-1: Summary of Historical Analytical Methods and Laboratory Protocols

Phase	Gold Method	Multi-Element Method	Sample Preparation	Laboratory	Accreditations
Newmont (1963-1964)	Fire assay (Au, Ag)	None reported	Unknown	Union Assay	Unknown
Newmont (1974)	AA (Au, Ag); Fire assay (Skyline & checks)	AA (Ag, Pb, Zn); As calorimetric	Unknown	Rocky Mountain (UT); Skyline (AZ); Union Assay	Unknown
Gold Resources (1974-1976)	Fire assay (Au, Ag)	Au, Ag, As, Hg, Cu (method unknown)	Unknown	Rocky Mountain; Union Assay	Unknown
Kerr Addison (1975)	Not specified	Cu, Zn, Au	Unknown	Vangeochem Lab (BC)	Unknown
Pioneer Nuclear (1979-1981)	Fire assay (Au; Au-Ag)	None reported	Unknown	Rocky Mountain; Union Assay; Cone Geochemical (CO)	Unknown
Pegasus (1983-1985)	Fire assay (Au)	Au, Ag, Hg ± Sb, As	Unknown	Union Assay; Rocky Mountain	Unknown
Permian / Pegasus (1984)	Fire assay (Au, Ag; 30 g)	None reported	Unknown	Union Assay; Rocky Mountain	Unknown
Noranda (1986-1989) – Soils & Rocks	Fire assay + AA finish (Au)	Ag, As, Au, Hg, Sb	Unknown	Chemex (NV); GSI (CA)	Unknown
Noranda Drilling (1986)	Fire assay (30 g; AA/gravimetric)	Pb, Zn (some holes; method unknown)	Unknown	Rocky Mountain; Assay Lab; GSI; Chemex	Unknown
Noranda Drilling (1987-1988)	Fire assay (30 g; gravimetric)	None reported	Unknown	ASI (NV); Chemex; GSI; Legend	Unknown
Noranda Drilling (1989)	Fire assay (30 g)	None reported	Unknown	Legend (NV)	Unknown
Pegasus (1990-1997)	HCL (primary); Fire assay (≥0.005 oz/t or selected); estimated FA for low grades	Multi-element (Au, Ag, As, Ba, Bi, Sb, Hg ± Cu, Pb, Zn, Mo)	Unknown	Black Pine Lab (on-site)	Not independent; unknown accreditation

Phase	Gold Method	Multi-Element Method	Sample Preparation	Laboratory	Accreditations
Western Pacific (2011-2012)	Fire assay (30 g; Au-AA23)	ICP-AES/MS (ME-MS41; 51 elements; aqua regia)	ALS prep (Elko, NV); no QA/QC inserted	ALS Minerals (Elko, Reno, Vancouver)	Not stated (ALS typically ISO-accredited, but not specified here)

Note: 1974 Rocky Mountain AA gold may represent cyanide-soluble gold only, not total gold.

11.1.2 Liberty Gold Surface Samples

Between 2017 and the Effective Date of this report, a total of 771 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 (kg). The samples were crushed to 70% passing 2 mm, split to obtain a 250 g 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 g fire assay with an AA finish (method code Au-AA23). Separate 1 g aliquots were analyzed for 51 major, minor, and trace elements by ICP-AES and MS following aqua-regia digestion (ALS method code ME-MS41).

Between 2017 and the Effective Date of this report, a total of 117 soil 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 3 kg. The samples were crushed to 70% passing 2 mm, split to obtain a 250 g 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 g fire assay with an AA finish (method code Au-AA23). Separate 1 g 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 mm. The crushed material was riffle split to obtain a 250 g 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.

Drilled samples from 2023 and after were 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. RC samples were collected by drill company personnel at the drill rig, dried on site and then shipped via Bill Evans Trucking to ALS Twin Falls prep lab.

The sample pulps were shipped by ALS to their assay laboratory in Reno, Nevada, where 30 g 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 g 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 g/t Au were re-assayed using a new 30 g 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.

In 2026 a silver re assay program of previously analyzed pulps was undertaken to establish a more robust silver assay result. Prior to 2026 samples were analyzed for silver using an aqua-regia (two-acid) digestion with AA finish. As aqua regia is not a total digestion for silver, the values are not sufficiently precise for use in Mineral Resource estimation. Liberty Gold pulled 6,254 pulps for re-assaying using four acid digestion.

The samples for re-assaying were completed by MSA Labs in Langley British Columbia, Canada. A 10 g aliquot of the received pulp was taken for analysis and digested using the sequential addition of nitric acid, perchloric acid and hydrofluoric acid and leached with hydrochloric acid. The digested sample was then mixed with deionized water and analyzed by Inductively Coupled Plasma-Optical Emission Spectroscopy (ICP-OES, MSA method code ICF-5Ag). Sample returning results greater than 100 ppm Ag were re-run using the ore-grade method (MSA code ICF-6Ag).

11.2 SAMPLE SECURITY

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

Liberty drill samples were stored at the Black Pine drill sites for a few days prior to transport. Drill samples were also stored at the Company's field 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 were stored in Liberty Gold's secure warehouse. Since 2021 all pulps and coarse reject have been relocated to the Liberty Gold exploration office adjacent to the Black Pine property.

11.3 QUALITY ASSURANCE/QUALITY CONTROL

Quality Assurance (QA) refers to systems and procedures implemented to maintain data quality during sampling, sample preparation, and analytical methods. Quality Control (QC) refers to the routine checks used to verify the quality

of the data. Together, QA/QC protocols help to ensure sample representivity, analytical accuracy, and analytical precision.

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 ft 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 11-2.

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 11-2: 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
BP-9	30	95	0.016	0.015	0.017
BP-11	110	185	0.009	0.008	0.006
BP-12	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 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 Sonic 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, having been examined in two batches for gold: 2017-2023 and 2024-2025, and an examination of results from the silver pulp re-assay program in 2024-2026. Results from the ongoing 2026 drill program were not included in the resource estimate and are not discussed here.

11.3.3 QA/QC Results

11.3.3.1 Au Certified Reference Materials

Certified Reference Materials (CRMs) evaluate accuracy of the assaying by detecting the differences between a result and an expected value, also known as a bias. Results of the regular submission of CRMs are used to identify potential issues with specific sample batches and long-term biases associated with the primary assay laboratory. Specific pass/fail criteria were used based on setting the CRM acceptance limits at the mean ±3 standard deviations (SD) as a failure limit threshold. The insertion of CRMs can also be useful for detecting sample switches and numbering issues that can occur with primary drill samples.

CRMs were used to monitor and evaluate the analytical accuracy and precision of the Liberty Gold drill sample assays performed at ALS.

A total of 16 CRMs have 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 11-3). 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 4,849 CRMs were inserted into the drill sample stream between 2017 and late 2025 and evaluated by SLR in temporal batches as shown in Table 11-3 and Table 11-4.

Table 11-3: Liberty Gold Certified Reference Materials 2024-2025

CRM	Source	Expected Value (Au g/t)	SD	No. of ALS Standard Analysis	Mean	No. of Failures	Comments
Au-19.09	MEG	0.711	0.032	9	0.7	0	Commercial, volcanic
Au-19.10	MEG	0.813	0.036	2	0.69	2	Commercial, siliceous
Au-21.01	MEG	0.407	0.023	2	0.41	0	Commercial, doped, siliceous matrix
LG.22.01	MEG	0.299	0.008	569	0.3	53	BP custom standard
LG.22.02	MEG	0.700	0.018	136	0.68	11	BP custom standard

Table 11-4: Liberty Gold Certified Reference Materials 2017-2023

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 Canyon 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.

From 2017 to 2023

The evaluation of CRMs during the 2017 to 2023 period involves analytical results for CRM PG13001X and indicates a consistent low bias in ALS gold assays relative to the certified value, with a notable proportion of results falling below the ±2 SD limits. Similar low-bias trends are observed in PG13002X, FGS2011A, and, to a lesser extent, LG19001, which represent the higher-grade standards used in the project. Temporal variation in performance suggests possible instrument drift, with partial improvement in late 2020 followed by deterioration in mid-2021.

Review of LG.21.01 results identified several extreme outliers (both high and low), primarily attributed to sample preparation or handling issues (e.g., sample switches or loss of gold prill) rather than analytical accuracy. A comparable issue was identified for LG.21.02, where mislabeling during preparation resulted in values consistent with LG.21.01; affected samples were subsequently corrected.

Overall, 133 failures were recorded from 4,209 CRM analyses (~3.2%). A significant proportion of these failures is attributable to the observed low analytical bias rather than random analytical error. Additional isolated high and low outliers across several CRMs were also linked to sample switches, representing a notable QA/QC concern (with respect to the analytical lab) requiring further investigation.

All CRM failures were reported to the laboratory and investigated. Where failures were attributed to sample handling issues, corrective actions were implemented. In cases where no procedural cause was identified, reanalysis of the failed CRM and adjacent samples was undertaken. Re-assay results generally fell within acceptable limits, and associated drill sample results remained consistent with original assays, supporting the interpretation that the observed failures are largely influenced by systematic low bias.

From 2024 to 2025

Limited insertions were available for CRMs Au.19.09, Au.19.10, and Au.21.01; therefore, the assessment focused on more frequently inserted standards, particularly LG.22.01 and LG.22.02, shown in Figure 11-1 and Figure 11-2 below.

All analytical work was conducted by ALS. Results for LG.22.01 (low-grade standard) indicate a general low-bias trend. A total of 53 results exceed the ±3 SD limits, predominantly during the period from April to July 2024. These failures include both high and low outliers relative to the certified value. In addition, a systematic trend of results below the mean (approaching the -3 SD limit) is observed in January 2025, which may be associated with sample preparation issues, as the affected samples were processed within a similar timeframe. The frequency of failures decreases through 2025; however, six isolated outliers remain, potentially related to sample mix-ups, and further warrant investigation.

The LG.22.02 CRM (medium-grade standard) shows 11 outliers, representing approximately 8% of total insertions. Three of these results fall significantly below the -3 SD limit and may be attributed to sample switches or mislabeling, while the remaining outliers require further investigation.

Overall, SLR considers that the CRM results support the validity of the assay data for use in mineral resource estimation, despite the noted issues, which are interpreted to be largely related to sample handling rather than systematic analytical errors.

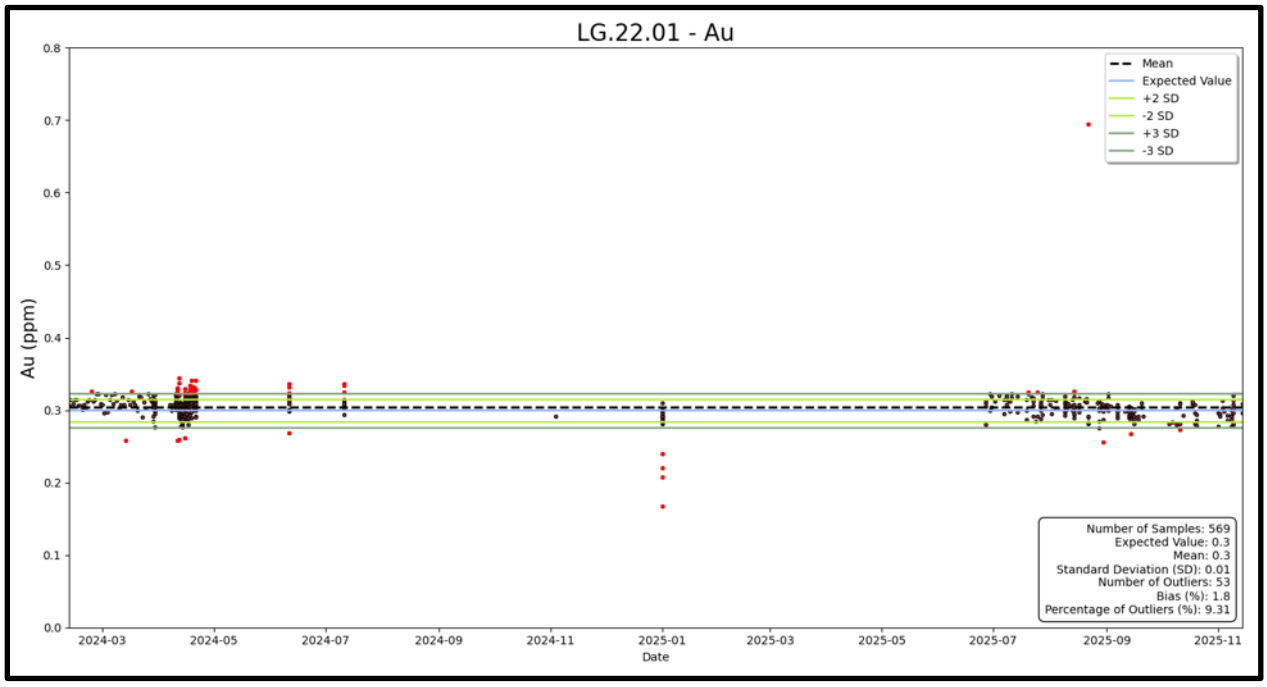


Figure 11-1: Control Chart of CRM LG.22.01 for Gold: 2024 to 2025

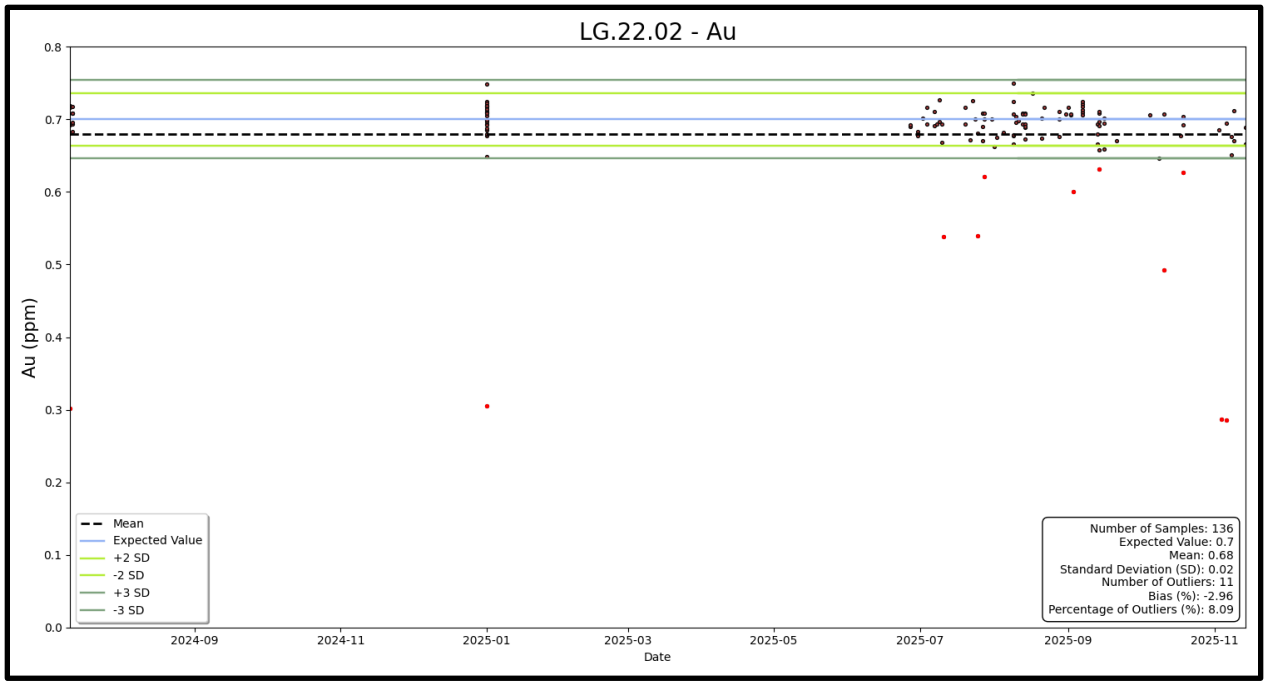


Figure 11-2: Control Chart of CRM LG.22.02 for Gold: 2024 to 2025

11.3.3.2 Ag Certified Reference Materials

During the 2024 to 2026 drilling and pulp re-assay programs, 3,946 samples were submitted for Ag analysis, with 3,275 samples sent to ALS and 671 to MSA. The dataset is supported by 1,451 reference sample submissions from five CRMs, three of which are commercial standards supplied by MEG of Carson City and Lamoille, Nevada (Au-19.09, Au-19.10, and Au-21.01). The remaining two CRMs are prepared by MEG, using material derived from Black Pine drill samples (LG.22.01 and LG.22.02). Overall CRM performance for the 2024 to 2026 QA/QC program is shown in Table 11-5.

Similarly to the review of Au CRMs, there were limited insertions of CRMs Au.19.09, Au.19.10, and Au.21.01 for Ag analysis; therefore, this assessment focused on LG.22.01 and LG.22.02. It is also understood that CRMs Au.19.09, Au.19.10, and Au.21.01 were sent to ALS and were analyzed using Aqua Regia. These three CRMs do not have Ag certified values for Aqua Regia and therefore all samples failed. It is also understood that samples sent to MSA were analyzed using Multi Acid Digestion. LG.22.01 and LG.22.02 have certified values for both Aqua Regia and Multi Acid digestion so plots were produced for both laboratories.

Table 11-5: CRM Performance for Ag in the 2024 to 2026 QA/QC Program

Lab	Element	Unit	CRM	Period Range	No. of Samples	Mean	EV	SD	No. of Outliers	Bias (%)	Failure Rate (%)	Upper Bound (2SD)	Lower Bound (2SD)	Upper Bound (3SD)	Lower Bound (3SD)
ALS	Ag	ppm	Au.19.09	2024-04-09 - 2024-10-09	9	24.54	36.7	1.68	9	-33.1	100	27.91	21.18	29.59	19.5
ALS	Ag	ppm	Au-19.10	2024-09-15 - 2024-09-26	2	23.9	35.11	1.61	2	-31.9	100	27.12	20.68	28.72	19.08
ALS	Ag	ppm	Au-21.01	2024-04-09 - 2024-09-10	2	100	241.6	14.9	2	-58.6	100	129.8	70.2	144.7	55.3
ALS	Ag	ppm	LG.22.01	2024-01-12 - 2026-12-02	856	0.75	0.81	0.12	1	-7.5	0.1	0.98	0.52	1.09	0.4
MSAL ABS	Ag	ppm	LG.22.01	2026-04-24 - 2026-11-05	105	1.49	1.41	0.17	1	5.8	0.9	1.83	1.15	1.99	0.99
ALS	Ag	ppm	LG.22.02	2024-01-29 - 2026-12-02	272	0.83	0.9	0.09	0	-8.3	0	1.01	0.65	1.1	0.56
MSAL ABS	Ag	ppm	LG.22.02	2026-04-14 - 2026-11-05	205	1.72	1.54	0.19	0	11.8	0	2.1	1.34	2.29	1.15

The Ag results of CRM LG.22.01, shown in Figure 11-3, indicate that both ALS and MSA laboratories performed well overall, with only one failure at each laboratory. The ALS dataset exhibits strong analytical control with a large sample size, low failure rate and stable temporal behavior; however, the plot does show a consistent negative bias of -7.49 %, indicating slight systematic underestimation. In contrast, the MSA plot demonstrated acceptable precision even with fewer data points, with a slight positive bias of +5.78%. Overall, both laboratories show acceptable performance, however SLR recommends the slight bias of each laboratory be monitored.

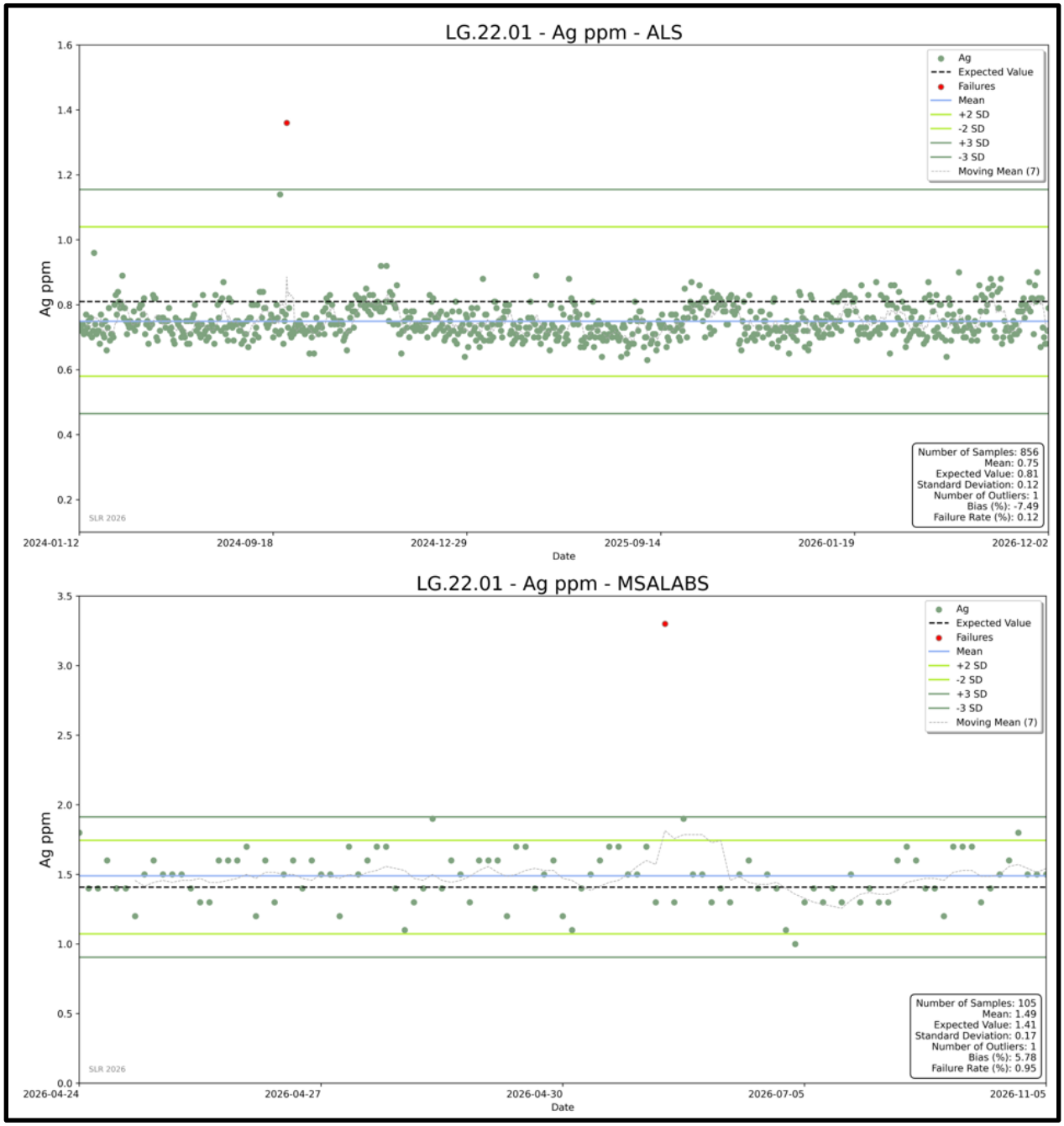


Figure 11-3: Control Chart of CRM LG.22.01 for Silver at ALS & MSA: 2024 to 2026

The Ag results of LG.22.02 are shown in Figure 11-4 below. Similarly to plots for LG.22.01, ALS shows strong analytical control, with tight clustering and zero failures; however, shows a systematic negative bias of -8.28%. This highlights consistent underestimation relative to the certified value, despite good precision and stability. The MSA plot for LG.22.02 shows all data falling within the 3SD threshold, however, shows a positive bias of 11.84% relative to the certified value. This bias is notably higher than the other plots and should be monitored closely.

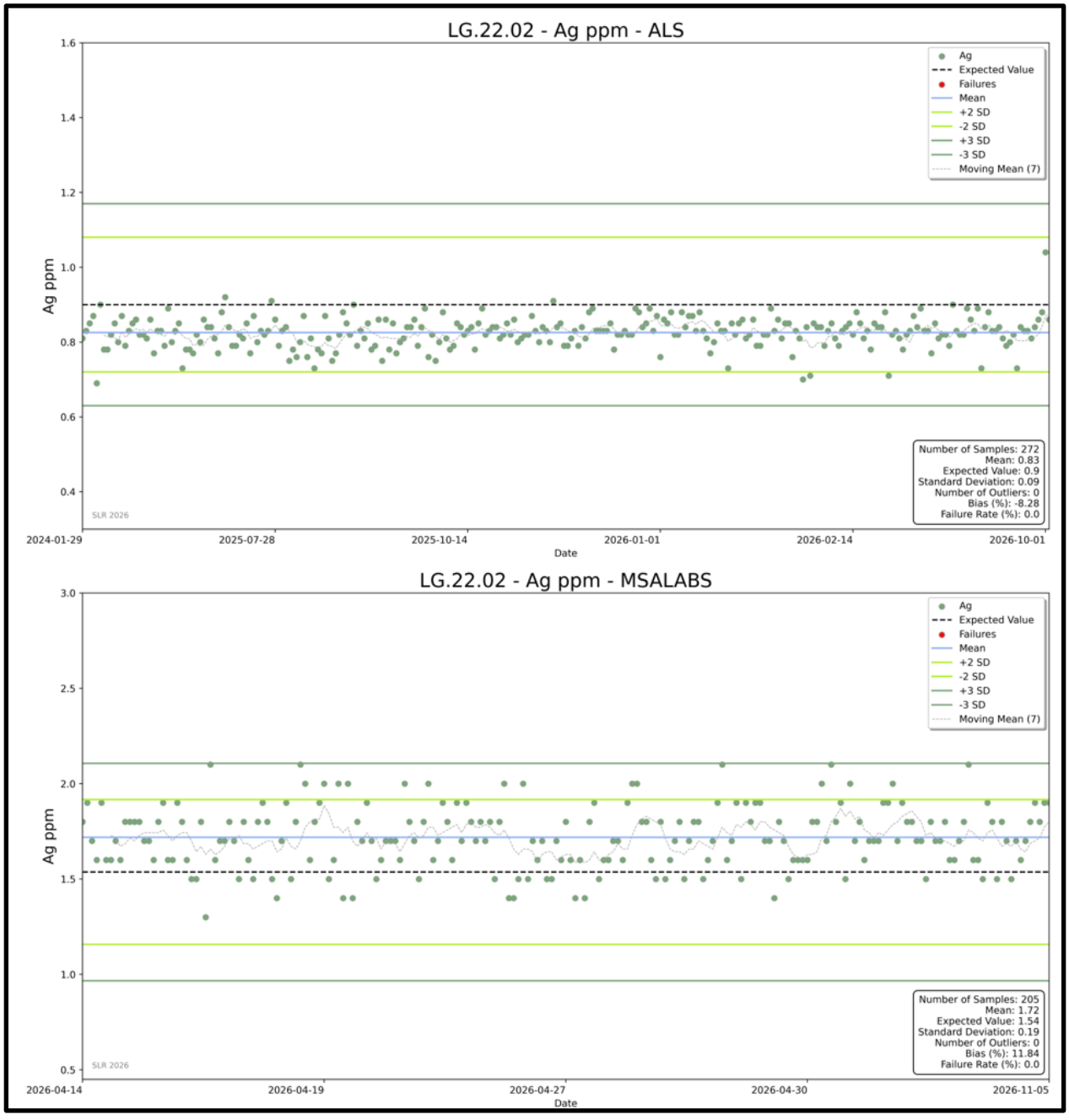


Figure 11-4: Control Chart of CRM LG.22.02 for Silver at ALS & MSA: 2024 to 2026

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.

11.3.3.3 Au Coarse Blanks

Blank samples are inserted into the assay stream to monitor potential contamination during sample preparation and analysis.

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/t Au; blanks with assays in excess of 0.025 g/t Au (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.

From 2017 to 2023

A total of 4,566 blanks were inserted between 2017 through 2023 programs, of which, 27 returned values greater than 0.025 ppm Au. Despite these exceedances, all results fall within ALS acceptance criteria (<1% of the preceding sample grade), indicating no significant contamination issues.

In cases of blank failures, internal reviews were conducted; however, no reanalysis was undertaken for RC samples, as the nature of RC chip sampling (full sample crushing) prevents replication of the original preparation stages.

From 2024 to 2025

A total of 684 blank samples were analyzed for Au (ppm) (Figure 11-5). The blanks are not certified reference materials; however, a threshold of 5X the lower detection limit (LD), equivalent to 0.025 ppm Au, was applied for evaluation. Based on this criterion, no significant contamination issues were identified.

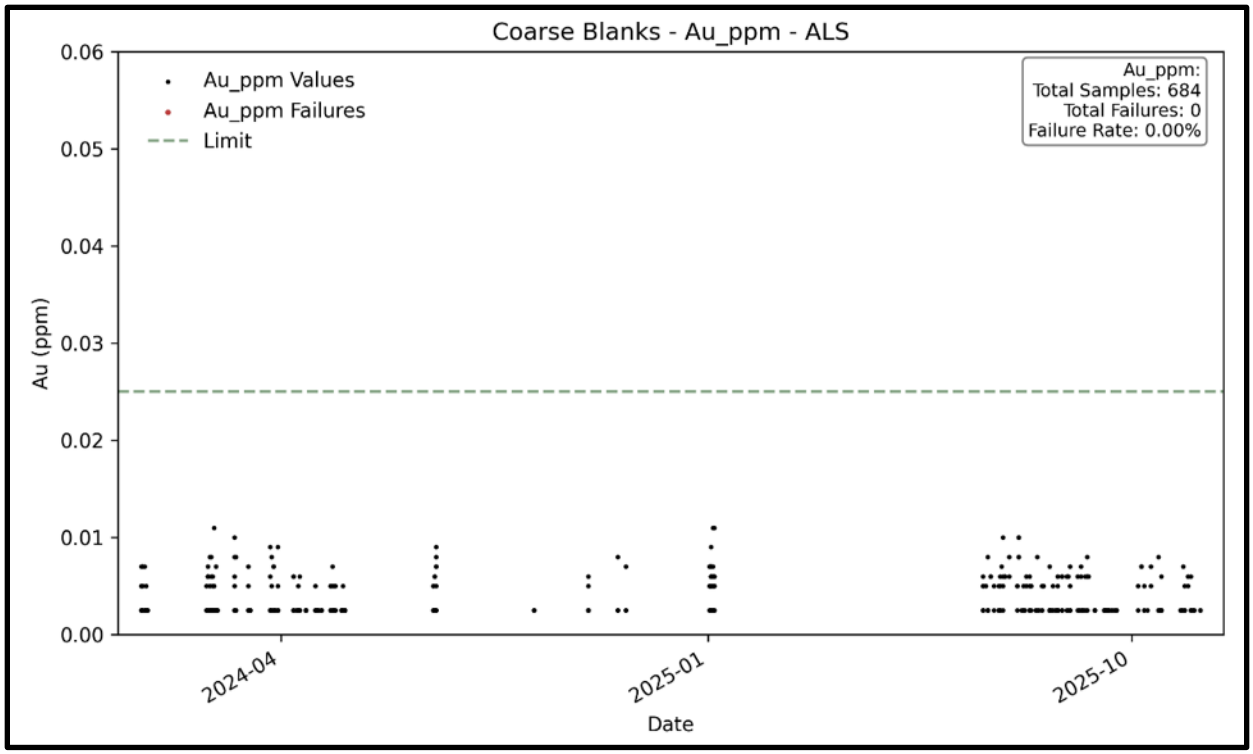


Figure 11-5: Control Chart of Blank for Gold (ppm) ALS: 2024 to 2025

11.3.3.4 Ag Blanks

Two types of blanks were assessed "Coarse" and "fine" throughout the 2024 to 2026 period. The coarse blanks material consists of Vigoro pond pebbles sourced from a hardware store and submitted to ALS. The fine blank is the certified reference material "CDN-BL-10T", obtained from CDN Resource Laboratories Ltd. with a maximum acceptable value of 0.5 ppm for four-acid Ag digestions. Fine blanks were submitted to MSA for analysis.

Both blanks were inserted by Liberty Gold at a frequency of approximately 1 in 33 samples. The blanks were both treated the same with the only difference being CDN-BL-10T is a pulp blank and is not crushed. 2024 to 2025 Ag Blank data is presented in Figure 11-6 below, split by laboratory.

Blank performance was assessed for both ALS and MSA using time-series plots of Ag concentrations. Failures were defined as blank results exceeding the established Limit of Detection (LoD) indicated on the plots multiplied by a factor of 1. It should be noted that six samples are viewed as mislabeled due to being classified as a coarse blank sent to MSA, however the reported assay result for this sample remains within acceptable limits. A summary table showing the performance of blanks at each laboratory is shown in Table 11-6.

Blank sample performance at both ALS and MSA is overall acceptable, with low rates of contamination. Each laboratory produced reported only one failure between the period of 2024 to 2025.

SLR is of the opinion that the blank datasets from both laboratories remain acceptable for QA/QC purposes.

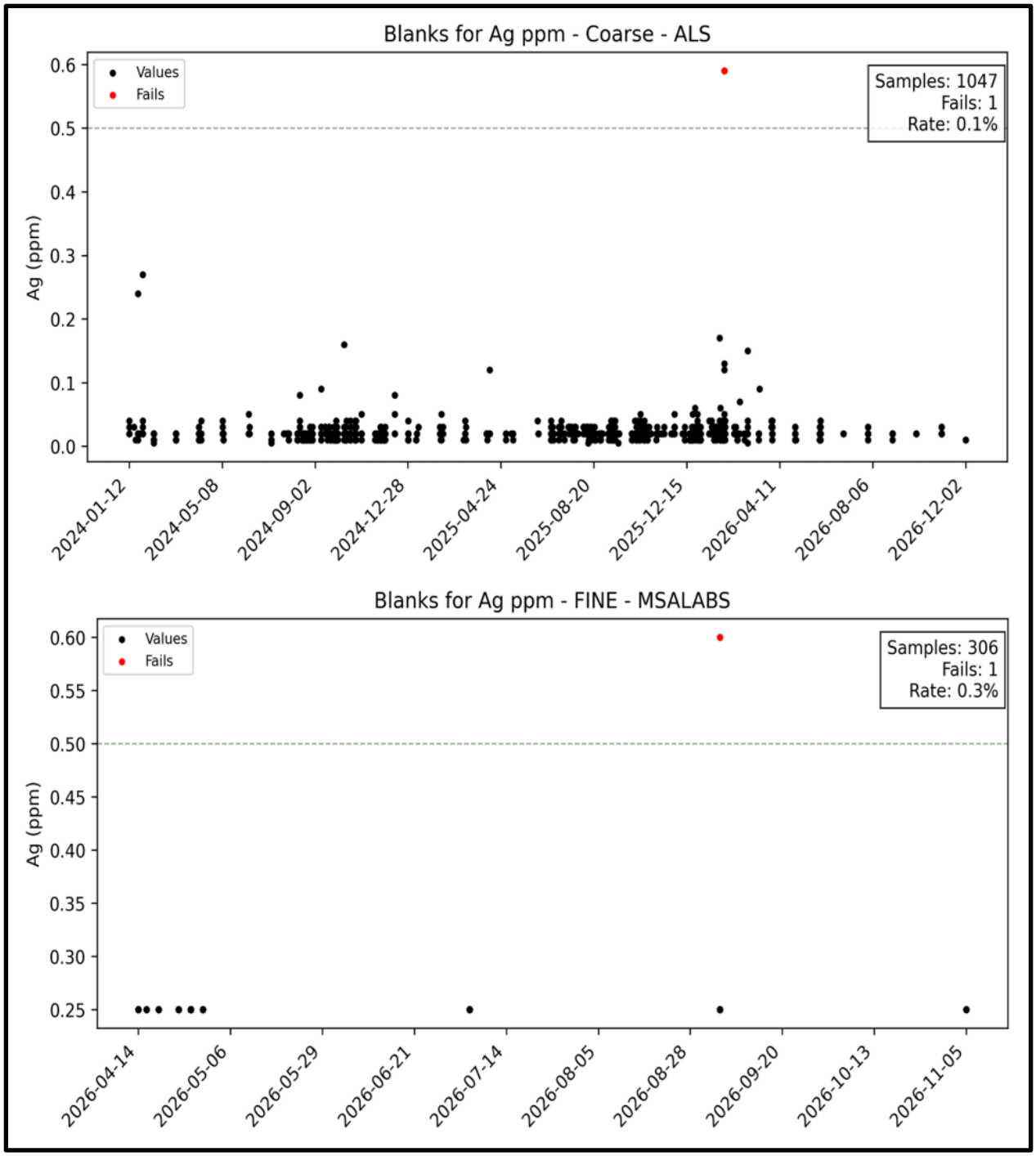


Figure 11-6: Control Chart of Blank for Silver (ppm) ALS & MSA: 2024 to 2025

Table 11-6: Blank Performance for Ag in the 2024 to 2026 QA/QC Program

Lab	Element	Type	Samples	Fails	Rate%	Limit	Min	Max	Mean
ALS	Ag	Coarse	1047	1	0.1	0.5	0	0.59	0.02
MSALABS	Ag	Coarse	6	0	0	0.5	0.25	0.25	0.25
MSALABS	Ag	Fine	306	1	0.33	0.5	0.25	0.6	0.25

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.

11.3.3.5 Au 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.

From 2017 to 2023

A total of 4,249 RC field duplicate pairs (2017–2023) with gold grades >0.01 ppm Au were evaluated, excluding one extreme outlier attributed to a probable sample switch. Overall, results show no significant analytical bias, with duplicate assays averaging 0.51% higher than original samples and an overall percent difference of 2.6%.

A total of 159 pairs (~3.7%) exceed ±30% variance. While most data fall within acceptable limits, several large discrepancies are likely related to sample switches, as supported by multi-element signatures. These occurrences represent a QA/QC concern and warrant further investigation with the laboratory.

From 2024 to 2025

SLR assessed duplicate data using scatter plots, for field duplicates for RC drilling campaign. The maximum acceptable failure rate was set at 10%.

A total of 685 duplicate pairs per analyte were evaluated. Field duplicates show the highest level of precision, with strong correlations across gold and silver elements and failure rates at or below the 10 percent threshold. Figure 11-7 illustrates the silver field duplicates, which show close alignment between pairs, and the rejected pairs correspond to high concentrations.

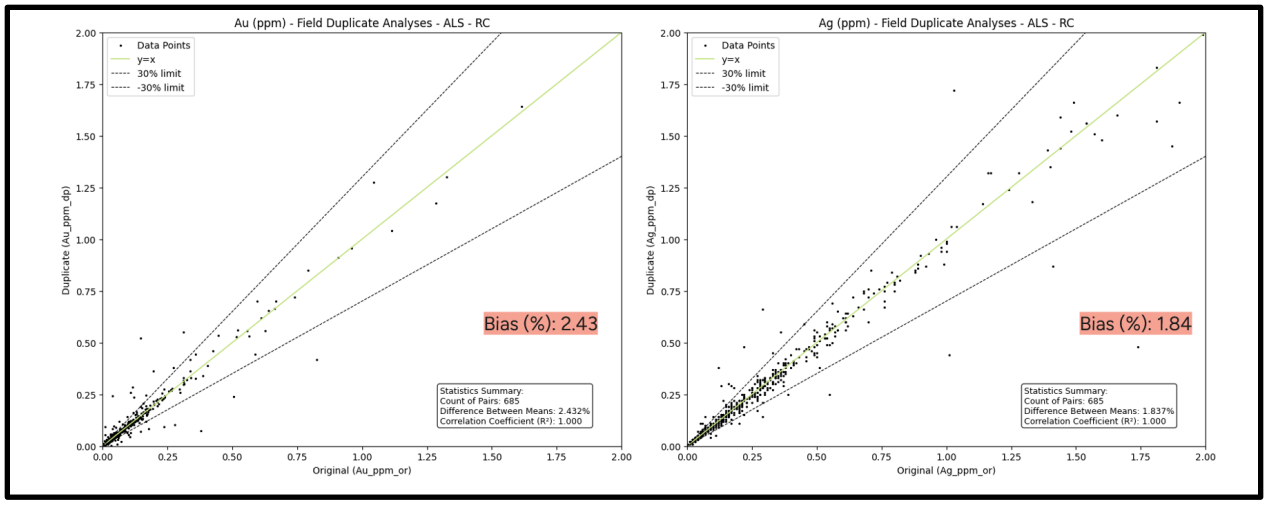


Figure 11-7: RC Field Duplicate Data: 2024-2025 Drilling Programs

In general, RC field duplicates show a high correlation (r ≥ 1), with a failure rate of 2.43 % and 1.84% for gold and silver respectively, moderated influenced by four outliers and predominantly associated with medium-grade samples (Figure 11-7). Despite this, the results are considered acceptable and suitable for use in the Mineral Resource estimation.

Additionally, a total of 26 field duplicate pairs were evaluated from the Sonic drilling campaign. The results show generally high correlation, with R^2 values of 0.93 for gold and 0.96 for silver, indicating acceptable to strong analytical precision.

However, the dataset exhibits notable variability, particularly for gold (Figure 11-8), where two samples with grades below 0.27 ppm Au show significant dispersion. In addition, minor scatters are observed in higher-grade silver samples, around 1.51 ppm Ag. These discrepancies may be attributable to potential sample mix-ups during preparation.

The overall high correlation is likely influenced by the presence of higher-grade samples (>1.18 ppm Au) within the dataset, which tend to reduce relative variability and improve statistical correlation.

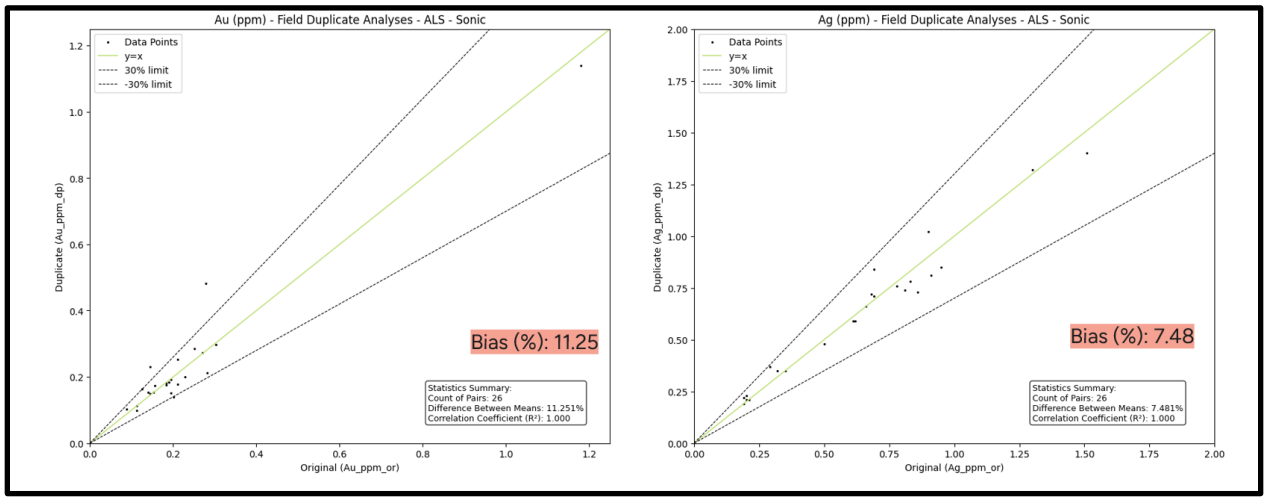


Figure 11-8: Sonic Field Duplicate Data: 2025 Drilling Programs

The QP has reviewed the performance and protocols of the RC field duplicates and finds them to be to industry standards. For Sonic drilling, the QP recommends the implementation of appropriate corrective measures to improve analytical precision under this drilling method 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.

Ag Duplicates

A description of the different types of duplicates used by Black Pine for Ag is provided in Table 11-7. Overall duplicate performance is provided in Table 11-8.

Table 11-7: Duplicate Types and Descriptions

Duplicate	Description
Field	A sample collected at the same time and from the same location as an original sample. Usually samples are ½ or ¼ core.
Pulp	The sample is obtained from splitting the pulverized material during sample preparation.

Table 11-8: Duplicate Performance for Ag in the 2024 to 2026 QA/QC Program

Lab	Dup Type	Element	Pairs	Fails	FailRate %	Corr	MeanΔ	Min	Max	Mean
ALS	FD	Ag	5,423	146	2.69	0.977	0	0	100	0.69
MSALABS	FD	Ag	6	1	16.67	0.984	0.24	0.67	8.8	3.28
MSALABS	PD	Ag	43	12	27.91	0.835	-0.1	0.42	8.4	2.5

Field Duplicates

SLR reviewed field duplicate performance for Ag samples submitted to ALS and MSA from 2017-2026 data provided. These assessments provide a measure of the combined effects of sample preparation precision, including crushing, splitting, and sample heterogeneity, and are an important component of the QA/QC program. Field duplicate performance for Ag was evaluated using HARD index and duplicate scatter plots shown in Figure 11-9. A total of 5,423 field duplicate pairs were assessed for ALS while 6 pairs were assessed for MSA. The HARD index plot for ALS shows very strong precision with the majority of data points falling within the acceptable 30% threshold. The scatter plot shows a strong positive linear relationship between the original and duplicate samples, with most pairs clustering closely to the y = x line and a correlation of 0.977. It is noted that there is increased variability at higher grades, however there is limited data to conclude analytical errors.

Due to the limited number of field duplicate pairs assessed at MSA, limited meaningful conclusions can be drawn, however, plots tentatively suggest positive lineal relationship, with variability at higher grades.

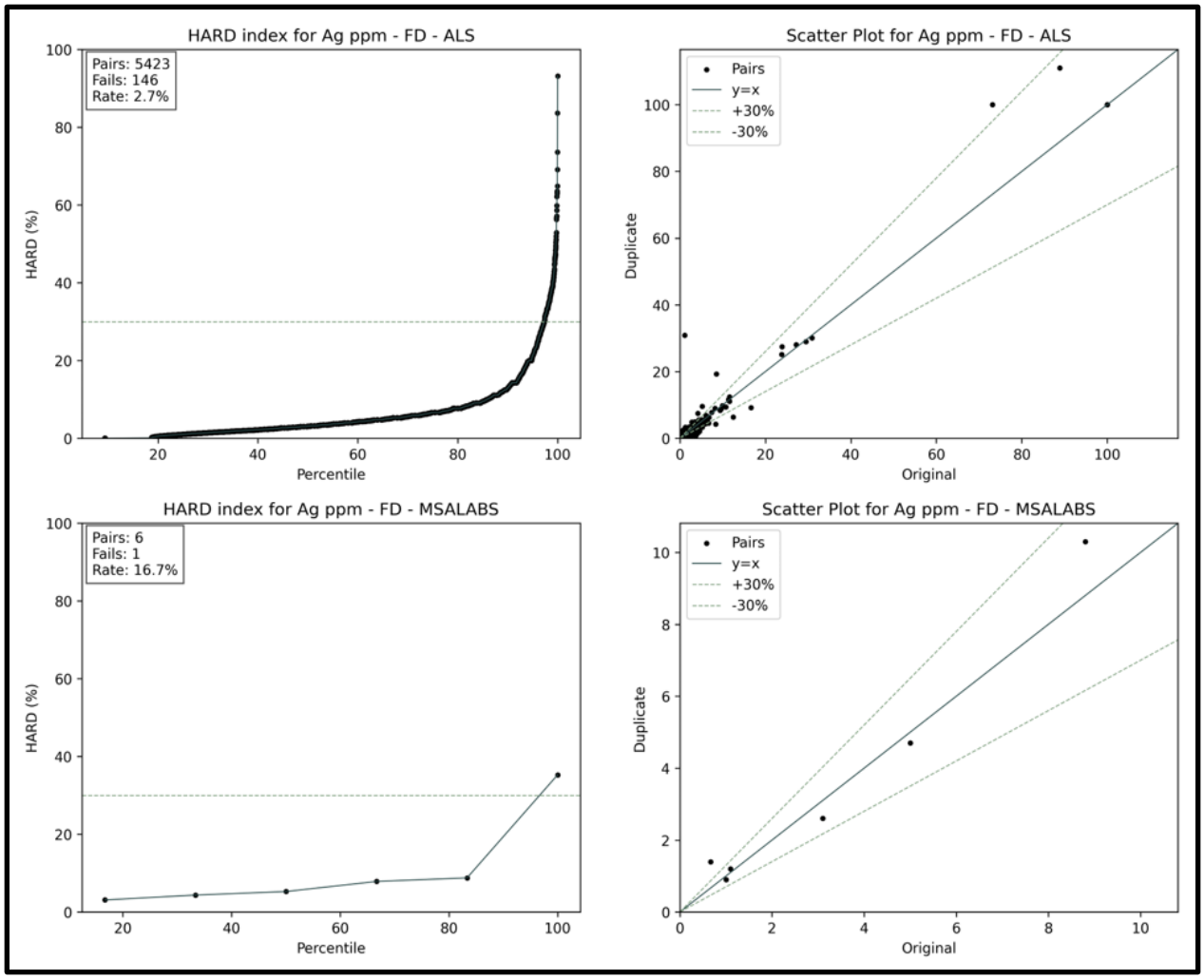


Figure 11-9: Silver Field Duplicate Data: 2017 - 2026

Pulp Duplicates

Pulp duplicate performance for Ag was evaluated to assess the repeatability of the sample preparation process, including crushing, splitting, and subsampling. HARD index distributions and duplicate scatter plots for samples submitted to MSA in 2026, shown in Figure 11-10. A total of 43 pulp duplicate pairs were reviewed. The HARD plot shows a higher failure rate of 27.9%, indicating poor precision relative to typical QA/QC tolerances. The strong upper-tail divergence suggests heterogeneity and/or procedural inconsistencies. The scatter plot indicates good agreement at low grades with more variability at higher grades.

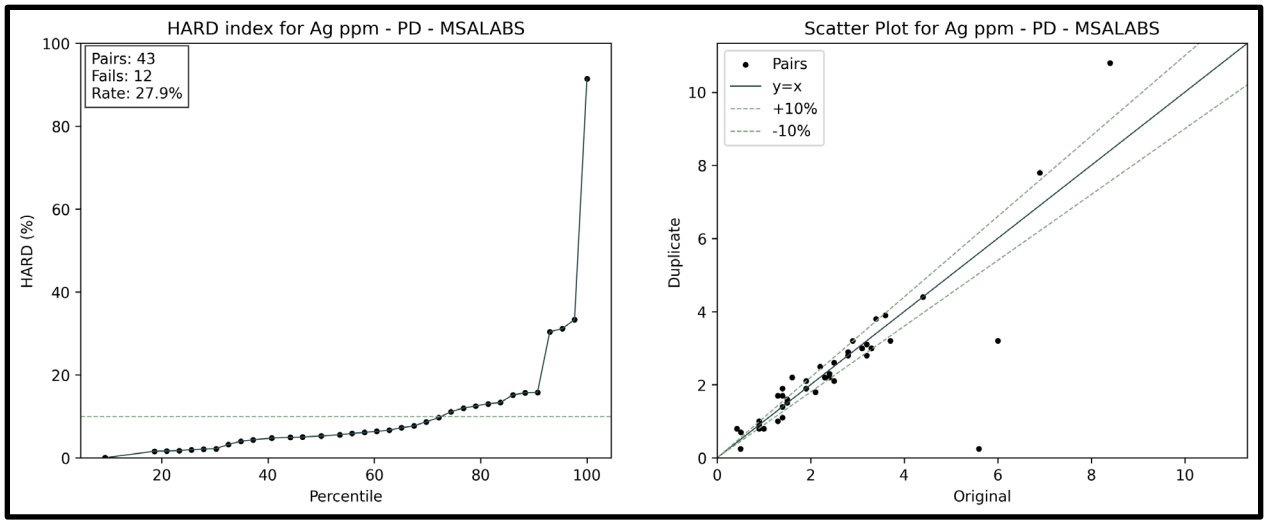


Figure 11-10: Silver Pulp Duplicate Data: 2026

11.3.3.6 Check Assays

From 2017 and 2019 to 2023 Drill Programs

A subset of drill sample pulps from ALS (2017-2019 to 2023) was re-assayed at Bureau Veritas (BV), an independent ISO 17025 accredited laboratory, to verify analytical accuracy. Approximately 7% of samples with grades >0.1 g/t Au were selected, resulting in 3,052 RC and 190 core pulp checks; lower-grade samples (<0.1 g/t Au) submitted in error were excluded from the evaluation due to increased variability near detection limits. Control samples submitted with the batch showed no issues, shown in Figure 11-11.

After excluding outliers (> 40% variance), 2,370 paired assays were compared. BV results show slightly lower mean gold values than ALS (0.433 ppm vs. 0.440 ppm) but exhibit a consistent positive bias relative to ALS at lower gold grades. Relative differences between paired samples are low (1.37% for 0.1-0.4 g/t Au, 2.24% for 0.4-1.0 g/t Au, and 0.24% for >1.0 g/t Au), supporting good overall agreement. The observed bias at lower grades is consistent with the low-bias trends identified in control sample analyses.

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.

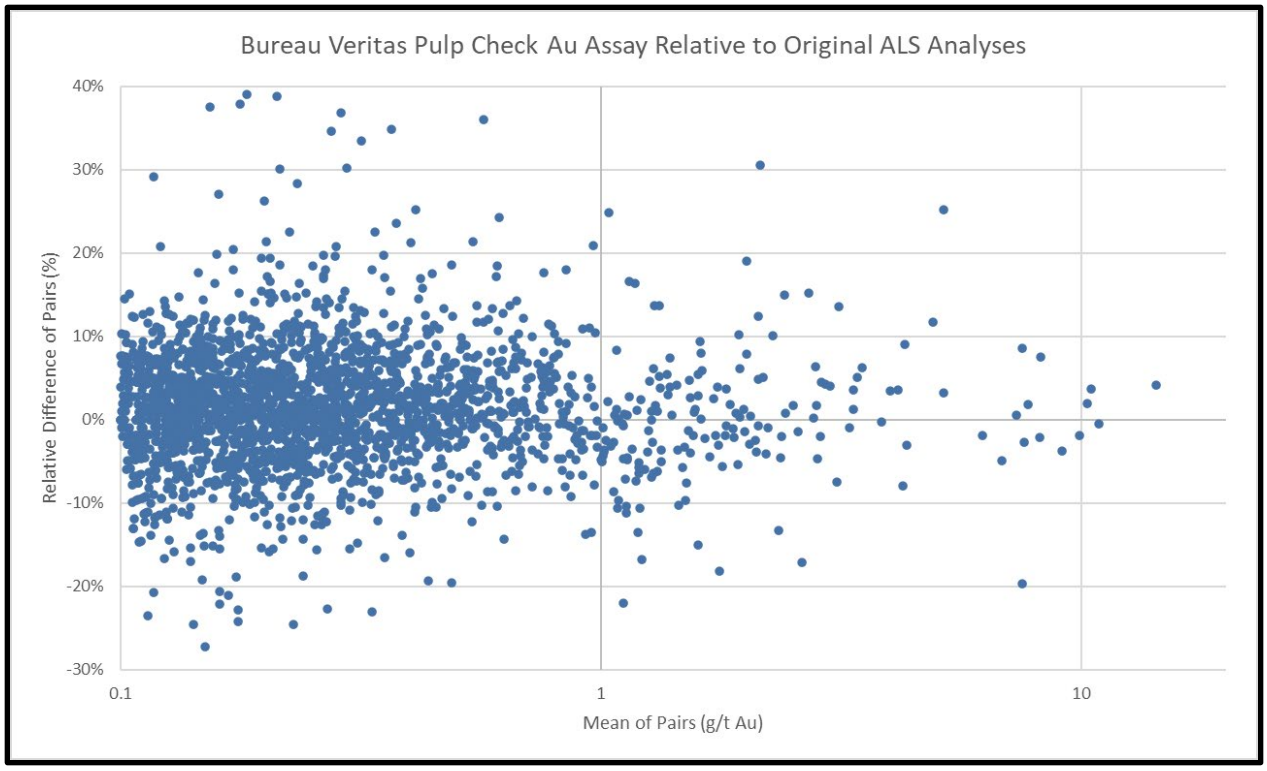


Figure 11-11: Check Assay Data Analysis

From 2024 to 2025 Drill Programs

Check assay data for SLR were not reviewed as part of the current verification process due to insufficient available information.

11.4 CYANIDE SOLUBLE GOLD ASSAY INVESTIGATION

In August 2021, it was noticed that results for cyanide soluble gold (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. Through a mutual agreement between ALS and Liberty, it was agreed that ALS would reassay all available affected sample intervals using coarse reject material stored on at the Liberty Gold exploration office.

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 SAMPLE BIAS ANALYSIS STUDY

In February 2025, SLR was retained to investigate a possible sample bias that a third party claimed may have contributed to an overstated Mineral Resource estimate for the Black Pine project. SLR was assisted by Lynda Bloom, M.Sc., P.Geo., President of Analytical Solutions Ltd. (ASL) and world expert on assay methods, sampling, and analytical procedures, in the statistical review and analysis of results.

The study arose following a review of the Project drill data (the 2024 Review) by an interested party as part of their due diligence for a potential transaction. The 2024 Review used a quantile-quantile (QQ) plot of paired Liberty Gold RC:vs diamond drilling sample data (2,134 pairs with a maximum separation distance of 15 m and an average separation distance of 8.5 m) and a comparison of the median gold values for the two data sets to conclude that the RC samples have a 13% positive bias over the diamond drilling hole samples. In consideration of the heavy reliance on RC samples in the June 1, 2024 MRE (approximately 30 times more meters drilled by RC than by diamond drilling), the review concluded that the 13% bias would translate into the MRE overstating the gold grade by a similar amount. This conclusion was influenced by the experience of the interested party at an orogenic gold deposit where studies had found that barren or low-grade fine material was being washed away during wet RC drilling practices which biased the sub-samples high due to the gold being associated with coarse fragments and the unequal (higher) preservation of those coarse fragments relative to fine material in the recovered sub-samples. The 2024 Review did not consider metallurgical studies specific to Black Pine or recognize that Black Pine is a Carlin-type deposit with different geological and mineralogical characteristics to an orogenic deposit.

SLR's approach included consideration of the geological features that contribute to the recovery of material during drilling and sampling, as well as the diamond and RC drilling and sampling procedures. With assistance from ASL, SLR created two paired datasets based on all Liberty Gold and historical data. One dataset paired the RC and DDH samples, while the other one paired RC and DDH samples composited to 3.048 m (10 ft) intervals. The pairs were generated using a Python script which found the nearest neighbor to the opposing drilling type. The pairs were then filtered to include only those within 10 m of each other. The 10 m distance was selected because there can be significant variability in the drill hole to drill hole assay values across 15 m (the distance used in the 2024 Review), however, a lower pair distance yielded few pairs for comparison. SLR used QQ and scatter plots to compare the data and concluded the following:

- There is no firm evidence that the RC drill hole assays are overrepresenting gold values at the Project.
- Complexities in the relationships between sample type, recovery, and gold grade may result in a high bias for RC versus DDH when compared using a simplified statistical approach. When the data is interrogated with various grade subsets, the apparent bias disappears. This may explain that the initial findings could be due to a selection bias as opposed to a sampling bias.
- Data below 0.1 ppm Au is hard to compare due to several detection limits for various generations of data.
- Low RQD values in DDH holes may bias gold values low due to lower core recoveries

SLR recommends updating the 1988 gold grain size fraction study described in Sletten et al, 2024, to confirm the results and expand it to show spatial representativeness, including samples from the largest areas, Discovery and

Rangefront. Additional RCH:DDH twins are recommended to be included in the next drilling campaign with the twinned data as close together as possible (ideally within 2 m).

11.6 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 oz 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 Mineral Resource estimation, interpretations, and conclusions summarized in this report.

For future drilling campaigns the SLR QP makes the following recommendations:

- An external QA/QC program (check assays at a secondary independent lab) was not implemented during 2024-2025 drilling campaigns. It is recommended that an external QA/QC check assay program be reinstated for future drilling campaigns. (Implement a check assays program with an umpire laboratory as a routine procedure, submitting pulp samples with representative grade distributions of the mineralization along with control samples. Approximately 5% of pulps should be sent for external analysis.)
- Maintain the use of CRMs that reflect the grades of the gold mineralization at Black Pine.
- Increase the insertion rate of CRMs to 1 in 25 samples as well as introducing a CRM with a higher Ag certified grade.
- Continue to monitor QA/QC data to quickly identify and correct any deviations and keep detailed records of QA/QC procedures, tests, results, and corrective actions for accountability and traceability.
- Implement and ensure that appropriate pulp blank material is submitted along with analytical samples to ensure that an assessment of sample contamination can be made for future campaigns.

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).

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.

2017 to 2023

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. This was a result of erroneously reported values from the lab and has since been rectified.
- 187 samples lack supporting assay certificates. This was a result of a typographical error from the lab where zero (0) was reported at the letter O. That information has since been rectified.

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 were a result of selection of a preferred value from multiple re-assays and have since been rectified.

2024 to 2025

SLR performed cross-validation procedures between the Black Pine assay database and ALS assay certificates. A total of 178 certificates spanning the years 2024 to 2025 were compiled and compared against Au and Ag values within the "Drilling_Samples.csv" assay database. Assay certificates were retrieved directly from the ALS Webtrieve system and used to validate the assay data contained within the database. A total of 10,335 samples were compared, representing 56% of the drilling sample dataset.

Most results align with the assay database value. A minor discrepancy was observed, where a recorded value of 127,000 ppm Ag was reported instead of the certified 127 ppm Ag. This difference is the result of a typographical error and has since been corrected.

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 QP SITE INSPECTION

Ms. Wilson 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 with respect to the evolving geologic understanding of the project and associated mineralization. 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.

Ms. Wilson 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.

As of the Effective Date of this report, Mr. Simmons visited the Black Pine Project site on June 3, 2019, October 18-19, 2019, June 22-23, 2020, May 3, 2021, August 1-2, 2023, June 2-4, 2025, August 13-15, 2025, and January 14-16, 2026. 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.

Mr. Carstensen conducted a site visit to the Black Pine property on October 31, 2023 and June 17, 2026. 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 rock storage facilities, mine facilities, access roads, and the plant and heap leach pad locations were reviewed.

12.3.1 Drill Hole Collars

Ms. Wilson 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.

12.3.2 Legacy Heap Leach Facility (LHLF)

The site was visited on June 6, 2025, by the Forte Dynamics team, including Mr. Donald Hulse, the QP for this analysis. Mr. Hulse is an SME-Registered Member, and has completed multiple resource estimates for leach pads, tailings deposits, and overburden storage deposits.

12.4 SUMMARY STATEMENT

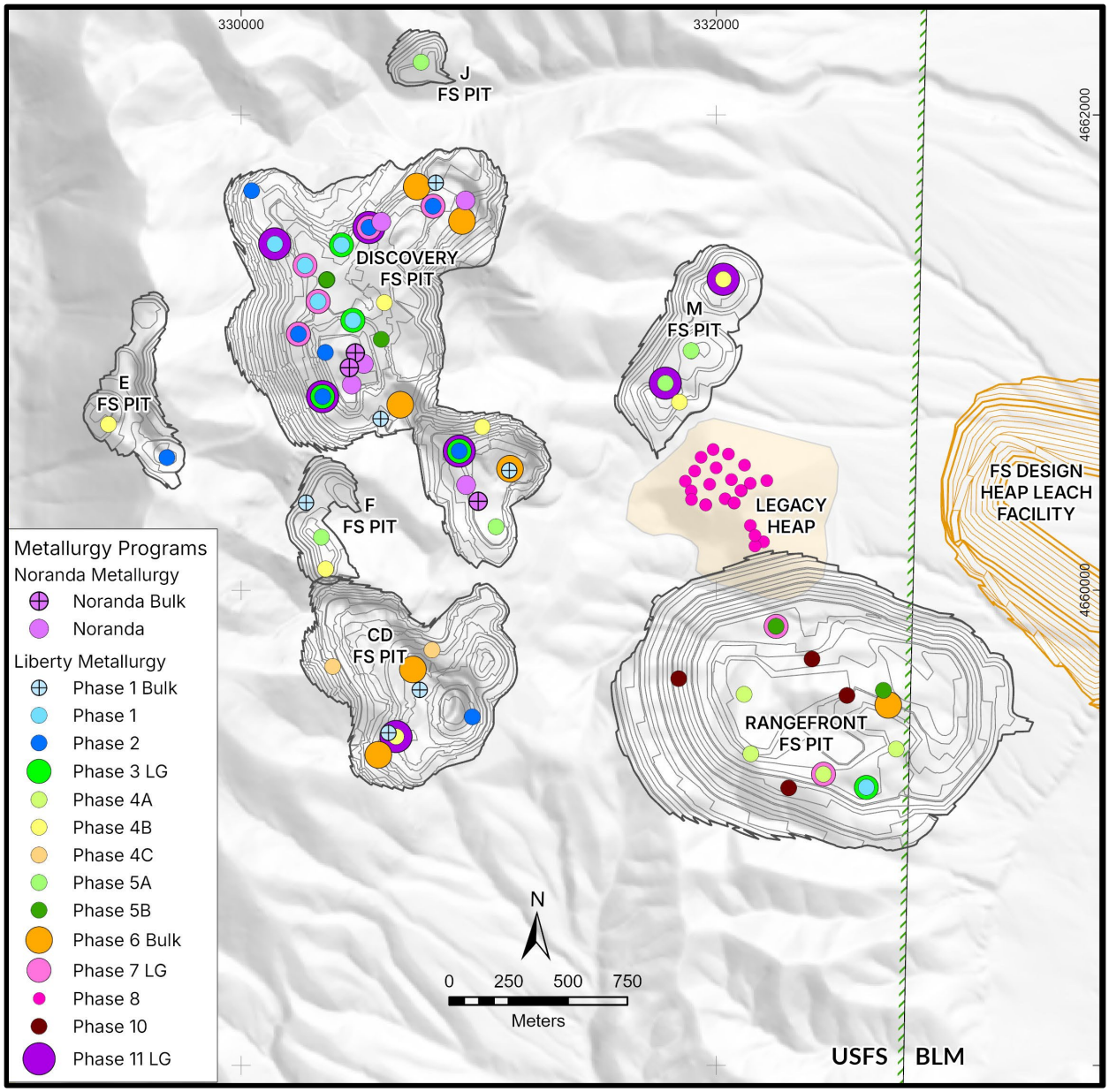
The modeling of the Black Pine resources is based on a database that includes 1,853 historical RC holes, 24 historical core holes, and 1,122 RC, 44 core holes and 26 sonic holes drilled by Liberty Gold.

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, a QP under 43-101, based on historical and Liberty Gold test results as cited. Mr. Simmons designed and supervised the various phases of the metallurgical program at Black Pine. In some cases the term “ore” is used in this section to refer to historical mineralized material tested by previous operators and consultants. Historical test parameters and results originally reported in imperial units have not been converted to metric units and in some cases Mr. Simmons prefers to use a mixture of imperial and metric units throughout the text. Metallurgical studies are described in greater detail in previous Black Pine technical reports, notably the 2024 Prefeasibility Study (PFS) and the reader is referred to these reports for more complete information and results.

A location map for all phases of Black Pine metallurgical testing conducted by Noranda and Liberty Gold is shown in Figure 13-1, and should be referenced for all phases of metallurgical testing discussed in Section 13 below. A summary of all test work completed for use in this feasibility study can be seen in Table 13-1. Note: Phase 6 ROM Bulk Sample, Phase 8, Phase 10, and Phase 11 testing is ongoing and is designed to refine accuracy for short to medium term operational mine/process planning. There is insufficient data to report results from these on-going and planned test programs at this time.



Source: Liberty Gold, 2026

Figure 13-1: Black Pine Metallurgical Testing Sample Location Map

Table 13-1: Summary of Metallurgical Test Work Completed - Black Pine Project

Sample Type	Count
Bottle-Rolls	
75 µm (DL)	220
75 µm (CIL)	212
1,700 µm (DL)	212
1,700 µm (CIL)	133
12.5 mm (DL)	25
Total Bottle Rolls	802
Columns	
12.5 mm (1/2")	82
25 mm (1")	136
>25 mm (>1")	9
Total Columns	227
Total Extraction Tests:	1,029
Total Samples:	216
Total Extraction Tests Completed:	1,029

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 the Black Pine Pre-Feasibility Study (PFS) and listed in chronological order below.

- Potter (1974)
- Ennis (undated – estimated 1975)
- Dawson (1980)
- Dix (1984 & 1990)
- Defilippi (1988)
- Yernberg (1998)
- Clemson (1998)

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 13-2). 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.

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 in 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 include: A Pit, Tallman Pit, Upper BX Pit, Lower F Zone, C/D Pit, and I-Pit.

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. Summaries of the Liberty Gold 2019-2020 bottle-roll and column leach test work are summarized in the PFS.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, from the Discovery Zone, and the area of historic drilling the in the eastern Rangefront area (KCA, 2020b).

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 a 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 of the PFS, the bulk sample and PQ core metallurgical composites reasonably sampled materials from the Discovery Zone Pola, Polb, Polc material types, 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 gaps that were minimally sampled in 2019 (Polc, Pold, and Pols).

13.3.1 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 the PFS:

- 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%.

13.3.2 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.

13.3.3 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. 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.

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.

13.3.4 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 weighted 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 weighted 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 weighted 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 weighted 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.

13.3.5 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 weighted 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 weighted average of 16.4%.

13.3.6 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).

- 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 consumption 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.

13.3.7 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).

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.

The eleven composites were subjected to the modified SMC Test at Hazen to generate data for SAG mill comminution parameters and crushing index (M_{ic}) by JKTech (Stepperud, 2020). The 2020 SMC Test® results for the 11 samples are given in Table 13-10 in the PFS. 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 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.

M_{ic} (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. M_{ic}, 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.

13.3.8 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 modeling 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 at various simulated heap heights.

It was 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 leach pad.

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. 3D geology modeling of clay

zones has been completed and is included in the Feasibility Study (FS) mine block models to facilitate mine plan process blending options.

13.4 BULK SAMPLE AND PHASE 1 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 13-2.

Table 13-2: Select Head Assays for Black Pine Mineralogy Samples

Sample ID	g/t Au		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 30 g fire assays – AAS finish.
3. Fe is the average of ICP & XRF assays.
4. Total organic carbon (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).

13.4.1 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 a P_{80} of approximately 100 µm for analysis. Selected assays of the samples are given in Table 13-2. They compare well generally with site assays, except for 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 assay method 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.

- Sulfide minerals are very insignificant, consisting of trace pyrite and very rare chalcopyrite and covellite. The latter is too rare to impact cyanide consumption.

13.4.2 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.

13.4.3 Gold Deportment:

13.4.3.1 Exposed Gold:

At a 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:
  - Contribute to 37-51% of the head grade of the five ores.
  - Are composed of 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.
  - 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:
  - Contributes to 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.

13.4.3.2 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/t Au. In the I-Pit sample, enclosed gold is substantial, representing approximately 0.3 g/t Au 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.

13.4.3.3 Refractory Gold:

- Ranges from 9% to 49% of the gold in the ores examined.
- 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.

13.4.3.4 Gold Preg-robbing by C-matter:

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

13.4.3.5 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.

13.4.4 AMTEL Mineralogy Conclusions

- A graphical summary of these deportment study results can be seen in Figure 13-2. The five samples from this stage of work are B073-7, BP73-10, BP78-12, BP87-25, and I Pit.
- Gold is present as tiny gold grains and even smaller colloidal particulates. The small size of gold grains and colloidal particulates contribute 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 with the exception of 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 clays is not an issue under CIL conditions.
- Predicted extractable gold from the deportment work versus achieved gold extraction in bottle-roll tests are tabulated below in Table 13-3. 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.

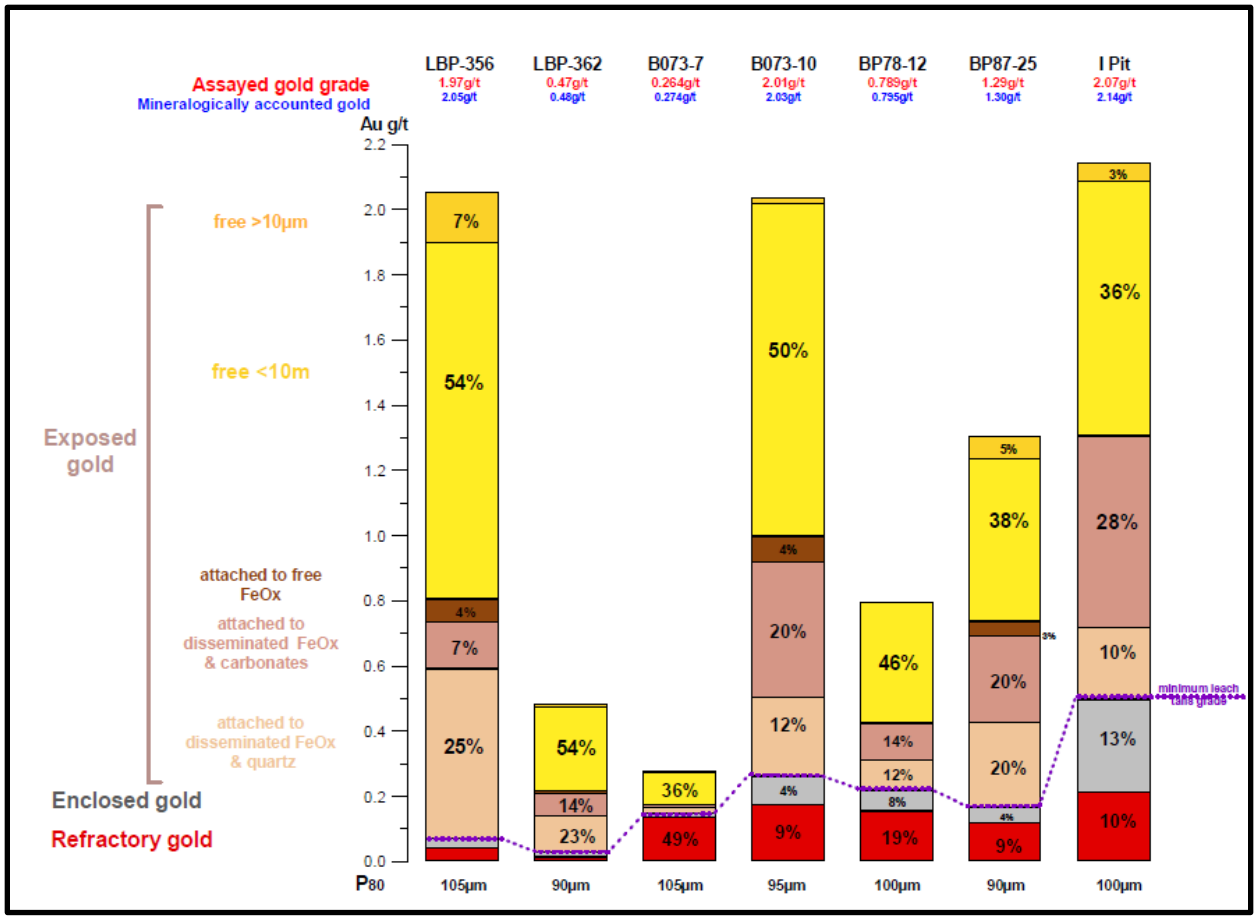


Figure 13-2: Deportment of Gold – Current and Past Ores

Table 13-3: 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

13.5 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 the 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.

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

13.5.1 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:

- 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 facility.
- 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 of 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%.

13.5.2 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 29 Black Pine metallurgical composites tested had NNP values >20 and are therefore considered to be non-acid producing.

13.5.3 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 13-4). 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 13-4: 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 target P_{80}'s and placed in columns of 10 cm and 15 cm diameters. During testing, the material was leached between 82 and 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 were 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.

13.5.4 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. In summary:

- 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 weighted 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 weighted 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/.
- Cyanide consumption for the CIL bottle-roll tests averaged 0.74 kg/t, and lime consumption averaged 0.51 kg/t.

13.5.5 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 weighted average of 78.8%.

13.5.6 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, with summarized results below:

- 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 weighted 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 weighted 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.

13.5.7 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 13-5 and the SMC comminution characterization Table 13-6.

Conclusions support the findings discussed in Section 13.3.7.

Table 13-5: 2021 Bond Abrasion Test Results

HRI No.	Client ID	Composite ID	Ai (g)
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 13-6: SMC Comminution Characterization Test Results

HRI No.	Client ID	Composite ID	sg	A	B	Black Pine Project - Comminution Characterization							
						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)
M_{ia} = coarse particle component
M_{ic} = crusher component
M_{ih} = high-pressure grinding roll component
SCSE = SAG circuit specific energy
sg = specific gravity of sample
t_a = low energy abrasion component of breakage

13.5.8 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 m in height, except for one (1) composite (BP214-55, FZ, Polc), which failed at 100 m. This was a marked improvement over Phase 1 as Phase 2 metallurgical core moved out of the Discovery Zone where Pola and Polb high-clay zones are located.

Load Permeability test apparatus and testing procedures are described in detail in the KCA 2021 report.

13.6 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 material received represented fifteen (15) individual composites. The samples received 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 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.

13.6.1 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:

- 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 of 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%.

13.6.2 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 diluted 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.

13.6.3 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 weighted 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 weighted 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.

13.6.4 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 weighted 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 weighted 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.

13.6.5 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 and summarized below:

- 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 weighted 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 weighted 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.

A somewhat unique characteristic of the Black Pine lower grade heap leach resources is that they commonly demonstrate clay and/or smoky carbon preg-borrowing characteristics. The clay or smoky carbon exists as 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. The CIL bottle-roll tests commonly have higher gold extractions than their corresponding direct leach bottle roll tests, a normal characteristic of preg-robbing or preg-

borrowing material. 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 and/or additional preg-robbing material is being liberated at finer crush sizes in the bottle roll tests. Figure 13-3 illustrates the extended leach curve witnessed in by column leach tests.

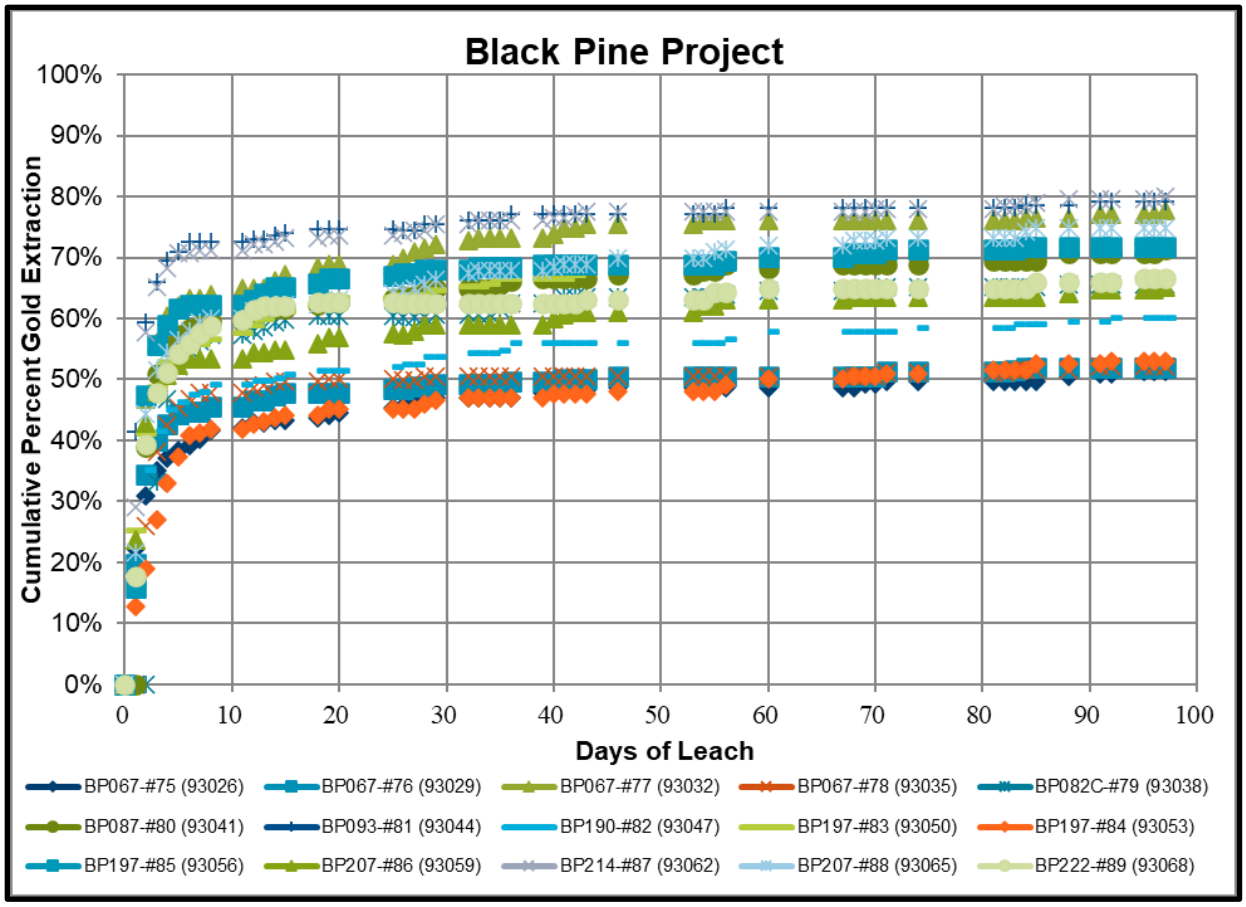


Figure 13-3: 2022 Low-Grade Variability Composites Gold Extraction vs. Days Under Leach for Column-Leach Tests

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 in height, except for composite (BP082-79), which failed at 25 m.

Refer to the KCA (2022) report for details on the load permeability test apparatus and testing procedures.

13.7 PHASE 3 MINERALOGY (RANGEFRONT POLA)

Two Reverse Circulation ore samples, representing Pola lithologic unit, from the Rangefront deposit were selected for gold deportment mineralogy study and shipped to AMTEL in London, Ontario, Canada. Select head assays for the two mineralogy samples are provided in Table 13-7.

Table 13-7: Select Head Assays for Rangefront Mineralogy Samples

Sample ID	g/t Au Independent	g/t Au Site	Fe (%) Independent	Fe (%) Site	TOC (%) Independent	TOC (%) Site
LBP 356	1.965 ±0.097	1.956	0.36	0.45	0.03	0.32
LBP 362	0.471 ±0.019	0.472	0.13	0.25	0.02	0.40

- Independent assaying of the samples was performed at ALS Vancouver. Client assays were provided.
- Independently determined gold grades are in very good agreement with the client assays.
- Independently determined organic carbon is significantly lower than the client assays. Detailed microscope examination of the material received corroborates the independent ALS values.

13.7.1 Mineralogy Summary

The ore samples were received coarsely crushed and were milled to P_{80} of 90 and 105 µm for analysis. The head assays of the samples, determined independently at ALS, are shown in Table 13-7. The head assays corresponding site assays are shown for validation. Gold assays determined independently compare very closely with those determined on site. Fe assays compare reasonably well. The TOC assays compared poorly; the TOC values determined independently are significantly lower than site.

13.7.2 General Mineralogy

- The mineralogy of the two ores examined is very simple. They contain primarily carbonates and quartz; minor quantities of clays/mica and barite; and only trace quantities of iron oxide/oxyhydroxide (FeOx) and pyrite.
- The carbonates are calcite and dolomite, with calcite the predominant carbonate. The carbonate content is 68 wt% in LBP 356 and 74 wt% in LBP 362.
- The quartz content is 21 wt% in LBP362 and 28 wt% in LBP 356.
- Mica/clays are in the form of illite/muscovite and lesser kaolinite. The total mica/clay content is 1.7 wt% in LBP362 and 3.5 wt% in LBP 356.
- Barite is in significant quantity only in LBP362; it comprises 3% of the sample weight.
- FeOx content is 0.2 wt% in LBP362 and 0.5 wt% in LBP 356. FeOx is present as hematite and goethite. In both ore samples, the very large majority of the FeOx particles are fine-grained and disseminated in quartz/clay composite particles. A direct FeOx association with carbonates is very rare.
- Sulfide minerals were rarely observed and generally as fine particles <20 µm. Pyrite particles are relatively more common; with rare arsenopyrite and chalcopyrite also present.

13.7.3 Gold Occurrence

Gold is present in two forms: gold grains and gold particulates. The differentiation is largely related to particle size.

- Gold grains: comprise free milling gold particles larger than 0.5 µm in diameter. At the P_{80} examined gold grains are largely free and to a lesser extent associated with rock particles.
- Colloidal-gold particulates: comprise tiny gold particulates smaller than 0.5 µm in diameter. They are predominantly associated with FeOx the FeOx/quartz/clay composite particles.

Both forms of gold are readily cyanidable if exposed.

13.7.4 Gold Deportment

From a cyanide perspective, gold was divided into exposed cyanidable at the examined P_{80}; enclosed cyanidable after grinding finer, and refractory to direct cyanidation which does not become exposed even after ultra-fine grinding.

The capability of C-matter to preg-rob gold was also determined. Gold deportment from a leach perspective is shown in Figure 13-2.

13.7.4.1 LBP 356 [1.97 g/t Au]

The deportment work (or mineralogic accounting for head grade) came within 104.5% of the assayed head grade.

- Exposed gold: at P_{80} of 105 µm, 97% of the gold is exposed and directly cyanidable. This gold is carried by gold grains (70%) and colloidal particulates (27%).
    - Exposed gold grains are composed of pure gold or are alloys with minor silver. The gold content ranges between 97 to 100 wt%; and the silver content between 0-3 wt%.
    - The majority of the gold grains are free; contributing 61% of the total gold or 1.25 g/t Au. A smaller quantity of gold is carried by gold grains attached to rock particles; these grains contribute 9% of the total gold.
    - Observed gold grains range in size from 1 to 53 µm. The large majority of gold particles are smaller than 10 µm in size. The average size of gold grains was measured at 5 µm.
    - Exposed colloidal gold particulates carry 27% of the gold in the sample. This gold is carried by the relatively larger gold particles, in the range of 0.2-0.5 µm.
    - Colloidal gold particulates are carried primarily by FeOx disseminated in rock and lesser by free FeOx. The average gold concentration in FeOx ranges between 98-126 ppm Au; and as high as 4,700 ppm.
- Enclosed gold: at P_{80} of 105 µm, only ~1% of the gold in LBP 356 remained enclosed (~0.03 g/t Au). This gold is mainly as colloidal gold particulates in likely less porous regions of the quartz/clay particles.
    - Size by size grade of rock particles shows that the grade of enclosed gold is largely constant irrespective of the size of the host particles. This indicates that grinding finer has minor impact to liberating more gold and that rock porosity is more important than grind in terms of exposing the gold for cyanidation.
- Refractory gold: contributes 2% of the gold in. This gold is carried by the smallest colloidal particulates which cannot be exposed even after ultra-fine-grinding.

13.7.4.2 LBP 362 [0.471 g/t Au]

The deportment of gold came within 102.4% of the assayed head grade.

- Exposed gold: at P_{80} of 90 µm, 94% of the gold is exposed and directly cyanidable. This gold is carried by gold grains (55%) and colloidal particulates (39%).
    - Exposed; gold grain composition is similar to LBP 356. Gold grains are close to pure gold with <1 wt% silver.
    - Exposed gold grains are primarily free. These grains are very small, with an average size of 4 µm. The largest gold grain observed measured 35 µm in size.
    - Exposed colloidal gold particulates are carried by FeOx disseminated in quartz/clay composites. Free FeOx particles observed only very rarely in this ore.
    - The average concentration of colloidal gold in FeOx ranged between 67-127 ppm Au; and as high as 4,500 ppm.
- Enclosed gold: at P_{80} of 90 µm, enclosed gold contributed ~3% of the gold, equivalent to 0.014 g/t Au.
    - Similar to LBP 356, grinding finer does not benefit gold liberation in any meaningful way.
- Refractory gold: contributes 3% of the gold in LBP 362, or 0.013 g/t Au.

13.7.5 Preg-robbing of Gold by C-matter

C-matter ability to preg-rob gold was determined from leach tests with and without activated carbon and with and without gold spike. Picking of C-matter for direct analysis was not possible. The test results show:

- C-matter is capable of preg-robbing gold. Preg-robbing is measurable even at very low TOC content.
- At similar TOC contents, C-matter in LBP 356 preg-robs more strongly than C-matter in LBP 362.
- At high activated carbon concentration, CIL reduced preg-robbing in LBP 356 and prevented preg-robbing in LBP 362.

13.7.6 Conclusions

- A graphical summary of these deportment study results can be seen in Figure 13-2. The five samples from this stage of work are LBP-356 and LBP-362.
- Gold is present in two forms: gold grains and colloidal gold particulates.
- Gold grains are nearly pure gold and generally very small, with average sizes of 4-5 µm. The small size is very conducive to fast leach kinetics.
- Colloidal particulates are carried by FeOx disseminated in quartz/clay composites.
- At P_{80} of 90 and 105 µm, 97% of the gold in LBP 356 and 94% of the gold in LBP 362 is exposed cyanidable. Enclosed and refractory gold components are both minor.
- At the grind fineness examined, gold does not appear to be strongly sensitive to grinding. Coarsening of the grind may be possible without great loss in recovery.
- At the grind fineness examined and based on gold occurrence alone, there is no impediment to achieving high gold recovery from both ores examined.
- Carbonaceous matter, present in both ores, is the limiting factor to gold recovery as it is capable of preg-robbing gold.
- C-matter preg-robbing capacity is greater in LBP 356 than LBP 362. Under CIL conditions, preg-robbing was reduced in LBP 356 and completely prevented in LBP 362.
- The TOC content of the samples examined was very low; and therefore, the preg-robbing of gold generally reduced. With increasing TOC content, losses of gold to preg-robbing are likely to be more substantial.

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. These procedures were the same as what is described in the original Phase 1 (KCA, 2020a) and Phase 2 (KCA, 2020b) final reports.

The variability composites were compiled from Black Pine large diameter PQ metallurgical core drilled 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 in-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 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, ICP multi-element analyses, whole-rock analyses and QXRD analyses:

- 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%.
- 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 within error of the analytical method and 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 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 2,078 ppm with an average of 90 ppm.
- Concentrations of the primary cyanide consumers were low and suggest low 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 76.8% and averaged 38.6%.

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 were 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, KCA 2023b, & KCA 2024a), 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 target P_{80}'s and placed in columns of 10 cm and 15 cm diameters. Columns 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 in Table 13-8. A total of 180 bottle roll tests are included in the summary table calculations:

Table 13-8: Summary 75 µm Bottle Roll Tests (60 comps, 120 tests)

Leach Test ID	Au Head ppm	Au Ext %	Reagents	
			Cyanide (kg/t)	Lime (kg/t)
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 Table 13-9. A total of 120 bottle roll tests is included in the summary table calculations:

Table 13-9: Summary 1,700 µm Bottle Roll Tests (60 comps, 120 tests)

Leach Test ID	Au Head ppm	Au Ext %	Reagents	
			Cyanide (kg/t)	Lime (kg/t)
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-Leach 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. Below is a high-level summary of the test results:

- 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 weighted 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 weighted 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/t 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.

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.

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. These procedures were the same as what is described in the original Phase 1 (KCA, 2020a) and Phase 2 (KCA, 2020b) final reports.

The variability composites were compiled from samples of large diameter PQ metallurgical core drilled in 2022 and 2023.

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 5A program was designed to test the new Back Range and J Zone resource areas and to in-fill gaps in current resources. A brief description of the Phase 5A 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 report entitled "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, ICP multi-element analyses, whole-rock analyses and QXRD analyses:

- 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. Selenium averaged 20 ppm and mercury 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 primary cyanide consumers were low. 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 to 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%.

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 were 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 were 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 Study time schedule. In 2025 laboratory column leach tests were performed on Phase 5A variability composites in Phase 5B testing and these results are reported below in Section 13.10.

The 1,700 µm and 75 µm direct leach and CIL bottle roll testing procedures are the same as all previous phases of testing 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 mm 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 g per liter of sodium cyanide and maintained at 0.50 g 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 pregnant solution was then siphoned off and filtered through a small column containing attrited 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 pregnant 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, 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 hrs. (10 days). The slurry was checked at 96, 120, 144, 168, 192, 216, and 240 hrs. for pH, 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-hr period, a closed-end perforated PVC pipe containing 150 g 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 hrs.) 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 mm, and three (3) 200-g portions were split out for triplicate gold and silver assays.
8. The remaining tailings material was wet screened at 0.075 mm. 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 mm. The dry screened minus 0.075-mm material was then combined with the wet screened minus 0.075 mm material.
9. Each separate size fraction was weighed, and the weights recorded. 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 Table 13-10. A total of 50 bottle roll tests is included in the summary table calculations:

Table 13-10: Summary 75 µm Bottle Roll Tests (25 comps, 50 tests)

Leach Test ID	Au Head ppm	Au Ext %	Reagents	
			Cyanide (kg/t)	Lime (kg/t)
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 Table 13-11. A total of 50 bottle roll tests are included in the summary table calculations:

Table 13-11: Summary 1,700 µm Bottle Roll Tests (25 comps, 50 tests)

Leach Test ID	Au Head ppm	Au Ext %	Reagents	
			Cyanide (kg/t)	Lime (kg/t)
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 13.9.5 of this report and in KCA 2024b. Results are summarized below:

- 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 weighted 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 weighted 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.

13.10 2025 PHASE 5B – COLUMN LEACH TESTING

Phase 5B Variability Composites column leach test work was commissioned after the Black Pine Pre-Feasibility Study was completed. Column leach head sample splits were retrieved from the original Phase 5A test program.

This Phase 5B summary reports only the new column leach test results as previous head analysis and bottle roll testing are summarized in Section 13.9 (above).

The Phase 5B Final KCA Test Report was completed in May 2025 (KCA 2025b) to include all test results conducted on Phase 5A and 5B core composites to consolidate all test work into one report (KCA Report I.D.: KCA0240016_01, dated 13 May 2025). One sample identified as BP1010C-#162, FZ,PMmx, J Zone (KCA Sample No. 98116 A) was not included in this test work due to insufficient material.

13.10.1 2025 Black Pine Phase 5B Column Leach Test Work

Column leach tests were conducted utilizing material crushed to 100% passing 37.5 mm or 100% passing 19.0 mm. The column feed material was blended with hydrated lime and/or cement prior to column loading. During testing, the material was leached for 64 or 66 days with a sodium cyanide solution.

The metal extractions are summarized in Table 13-12. Gold extractions are presented graphically Figure 13-4. The detailed results of the column leach tests can be found in KCA 2025b.

Column test extraction results in Table 13-12 are based upon carbon assays and the calculated head (carbon assays + tail assays).

- Gold extractions ranged from 37.6% to 90.8% over a 64-66-day leach period.
- Extractions were based on calculated heads which ranged from 0.125 to 4.682 ppm.
- Silver extractions ranged from 13.7% to 72.3% based on calculated heads which ranged from 0.43 to 211.0 ppm.

Sodium cyanide consumptions ranged from 0.3 to 0.91 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 consumption. Samples were blended with 0.00 to 1.7 kg/t hydrated lime and/or agglomerated with 0.00 to 4 kg/t cement.

Table 13-12: Summary of Metal Extractions and Chemical Consumptions

Comp ID	Geology		Column Leach Data						Au Balance		Ag Balance		Reagents		
	Mine Area	Formation	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)	NaCN kg/t	Lime kg/t	Cement kg/t
Phase 5B - Backrange, J Zone, Tallman, M Zone & F Zone Variability Core Composites															
BP956C-150	Backrange	Polb,Fz	25,000	23,000	9.3	Fail 100m	0.5	64d	64.3	0.412	42.2	1.92	0.55	0.75	
BP956C-151	Backrange	Fz,Polb	12,500	10,500	26.8	N/A	0.5	64d	85.5	2.107	54.7	4.75	0.61	0.00	2.3
BP956C-152	Backrange	Fz,Polb	12,500	12,500	23.7	N/A	0.5	64d	80.1	4.689	36.6	4.94	0.80	0.00	4.0
BP956C-153	Backrange	Polb,Fz	25,000	26,600	4.3	Pass	0.5	64d	89.6	1.116	51.7	2.21	0.57	0.50	
BP956C-154	Backrange	PMmx,Fz	12,500	13,000	9.3	N/A	0.5	64d	76.5	0.306	65.7	1.02	0.67	0.75	
BP968C-155	Backrange	Polb,FZ	25,000	22,600	6.6	Pass	0.5	64d	61.8	0.340	53.2	2.22	0.64	0.76	
BP968C-156	Backrange	FZ,Polb	25,000	17,200	14.6	Fail 50m	0.5	64d	85.4	4.682	68.4	3.26	0.91	0.00	2.3
BP968C-157	Backrange	Polb	25,000	24,200	4.9	Pass	0.5	64d	79.6	1.355	51.0	1.72	0.55	0.50	
BP969C-158	Backrange	Polb	12,500	12,900	5.2	N/A	0.5	64d	37.6	0.125	49.9	1.52	0.60	0.75	
BP1010C-159	J Zone	Pola,Tl	12,500	12,300	10.6	N/A	0.5	64d	81.2	0.527	21.5	2.29	0.64	0.75	
BP1010C-160	J Zone	Pola	25,000	25,200	2.6	Pass	0.5	64d	64.7	0.337	13.7	1.14	0.50	1.02	
BP1010C-161	J Zone	Pola,FZ	12,500	11,100	9.9	N/A	0.5	64d	81.0	0.919	15.8	1.41	0.65	0.76	
BP1010C-162	**Sample Lost**														
BP1008C-163	Tallman	Polc	25,000	25,800	2.5	Pass	0.5	66d	81.0	0.400	41.2	0.48	0.54	0.51	
BP1008C-164	Tallman	Polc,FZ	25,000	27,600	1.7	Pass	0.5	66d	71.0	0.276	30.1	0.43	0.53	0.50	
BP1008C-165	Tallman	Polc	25,000	24,400	2.7	Pass	0.5	64d	83.5	1.204	33.2	0.60	0.54	0.50	
BP980C-166	M Zone	FZ,Polc	12,500	10,200	13.9	N/A	0.5	66d	73.6	0.747	41.3	142.5	0.7	1.7	4.0
BP980C-167	M Zone	Polc,FZ	25,000	25,200	3.1	Pass	0.5	66d	82.6	0.242	49.9	14.9	0.3	1.5	
BP980C-168	M Zone	PMmx,FZ	12,500	12,100	5.9	N/A	0.5	66d	86.6	0.461	67.6	3.64	0.58	0.75	
BP979C-169	M Zone	Polc	25,000	24,500	4.5	Pass	0.5	66d	72.8	0.309	15.8	2.46	0.52	0.50	
BP979C-170	M Zone	Polc	25,000	23,800	3.3	Pass	0.5	66d	77.5	0.883	24.9	211.1	0.5	0.5	
BP979C-171	M Zone	Polc,FZ	25,000	25,400	5.4	Pass	0.5	66d	90.8	0.840	66.0	9.08	0.46	0.52	
BP1001C-172	F Zone	Pols	25,000	25,000	3.3	Pass	0.5	66d	67.3	0.837	23.3	1.04	0.55	0.76	
BP1001C-173	F Zone	Pols,FZ	25,000	24,000	9.4	Pass	0.5	66d	56.1	0.244	29.4	1.08	0.53	0.75	
BP1001C-174	F Zone	PmMx,FZ	12,500	10,500	24.6	Fail 100m	0.5	66d	81.8	0.187	72.3	0.94	0.54	0.00	4.0

*1. Load Permeability Tests were conducted at 25, 50, 75, and 100 m simulated heap height. Pass indicated no problem up to 100 m. N/A means column residue was not tested. Fail indicates the simulated heap height (in metes) for permeability failure.

Extraction results contained in Figure 13-4 were based upon the daily solution assays vs. the calculated head (solution assays + tailings assays).

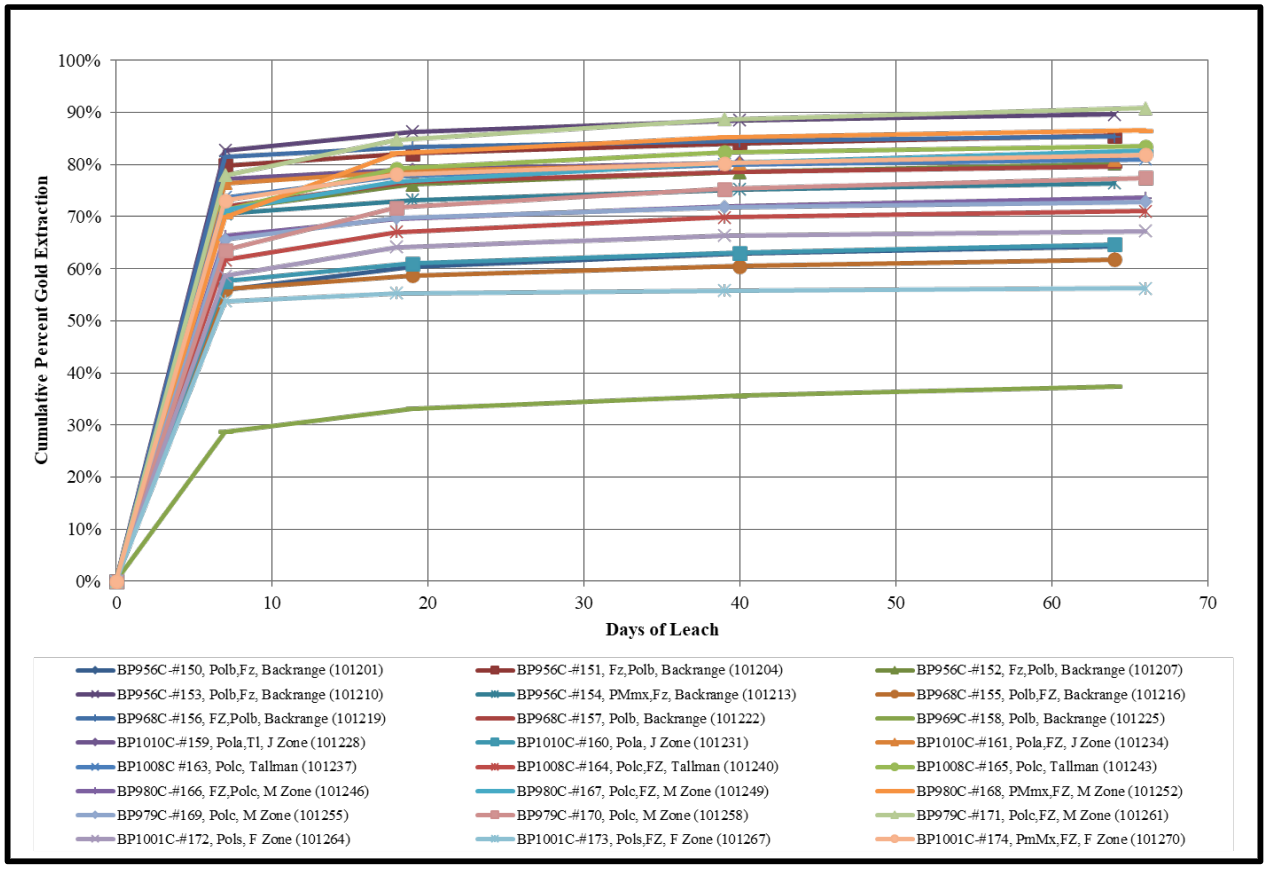


Figure 13-4: 2025 Phase 5A Gold Extraction vs. Days Under Leach for Column-Leach Tests

13.11 2026 PHASE 6 – RUN-OF-MINE BULK SAMPLE TESTING

The Phase 6 Run-of-Mine (ROM) Bulk Sample program includes bottle rolls, small column tests, and 4 ft-diameter by 20 ft-tall column leach tests are ongoing at KCA. This work will not be completed until late 2026. Results will be published once the program is finished, and a final report is issued.

13.12 2026 PHASE 7 – VARIABILITY COMPOSITE TESTING (LOW-GRADE)

In April 2025 Liberty Gold delivered forty-four (44) 5-gal buckets containing broken PQ core material representing nine (9) individual low-grade variability composites 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. These procedures were the same as used in previous Phases of work at KCA and are documented in the KCA Phase 7 Final Report (KCA 2025b). The variability composites represent resource materials from the Discovery Zone and Rangefront deposits.

The PQ core intervals samples for Phase 7 were selected from remaining metallurgical core that was not sampled in previous Phases of work. No new drilling was required.

Splits of the nine (9) composite heads were delivered to two separate laboratories for additional geo-metallurgical and environmental characterization analysis:

- ALS for ICP and gold cyanide solubility analysis
- FLS for XRD and Whole-Rock analysis

The Phase 7 - 2025 metallurgical test program was designed to test cut-off grade (0.10-0.15 ppm Au range) material to more accurately model gold recovery at lower gold grades than previously tested.

13.12.1 2025 Phase 7 Black Pine Variability Composite Head Assays

Head assay details and geo-metallurgical characterization results are in the KCA 2025b 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 13-13:

- Gold grades ranged from 0.098 ppm to 0.310 ppm and averaged 0.153 ppm.
- Silver grades ranged from 0.55 ppm to 2.23 ppm and averaged 1.29 ppm.
- Organic carbon ranged from <0.10% to 0.28% and averaged 0.19%.
- Sulfide sulfur ranged from <0.01% to 0.05% and averaged <0.02%.
- Preg-robbing analyses ranged from 0.0% to 14% (using a 1 ppm spike). Only two composites (BP214-#200 and BP207-#20365) were >0.0%.
- Preg-robbing values <10% are considered within the error band of the test procedure and are classified as non-preg-robbing.
- Copper values were very low, ranging from 17 ppm to 105 ppm and averaged 36 ppm.
- Gold cyanide solubility ranged from 24.8% to 82.6% and averaged 63.3%. Note: A number of composites show low cyanide solubility, lower than achieved in bottle roll and column leach testing, potentially due to several factors: 1. The accuracy of the cyanide solubility procedure applied to samples with very low gold grade, 2, Preg-borrowing clays and smoky carbon and/or 3. Gold encapsulated in high carbonate rocks.
- Concentrations of the deleterious elements were as follows: selenium averaged 44 ppm; mercury ranged from 1 ppm to 4.2 ppm with an average 2.3 ppm; and arsenic levels were low, ranging from 26 ppm to 139 ppm with an average of 65 ppm.
- Concentrations of the primary cyanide consumers were low and suggest minimum potential for affecting cyanide consumption rates. Copper averaged 36 ppm, nickel averaged 44 ppm and zinc averaged 117 ppm.
- Whole-rock silica (SiO_2) content ranged from 13.7% to 55.7% and averaged 37.1%.

Table 13-13: 2025 Phase 7 Variability Composite Head Assays by KCA and ALS

KCA Sample No.	Composite ID	ALS			KCA												Preg-robb Assays		
		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 ppm	Au-w/o spike ppm	Au PR %
Phase 7 - Cut-off Grade Variability Composites																			
102209 A	BP222-#197	0.138	0.105	76.1	1.32	0.48	36.4	36	2.66	7.4	5.27	0.18	5.09	0.03	0.01	0.02	1.06	0.06	0.0
102210 A	BP073-#198	0.310	0.165	53.2	2.23	0.66	29.6	40	6.80	17.0	3.48	0.28	3.20	0.02	0.01	0.01	1.11	0.11	0.0
102211 A	BP082-#199	0.109	0.033	29.8	1.51	0.64	42.4	26	4.92	18.9	3.68	0.21	3.47	0.02	0.01	0.01	1.04	0.04	0.0
102212 A	BP214-#200	0.111	0.028	24.8	1.85	1.00	54.0	27	6.94	25.7	7.75	0.21	7.54	0.02	0.01	0.01	0.90	0.03	13.0
102213 A	BP207-#201	0.207	0.180	87.0	0.79	0.38	48.2	105	88.20	84.0	9.73	0.10	9.63	0.10	0.05	0.05	1.05	0.05	0.0
102214 A	BP207-#202	0.156	0.115	73.7	1.37	0.64	46.7	21	1.32	6.3	6.01	0.16	5.85	0.05	0.02	0.03	1.07	0.07	0.0
102215 A	BP222-#203	0.121	0.100	82.6	0.55	0.24	43.8	32	6.46	20.2	9.00	0.13	8.87	0.01	0.01	<0.01	0.90	0.04	14.0
102216 A	BP541-#204	0.098	0.070	71.4	0.96	0.28	29.2	18	0.78	4.3	6.12	0.17	5.95	0.02	0.01	0.01	1.05	0.05	0.0
102217 A	BP991-#205	0.124	0.075	60.5	1.03	0.48	46.7	17	0.62	3.6	6.62	0.27	6.35	0.03	0.01	0.02	1.05	0.05	0.0

Source: GL Simmons Consulting LLC, 2025

13.12.2 Phase 7 – Direct Leach and CIL Bottle Roll Tests on 75 µm Composite Samples

Fine-milled bottle-roll leach tests were completed on each of the 9 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 2025b.

- The Direct Leach gold head grades for the composites ranged from 0.089 ppm to 0.264 ppm gold, with an average of 0.145 ppm. Gold extraction from this material ranged from 33.7% to 65.9%, with a weighted average of 54.1%.
- The CIL leach gold head grades for the composites ranged from 0.121 ppm to 0.239 ppm gold, with an average of 0.161 ppm. Gold extraction from this material ranged from 39.1% to 79.5%, with a weighted average of 62.6%.
- Cyanide consumption for the Direct Leach bottle-roll tests averaged 0.31 kg/t and lime consumption averaged 0.89 kg/t; and
- Cyanide consumption for the CIL bottle-roll tests averaged 0.72 kg/t and lime consumption averaged 0.72 kg/t.

13.12.3 Phase 7 – Direct Leach and CIL Bottle-Roll Tests on 1,700 µm Composite Samples

Coarse bottle-roll leach tests were completed on each of the 9 composites. The coarse bottle-roll test procedure and results are described in detail in KCA 2025b.

- The Direct Leach gold head grades for the composites ranged from 0.082 ppm to 0.224 ppm gold, with an average of 0.129 ppm. Gold extraction from this material ranged from 36.3% to 69.7%, with a weighted average of 58.5%.
- The CIL leach gold head grades for the composites ranged from 0.117 ppm to 0.206 ppm gold, with an average of 0.141 ppm. Gold extraction from this material ranged from 39.7% to 78.35%, with a weighted average of 60.5%.
- Cyanide consumption for the Direct Leach bottle-roll tests averaged 0.12 kg/t and lime consumption averaged 0.78 kg/t; and
- Cyanide consumption for the CIL bottle-roll tests averaged 1.15 kg/t and lime consumption averaged 0.50 kg/t.

13.12.4 Phase 7 – Column-Leach Tests on Composite Samples

All 9 composites were subjected to laboratory column-leach testing at KCA at a target P_{80} of 25 mm. Column test procedures are described in detail in KCA 2025b. Column test gold and silver extractions are based upon loaded carbon assays and tails screen assays and are in Table 13-14.

- Calculated gold head grades ranged from 0.098 ppm to 0.139 ppm and averaged 0.111 ppm. Gold extractions ranged from 32.8% to 80.2%, with a weighted average of 59.3%.
- Calculated silver head grades ranged from 0.328 ppm to 2.128 ppm and averaged 0.953 ppm. Silver extractions ranged from 7.4% to 26.8%, with a weighted average of 19.9%.
- Cyanide consumptions ranged from 0.64 kg/t to 0.93 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 0.50 kg/t to 1.50 kg/t and averaged 1.1 kg/t on 7 of the 9 columns. Two columns (BP222-#197 and BP073-#198) were agglomerated with 4.0 kg/t cement and did not require any lime to maintain alkalinity.

Gold extraction plotted versus days under leach is shown graphically in Figure 13-5 and it should be noted that these results are based upon column solution balances.

Table 13-14: 2025 Variability Column Test Results

KCA Sample No.	Comp ID	Test No	Geology		Coarse-Crush (25 mm) Column Data						Au Balance		Ag Balance		Reagents		
			Mine Area	Formation	Target P_{80} (mm)	Screen P_{80} (mm)	% -200M	Load Perm Tests*1	NaCN (g/l)	Time (days)	Au Ext %	Calc Hd Au (ppm)	Ag Ext %	Calc Hd Ag (ppm)	NaCN kg/t	Lime kg/t	Cement kg/t
Phase 7 - Cut-off Grade Variability Core Composites																	
102209 A	BP222-#197	102226	DZ	Pola	25.0	25.2	16.0	N/A	0.5	92d	61.4	0.139	21.3	1.010	0.90	-	4.0
102210 A	BP073-#198	102229	DZ	Polb	25.0	22.9	24.9	N/A	0.5	92d	32.8	0.129	16.8	2.128	0.93	-	4.0
102211 A	BP082-#199	102232	DZ	Polc	25.0	23.1	21.3	N/A	0.5	92d	41.1	0.100	26.8	1.087	0.90	1.0	-
102212 A	BP214-#200	102235	DZ	Pold	25.0	24.7	2.4	N/A	0.5	92d	69.9	0.098	23.1	1.379	0.64	1.5	-
102213 A	BP207-#201	102238	DZ	Pola,CalFm	25.0	25.1	2.7	N/A	0.5	92d	67.5	0.089	52.6	0.542	0.66	1.0	-
102214 A	BP207-#202	102241	DZ	Polc	25.0	24.5	4.8	N/A	0.5	92d	67.6	0.123	11.6	1.129	0.90	1.0	-
102215 A	BP222-#203	102244	DZ	Pold	25.0	24.8	1.5	N/A	0.5	92d	80.2	0.116	9.3	0.328	0.60	1.5	-
102216 A	BP541-#204	102247	Rangefront	Pola	25.0	25.0	5.2	N/A	0.5	92d	65.4	0.098	7.4	0.599	0.80	1.0	-
102217 A	BP991-#205	102250	Rangefront	Ppos	25.0	23.9	3.9	N/A	0.5	92d	51.3	0.108	8.2	0.372	0.70	0.5	-

Source: GL Simmons Consulting LLC, 2025

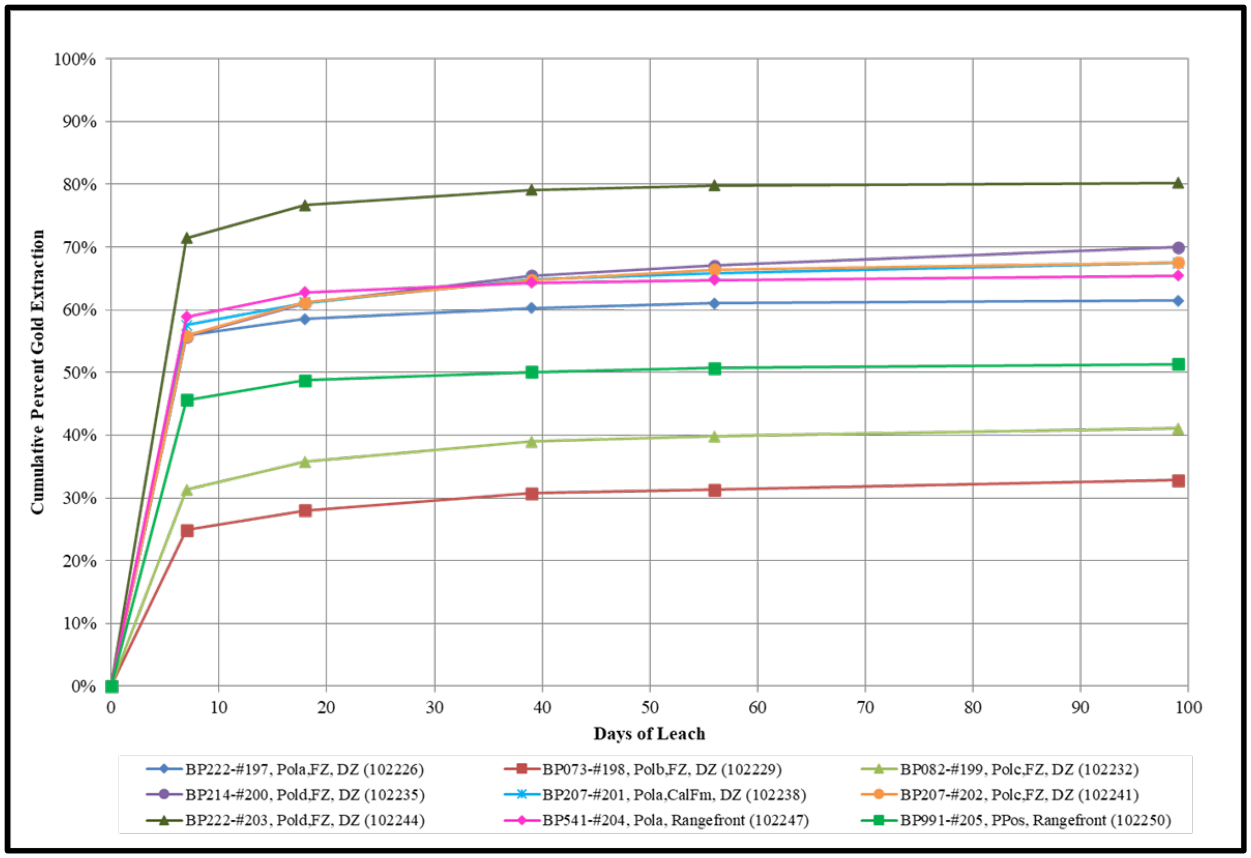


Figure 13-5: Phase 7 Gold Extraction vs. Days Under Leach for Column-Leach Tests

13.13 GOLD RECOVERY METHODOLOGY AND COMMERCIAL SCALE RECOVERY MODELS

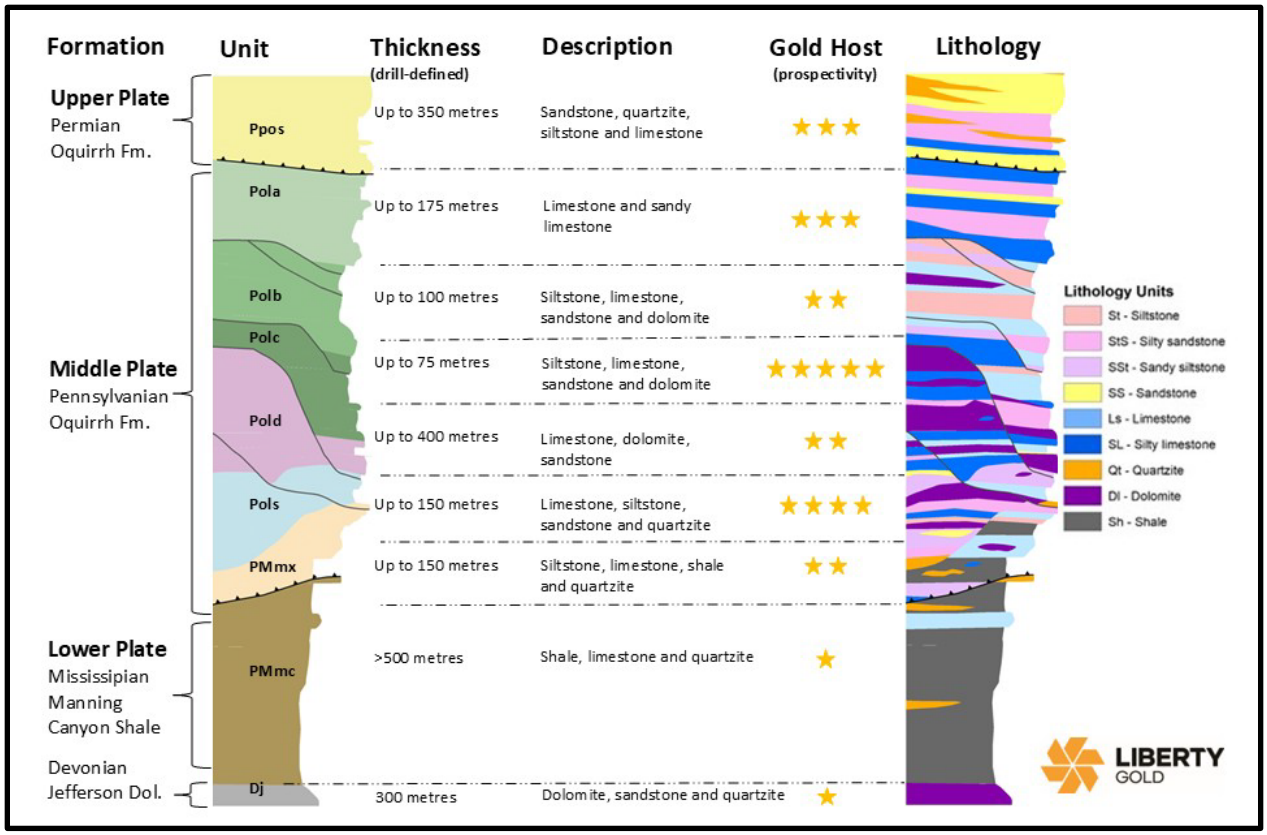
13.13.1 Gold and Silver Recovery Methodology

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: Incorporate steps 1-4 into final recovery models that reflect modeled test data met balances, commercial scale inefficiencies and deductions for solution losses.

13.13.2 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 13-6. 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.



Source: Liberty Gold, 2026

Figure 13-6: Black Pine Lithology

To facilitate gold and silver recovery modeling of the various lithologic units the following 3D gold cyanide solubility domains 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%. This mineralized material is always classified as waste, to be discarded with other mine waste or stockpiled separately.

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 degrees of gold cyanide solubility related to the following characteristics.

Factors that positively impact gold cyanide solubility:

- Medium to high-grade mineralization
- Decalcification in low Mg-rocks (higher porosity and permeability)

Factors that negatively impact gold cyanide solubility:

- Low grade, near economic cut-off, has naturally lower cyanide solubility, especially in Mg-elevated dolomitic rocks due to gold in unfractured carbonate rock mass.
- 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.
- Preg-borrowing residual "smoky" carbon exists at Black Pine that acts very similar to the preg-borrowing clays.
- Very minor sulfides, containing gold in solid solution, are present that inhibit gold cyanide solubility.
- 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, that could affect cyanide consumption.
- Ultra-fine encapsulation of gold in carbonates or quartz

ROM recovery models have been developed for heap leach P_{80} feed sizes ranging up to 250 mm (10 in). To assist in estimating the commercial ROM feed size, for each MET unit, all metallurgical cores have been visually logged to estimate the percentage of +150 mm (+6 in) core in all variability composites that were tested. Refer to Table 13-15 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.

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 into gravel sized fragments via either blasting or simple load/haul/dump mechanical work.

The assumed P_{80} feed sizes associated with each ore type have been validated with detailed core logging and fragmentation modeling completed as part of this Feasibility Study. Drill and blast designs have been aligned with these P_{80} targets.

Black Pine is noted for its consistently flat gold extraction vs. P_{80} feed size response. This is primarily due to gold being deposited in fractures, and in strongly decalcified and/or brecciated carbonate rocks. However, where dolomite is present (visually, or in rocks with Mg content >2.0%) rocks are proven to be harder to break and have fewer natural fractures as evident in Table 13-15. In these cases, 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 13-15: ROM P_{80} Modeling

	Feet	Ft(%)	Mg %	+6" (%)	Modeled P_{80}
Ppos Met-1: AuCN >65%	**828**	**8.3**	**0.28**	**7.9**	
Discovery Zone Ppos Met-1: AuCN >65%	116.9	1.2	0.11	1.2	**100 mm**
Rangefront Ppos Met-1: AuCN >65%	711.3	7.1	0.31	9.0	
Pola Met-1: AuCN >65%	**2,271**	**22.6**	**0.73**	**17.1**	
Discovery Zone Pola Met-1: AuCN >65%	143.7	1.4	0.55	8.6	**150 mm**
E Pit Zone Pola Met-1: AuCN >65%	64.3	0.6	1.35	28.2	
J Zone Pola Met-1: AuCN >65%	1611.9	16.1	0.72	18.1	
Rangefront Pola Met-1: AuCN >65%	451.5	4.5	0.75	14.5	
Polb Met-1: AuCN >65%	**813**	**8.1**	**0.50**	**13.1**	
Discovery Zone Polb Met-1: AuCN >65%	451.5	4.5	0.75	14.5	**125 mm**
E Pit Zone Polb Met-1: AuCN >65%	107.5	1.1	0.28	7.8	
M Zone Polb Met-1: AuCN >65%	31.0	0.3	0.25	13.9	
Backrange Polb Met-1: AuCN >65%	222.9	2.2	0.13	12.6	
Polc Met-1: AuCN >65%	**1,928**	**19.2**	**1.72**	**33.7**	
Discovery Zone Polc Met-1: AuCN >65%	1586.0	15.8	1.85	34.7	**200 mm**
M Zone Polc Met-1: AuCN >65%	342.0	3.4	1.14	29.0	
Pold Met-1: AuCN >65%	**1,071**	**10.7**	**4.63**	**41.5**	
Discovery Zone Pold Met-1: AuCN >65%	625.1	6.2	3.99	45.4	**250 mm**
F Zone Pold Met-1: AuCN >65%	52.5	0.5	4.82	56.9	
C/D & I Pit Pold Met-1: AuCN >65%	393.2	3.9	5.63	33.3	
Pols Met-1: AuCN >65%	**1,246**	**12.4**	**1.66**	**28.4**	
Discovery Zone Pols Met-1: AuCN >65%	540.9	5.4	2.12	31.9	**200 mm**
F Zone Pols Met-1: AuCN >65%	263.6	2.6	1.40	23.4	
C/D & I Pit Pols Met-1: AuCN >65%	441.9	4.4	1.23	27.1	
PmMx All Zones Met-1: AuCN >65%	**262.1**	**2.61**	**1.08**	**27.6**	
PmMx All Zones Met-1: AuCN >65%	262.1	2.6	1.08	27.6	**200 mm**
ALL Zones Met-2: AuCN >50%, <65%	**679**	**6.8**	**1.07**	**17.6**	
Discovery Zone Polb Met-2: AuCN >50%, <65%	421.3	4.2	0.79	13.5	**150 mm**
Discovery Zone Polc Met-2: AuCN >50%, <65%	66.2	0.7	1.78	25.3	
Backrange Polb Met-2: AuCN >50%, <65%	142.2	1.4	0.13	10.4	
J Zone Pola Met-2: AuCN >50%, <65%	49.7	0.5	5.15	62.7	
C/D Pit Pols Met-2: AuCN >50%, <65%	N/A	N/A	N/A	N/A	
ALL Zones Met-3: AuCN >25%, <50%	**324**	**3.2**	**2.53**	**32.4**	
Rangefront Ppos Met-2: AuCN >50%, <65%	58.2	0.6	0.16	2.6	**200 mm**
Discovery Zone Polc Met-3: AuCN >25%, <50%	114.0	1.1	1.01	37.2	
Discovery Zone Pold Met-3: AuCN >25%, <50%	113.5	1.1	5.84	45.2	
Backrange Polb Met-3: AuCN >25%, <50%	38.2	0.4	0.89	25.2	

13.13.2.1 Durability Modeling

The Black Pine ore deposits contain discrete high-clay and blocky ore zones of mineralization.

High-clay ores have the potential to blind or plug the leach pad and retard or block solution flow through the heap, resulting in either a delay or loss of predicted gold recovery.

Blocky ores (generally high-Mg dolomitic material) have little fines and, if placed on the leach pad without blending, can result in solution channeling and reduced gold recovery.

High-clay and blocky ore types are planned to be blended at the mining bench face or on the leach pad before applying cyanide leach solution. Ideal or optimized gold heap leaching is achieved when the leach pile is equally saturated with cyanide solution such that the rate of solution percolation through the heap is relatively consistent from lift to lift.

Models for clay and blocky material have been developed and are included in the geology and mine 3D block models. The clay model was developed by identifying commonalities between samples that failed compacted permeability testing at any simulated height. A sample is considered to have failed the compacted permeability test if the flow rate is less than 10x the assumed application rate onto the heap at any pressure level during the simulated compaction, the slump is over 15%, or if any agglomeration is required for adequate flow in the column. Agglomeration in the column test does not mean agglomeration is required at ROM scale due to the significant difference in particle size. The ratio of aluminum to magnesium (Al/Mg) predicts compacted permeability failure of metallurgical samples well and can be further improved by including visually identified clay intervals from drill hole sample logs. Material with an Al/Mg ratio greater than 1.4 or clay intervals with greater than 3 intensity (on a 1 to 5 scale) not only fails compacted permeability testing but also exhibits increased swelling clay concentrations and a higher percentage of minus-200 mesh material. A Clay wireframe domain was built in Leapfrog Geo using an indicator interpolate estimation defined by Al/Mg > 1.4 or Clay > 3. The Clay wireframe was then evaluated to the block model to code individual blocks within the wireframe as Clay. These Clay blocks were required to be blended as detailed in Section 16.10.

The clay model was validated against a historic area of A Pit where mining records from the 1990s indicated that significant clay material was encountered, and the current model would have predicted problematic clay material in that area.

Blocky material was additionally coded into the model as blocks with magnesium greater than or equal to 2%, which is based on the fact that the blocky material is dominantly dolomitic. Blocky material will require tighter drill spacing to achieve adequate fragmentation and will require blending to reduce solution channeling on the heap. A Blocky wireframe domain was built in Leapfrog Geo using an indicator interpolate estimation defined by Mg ≥ 2% within carbonate lithologies (see Section 7.2). The Blockly wireframe was then evaluated to the block model to code individual blocks within the domain as Blocky. Model blocks coded as Blocky were required to be blended as detailed in Section 16.10.

Material not coded as Clay or Blocky was determined to be Durable; therefore, three material types exist in the block model with respect to durability. Durable material can be stacked onto the heap without any constraints on blending.

13.13.3 Gold and Silver Recovery Model Equations for ROM Heap Leaching

13.13.3.1 Gold Recovery

Gold recovery equations for MET1, MET2, and MET3 (oxide ore types) are shown in Table 13-16. A conventional Power Law Fit Equation is used to model the final commercial scale gold recovery for the various ore types. HG in the gold recovery equation column refers to the gold head grade in the mine block model.

Table 13-16: Black Pine – ROM Gold Recovery Equations

Black Pine ROM Heap Leach Au Recovery Equations		*REC% =a+b*HG^c*
MET Code	**Lithology and P_{80}**	**Equation**
MET1	Pola: DZ ROM (150 mm)	= 95.23 + -16.88 × HG^(-0.408)
MET1	Pola: E Pit + J-Zone ROM (150 mm)	= 81.38 + -1.65 × HG^(-1.000)
MET1	Pola: Rangefront ROM (150 mm)	= 96.35 + -11.02 × HG^(-0.527)
MET1	Polb: DZ, E-Pit & M-Zone ROM (125 mm)	= 72.76 + -0.69 × HG^(-1.000)
MET1	Polb: Backrange ROM (125 mm)	= 79.23 + -1.41 × HG^(-1.000)
MET1	Polc: DZ & M-Zone ROM (200 mm)	= 94.05 + -16.89 × HG^(-0.326)
MET1	Pold: DZ ROM (250 mm)	= 89.95 + -11.69 × HG^(-0.306)
MET1	Pold: F-Zone + C/D/I Pit ROM HL (250 mm)	= 56.01 + -0.39 × HG^(-1.000)
MET1	Pols: DZ + F-Zone + C/D/I Pit ROM (200 mm)	= 96.43 + -18.58 × HG^(-0.509)
MET1	PPos: DZ + Rangefront ROM (100 mm)	= 95.59 + -11.09 × HG^(-0.458)
MET1	PmMx: Met-1 ROM (200 mm)	= 79.05 + -1.01 × HG^(-0.619)
MET2	ALL Zones: Met-2 ROM (150 mm)	= 63.92 + -1.61 × HG^(-1.000)
MET3	ALL Zones: Met-3 ROM (200 mm)	= 48.37 + -1.55 × HG^(-1.000)

Three example plots are provided below in Figure 13-7, Figure 13-8, and Figure 13-9 to demonstrate Gold Recovery Variability vs. Gold Head Grade for the following three Black Pine ore types: MET-1 Pola: DZ ROM (150 mm), MET-1 Pold ROM (250 mm) and MET-2 ALL ZONES ROM (150 mm).

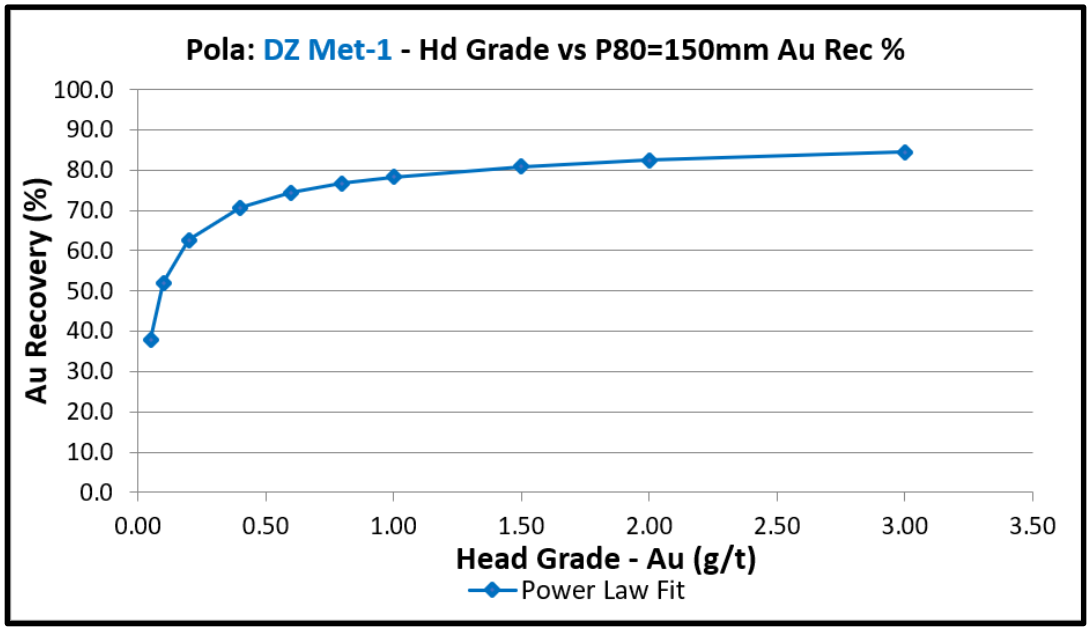


Figure 13-7: DZ Met-1 Pola – Gold Recovery vs Head Grade

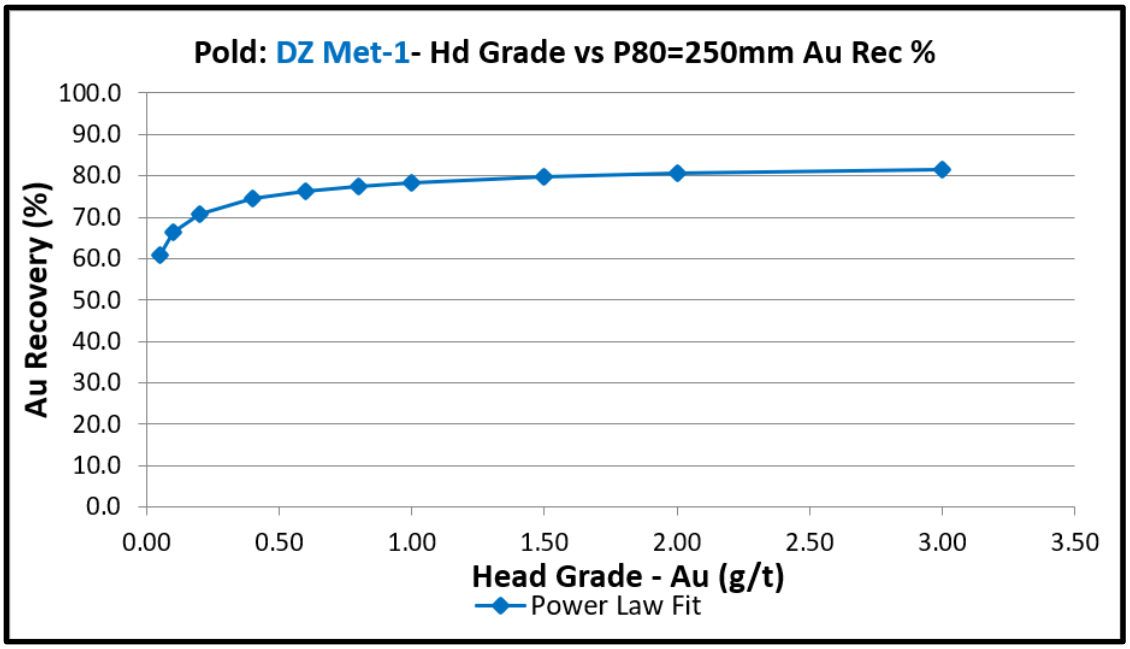


Figure 13-8: DZ Met-1 Pold - Gold Recovery vs Head Grade

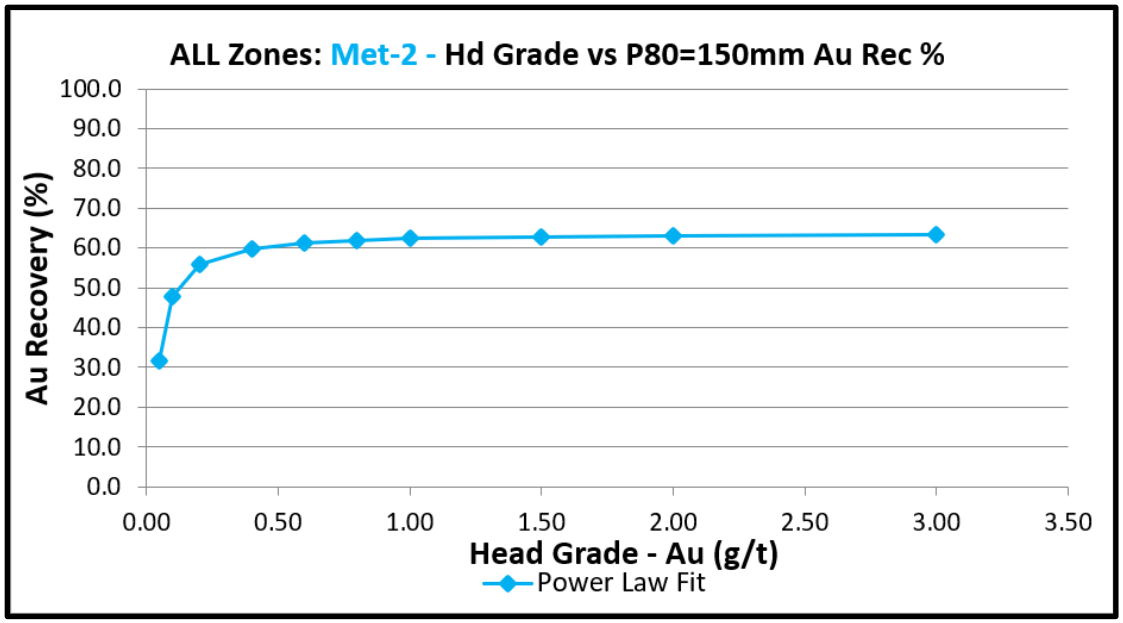


Figure 13-9: ALL Zones Met-2 - Gold Recovery vs Head Grade

13.13.3.2 Silver Recovery

Black Pine silver head grades are low, with the exception of M-Zone and Tallman. Modeled zones for silver are different than for gold, see Table 13-17.

Table 13-17: Black Pine – ROM Silver Recovery Equations

Black Pine ROM Heap Leach Ag Recovery Equations		*REC% =a+b*HGc*
MET Code	**Lithology and P_{80}**	**Equation**
MET1	DZ,Epit+FZone+C/D/I Pit+JZone - Met-1	= 14.32 + -6.79 × HG^(-0.640)
MET 1, 2 & 3	MZone + Tallman - ALL Met Zones	= 83.16 + -70.39 × HG^(-0.190)
MET1	Rangefront - Met-1	= 26.13 + -22.29 × HG^(-0.190)
MET 1, 2 & 3	Backrange - ALL Met Zones	= 38.53 + -3.42 × HG^(-1.000)
MET2 (no MZ or BR)	ALL Deposits - Met-2 (Except MZone and Backrange)	= 73.92 + -59.74 × HG^(-0.140)
MET3 (no MZ or BR)	ALL Deposits - Met-3 (Except MZone and Backrange)	= 15.11 + -1.17 × HG^(-0.990)

13.14 REAGENT CONSUMPTIONS

Reagent consumptions and requirements, including cyanide and lime, were estimated by M3 based on metallurgical test work completed to date for the Black Pine materials and are summarized below.

13.14.1 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 leach pad 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 0.20 kg/t of ore. As part of the feasibility study, a heap leach solution model was developed for all years of production. That model yielded estimated heap leach solution pad application flow rates for all years of production. This model allowed for the transition from calculating cyanide consumption using ore tonnage to calculating cyanide consumption using process solution flow rates which is commonplace in operation. An estimate of 200 ppm cyanide concentration in the application solution was considered as the design operating point, with a residual cyanide concentration of 25 ppm in the solution discharging from the pad, for a resultant cyanide addition rate of 175 ppm (0.175 kg/t of solution).

Based on the total estimated cyanide addition, the resultant cyanide addition rate per tonne of ore can be back-calculated as 0.23 kg/t of ore. This is higher than the equivalent 0.175 kg/t of solution, which is due to LOM application solution to ore ratio being greater than 1.00. This is also slightly more conservative than the cyanide consumption rate estimated via laboratory testing and subsequent scaling for commercial operation.

13.14.2 Lime Consumption

Lime requirements for pH control ranged from 0.50 to 1.52 kg/t and averaged 0.97 kg/t. Converting to commercial quicklime requirements, 1.00 kg/t is estimated for ROM ore pH control.

13.15 SUMMARY

The QP is of the opinion that samples tested are sufficiently representative to accurately support the conclusions summarized herein. Additional work is on-going and some work is planned to fill gaps on new resource additions, refine operations knowledge on low-grade (cut-off grade) composites in Rangefront and evaluation of the economic potential of re-leaching at least a portion of the Legacy Heap Leach Facility (LHLF) materials and Legacy Rock Dumps.

Summary of the five (5) additional metallurgical programs that are either in progress or planned to be completed:

- Phase 6 - Seven ROM large diameter column (4 ft x 20 ft) are in-process at KCA
- Phase 8B - LHLF sonic drill samples are being tested at KCA to evaluate the potential for re-leaching at least a portion of this material (ongoing)
- Phase 10 - New Rangefront Variability Met Testing, covering gaps in new resource definition drilling (ongoing)
- Phase 11 - Additional Rangefront low-grade met testing (ongoing)
- Phase 12 - Met testing on Legacy Rock Dumps planned once current drilling of these resources is completed. (Planned)

14 MINERAL RESOURCE ESTIMATES

14.1 SUMMARY

Mineral Resources for the Black Pine Project were estimated in accordance with NI 43-101 guidelines. Updating of the gold estimation domains was completed by Liberty Gold and SLR between October and November 2025 and reviewed by SLR prior to use in estimation. Estimation of the gold Mineral Resources was completed by SLR between November 2025 and December 2025. Modeling and estimation of the silver Mineral Resources took place in May 2026. Estimation of the legacy heap leach facility (LHLF) was completed by Forte Dynamics (now part of SLR) in November 2025. All work was completed by or under the supervision of the SLR QPs.

A Mineral Resource estimate with an effective date of January 31, 2026, was published in a News Release dated February 10, 2026 (Liberty Gold Corp 2026). This section details the gold estimate supporting the February 10, 2026 news release, as well as the addition of a silver resource in the M and Discovery Zones, and the inclusion of the LHLF following completion of a sonic drill program and block model estimation by Forte Dynamics in 2025. The HLF, which remains following Pegasus' mining operations from 1991 to 1997, has been reclaimed, though recent studies suggest that there remains recoverable gold within it. The Mineral Resource estimate presented here supersedes any previously stated Mineral Resources for the Black Pine Property. The Effective Date of the Resource Estimate is April 30, 2026.

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 with 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 re-blocked to 10 m x 10 m x 5 m 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 Mineral Resources, with the exception of the LHLF, were reported within a conceptual open pit shell generated in Whittle, at a 0.1 g/t gold cut-off grade. The HLF was reported in its entirety at the same cut-off grade. 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 – April 30, 2026

Material Type	Zone	Classification	Tonnage (kt)	Average Grade		Contained Metal	
				Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
In-Situ	CD	Indicated	37,614	0.28	—	336	—
		Inferred	12,223	0.2	—	80	—
	Discovery	Indicated	188,399	0.35	1.35	2,097	8,147
		Inferred	27,503	0.19	1.03	169	912
	E	Indicated	9,657	0.26	—	82	—
		Inferred	11,825	0.22	—	82	—
	F	Indicated	6,543	0.32	—	66	—
		Inferred	1,217	0.24	—	10	—
	J	Indicated	3,601	0.33	—	38	—
		Inferred	12,731	0.25	—	104	—
	M	Indicated	7,324	0.51	2.57	120	606
		Inferred	5,406	0.21	1.25	36	217
	Rangefront	Indicated	237,679	0.26	—	1,998	—
		Inferred	59,787	0.19	—	371	—
Total In-Situ		**Indicated**	**490,816**	**0.30**	—	**4,737**	**8,754**
		Inferred	**130,691**	**0.2**	—	**851**	**1,133**
Backfill and Waste Dumps		**Indicated**	**0**	—	—	**0**	—
		Inferred	**18,144**	**0.25**	—	**145**	—
HLF		**Indicated**	**30,844**	**0.19**	—	**190**	—
		Inferred	**0**	—	—	—	—
Total		**Indicated**	**521,660**	**0.29**	—	**4,927**	**8,754**
		Inferred	**148,835**	**0.21**	—	**996**	**1,133**

Notes:

1. The Mineral Resource has an effective date of April 30, 2026.
2. The Mineral Resource was prepared by Ms. Valerie Wilson, P.Geo., Principal Resource Geologist at SLR Consulting (Canada) Inc. (SLR) but for the LHLF estimate, which was prepared by Mr. Don Hulse, RM-SME of Forte Dynamics. Ms. Wilson is the Qualified Person (QP) responsible as defined under NI 43-101 for the Mineral Resource estimate but not for the LHLF material; Ms. Wilson is independent of Liberty Gold. Mr. Don Hulse is the QP for the LHLF estimate and is independent of Liberty Gold. CIM (2014) definitions were followed for Mineral Resources.
3. Mineral Resources are estimated at a gold cut-off grade of 0.10 g/t using a long-term gold price of US$2,800 per ounce.
4. Gold is reported for all blocks of in-situ, backfill, waste dump, or LHLF material. The waste dumps and backfill are limited to the CD, Discovery, F, J, and Rangefront zones. The LHLF is within the Rangefront zone, with the northern portion crossing into M Zone.
5. Silver is reported only for those blocks that meet or exceed the gold cut off grade of 0.10 g/t Au. It is limited to the Discovery and M zones, areas which are supported by analytical results from 4-acid digestion at a drill spacing threshold for gold class criteria.
6. Mineral Resources are estimated using variable recoveries derived from metallurgical studies.
7. Bulk density is variable by rock type and ranges between 2.43 t/m^3 and 2.63 t/m^3 in the mineralized domains. Quaternary deposits, LHLF, Backfill, and Waste Dumps were assigned a value of 1.8 t/m^3.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are inclusive of Mineral Reserves.
9. In-situ, backfill, and waste dump material are reported within conceptual open pit shells. Silver grades were not used in the pit optimization.
10. LHLF material is reported within the LHLF wireframe above a 0.10 g/t Au cut-off grade. Silver is not considered to be recoverable within the LHLF and is not reported.
11. The QPs for this Mineral Resource estimate are not aware of any environmental, permitting, legal, title, taxation, socio economic, marketing, political, or other factors that could materially affect the Mineral Resource Estimates presented in this news release.
12. Numbers may not add or multiply correctly due to rounding.

14.2 COMPARISON TO PREVIOUS RESOURCES

Several changes in economic and technical parameters were made to better align with current metal prices and inputs used for the feasibility study. Key changes relative to the previous Mineral Resource estimate are:

- Additional drill holes, with the majority of new drilling in the Rangefront and Discovery areas. The infill drilling was designed to convert Inferred resources to Indicated, and to expand the resource at Rangefront.
- Updated metallurgical domains and recovery equations for gold.
- An increase in gold price from $2,000/oz to $2,800/oz.
- Small changes to mining cost inputs during pit optimization to align with FS work.
- Addition of a 30 m low-grade buffer surrounding the gold estimation domains. This buffer was included in the Inferred category.
- The changes detailed above were captured in a Mineral Resource estimate effective January 31, 2026 (Liberty Gold Corp 2026). The block model supporting the April 30, 2026 Mineral Resource is unchanged but for the inclusion of the gold in the Indicated category within the LHLF following completion of a sonic drilling program in 2025, and the inclusion of silver following a re-assay program.

A comparison of the April 30, 2026 Mineral Resource estimate with the one from June 1, 2024 is presented in Table 14-2. Silver is not included in the comparison as it was not reported in the previous resource.

Table 14-2: Comparison of Previous and Current Mineral Resource Estimates

Class	June 1, 2024			April 30, 2026			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	402,564	0.32	4,163	521,660	0.3	4,927	30%	-6%	18%
Inferred	97,680	0.23	712	130,691	0.2	996	34%	-13%	40%

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 this determination. Capital costs, including sustaining capital, have been excluded. The parameters used to determine the cut-off grade are listed in Section 14.12.1 – Pit Optimization.

14.4 RESOURCE DATABASE

Within the Black Pine property, Liberty Gold and its predecessors have completed 3,069 holes (1,853 historical RC holes, 24 historical core holes, 1,122 Liberty Gold RC holes, 44 Liberty Gold core holes, and 26 Liberty Gold sonic holes). 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 m in the center of the Discovery, C, D, E, and F areas, to an average of 50 m in the Backrange, J, M, and Rangefront areas. Drill hole spacing is up to 200 m 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 removed from the compositing (and estimation) process. Table 14-3 describes the drilling and meterage used for the Mineral Resource estimate for gold.

Table 14-3: Summary of Drill Holes supporting the Block Model Estimation by Company

Company	Core		RC		Sonic*	
	Number of Holes	Meters (m)	Number of Holes	Meters (m)	Number of Holes	Meters (m)
Gold Resources	3	135	13	1,083	-	-
Newmont Mining	-	-	37	3,119	-	-
Noranda	4	245	532	51,366	-	-
Pegasus Mining	17	1,230	1,205	124,692	-	-
Pioneer Nuclear	-	-	28	2,458	-	-
Western Pacific Resources	-	-	38	7,920	-	-
Historical Total	**24**	**1,609**	**1,853**	**190,638**	**0**	**0**
Liberty Gold	44	8,286	1,122	269,627	26	1,448
Project Total	**68**	**9,895**	**2,975**	**460,265**	**26**	**1,448**

*Note: The 2025 sonic drilling program was designed to test the LHLF. 24 of the 26 holes were drilled in the leach pad. The remaining 2 holes targeted a waste dump in the CD area.

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 Mineralization

Silver was historically recovered as a by-product of the gold mining (Annual Report on Form 10-K for the Fiscal Year Ended December 31, 1997). Silver occurs at Black Pine in low concentrations and is associated with at least two distinct mineralizing events. The dominant, and more extensive silver event is closely correlated with the stratabound gold mineralization. An earlier silver event associated with base metals including Pb, Zn, Cd and is interpreted to occur in structures oriented at higher angles relative to the stratabound system.

14.4.2 Silver Re-Assaying

Historical operators selectively assayed for silver using fire assay (FA) analysis, whereas Liberty Gold routinely analyzed all holes for silver using an aqua-regia (two-acid) digestion with AA finish. However, this method is not a total digestion for silver and not sufficiently precise for use in Mineral Resource estimation. To better quantify silver grades, Liberty Gold re-assayed 6,254 pulps in 2026 using four-acid digestion. Pulps were selected based on grade, drill hole location, and the availability and condition of the original pulp material. The Discovery and M Zones were prioritized as they contain the majority of the Project's silver mineralization. The objective was to increase spatial coverage of fire assay and four-acid digestion samples to match the drill spacing supporting a classification of Indicated, 60 m or less, to allow the silver estimate to be classified and reported alongside the gold estimate.

Table 14-4 lists the number of holes and meters used to inform the silver estimate, as well as the analytical techniques used.

Table 14-4: Summary of Drill Holes Supporting the Block Model Silver Estimation by Company and Method

Company	Number of Drill Holes (RC and Core)	Total Meters Drilled (m)	Meters Sent for Assay (m)[1]	Digestion/Method[2]	Laboratory Code
Gold Resources	16	1,218	1,132	FA	-
Newmont Mining	37	3,119	3,012	FA	-
Noranda	536	51,611	49,943	FA	-
Pegasus Mining	1222	125,922	121,141	FA	-
Pioneer Nuclear	28	2,458	2,352	FA	-
Western Pacific Resources	38	7,920	6,073	FA, 2-acid ICP/MS	ALS ME-MS41
Liberty Gold[3]	1,166	279,361	265,401	2-acid ICP-MS, ICP-AES	ALS ME-MS41, Ag-OG46
			9,491[4]	4-acid ICP-OES	MSA ICF-5Ag, ICF-6Ag

Notes:
1. The total meters sent for assay may differ from total meters drilled due to failed drill holes, unsampled core, or other reasons samples were not sent to a laboratory.
2. FA: Fire Assay, ICP-MS: Inductively Coupled Plasma Mass Spectrometry, ICP-AES: Inductively Coupled Plasma Atomic Emission Spectrometry.
3. Liberty Gold sonic drill holes are not listed in this table because silver is not considered recoverable from the HLF.
4. The number of meters listed under the "4-acid" method are also counted in the "2-acid" row. These are the pulps Liberty Gold selected to re-assay.

14.4.3 Derivation of Silver Values Using 2-Acid/4-Acid Linear Regression

Although the silver four-acid re-assaying program was robust, gaps remained in the dataset, and the four-acid results alone were not sufficient to support a standalone Mineral Resource estimate. To evaluate the relationship between the two analytical methods 6,254 paired four-acid and two-acid assays were statistically compared.

To estimate silver in the block model, SLR first considered a hierarchical dataset in which the four-acid silver value was used when available, and the original two-acid silver value otherwise. This approach is conservative as the comparison of the four-acid and two-acid paired samples shows the two-acid digestion returns only about 67% of the silver concentration measured by the four-acid method (Figure 14-1).

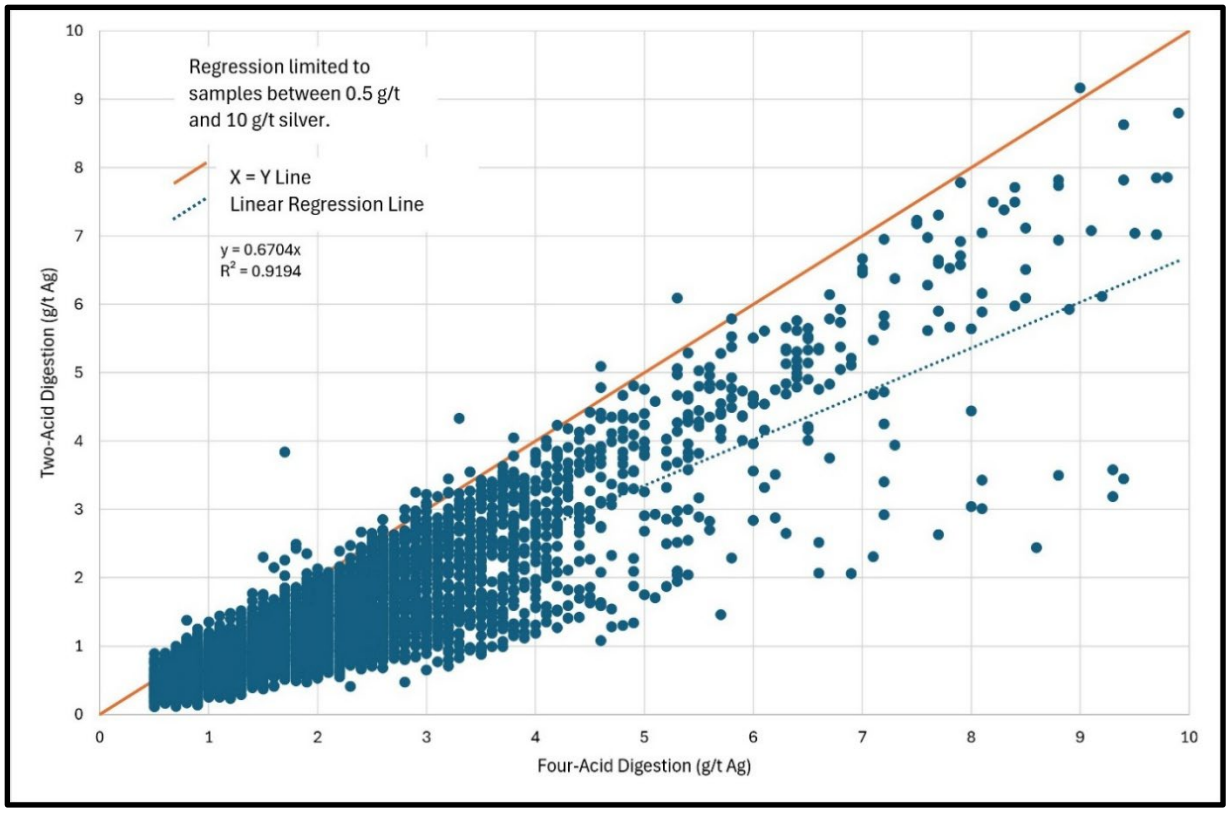


Figure 14-1: Silver Linear Regression of Two-Acid vs Four-Acid Digestions

By restricting the grade range to samples that represent the bulk of the database (0.5 to 10 g/t Ag) a linear regression was applied to the two-acid values to better approximate true values. For samples at or near the detection limit (0.5 g/t Ag) as well as those above 10 g/t Ag, uncertainties were too great to justify applying the regression; therefore, the original two-acid values were retained. The summary calculations and hierarchy used to assign silver values in the Mineral Resource are presented in Table 14-5.

Table 14-5: Handling of Silver Analytical Techniques in the Database

2A Regression Calculation	Value Applied
if Ag 2A <0.5	Ag 2A
if Ag 2A <10	Ag 2A /0.6704
if Ag 2A ≥ 10	Ag 2A
Silver Calculation Hierarchy	**Value Applied**
if FA is valid	FA
if 4A is >0.5	4A
if 4A is <0.5 and 2A regression is <0.5 and >0.25	2A regression
if 4A is <0.5 and 2A regression is <0.25	0.25
if 4A is <0.5 and 2A regression is >0.5	0.25
if 4A is <0.5 and 2A regression is blank	0.25
otherwise	2A regression

Notes:
1. FA = Fire Assay (historical samples only) 2A= two-acid, 4A= four-acid.
2. Historical sample detection limits varied but there was a significant population of samples a detection limit of 0.343 g/t (0.010 opt). For these samples, a value of 0.25 g/t Ag was assigned.

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 also hosts mineralization in the upper plate. In aggregate, favorable host stratigraphy of the middle plate comprises 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, middle, and lower, structural plates, all of which were created using Leapfrog software with extensive explicit controls applied. This geological modeling covers the full extent of the mineralization at Black Pine. The digital surfaces and solids were used extensively as guides for the modeling 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:

- Backfill: Insitu material remobilized during historical mining activities (see Section 14.5.6)
- Legacy Waste Dumps: Insitu material remobilized during historical mining activities (see Section 14.5.6)
- 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 14-2 shows the different areas across the Project and the location of the LHLF, which is described further in Sections 14.5.2.3 and 14.16.

14.5.2.1 Gold Domains

Initial gold mineralization domains were prepared by Liberty Gold geologists, and were reviewed, modified, and then adopted by SLR for use in resource estimation. 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 Au. Enclosed within the medium-grade domains, high-grade domains were built targeting material above 0.5 g/t Au. The exception is Rangefront, which targeted higher grade material above 0.3 g/t Au. These modeling cut-off grades were chosen to reflect the natural grade boundary above which continuous higher-grade material could be captured.

To account for more discontinuous mineralization outside of the medium- and high-grade domains, a low-grade domain was created based on a buffer of 30 m surrounding the medium grade domains.

Within the domains, internal-low grade material (just below, equal to, or just above the targeted gold grade) was modeled 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.

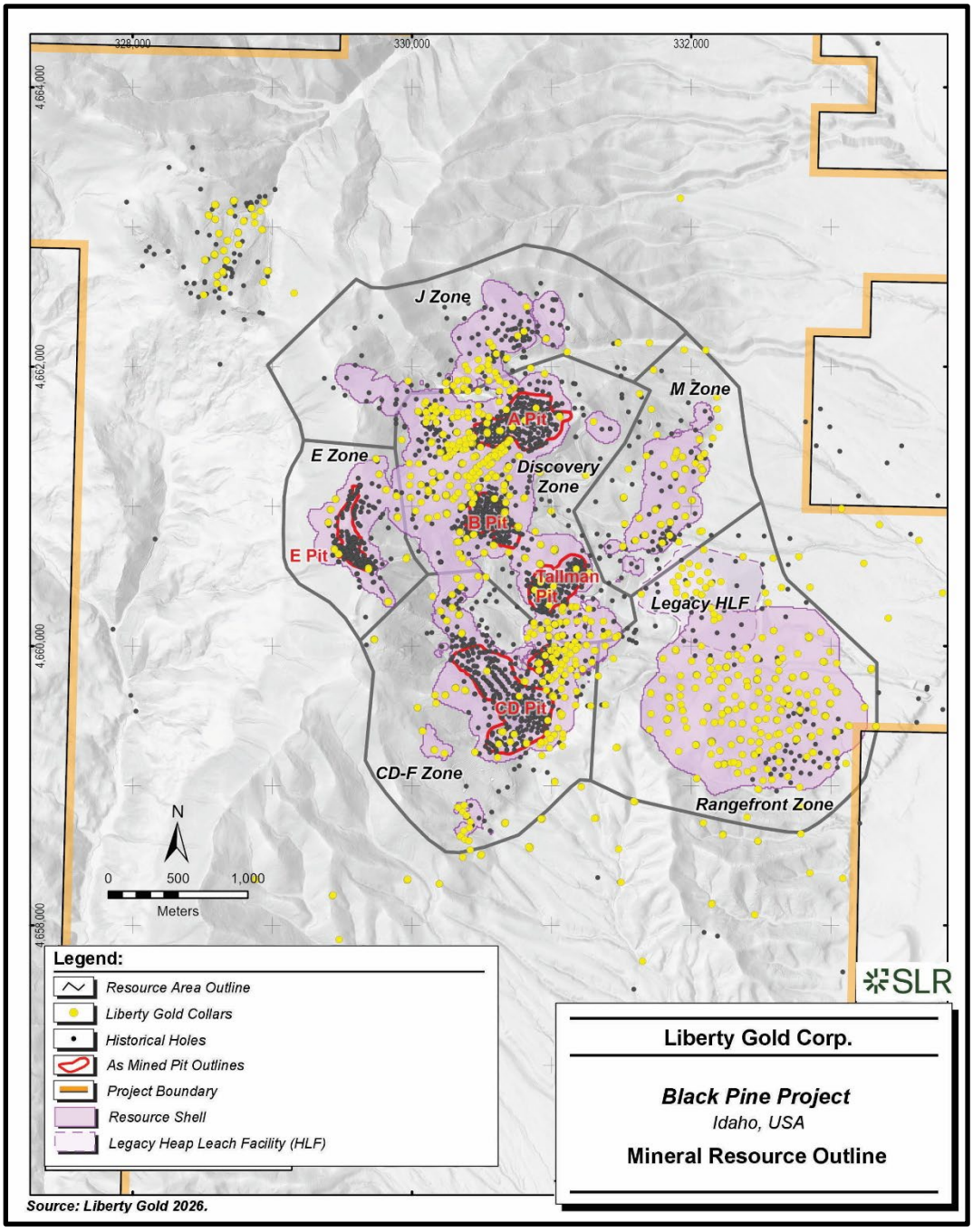


Figure 14-2: Black Pine Project Mineral Resource Outline

While the modeled mineralization lies dominantly within the bounds of the middle structural-lithological plate, 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 down 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 modeled 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 285 individual mineralized domain wireframes were created for the Project (Table 14-6). Cross-sections showing examples of the gold mineral-domain modeling are shown in Figure 14-3 and Figure 14-4. In addition to the high and medium grade domains, other domains used in the estimation process include:

- Low-grade domains based on a 30 m buffer around the medium-grade domains.
- Internal-low grade domains described in Section 14.5.5.
- Wireframes representing the Qal unit from the geological model described in Section 7.
- Wireframes representing the backfill and waste dumps described in Section 14.5.6.

Table 14-6: Black Pine Project Gold Mineralization Domain Wireframes

Area	Number of Medium Grade Wireframes	Number of High-Grade Wireframes
Discovery	14	45
Rangefront	13	23
CD	17	54
F	3	8
E Pit	15	15
M Zone	11	14
J Zone	12	9
Total Medium and High-Grade Wireframes	253	

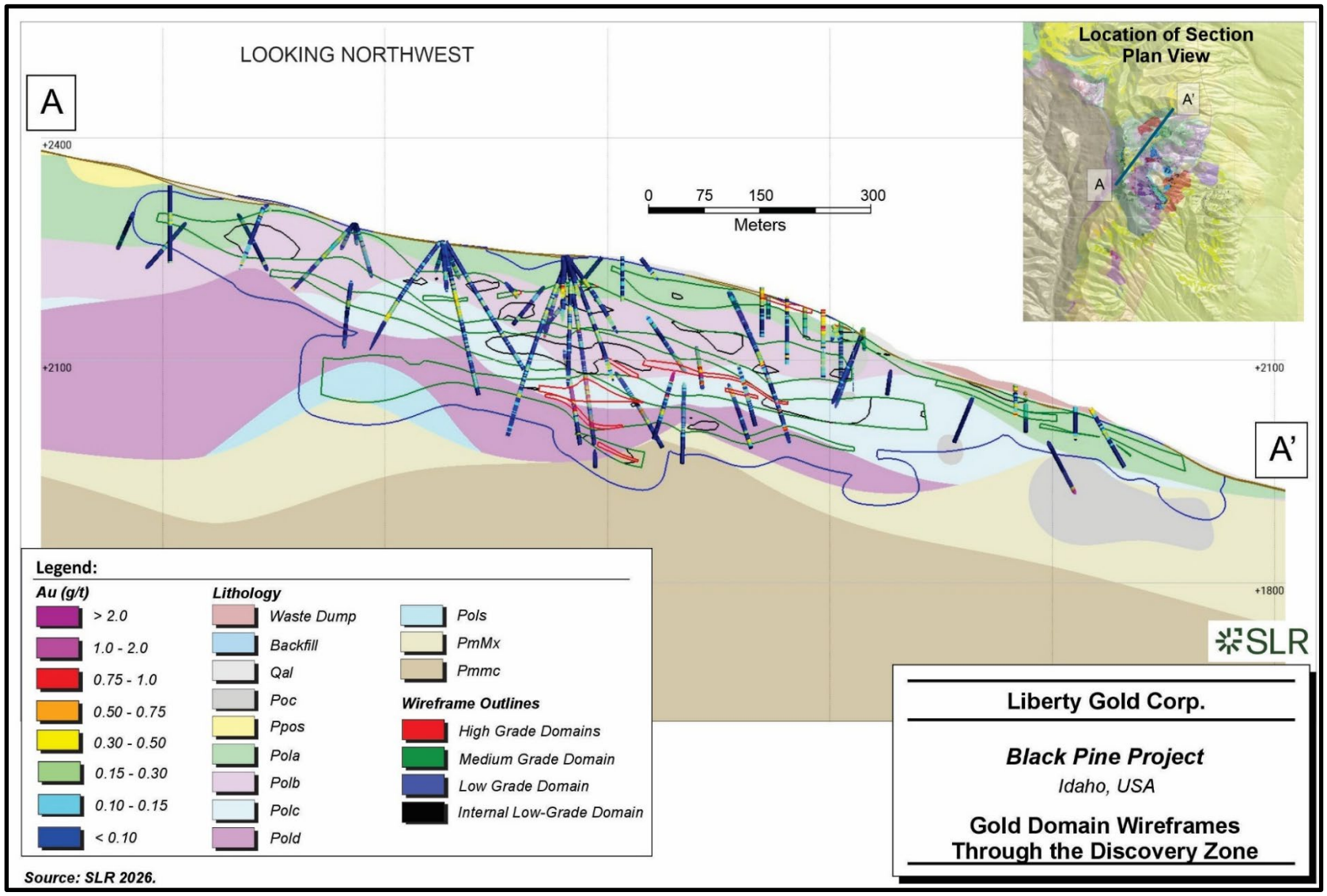


Figure 14-3: Gold Domain Wireframe Model through Discovery Zone

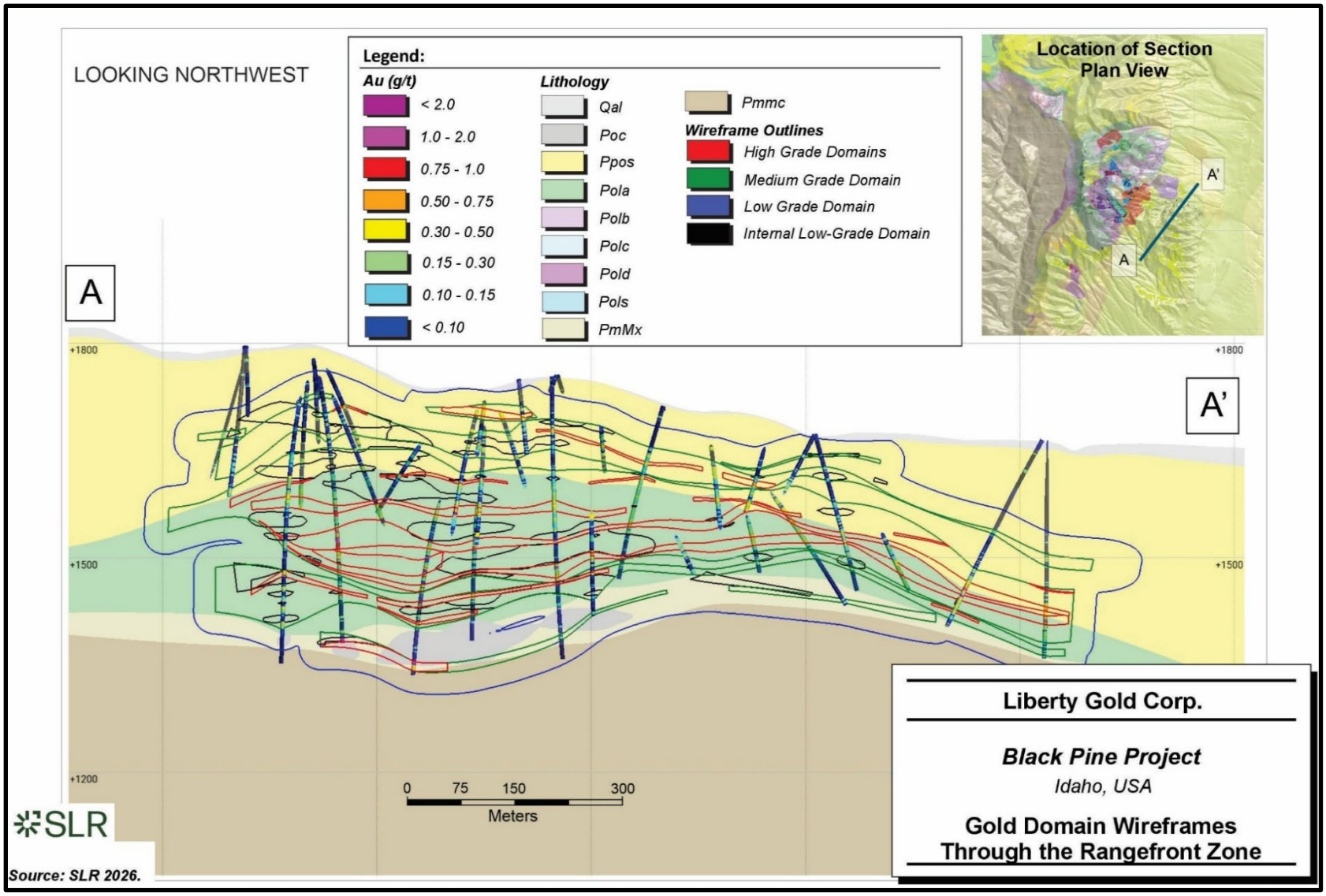


Figure 14-4: Wireframe Model through Rangefront Zone

14.5.2.2 Silver Domains

The controls on silver mineralization are not as well understood as those for gold. The main mineralizing event correlates with stratigraphy and gold mineralization. A secondary trend is visible correlating with high-grade silver at higher angles, cross-cutting the stratigraphy in some areas. SLR used Leapfrog software to generate grade shells for silver and compared those to the gold grade shells, domains, and stratigraphy. Since the majority of the mineralization followed the same trends as the gold, it was decided that the medium-grade gold domains would be suitable for estimation of silver. Statistical analysis did not show a significant high grade silver population, therefore the high-grade domains were not used. A 30 m buffer was created around the medium-grade domains to capture any isolated samples, defining the low-grade domain. The silver domains in the Discovery zone are shown in Figure 14-5 along with the silver-grade shells that depict the mineralized trends.

Table 14-7 lists the number of domains used in the silver estimation. The table does not list the low-grade buffer or the Qal unit from the geological model, though both were included in the estimation process.

Table 14-7: Black Pine Project Silver Mineralization Domain Wireframes

Area	Number of Medium Grade Wireframes
Disco	14
M Zone	11
Total Silver Wireframes	25

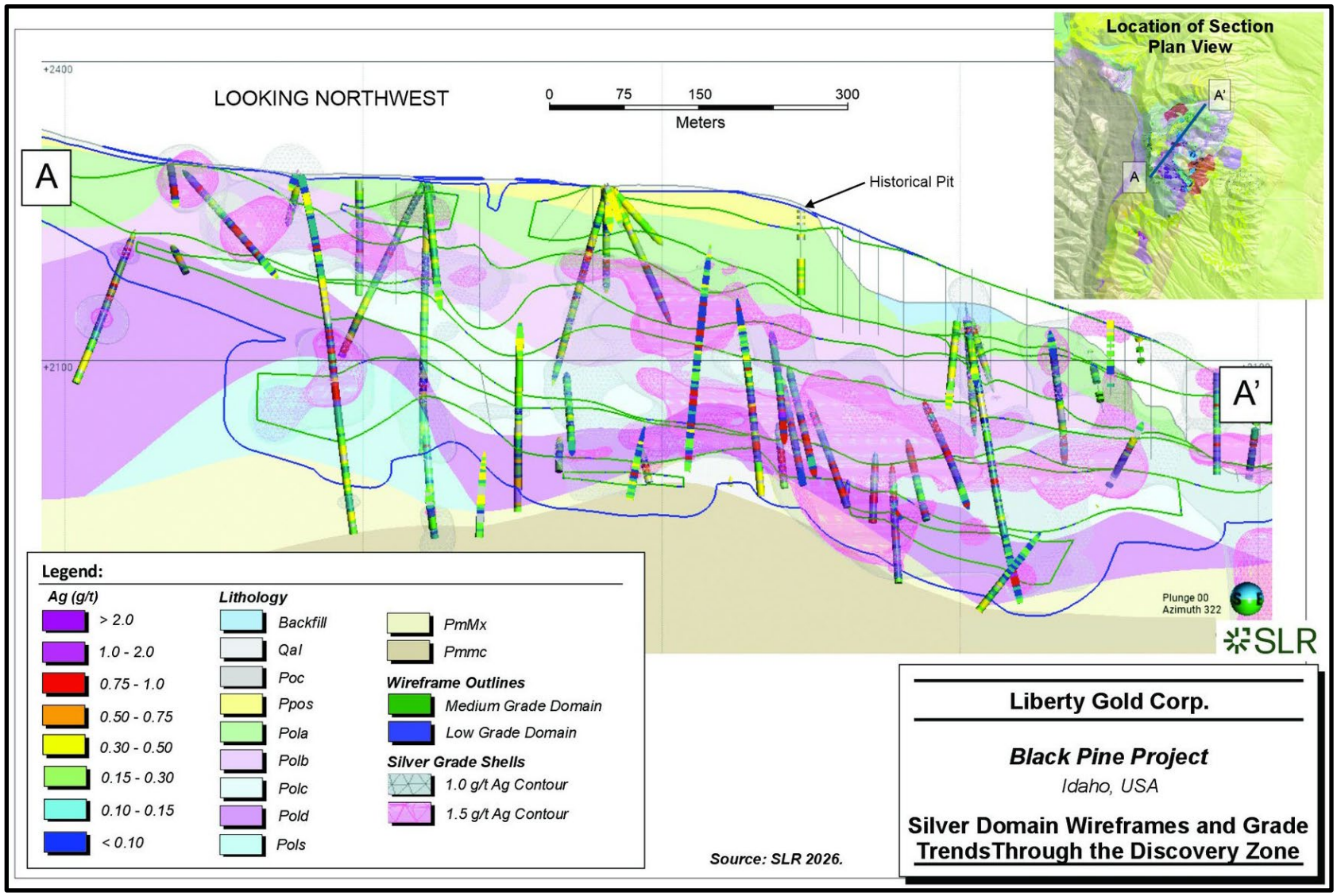


Figure 14-5: Silver Domain Wireframes and Grade Trends Through the Discovery Zone

14.5.2.3 Legacy Heap Leach Facility

Pegasus acquired the Black Pine property in 1990 and constructed a cyanide heap-leach pad and gold recovery plant. Extraction of mineralized material began in October 1991, with the first gold poured in January 1992 (Pegasus, 1993). A total of six pits were mined between 1991 and 1997, totaling approximately 26.5 Mt of waste rock and 31 Mt of ore (Sawyer, 1997). Reclamation of the heap-leach pad began after mining ceased in 1997.

Previous studies (See Section 6.6.3) suggest that recoverable gold remains in the reclaimed leach pad. The estimate of the LHLF was completed by Forte Dynamics and is described in Section 14.16.

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 under and 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 modeled 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% recovery for gold prior to pit optimization and the areas are excluded from the Mineral Resource statement. Though silver was not included in the pit optimization, the areas were also coded as 0% recovery for silver.

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 sulfides or carbonaceous materials. Generally, these low-cyanide-solubility occurrences are associated with 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 modeled, they do influence the metallurgical recoveries of the Polb unit that are coded into the model. Figure 14-6 shows the carbon (Poc, modeled in gray) along with the mineralized domains, and the lithology model.

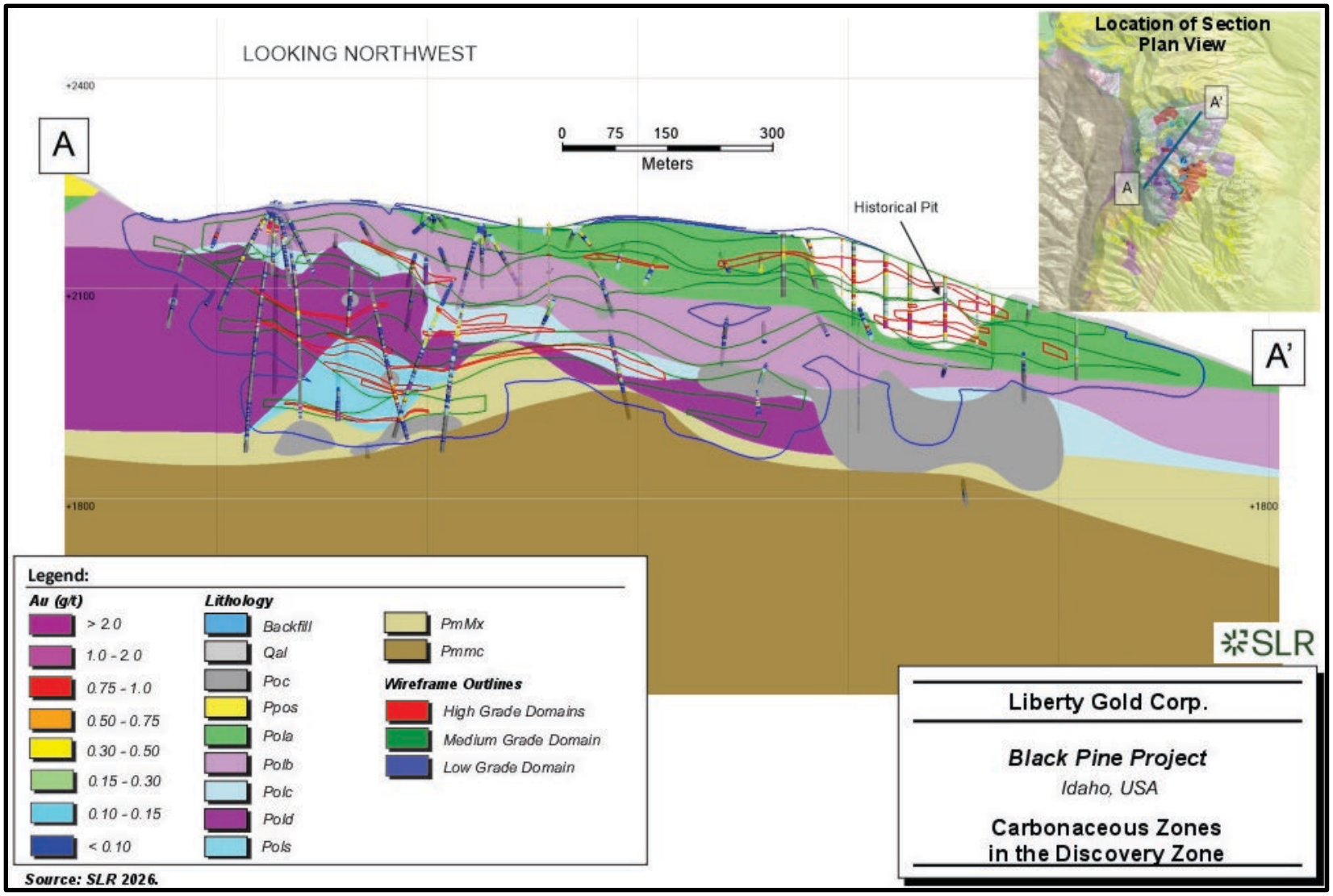


Figure 14-6: Lithologic Model Showing Carbonaceous Zones (Poc)

14.5.4 Geometallurgical Model

At Black Pine, eight historical and 204 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 modeling. 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 modeling 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 Gary Simmons.

Figure 14-7 shows the geometallurgical domains and the cyanide solubility ranges for each domain are listed in Table 14-8.

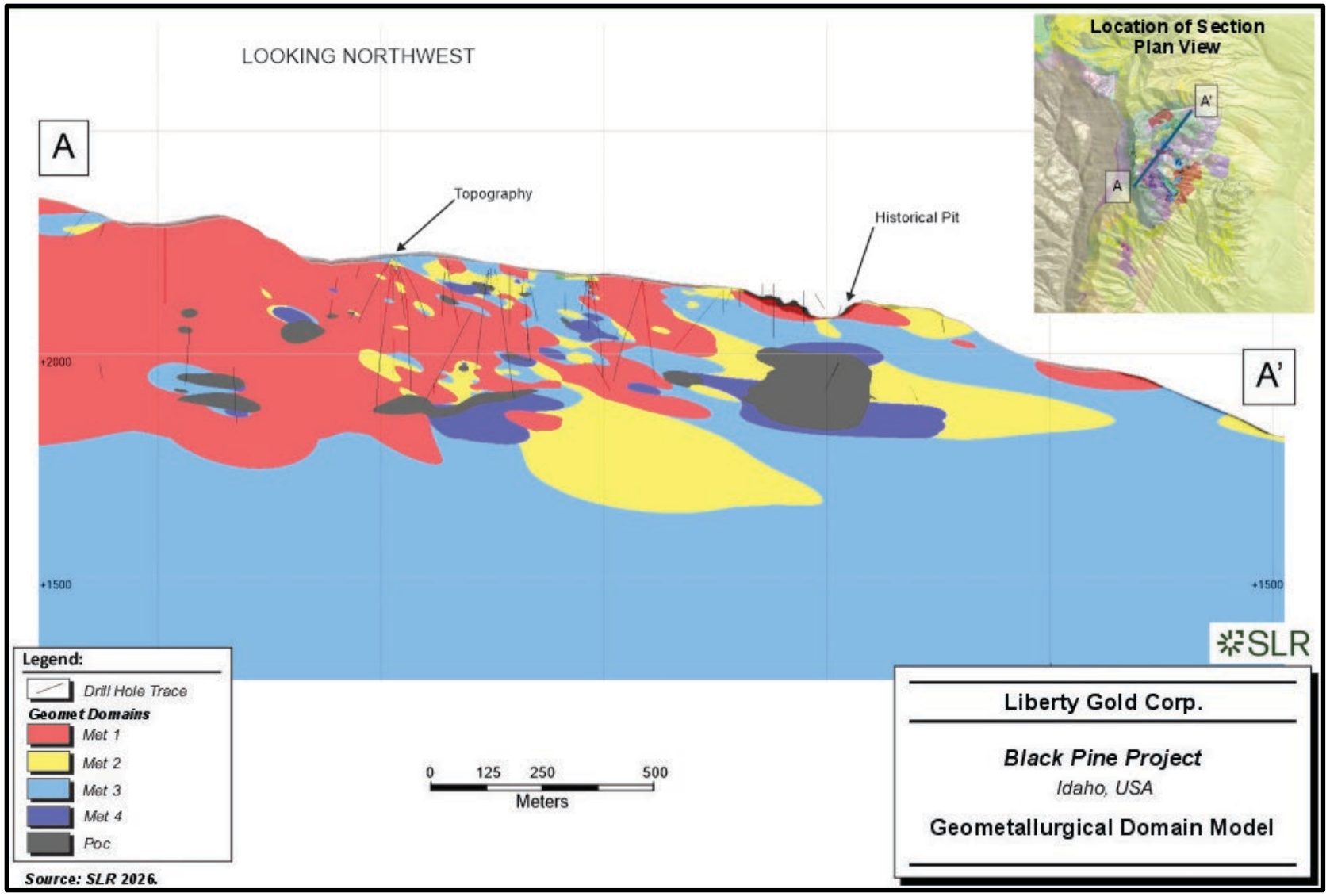


Figure 14-7: Geometallurgical Domain Model

Table 14-8: 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 Internal 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 "internal 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 internal low-grade domains further segregate the just-below cut-off grade samples and volumes and prevent smearing of the higher grades. Figure 14-8 shows an example of the internal low-grade indicator domains contained within the medium-grade domains.

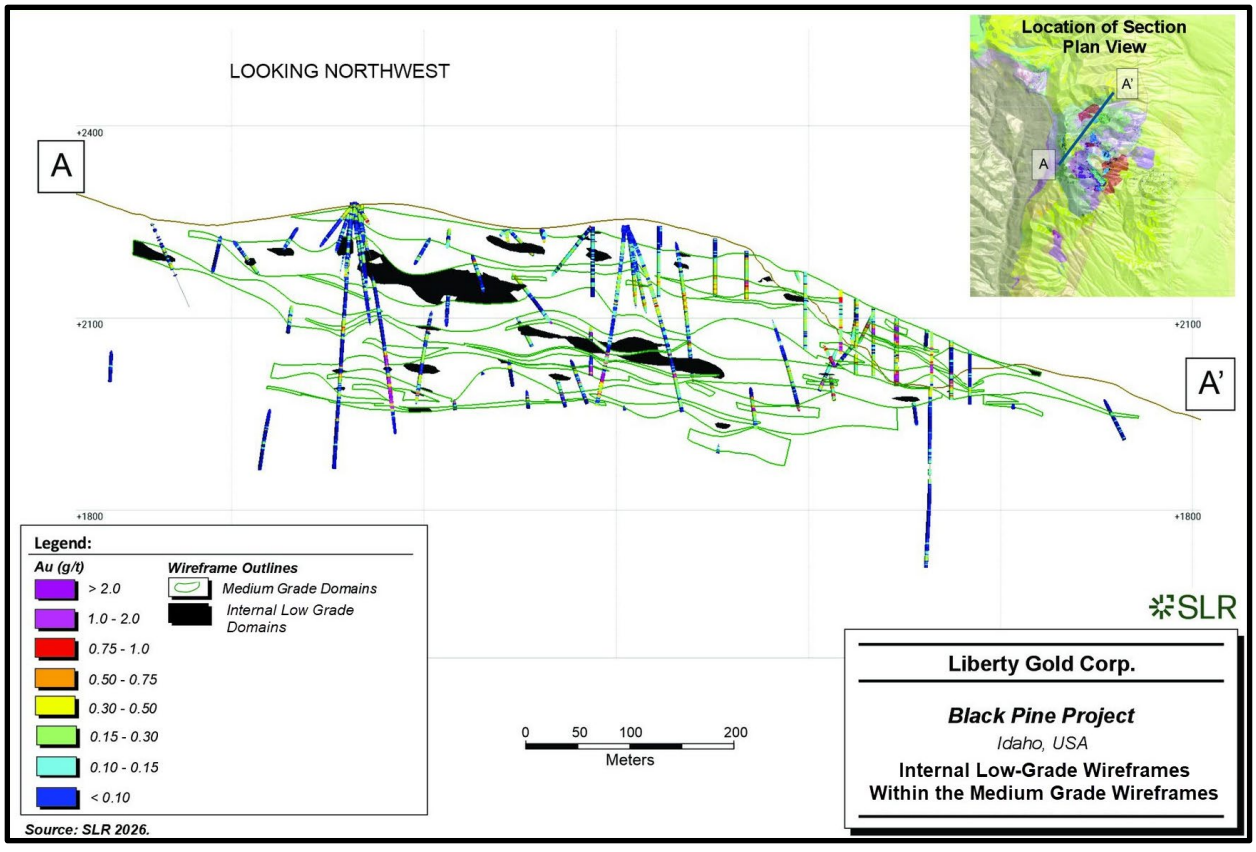


Figure 14-8: Internal Low- Grade Indicator Domains within the Medium Grade Domains

14.5.6 Backfill and Legacy 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 topographic surface 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 Project. The topographic surfaces were used to code the block models so that the Mineral Resources were limited to in-situ bedrock 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 within 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 14-9 shows the waste dumps.

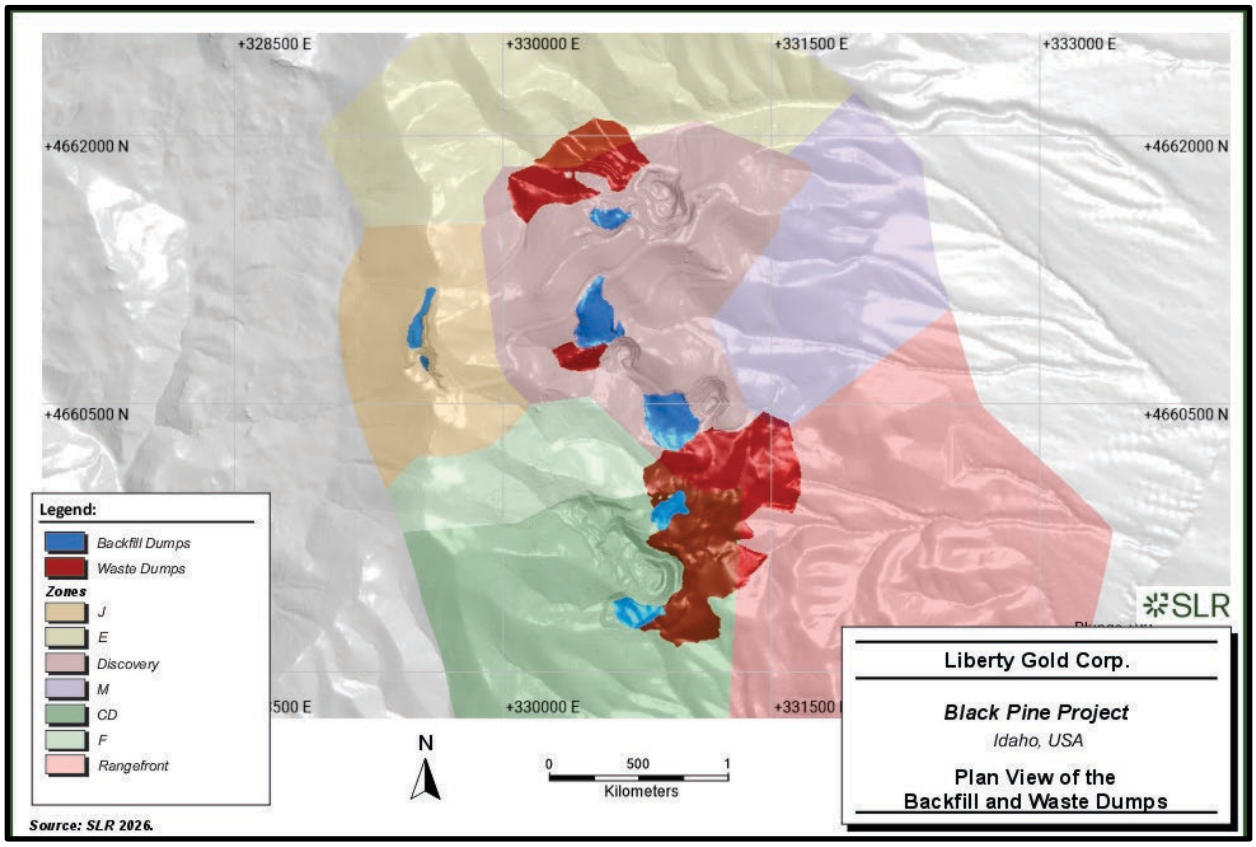


Figure 14-9: Backfill and Legacy Waste Dumps

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 gold 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 14-10, Figure 14-11, Figure 14-12, and Figure 14-13 as applied to the data set for the mineralized domains. The gold capping values used, and the uncapped and capped gold assay statistics by domain group within the respective areas, are summarized in Table 14-9.

Table 14-10 shows the uncapped and capped silver assay statistics, as well as the capping values for the Discovery and M Zone areas. Silver was not estimated in the other areas.

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.

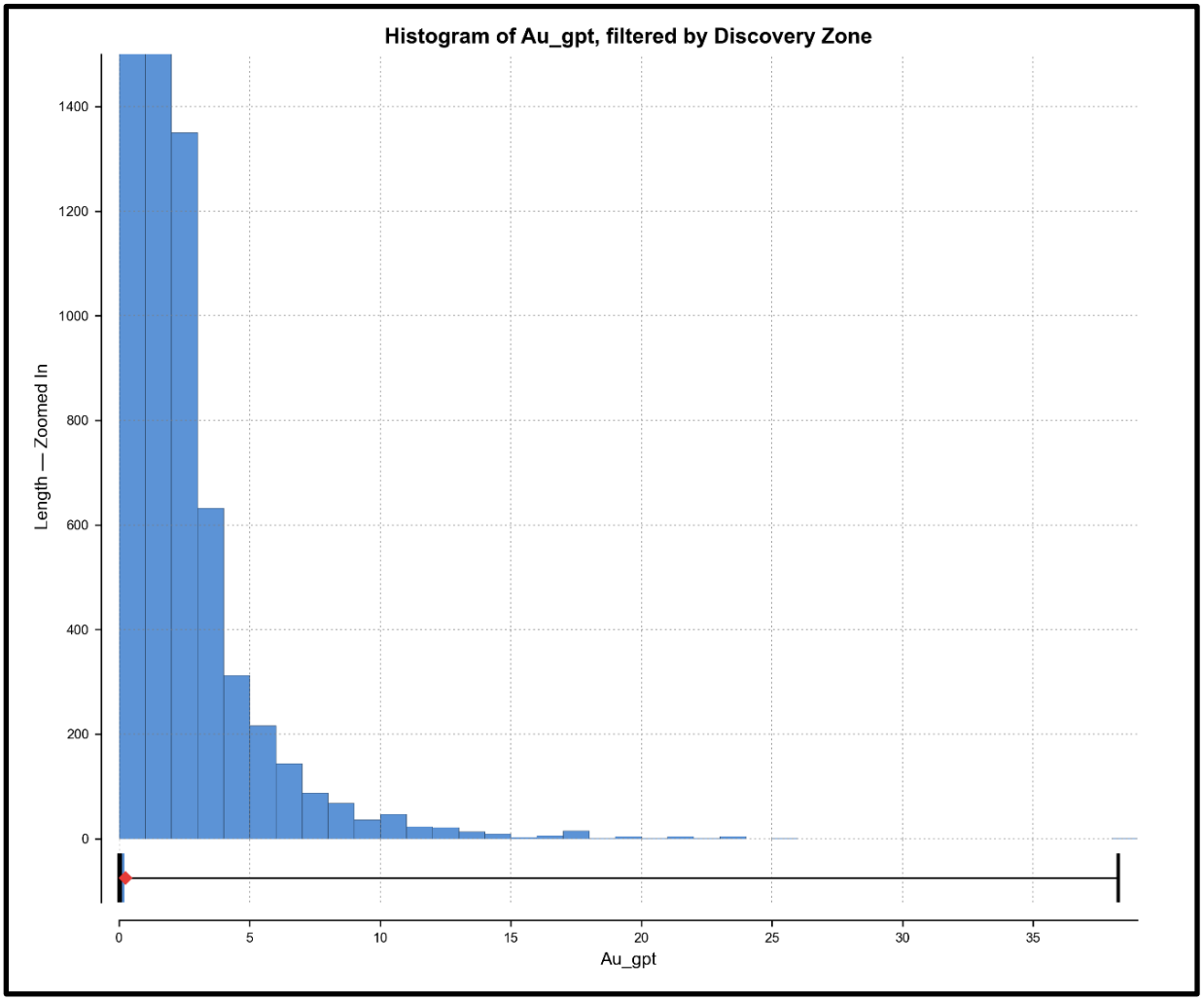


Figure 14-10: Histogram for Discovery Gold Assays (zoomed in on y-axis)

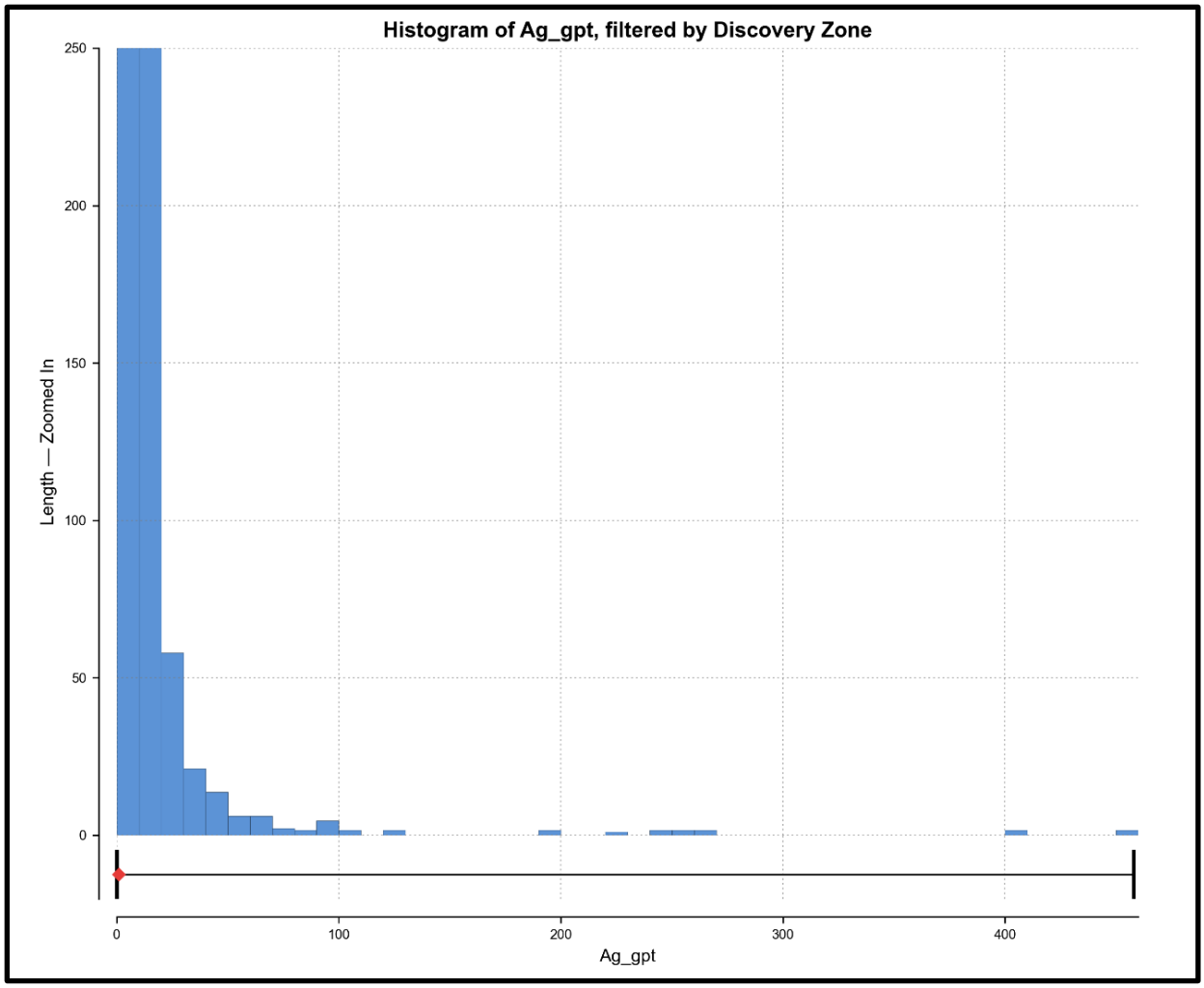


Figure 14-11: Histogram for Discovery Silver Assays (zoomed in on y-axis)

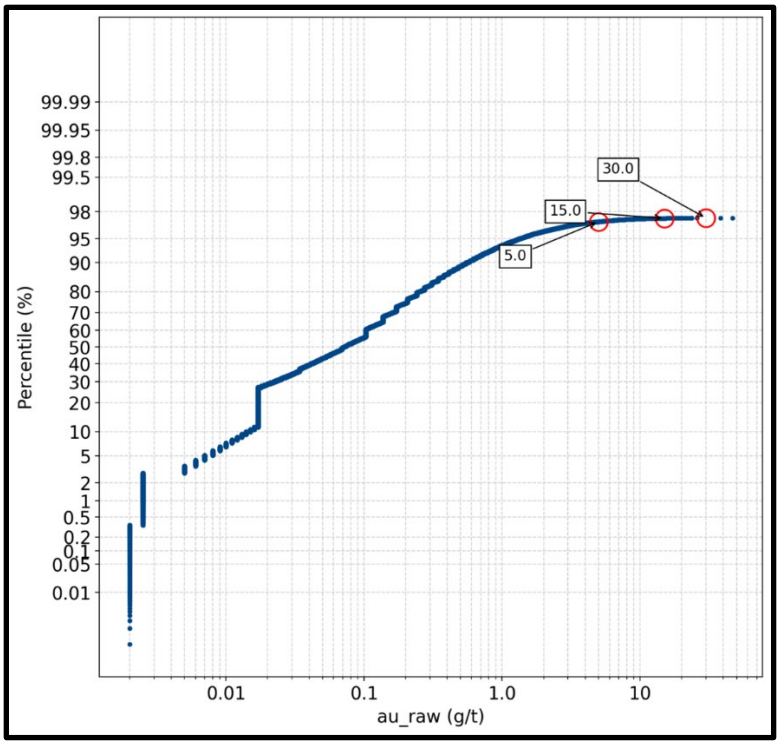


Figure 14-12: Log Probability Plots for Discovery Gold Assays

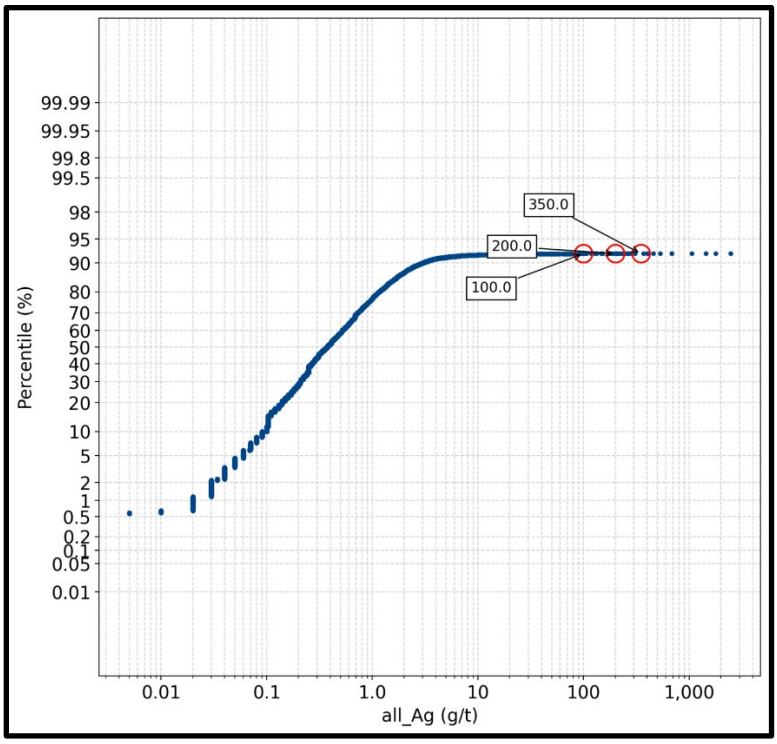


Figure 14-13: Log Probability Plots for Discovery Silver Assays

Table 14-9: Descriptive Statistics of Uncapped vs Capped Gold Assays

Area	Domain1	Length (m)	Uncapped			Capped			Capping Value (g/t)	Number of Caps
			Mean (g/t)	CV	Maximum (g/t)	Mean (g/t)	CV	Maximum (g/t)		
CD	HG	4,871	0.89	1.02	25.27	0.88	0.83	8.00	8.00	3
	MG	18,057	0.23	0.61	2.43	0.23	0.60	2.00	2.00	3
	LG	15,509	0.04	2.47	6.49	0.04	1.45	1.00	1.00	5
	ULG	4,830	0.05	1.08	2.43	0.05	1.05	1.00	1.00	1
	Backfill	404	0.11	1.21	1.27	-	-	-	uncapped	-
	Waste Dump	657	0.21	2.53	8.22	-	-	-	uncapped	-
F	HG	601	1.13	1.67	17.35	1.03	1.23	8.00	8.00	7
	MG	2,513	0.22	0.72	1.75	0.22	0.72	1.75	2.00	-
	LG	3,854	0.04	1.63	1.51	0.04	1.45	1.00	1.00	2
	ULG	556	0.06	1.02	1.71	0.06	0.92	1.00	1.00	1
	Waste Dump	82	0.16	0.71	0.58	-	-	-	uncapped	-
E	HG[2]	608	1.45	2.20	35.52	1.14	0.95	5.00	5.00	12
	MG	6,986	0.23	1.15	7.65	0.23	1.02	3.00	3.00	11
	LG	9,756	0.05	1.54	2.74	0.05	1.39	1.00	1.00	2
	ULG	1,954	0.05	1.29	1.51	0.05	1.24	1.00	1.00	1
	3200	245	4.56	1.49	46.70	4.31	1.31	25.00	25.00	3
	3201	571	2.63	1.51	26.71	2.63	1.50	25.00	25.00	1
Rangefront	HG	9,820	0.62	1.17	11.15	0.61	1.06	5.00	5.00	29
	MG	30,505	0.16	0.65	3.31	0.16	0.61	1.00	1.00	25
	LG	33,159	0.05	1.07	3.43	0.05	0.86	1.00	1.00	4
	ULG	11,717	0.07	0.78	0.97	0.07	0.78	0.97	1.00	-
J	HG	724	0.97	1.02	9.12	-	-	-	uncapped	-
	MG	3,429	0.23	1.05	4.56	0.22	0.93	2.00	2.00	5
	LG	7,345	0.04	1.71	1.51	0.03	1.51	1.00	1.00	1
	ULG	771	0.05	1.09	1.06	0.05	1.09	1.00	1.00	1
	Waste Dump	243	0.16	0.61	0.58	-	-	-	uncapped	-
M	HG	1,056	1.51	1.64	46.70	1.40	1.01	8.00	8.00	9
	MG	3,934	0.24	1.21	5.14	0.24	1.14	3.00	3.00	4
	LG	13,440	0.04	5.15	9.59	0.04	1.68	1.00	1.00	19
	ULG	735	0.07	1.32	2.73	0.07	1.17	1.00	1.00	1
Discovery	HG	13,780	1.55	1.31	38.26	1.54	1.27	20.00	20.00	4
	MG	13,766	0.06	1.64	5.21	0.06	1.29	1.50	5.00	9
	LG	63,638	0.24	1.12	10.83	0.24	1.05	5.00	1.50	15
	ULG	52,339	0.05	1.49	4.77	0.05	1.32	1.50	1.50	8
	Backfill	977	0.16	0.80	2.14	-	-	-	uncapped	-
	Waste Dump	1,167	0.15	0.90	1.47	-	-	-	uncapped	-

Notes:
1. 'Qal' (not listed) was assigned the Low-Grade Cap value for all areas.
2. The "HG" domain in the E-Pit area excludes domains 3200 and 3201, as these high-grade zones were estimated separately.

Table 14-10: Descriptive Statistics of Uncapped vs Capped Silver Assays

Area	Domain	Length (m)	Uncapped			Capped			Capping Value (g/t)	Number of Caps
			Mean (g/t)	CV	Maximum (g/t)	Mean (g/t)	CV	Maximum (g/t)		
Disco	MG	54,655.60	1.39	3.43	458.00	1.07	1.86	55.00	55.00	15
	LG	38,973.30	1.10	1.45	55.50	0.84	1.62	55.00	55.00	1
M	MG	5,389.00	4.53	6.42	1435.00	3.62	3.19	110.00	110.00	22
	LG	12,670.60	1.64	12.44	1776.00	1.13	2.44	110.00	110.00	2

14.7 COMPOSITING

Composites were created from the capped, raw assay values using the downhole compositing function of Leapfrog's modeling 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 ft) 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

Area	Domain	3.048 m Composites					
		Length (m)	Mean (g/t)	CV	Minimum (g/t)	Median (g/t)	Maximum (g/t)
CD	HG	4,901	0.88	0.72	0.02	0.70	6.10
	MG	18,284	0.23	0.51	0.02	0.20	1.65
	LG	15,877	0.04	1.27	0.00	0.02	0.99
	ULG	4,896	0.05	0.68	0.00	0.05	0.36
	Backfill	2,006	0.29	1.29	0.00	0.18	3.50
	Waste Dump	707	0.21	2.19	0.00	0.13	5.25
F	HG	601	1.03	1.13	0.08	0.70	8.00
	MG	2,538	0.23	0.59	0.02	0.19	1.15
	LG	3,950	0.04	1.20	0.00	0.02	0.69
	ULG	588	0.06	0.63	0.01	0.06	0.30
	Waste Dump	83	0.16	0.60	0.03	0.14	0.40
E	HG*	614	1.10	0.77	0.02	0.79	4.35
	MG	7,005	0.23	0.85	0.02	0.18	3.00
	LG	9,888	0.05	1.40	0.00	0.02	2.74
	ULG	1,987	0.05	0.81	0.01	0.03	0.36
	3200	248	4.28	1.24	0.02	2.37	24.33
	3201	585	2.60	1.36	0.02	1.25	20.40
Rangefront	HG	9,734	0.61	0.95	0.01	0.44	5.00
	MG	30,286	0.16	0.50	0.00	0.15	1.00
	LG	33,237	0.05	0.75	0.00	0.04	0.88
	ULG	11,716	0.07	0.62	0.00	0.06	0.65
J	HG	724	0.97	0.81	0.07	0.77	5.68
	MG	3,453	0.22	0.79	0.02	0.17	1.88
	LG	7,513	0.04	1.39	0.00	0.02	0.73
	ULG	775	0.05	0.63	0.02	0.04	0.16
	Waste Dump	249	0.16	0.55	0.02	0.14	0.47
M	HG	1,063	1.39	0.87	0.02	1.01	8.00
	MG	4,005	0.24	0.92	0.01	0.17	2.22
	LG	13,902	0.04	1.45	0.00	0.02	1.00
	ULG	735	0.07	0.90	0.01	0.05	0.61
Disco	HG	13,865	1.55	1.16	0.00	0.99	20.00
	MG	64,421	0.24	0.93	0.00	0.19	5.00
	LG	54,482	0.05	1.21	0.00	0.03	1.50
	ULG	13,965	0.06	1.05	0.00	0.05	1.50
	Backfill	1,258	0.22	1.56	0.02	0.14	4.40
	Waste Dump	1,313	0.16	0.89	0.00	0.12	1.29

*High-grade (HG) domain statistics for E zone exclude the 3200 and 3201 domains.

Table 14-12: Summary of Capped Silver Composite Statistics by Domain Group

Area	Domain	3.048 m Composites					
		Length (m)	Mean (g/t)	CV	Minimum (g/t)	Median (g/t)	Maximum (g/t)
Disco	MG	56,432	1.34	1.49	0.00	0.92	55.00
	LG	39,141	1.10	1.31	0.00	0.70	27.75
M	MG	5,441	3.62	2.99	0.00	1.25	110.00
	LG	12,710	1.39	1.95	0.00	0.80	110.00

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 14-14 shows the experimental variogram and model for the “203” low-grade domain within the Discovery Zone.

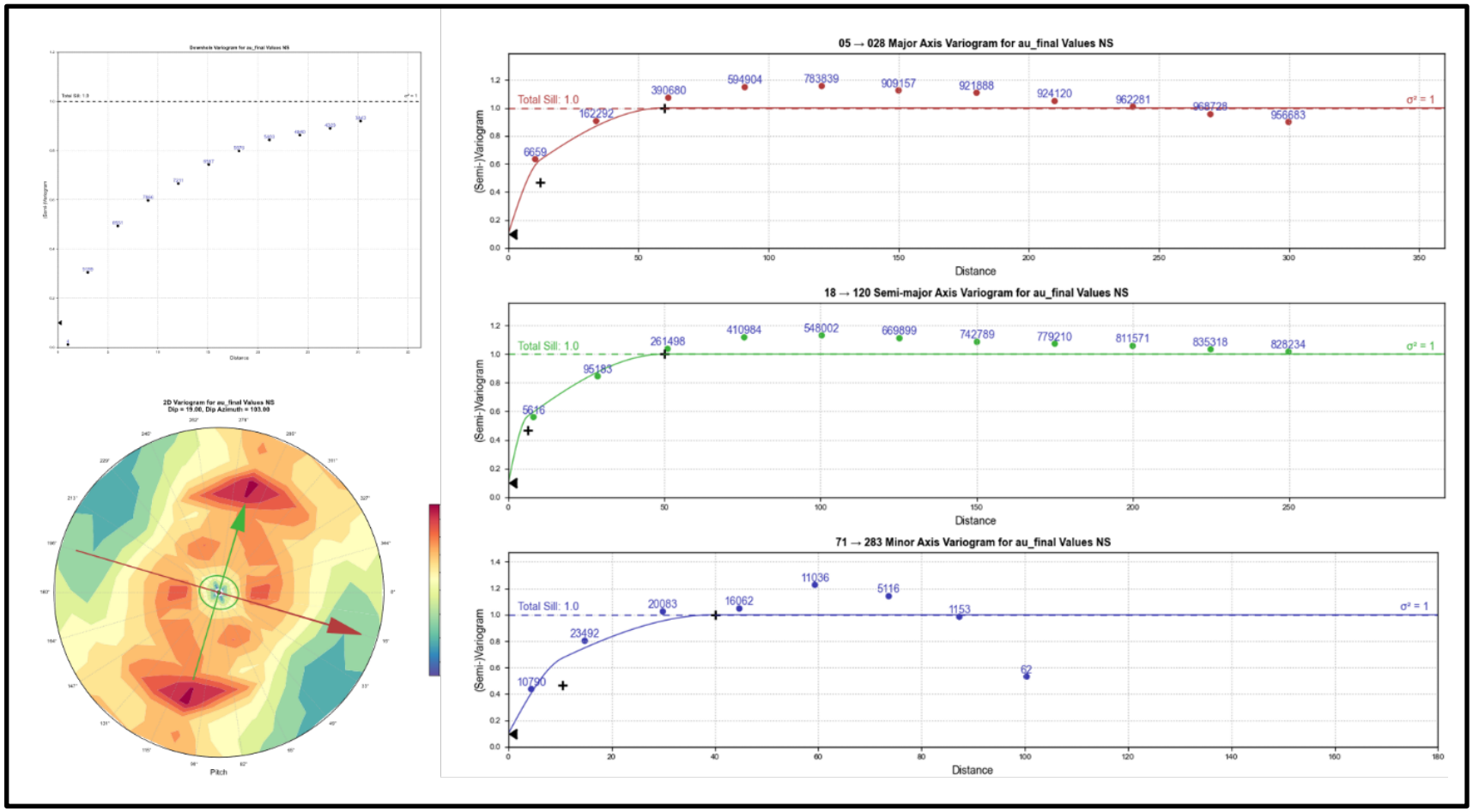


Figure 14-14: Variogram Model – Discovery Low Grade Domain 203

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.

Search ellipsoids 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.

14.9.1 Gold

Similar interpolation parameters were used for all gold domains. The medium- and high-grade domains used the same ellipsoid ranges, minimum and maximum number of composites, and maximum number of composites per hole. The low-grade domain used a larger Pass 3 ellipsoid range to account for the wider spaced holes around the edges of the deposit. Because the low-grade domain was built using a 30 m buffer around the medium-grade and was not informed by geological features, it included the use of a search restriction to prevent smearing of high-grades beyond a distance of 40 m. The internal-low grade domains were built within, and overprinting the medium grade domains, used soft boundaries with a distance of 5 m. Hard boundaries were used for all other gold domains (Figure 14-15).

14.9.2 Silver

Boundary analysis using the silver values within the medium- and low-grade domains showed softer boundaries than those for gold (Figure 14-15). Similar interpolation parameters were used for the silver estimation as were used for gold, but with the implementation of 10 m soft boundaries. Due to sparse data around the edges of the deposit, the ellipsoid ranges for Pass 3 were relaxed for both the medium- and low-grade domains up to a maximum of 420 m x 420 m x 120 m.

14.9.3 Backfill and Legacy Waste Dumps

The post-mining domains were estimated separately with the parameters adjusted to include only holes drilled by Liberty Gold.

Table 14-13: Grade Interpolation Parameters

Value	Domain	Boundary	Pass	Ellipse Size (m)	Ellipsoid Directions[1]	Min Comps	Max Comps	Max Comps per Hole	Search Restriction
Au (All Areas)	MG and HG	Hard	1	80x80x10	VO	3	12	2	No
		Hard	2	160x160x15	VO	3	12	2	No
		Hard	3	320x320x20	VO	1	12	2	No
	LG	Hard	1	80x80x10	VO	3	12	2	Yes, 40 m, 1 g/t
		Hard	2	160x160x15	VO	3	12	2	Yes, 40 m, 1 g/t
		Hard	3	360x360x60	VO	1	12	2	Yes, 40 m, 1 g/t
	ULG	Soft (5 m)	1	80x80x10	VO	3	12	2	No
		Soft (5 m)	2	160x160x15	VO	3	12	2	No
		Soft (5 m)	3	320x320x20	VO	1	12	2	No
	Backfill/Waste Dump	Hard	1	60x60x10	6, 90, 90	3	20	2	No
		Hard	2	90x90x15	6, 90, 90	3	20	2	No
		Hard	3	90x90x15	6, 90,90	1	20	2	No
Ag (Disco and M Zone only)	MG	Soft (10 m)	1	80x80x10	VO	3	12	2	No
		Soft (10 m)	2	160x160x15	VO	3	12	2	No
		Soft (10 m)	3	320x320x20 to 380x380x80	VO	1	20	2	No
	LG	Soft (10 m)	1	80x80x10	VO	3	12	2	Yes, 40 m, 1 g/t
		Soft (10 m)	2	160x160x15	VO	3	12	2	Yes, 40 m, 1 g/t
		Soft (10 m)	3	420x420x120	VO	1	20	2	Yes, 40 m, 1 g/t

Notes:
1. Ellipsoid directions are either shown in degrees (dip, dip direction, pitch) or based on VO (Variable Orientation in Leapfrog software).

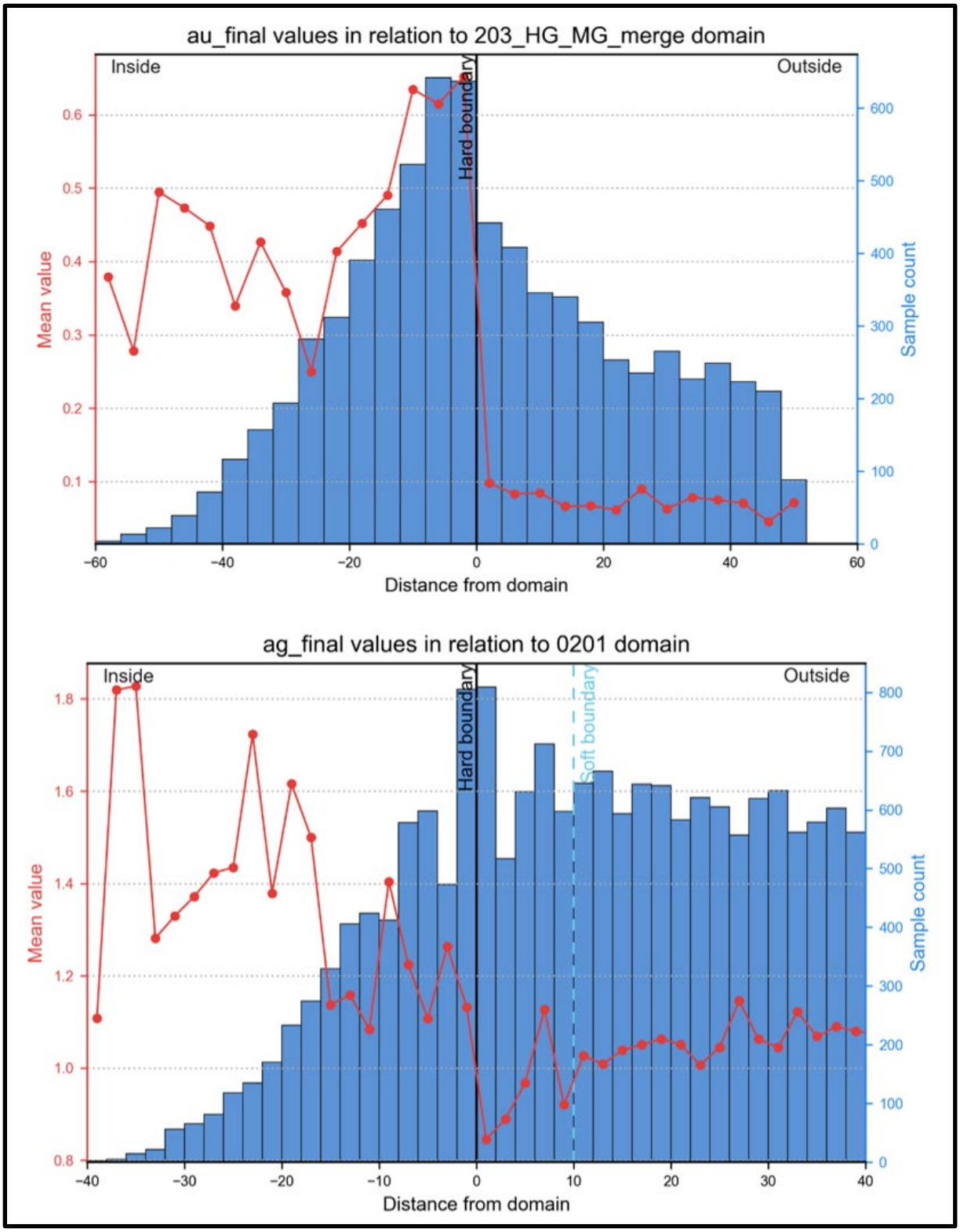


Figure 14-15: Example of Contact Plots Used for Boundary Analysis (Au and Ag)

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)	
CD, F	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	329,500	4,658,400	1,500	0
Discovery	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	329,800	4,659,900	1,700	0
E Pit	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	329,100	4,660,200	2,100	0
J Zone	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	329,100	4,661,500	1,700	0
M Zone	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	331,200	4,660,400	1,500	0
Rangefront	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	331,500	4,658,800	1,250	0
Combined Final	Sub-blocked	10 (2.5)	10 (2.5)	5 (1.25)	327,900	4,658,400	1,250	0
Combined Final	Re-Blocked	10	10	5	327,900	4,658,400	1,250	0

Notes:
1. Silver estimation was conducted on a stitched sublock model for the Discovery and M Zones The block sizes shown in table stayed the same. The origin was X-axis: 327,900, Y-axis 4,658,400, Z-axis 2,500.

14.11 BULK DENSITY

Bulk density measurements were performed on a total of 1,293 core samples to understand the variations in density across lithogic and alteration/mineralization types. 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. The QP is of the opinion that the distribution of measurements is representative of the variability within the lithological units and that the density values have been applied correctly within the block model.

Table 14-15: Block Model Densities by Lithology and Gold Domain

Lithology Unit	Domain	Density Value (g/cm³)
Poc	MG or HG min zone	2.45
	All Other Domains	2.43
PPos	MG or HG min zone	2.44
	All Other Domains	2.46
Pola	MG or HG min zone	2.48
	All Other Domains	2.53
Polb	MG or HG min zone	2.43
	All Other Domains	2.43
Polc	MG or HG min zone	2.44
	All Other Domains	2.53
Pold	MG or HG min zone	2.57
	All Other Domains	2.62
Pols	MG or HG min zone	2.50
	All Other Domains	2.59
PmMx	MG or HG min zone	2.53
	All Other Domains	2.59
PMmc	MG or HG min zone	2.60
	All Other Domains	2.63
QAL		1.80
Waste		1.80
Backfill		1.80

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 14-16 shows the conceptual pit shells with a 0.10 g/t Au grade cut-off in relation to the block model.

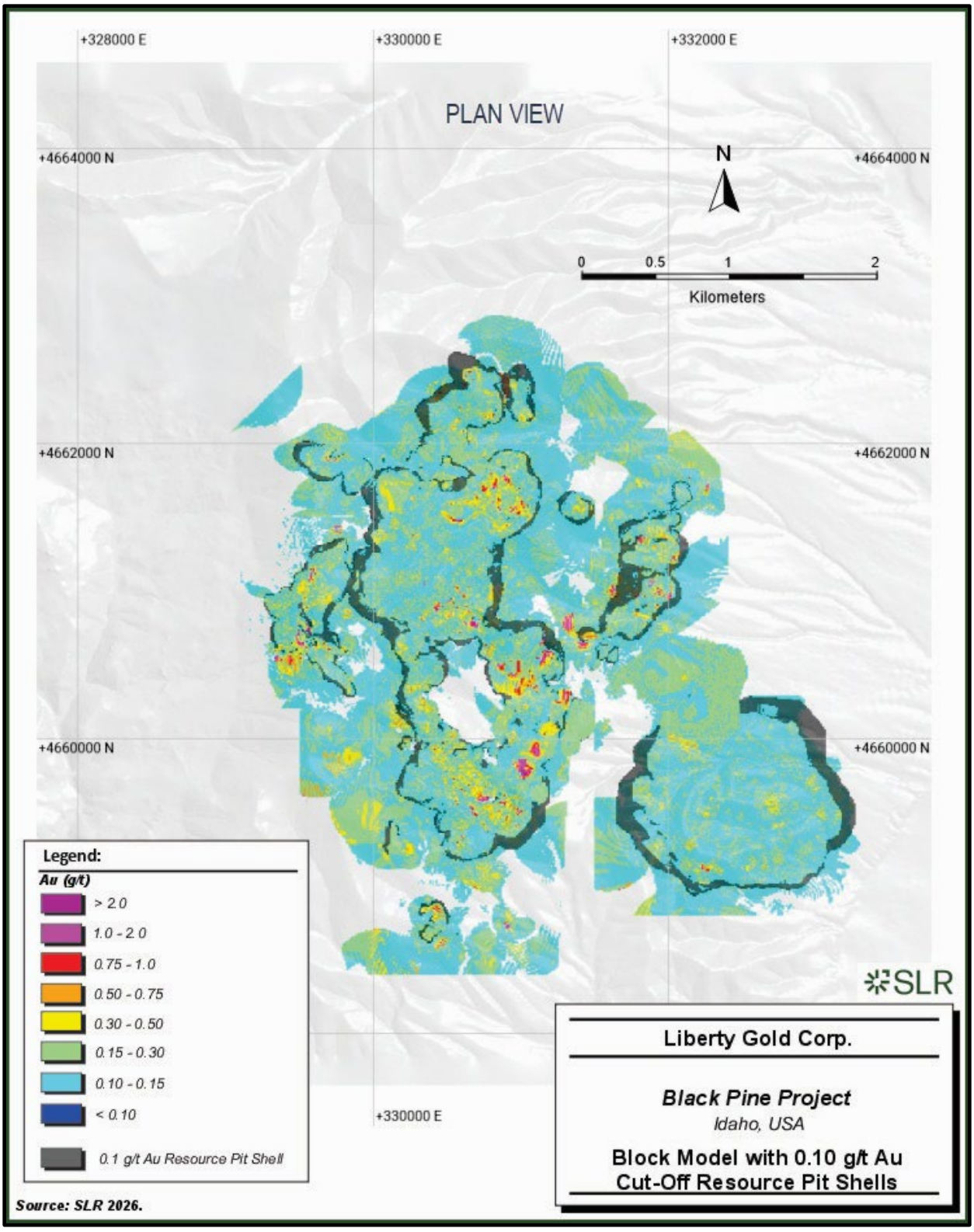


Figure 14-16: Block Model with 0.10 g/t Au Cut-Off Resource Pit Shells

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 14-16. 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 grades, density, lithology, and mineral types
- Process and mining costs
- Downstream costs, such as gold refining, royalties, and freight

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	Value
Gold Price	US$/oz	2,800
Base Mining Cost	$/t mined	2.5935
Process Cost		
Process Cost	$/t leached	2.098
Site General Cost	$/t leached	0.8
Recoveries		
Processing Gold Recovery	%	based on equations from Met testing
Other Costs		
Dore Freight, Security & Insurance	$/oz produced	2.2
Royalties (0.25% of Au Price)	$/oz produced	7

Note: Gold recovery assigned in Block Model

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 modeling 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

All remaining blocks contained within the wireframe model and estimated within the block model were limited to an Inferred classification.

The backfill dumps, waste dumps, and 30 m low-grade buffer domain 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 14-17.

In the SLR QP's opinion the classification of Mineral Resources is reasonable and appropriate for disclosure.

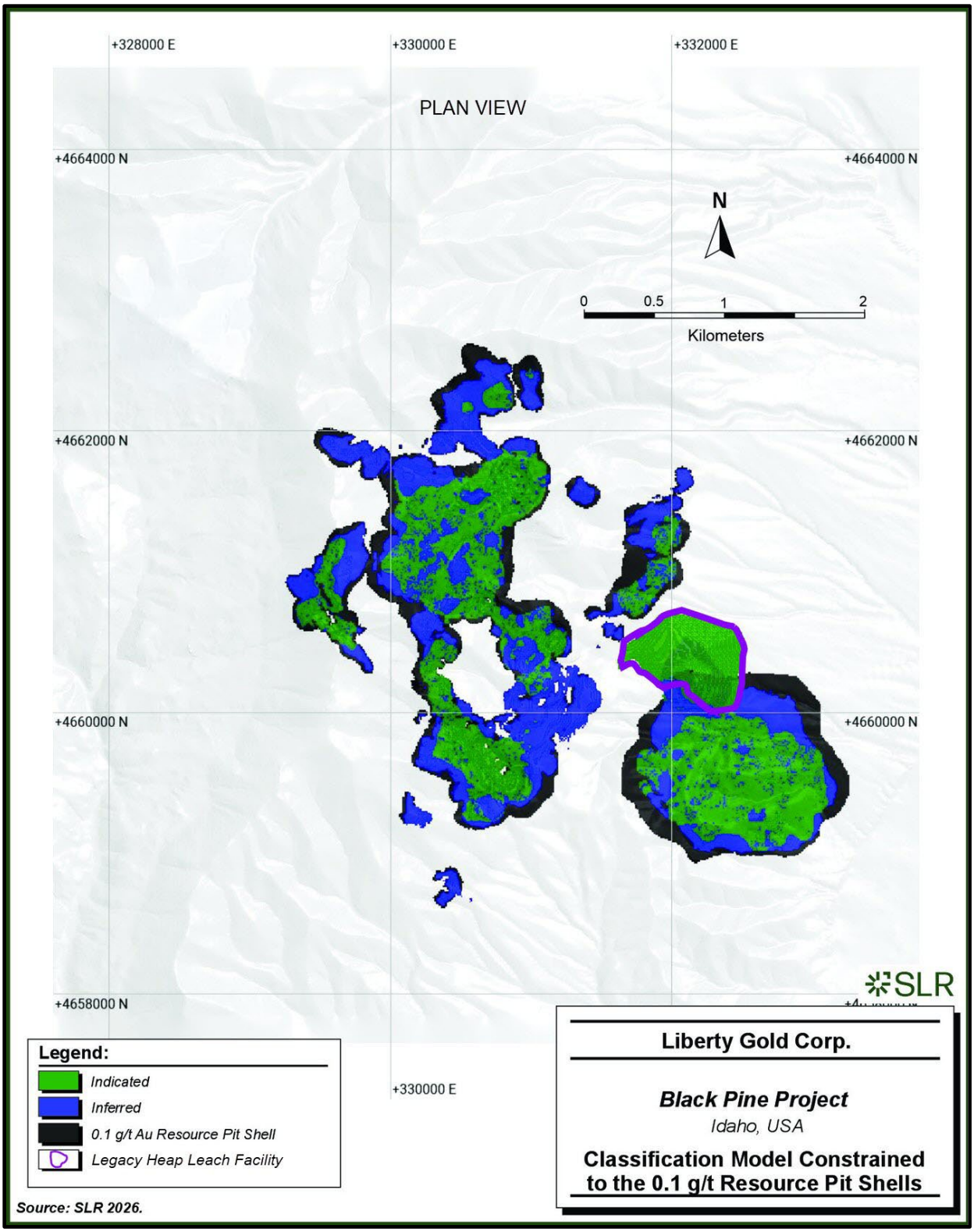


Figure 14-17: Classification Model Constrained to the 0.1 g/t Au Resource Pit Shells

14.14 BLOCK MODEL VALIDATION

14.14.1 Statistics

The SLR QP reviewed and validated the block model using various modeling 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 14-17. The comparison of the estimated silver block 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. There are instances where the internal low-grade domain blocks have a higher maximum grade than the composites within the same domain, which is due to the 5 m one-way soft boundary that was used for estimation. This is not considered problematic, as the internal low-grade domains are nested within the medium-grade domains and are solely used to constrain the extents of the samples at or near the cut-off grade.

Table 14-17: Comparison of Block and Composite Gold Grades

Zone	Domain Group	Type	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	CV
CD	HG	Blocks	298,529	0.00	5.43	0.81	0.58
		Comps	1,836	0.02	6.10	0.88	0.72
	MG	Blocks	887,935	0.00	1.51	0.19	0.43
		Comps	6,249	0.02	1.65	0.23	0.51
	LG	Blocks	1,381,812	0.00	1.81	0.04	0.85
		Comps	5,322	0.00	0.99	0.04	1.27
	ILG	Blocks	115,168	0.00	1.10	0.11	0.67
		Comps	1,644	0.00	0.36	0.05	0.68
	Backfill	Blocks	57,747	0.00	1.19	0.11	0.90
		Comps	675	0.00	3.50	0.29	1.29
	Waste Dump	Blocks	57,461	0.00	5.23	0.14	1.50
		Comps	269	0.00	5.25	0.21	2.19
F	HG	Blocks	28,450	0.00	7.18	0.91	0.66
		Comps	219	0.08	8.00	1.03	1.13
	MG	Blocks	149,199	0.00	1.11	0.20	0.44
		Comps	868	0.02	1.15	0.23	0.59
	LG	Blocks	300,968	0.00	0.67	0.04	0.75
		Comps	1,313	0.00	0.69	0.04	1.20
	ILG	Blocks	17,377	0.01	0.75	0.10	0.58
		Comps	198	0.01	0.30	0.06	0.63
	Waste Dump	Blocks	7,038	0.03	0.33	0.14	0.50
		Comps	28	0.03	0.40	0.16	0.60
Disco	HG	Blocks	632,108	0.00	17.59	1.33	0.78
		Comps	4,882	0.00	20.00	1.55	1.16

Zone	Domain Group	Type	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	CV
	MG	Blocks	2,221,950	0.00	4.83	0.22	0.61
		Comps	21,669	0.00	5.00	0.24	0.93
	LG	Blocks	2,793,575	0.00	1.45	0.05	0.82
		Comps	18,261	0.00	1.50	0.05	1.21
	ILG	Blocks	219,383	0.00	4.47	0.11	0.74
		Comps	4,779	0.00	1.50	0.06	1.05
	Backfill	Blocks	128,421	0.00	3.74	0.18	1.55
		Comps	416	0.02	4.40	0.22	1.56
	Waste Dump	Blocks	87,787	0.00	1.04	0.14	0.88
		Comps	437	0.00	1.29	0.16	0.89
E	HG	Blocks	23,870	0.00	3.61	0.94	0.60
		Comps	414	0.02	4.35	1.10	0.77
	E Pit_3200	Blocks	4,050	0.00	23.29	3.89	1.04
		Comps	173	0.02	24.33	4.28	1.24
	E Pit_3201	Blocks	6,781	0.14	14.40	2.35	0.93
		Comps	380	0.02	20.40	2.60	1.36
	MG	Blocks	279,645	0.00	2.85	0.20	0.60
		Comps	4,626	0.02	3.00	0.23	0.85
	LG	Blocks[1]	590,919	0.00	0.00	0.00	0.00
		Comps	6,290	0.00	2.74	0.05	1.40
	ILG	Blocks	30,808	0.00	4.90	0.11	1.20
		Comps	1,434	0.01	0.36	0.05	0.81
J	HG	Blocks	49,018	0.00	4.48	0.93	0.54
		Comps	483	0.07	5.68	0.97	0.81
	MG	Blocks	493,285	0.00	1.86	0.20	0.59
		Comps	2,220	0.02	1.88	0.22	0.79
	LG	Blocks	1,014,117	0.00	0.66	0.04	1.02
		Comps	4,525	0.00	0.73	0.04	1.39
	ILG	Blocks	22,557	0.02	0.12	0.04	0.47
		Comps	534	0.02	0.16	0.05	0.63
	Waste Dump	Blocks	76,804	0.02	0.46	0.18	0.42
		Comps	91	0.02	0.47	0.16	0.55
M	HG	Blocks	102,161	0.00	7.99	1.39	0.62
		Comps	712	0.02	8.00	1.39	0.87
	MG	Blocks	606,679	0.00	2.15	0.20	0.72
		Comps	2,658	0.01	2.22	0.24	0.92
	LG	Blocks	1,269,228	0.00	1.00	0.03	1.09
		Comps	8,815	0.00	1.00	0.04	1.45
	ILG	Blocks	35,780	0.01	2.12	0.16	1.38
		Comps	572	0.01	0.61	0.07	0.90
Rangefront	HG	Blocks	1,273,669	0.00	4.99	0.59	0.72
		Comps	3,407	0.01	5.00	0.61	0.95
	MG	Blocks	2,967,799	0.00	0.97	0.15	0.38

Zone	Domain Group	Type	Count	Minimum (g/t Au)	Maximum (g/t Au)	Mean (g/t Au)	CV
		Comps	10,308	0.00	1.00	0.16	0.50
	LG	Blocks	3,517,592	0.00	0.88	0.04	0.75
		Comps	11,111	0.00	0.88	0.05	0.75
	ILG	Blocks	468,053	0.00	0.72	0.09	0.55
		Comps	3,893	0.00	0.65	0.07	0.62

[1] The LG domain in E zone does not exist as there was uncertainty about low-grade continuity. The results for that mineralized zone is that all grade outside of the existing domains was not included in any resource classification.

Table 14-18: Comparison of Block and Composite Silver Grades

Zone	Domain Group	Type	Count	Minimum (g/t Ag)	Maximum (g/t Ag)	Mean (g/t Ag)	CV
Disco	MG	Blocks	228,940	0.00	45.45	1.26	1.02
		Comps	18,796	0.00	55.00	1.34	1.49
	LG	Blocks	303,212	0.00	23.73	0.82	0.98
		Comps	13,077	0.00	27.75	1.10	1.31
M	MG	Blocks	43,224	0.00	108.03	2.00	2.12
		Comps	1,858	0.00	110.00	3.62	2.99
	LG	Blocks	154,703	0.00	97.49	0.85	1.19
		Comps	4,230	0.00	110.00	1.39	1.95

14.14.2 Swath Plots

SLR produced comparative statistics, including NN and ID^2 estimations, and swath plots for all deposits. Swath plots generally demonstrated good correlation, with domain and the high-grade zones within it. Swath plots for gold values in Discovery zone domain 203 are shown in Figure 14-18. Swath plots for silver values in Discovery zone domain 201 are shown in Figure 14-19.

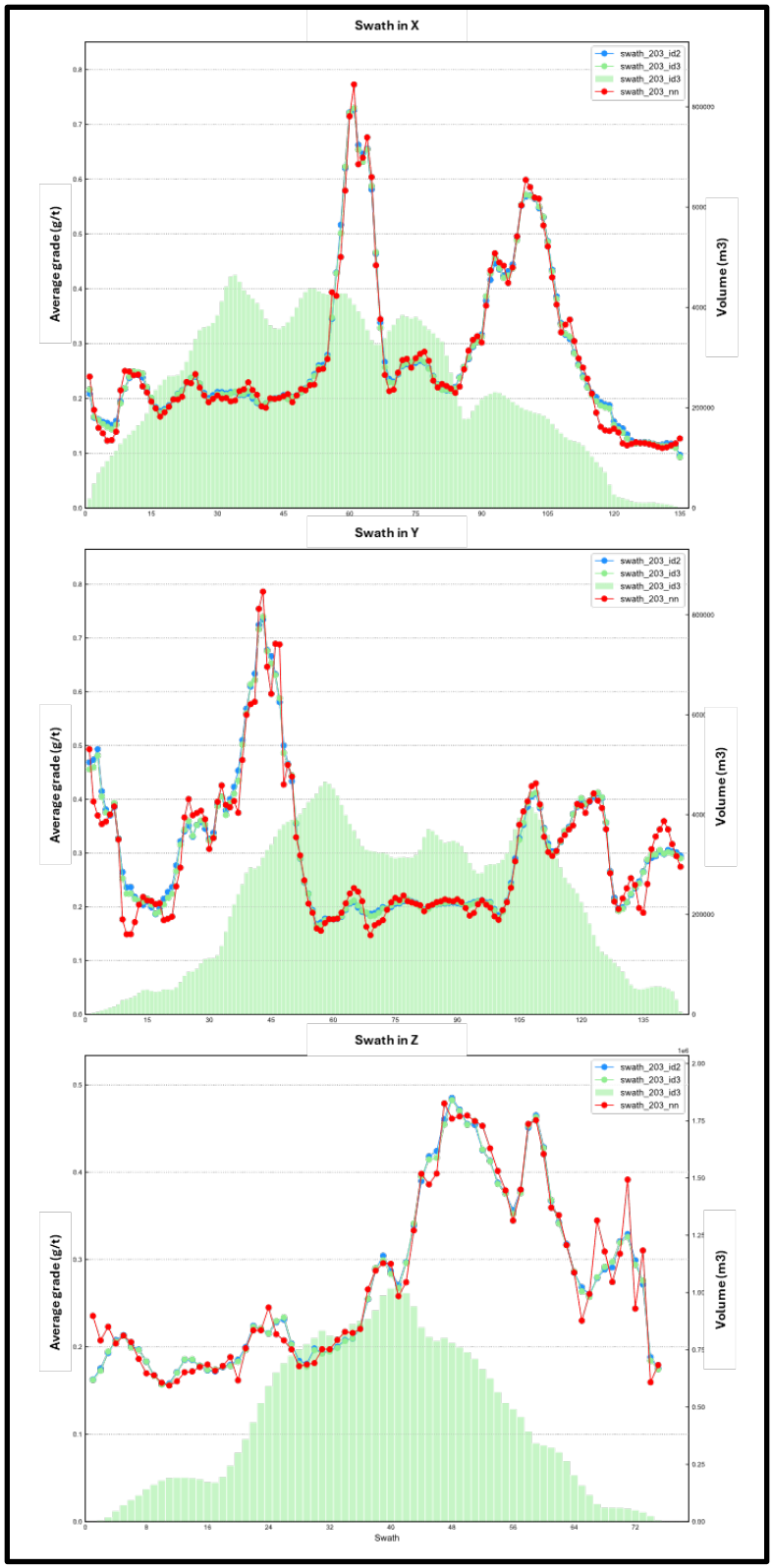


Figure 14-18: Gold Swath Plots Through Discovery 203 Zone

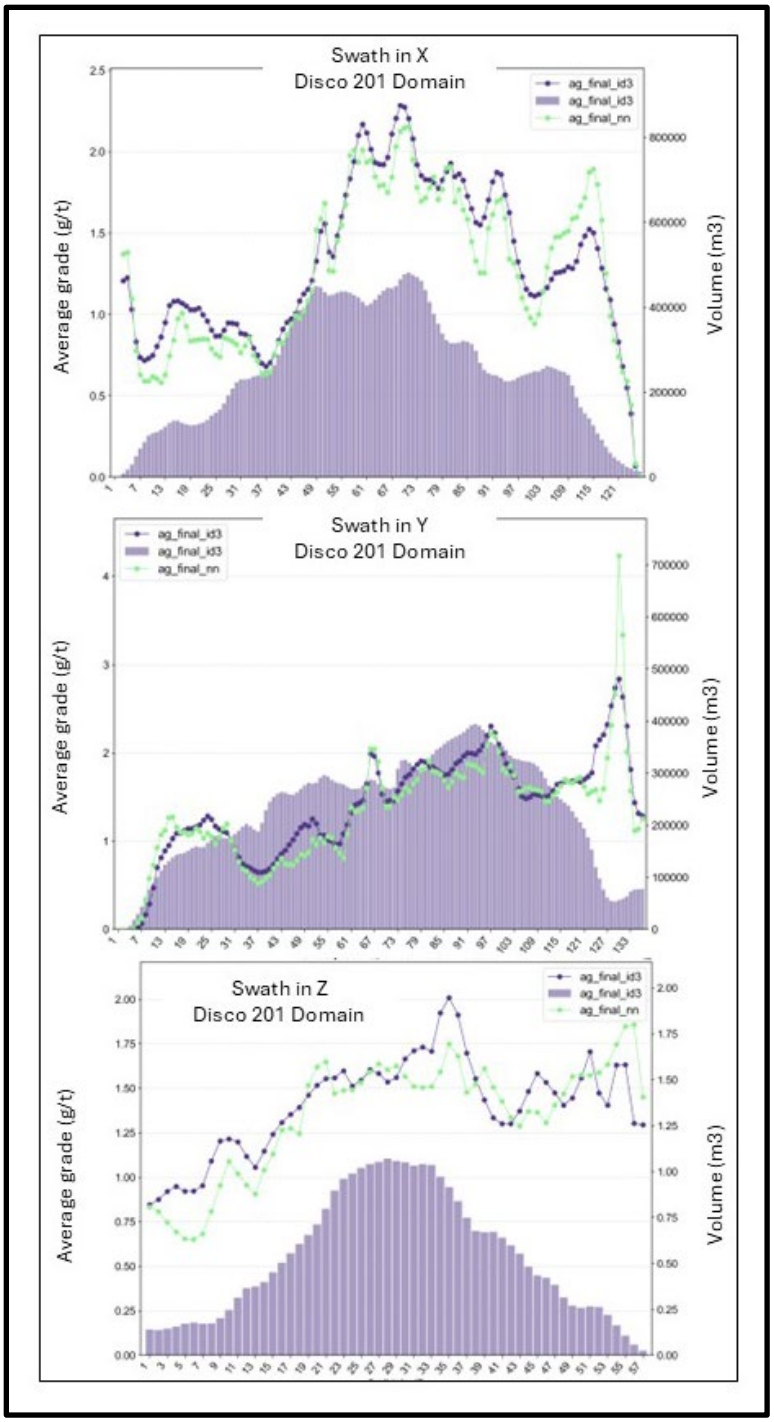


Figure 14-19: Silver Swath Plots Through Discovery 201 Zone

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 14-20 through Figure 14-24 show various cross sections through different parts of the Project.

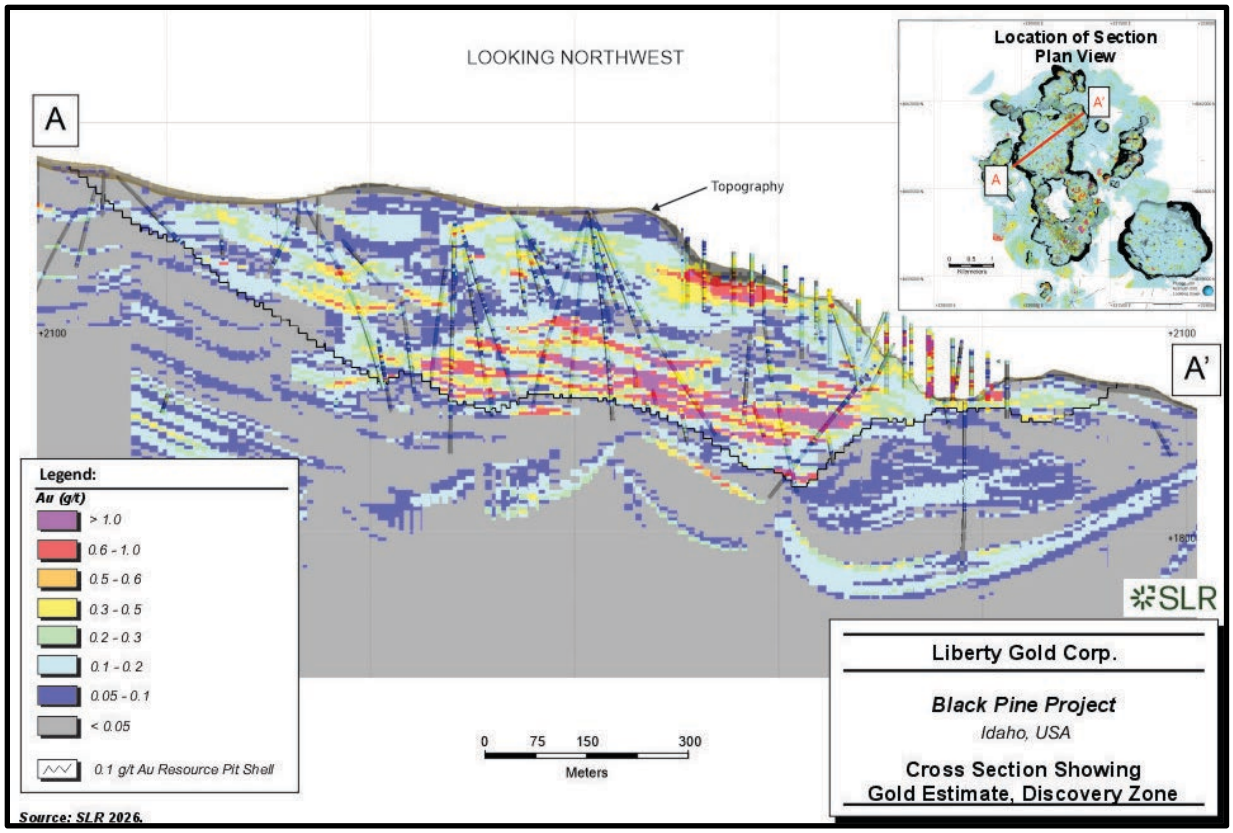


Figure 14-20: Cross Section Showing Gold Estimate Through Discovery Zone

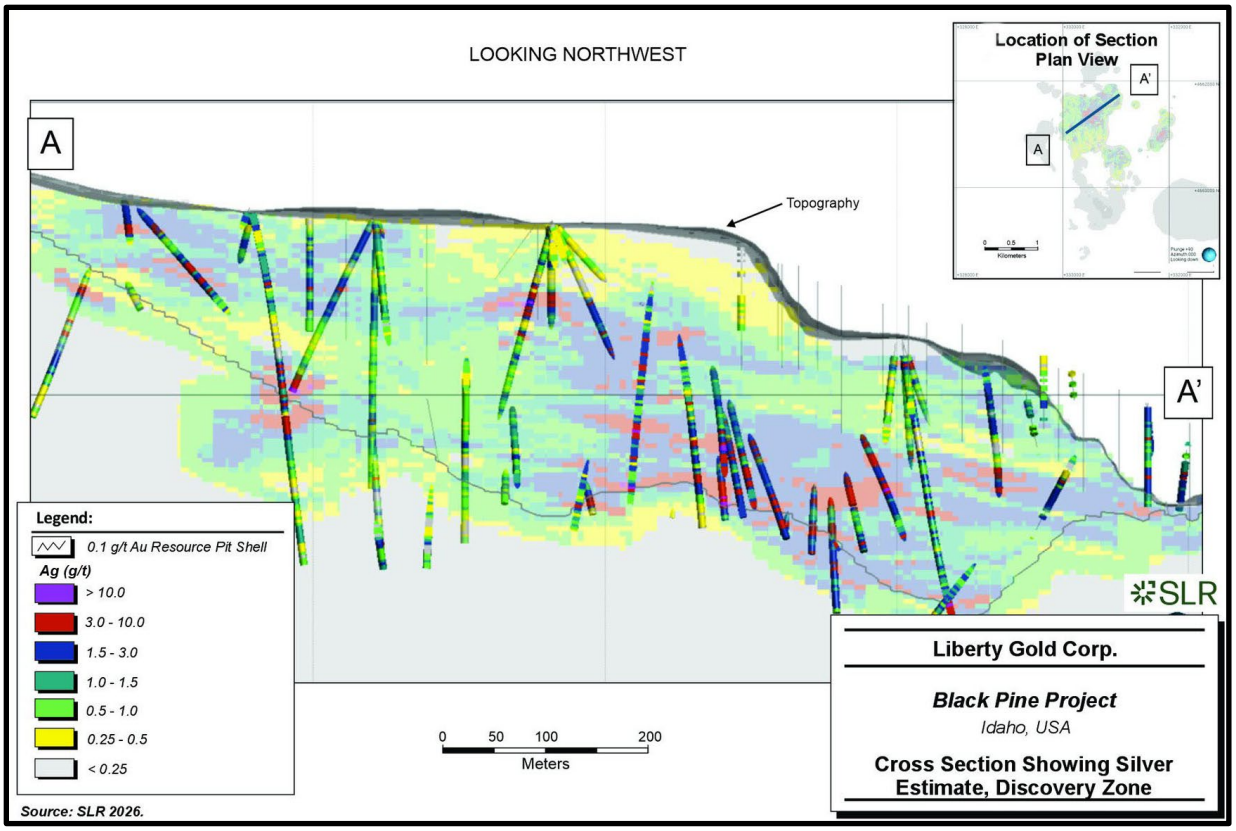


Figure 14-21: Cross Section Showing Silver Estimate, Discovery Zone

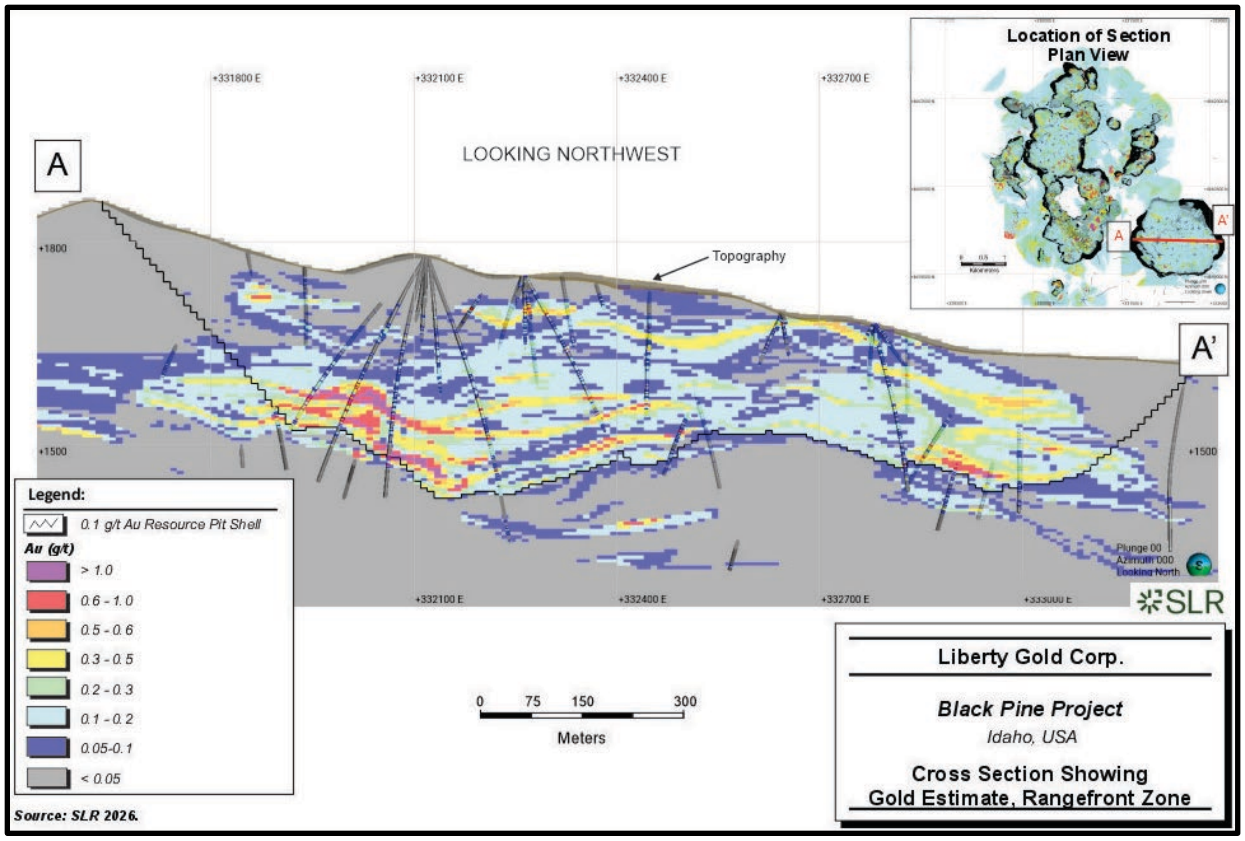


Figure 14-22: Cross Section Showing Gold Estimate Through Rangefront Zone

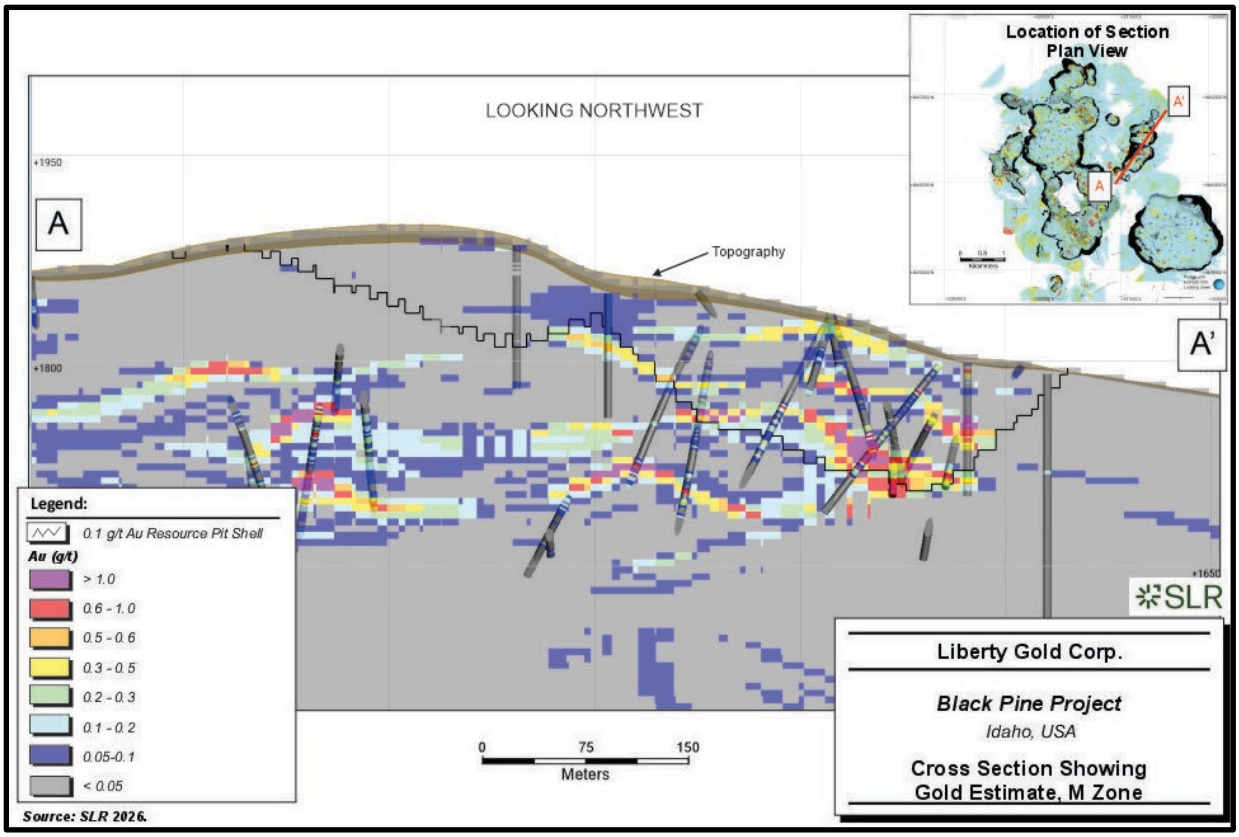


Figure 14-23: Cross Section Showing Gold Estimate Through M zone

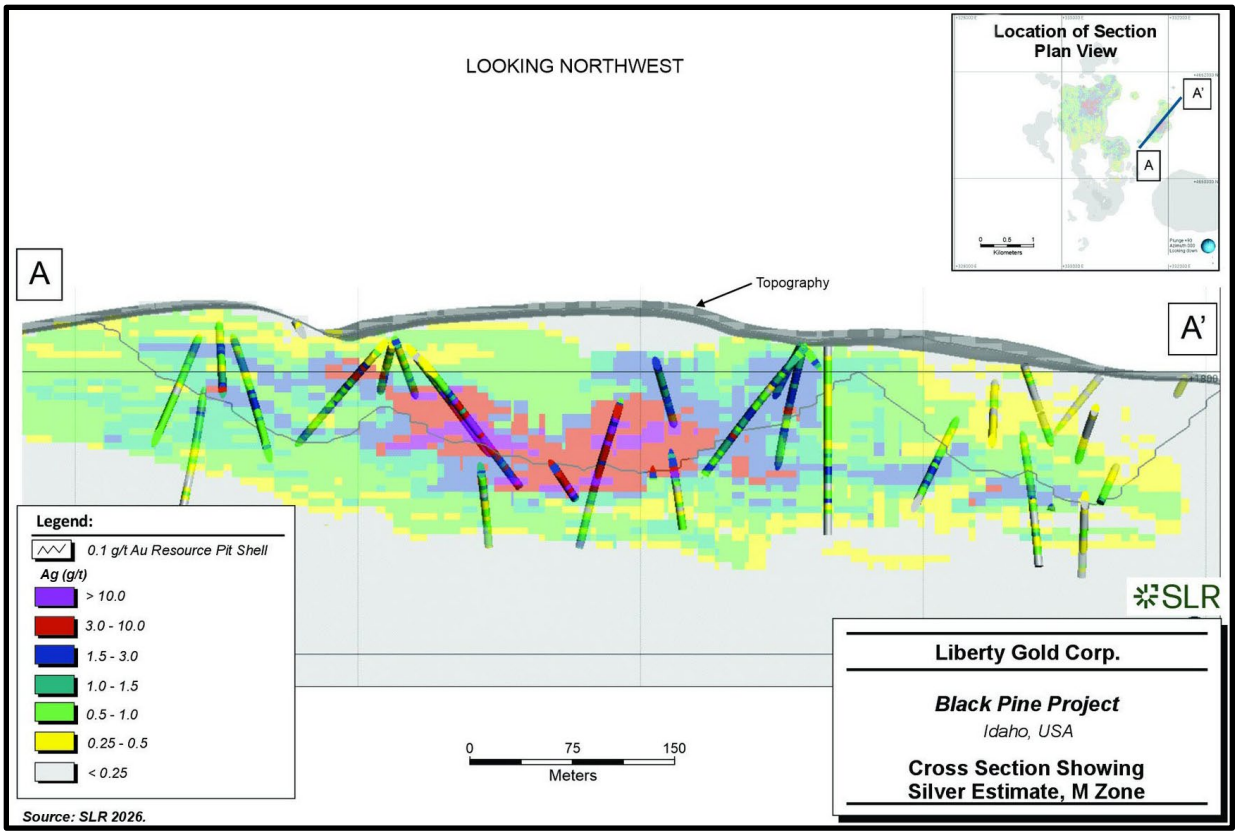


Figure 14-24: Cross Section Showing Silver Estimate, M Zone

14.14.3.1 Blast Data Visual Reconciliation

A lack of production records by pits and zones means direct reconciliation cannot be performed. However, blast hole records do exist for some of the historical pits. SLR visually compared gold assay results from blast hole data to the updated block model and found the comparisons to be favorable. Figure 14-25 is a cross section through the historical CD pit comparing gold values from historical blast holes with the block model.

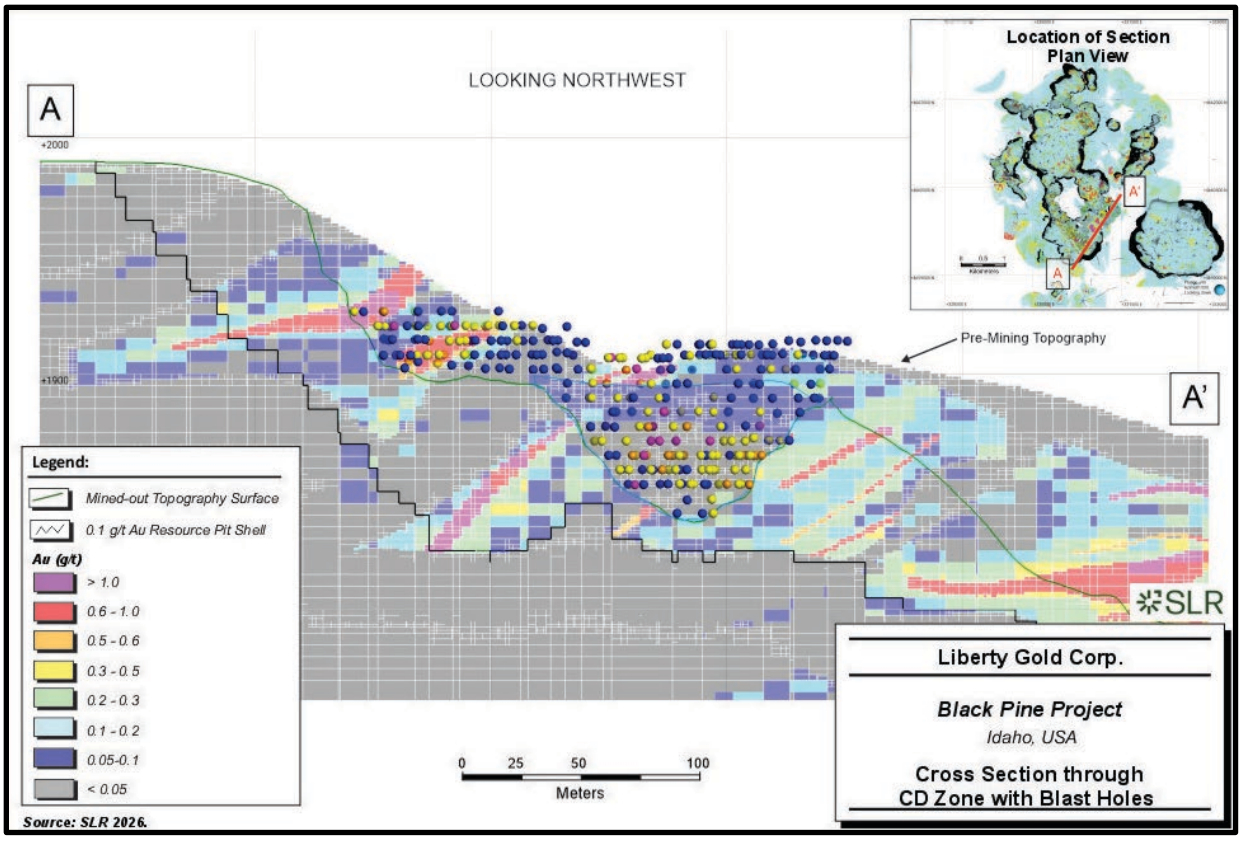


Figure 14-25: Cross Section through CD Zone with Blast Holes

14.15 GRADE TONNAGE SENSITIVITY

Figure 14-26 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 is 1,157 Mt.

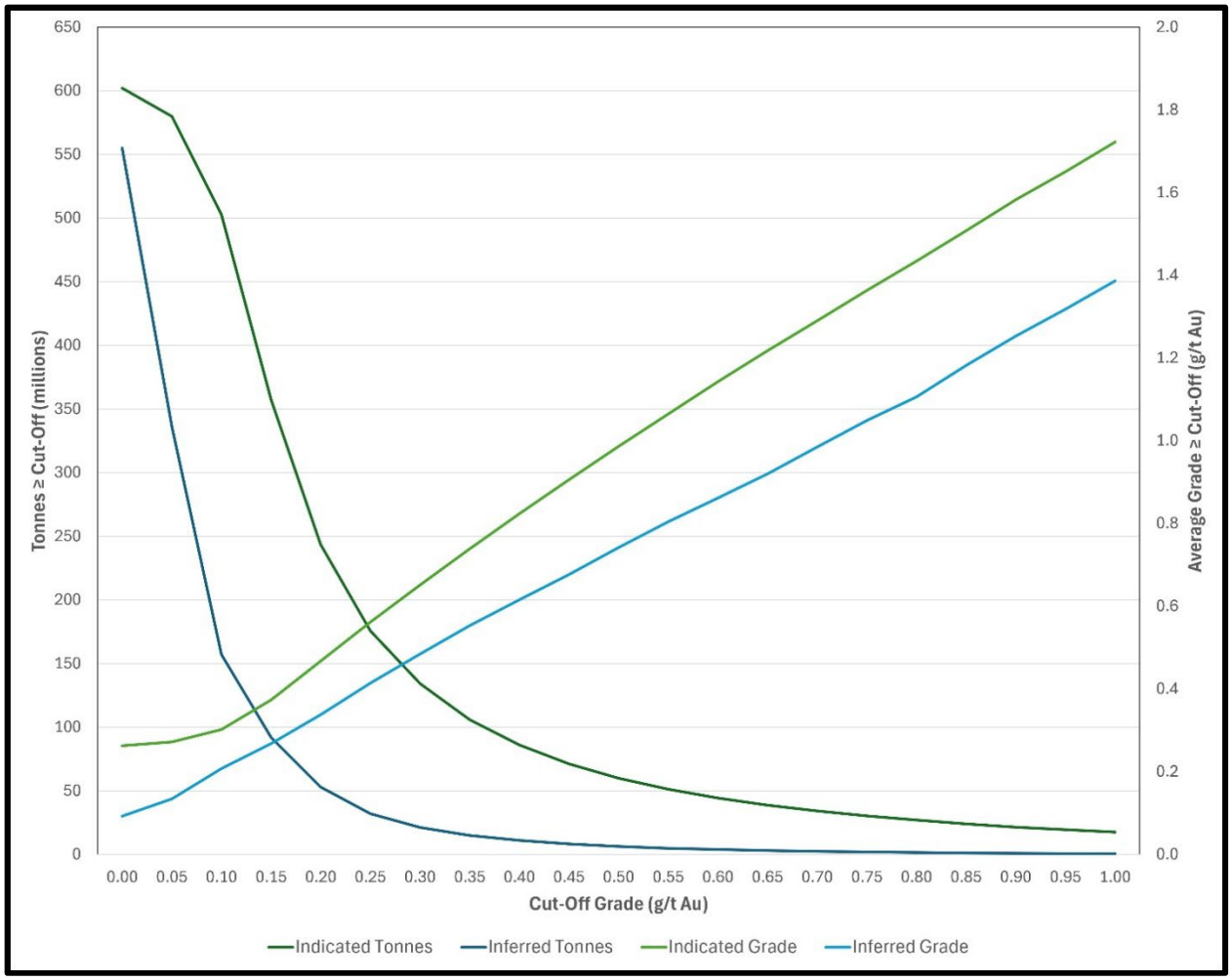


Figure 14-26: Indicated and Inferred Gold Mineral Resource Grade Tonnage Curves

14.16 ESTIMATION OF THE LEGACY HEAP LEACH FACILITY

In April 2025, Liberty Gold requested Forte Dynamics' (now part of SLR) services in preparing a sonic drill program for the Legacy Heap Leach Facility (LHLF). The objective of this work was to collect sufficient data to develop a Mineral Resource estimate of the residual gold in the HLF, for inclusion in the feasibility study. The QP for the HLF estimate is Mr. Don Hulse of Forte Dynamics.

14.16.1 Drill Program

Twenty-four sonic drill holes were planned to provide an even distribution of samples to characterize the grades and residual gold in the LHLF. Two additional sonic holes were drilled to test the waste dump in the CD area. Samples in the HLF were analyzed to determine both contained metal and the potentially recoverable gold. As the site has been remediated, only limited work was allowed and restrictions to the program included:

- No construction of roads or drill pads
- Drilling only on the top surface of the HLF
- Dry drilling only

- No work on the side slopes
- Stop all holes 10 m above the estimated liner elevation (based on the pre-mining topography)
- Backfill all holes with bentonite prior to completion of the project
- File abandonment reports
- Provide a plastic liner at each drill site to catch and collect spills

Sonic drilling was selected for this work as it has become a standard for drilling leach pads and mine dumps. It extracts a largely intact sample with minimal disturbance of the vertical column of the material. Samples were recovered from the core barrel in 3.048 m (10 ft) intervals. Core was logged, photographed, and select recovered materials were composited using the 'cone and quarter' method prior to shipping samples for laboratory analysis.

14.16.2 Laboratory Analysis and Testing

457 samples covering 1,393 m (4,570 ft) were sent to the ALS Laboratories preparation facility in Idaho Falls, Idaho. Analysis was completed by ALS Minerals in Sparks, Nevada, consisting of the following:

- Each head sample was split to obtain representative sub-splits for assay and specialty testing.
- The primary assay split was ring-and-puck pulverized
- Gold was assayed by 1 assay-ton fire assay with atomic absorption (AA) finish.
- Silver was assayed by 1 assay-ton fire assay with four-acid digestion and AA finish.
- Multi-element analysis was completed by four-acid digestion with ICP-OES finish.
- A verification split was submitted for gold and silver by PhotonAssay.
- A separate split was analyzed for carbon/sulfur speciation by LECO CS-230, including 33% HCl
- leach treatment.
- Cyanide solubility was determined on a split using a 24-hour cyanide shake test (15 g sample,30 mL solution, 0.50% NaCN) for gold and silver.
- Preg-robbing potential was evaluated on a split using a shake leach with a 1 ppm gold spike.

14.16.3 Statistical Analysis and Estimation

The statistics for the gold values are shown in Table 14-19. The analysis focused on total gold (Au) and cyanide soluble gold (AuCN). Silver was analyzed however with a mean of 0.41 g/t and maximum of 1.51 g/t, it was considered to have no economic impact and was excluded from reporting. Several methods were used to determine bulk density. The method that yielded the highest confidence was to compare the estimated volume of the HLF with the tonnage deposited based on historical production records. This provided an estimated bulk density value of 1.8 t/m^3.

Table 14-19: Descriptive Gold Statistics for the Sonic Drill Holes

Value	Number of Missing Samples	Number of Samples	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	CV
Au	4	455	0.06	0.62	0.19	0.43
Au CN	33	426	0.03	0.51	0.10	0.61

Based on statistical plots, the gold values were capped at 0.40 g/t. Cyanide soluble gold was capped at 0.30 g/t. Variograms were generated to determine if there was any horizontal continuity, as each lift of the pad was placed with material physically proximal in the open pit. The variograms have a nugget of approximately 25% with vertical and horizontal ranges of 21 m and 168 m, respectively.

Ordinary Kriging (OK) was used, with a two-pass search strategy. The ellipsoids measured 200 m x 200 m x 30 m (first pass) and 500 m x 500 m x 75 m (second pass).

The values were estimated using a regular block model in MicroModel™ mining software, with block size of 10 m x 10 m x 2 m. To align with the block model containing the in-situ resource, the backfill, and the waste dumps, the HLF block model values were merged into the block model described in Section 14.10 using Leapfrog Geo software. The HLF was reported in the Indicated category.

14.17 MINERAL RESOURCE ESTIMATE REPORTING

The Project resource estimate is summarized by material type (in situ, backfill, waste dumps, and HLF) in Table 14-20. The in-situ material, as well as the backfill and waste dumps, are reported at a 0.10 g/t gold cut-off grade within an optimized resource pit shell. Silver is reported only for those blocks with the pit shell that meet or exceed the 0.10 g/t gold cut-off grade. Due to the requirement for the HLF material to remain in a containment facility, any potential mining scenario must consider mining and stacking all of the material on the new leach pad. The heap leach pad is reported in its entirety at a 0.10 g/t gold cut-off grade.

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 April 30, 2026. 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 26, 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 14-20: Summary of Mineral Resources as of April 30, 2026

Material Type	Area	Classification	Tonnage (kt)	Average Grade		Contained Metal	
				Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
In-Situ	CD	Indicated	37,614	0.28	—	336	—
		Inferred	12,223	0.20	—	80	—
	Discovery	Indicated	188,399	0.35	1.35	2,097	8,147
		Inferred	27,503	0.19	1.03	169	912
	E Pit	Indicated	9,657	0.26	—	82	—
		Inferred	11,825	0.22	—	82	—
	F	Indicated	6,543	0.32	—	66	—
		Inferred	1,217	0.24	—	10	—
	J Zone	Indicated	3,601	0.33	—	38	—
		Inferred	12,731	0.25	—	104	—
	M Zone	Indicated	7,324	0.51	2.57	120	606
		Inferred	5,406	0.21	1.25	36	217
	Rangefront	Indicated	237,679	0.26	—	1,998	—
		Inferred	59,787	0.19	—	371	—
Total In-Situ		**Indicated**	**490,816**	**0.30**	—	**4,737**	**8,754**
		Inferred	**130,691**	**0.20**	—	**851**	**1,133**
Backfill and Waste Dumps		**Indicated**	**0**	—	—	**0**	—
		Inferred	**18,144**	**0.25**	—	**145**	—
HLF		**Indicated**	**30,844**	**0.19**	—	**190**	—
		Inferred	**0**	—	—	—	—
Total		**Indicated**	**521,660**	**0.29**	—	**4,927**	**8,754**
		Inferred	**148,835**	**0.21**	—	**996**	**1,133**

Notes:

1. The Mineral Resource has an effective date of April 30, 2026.
2. The Mineral Resource was prepared by Ms. Valerie Wilson, P.Geo., Principal Resource Geologist at SLR Consulting (Canada) Inc. (SLR) but for the LHLF estimate, which was prepared by Mr. Don Hulse, RM-SME of Forte Dynamics. Ms. Wilson is the Qualified Person (QP) responsible as defined under NI 43-101 for the Mineral Resource estimate but not for the LHLF material; Ms. Wilson is independent of Liberty Gold. Mr. Don Hulse is the QP for the LHLF estimate and is independent of Liberty Gold. CIM (2014) definitions were followed for Mineral Resources.
3. Mineral Resources are estimated at a gold cut-off grade of 0.10 g/t using a long-term gold price of US$2,800 per ounce.
4. Gold is reported for all blocks of in-situ, backfill, waste dump, or LHLF material. The waste dumps and backfill are limited to the CD, Discovery, F, J, and Rangefront zones. The LHLF is within the Rangefront zone, with the northern portion crossing into M Zone.
5. Silver is reported only for those blocks that meet or exceed the gold cut off grade of 0.10 g/t Au. It is limited to the Discovery and M zones, areas which are supported by analytical results from 4-acid digestion at a drill spacing threshold for gold class criteria.
6. Mineral Resources are estimated using variable recoveries derived from metallurgical studies.
7. Bulk density is variable by rock type and ranges between 2.43 t/m^3 and 2.63 t/m^3 in the mineralized domains. Quaternary deposits, LHLF, Backfill, and Waste Dumps were assigned a value of 1.8 t/m^3.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are inclusive of Mineral Reserves.
9. In-situ, backfill, and waste dump material are reported within conceptual open pit shells. Silver grades were not used in the pit optimization.
10. LHLF material is reported within the LHLF wireframe above a 0.10 g/t Au cut-off grade. Silver is not considered to be recoverable within the LHLF and is not reported.
11. The QPs for this Mineral Resource estimate are not aware of any environmental, permitting, legal, title, taxation, socio economic, marketing, political, or other factors that could materially affect the Mineral Resource Estimates presented in this news release.
12. Numbers may not add or multiply correctly due to rounding.

15 MINERAL RESERVE ESTIMATES

The Black Pine 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) for this report.

The Mineral Reserve consists of Indicated blocks above a cut-off of 0.05 g/t Au recoverable grade and contained within the ultimate pit design. No Measured Mineral Resources are classified in the Black Pine deposits, so there are no Proven Reserves. Pits were designed in accordance with geotechnical recommendations, and are based on economic calculations using metal prices, costs, and recoveries.

15.1 BLACK PINE MINERAL RESERVE ESTIMATE

The Mineral Reserves for the Black Pine Project are listed in Table 15-1 with the gold grade (Au) estimates based on the gold grades in the block model.

Table 15-1: Proven and Probable Reserves – Black Pine Project

Reserve Class	Tonnes (Mt)	Au (g/t)	Contained Ounces (Mozs)
Proven	0.0	-	-
Probable	433.3	0.29	4.04
Proven and Probable	**433.3**	**0.29**	**4.04**

Notes:
1. The Mineral Reserve estimate was prepared by AGP, Barrie ON, Canada and has an effective date of August 1, 2026. 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 an incremental net positive revenue cut-off with a minimum 0.05 g/t Au Recoverable Grade.
4. The cut-off grades are based on a gold price of US$3,100/Au oz.
5. Metallurgical recovery of gold is based on a variable gold leach recovery model derived from extensive metallurgical studies.
6. Modeled carbonaceous units and lower recovery metallurgical domains have been treated as waste. Overall gold leach recovery averages 70.2%.
7. Units are metric tonnes, metric grams & troy ounces; “Au” = gold.
8. The estimate of Mineral Reserves may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

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);
- Gold recovery (%);
- Mineral Resource classification (Measured, Indicated, Inferred);
- Block density (t/m^3);
- Metallurgical Zones (gold cyanide solubility zones, clay).

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, mining cost, incremental ore haulage cost, and block value per ton.

15.3 MINE RECOVERY AND DILUTION

No mining dilution or ore loss is applied to the Mineral Reserves.

15.4 WALL SLOPE ANGLES AND BENCH CONFIGURATION

A feasibility level geotechnical investigation for the Black Pine 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. Further information on the geotechnical parameters used can be found in Section 16.3.

15.5 CUTOFF 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 net revenue generated by processing that block is greater than the incremental costs for that block, excluding the base mining cost. A 0.05 g/t Au recoverable grade was set as a minimum cutoff for ore. Further information on how the Cutoff Grade was calculated can be found in Section 16.4

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 15-2 and Table 15-3. Geotechnical parameters for slope design sectors were coded into the reserve model based on recommended pit slope design criteria from Knight Piésold Ltd. consulting. Pit restrictions include a 15-m offset from the measured groundwater level in the Rangefront and CD pits. Optimized pit shells were generated using the block values, pit slope criteria, and associated mining, process, incremental ore haul, 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.

Conceptual pit designs developed for Black Pine are based on the $2,250/Au oz optimization shell which corresponds with a Revenue Factor of 0.9. 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 Piésold. The ramp gradient is 10% with 37.5 m or 22.8 m ramp widths dependent on the size haul truck that is recommended for that pit. 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 Black Pine open pits are split into seven areas. The larger Discovery and Rangefront pits, plus M Pit, J Pit, E Pit, F Pit, and CD pits will be developed as 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	$2,500*
	Royalty	%	0.25%
Dore	Payable	%	100
	Selling Cost	USD/oz	$10.00
Metallurgical	Gold Recovery	%	Varies AURCV[1]
Mining	Base to Pit Exit	$/t mined	$1.62
	Haulage	$/t mined	varies
	Sustaining Capital	$/t mined	$0.26
	Incremental Cost below Exit	$/t mined/ 10 m	$0.026
	Incremental Cost above Exit	$/t mined/ 10 m	$0.023
Processing	Leaching Cost	$/t ore	$1.88
	Incremental Ore Haul	$/t ore	OMCST[2]-WMCST[3]
G&A	General and Administrative	$/t ore	$0.77
Production Assumptions	Leach Feed Rate	t/year	18.25 Mt
	Mining Dilution	%	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

*The $2,250/oz optimized pit shell was selected for the ultimate pit limit.
[1]AURCV - 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	0.38	1.22	1850
F	9.3	26.9	0.38	1.22	1990
E	23.2	69.7	0.95	2.85	2300
Discovery	15.3	34.4	0.53	1.19	1990
J	8.7	29.3	0.36	1.20	2000
Rangefront	29.6	19.4	1.03	0.67	1630
M	7.0	17.4	0.28	0.71	1790
Default	9.3	26.9	0.38	1.22	1850

15.7 MINE SCHEDULE

The mine schedule plans to deliver 433 Mt of leach feed grading 0.35 g/t Au in the first 5 years, 0.30 g/t Au in the first 10 years, and 0.29 g/t Au over the mine life. The mine life includes one year of pre-production mining and 15 years of in-pit mining. The waste rock tonnage from all pits totaling 582 Mt will be placed into rock storage facilities or backfilled into mined out pits. The overall strip ratio is 1.3:1. Processing of stockpiled ore will continue for 1 year after active mining is complete.

The leach production schedule can place a maximum of 30.0 M tpa (82 kt/d) in Years 1 and 2 and up to 37.0 M tpa (101 kt/d) from Year 3 onward. Leach ore placement varies by year and is dependent on the annual total material movement allowed throughout the project life.

A detailed production schedule was developed using the reserves and built-up costs to support the financial evaluation of the project.

16 MINING METHODS

16.1 OVERVIEW

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 onto the pad. Underground mining was not considered for this study.

The mine schedule for open pit mining consists of 433 Mt of leach feed grading 0.35 g/t Au in the first 5 years, 0.30 g/t Au in the first 10 years, and 0.29 g/t Au over a mine life of 16 years. The open pit waste rock tonnage totals 582 Mt and will be placed into rock storage areas (RSF's). Constructed RSF's will receive 348 Mt of waste rock and 202 Mt of waste rock will be backfilled into mined pits. If included in the current MPO permitting effort, the 32 Mt of legacy heap leach material will be stockpiled on to the lined heap leach facility starting in Year 8 and finishing in Year 16. The projected mine life includes one-year of pre-production followed by 15 years of production mining. Met 1 and Met 2 material is only stockpiled during pre-production and reclaimed to the heap leach pad in Year 1. Low recovery leach ore (Met 3) will be stockpiled throughout the mine life and sent to the leach pad in Years 15 and 16 after in-pit mining is complete. Removal and placement of 4 Mt of legacy heap leach material onto the lined facility occurs in Year 8 during Rangefront mining, with the remaining 28 Mt scheduled for Year 16.

The leach production rate is 27.1 M tpa starting in Year 1 and varies between 16.0 M tpa and 29.4 M tpa through Year 10 due to pit sequencing and total material movement limits. The last five years of full production, projects 37.0 M tpa placed on the pad in Years 11-13, and 32.6 Mt in Years 14-15. Stockpile reclaim includes 8.0 Mt of Met 1 and Met 2 material in Year 1 with 10.1 Mt and 17.6 Mt of Met 3 stockpile material reclaimed to the pad in years 15 and 16, respectively. A total of 35.7 Mt of ore will be stockpiled with a peak stockpile capacity of 27.7 Mt reached in Year 15.

The mine plan presented in this Feasibility Study incorporates feasibility-level refinements to mining rates, sequencing and equipment requirements while remaining consistent with the open-pit, run-of-mine development concept described in the Mine Plan of Operations. The mine plan is intended to support the Mineral Reserve estimate and economic analysis presented herein and is not intended to replace, supersede, amend or otherwise modify the Mine Plan of Operations or any active permitting review.

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 was 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
AU	0	20	0.0001	g/t	Gold Grade Undiluted
AURCV	0	5	0.0001	factor	Leach Recovery Gold
CLASS	0	5	1	code	Resource Classification
REGON	0	10	1	code	Area Region
GEOMT	0	10	1	code	Metallurgical Code
LITHO	0	20	1	code	Lithology
CLAY	0	20	1	code	Clay Code
DENSE	0	3	0.000001	g/cc	Density
PMTOP	0	1	0.0001	0 or 1	Topography code
TOPRG	0	1	0.0001	factor	Topography

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)	bpfs10.dat
MinePlan file 15 (model file)	bpfs15.p02
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 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
AU	0	20	0.0001	g/t	Gold Grade Undiluted
AURCV	0	100	0.0001	factor	Leach Recovery Gold
AG	0	100	0.0001	g/t	Silver Grade Undiluted
AGR	0	100	0.0001	factor	Leach Recovery Silver
RAUD	0	20	0.0001	g/t	Recovered Gold Grade AU*AURCV
RAGD	0	100	0.0001	g/t	Recovered Silver Grade AG*AGRCV
CLASS	0	5	1	code	Resource Classification
REGON	0	10	1	code	Area Region
GEOMT	0	10	1	code	Metallurgical Code
LITHO	0	20	1	code	Lithology
CLAY	0	20	1	code	Clay Code
DENSE	0	3	0.0001	g/cc	Density
PMTOP	0	1	0.0001	0 or 1	Topography code
TOPRG	0	1	0.0001	factor	Topography
TOPO%	0	100	0.0001	%	Percent Topography
SLPCD	0	25	1	Code	Slope Region code
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
OSA	0	90	0.1	degrees	Overall angle for pit optimization
RQDCD	0	10	1	code	RQD Code added 1/22/2026
MINED	0	5	1	-	Code for dilution script
BLOKT	0	2000	0.001	tonnes	Non-diluted block tons
OWFG	0	2	1	-	Ore: waste Flag; if >0.1 g/t Au cut-off
DTON	0	2000	0.001	tonnes	Diluted block tonnes from script
DDEN	0	5	0.0001	g/cc	Density from dilution script
AUDM	0	20	0.0001	g/t	Mining dilution applied to AUD
RAUDM	0	20	0.0001	g/t	Recoverable gold grade diluted
AGDM	0	100	0.0001	g/t	Mining dilution applied to AGD
RAGMD	0	100	0.0001	g/t	Recoverable Silver grade diluted
CFPC1	-25	100	0.01	$/t	Cash Flow per tonne
CFPC2	-25	100	0.01	$/t	Cash Flow per tonne
DEST1	0	5	1	code	Destination Code -CFPC1
DEST2	0	5	1	code	Destination Code -CFPC2
OMCST	0	20	0.001	$/t	Ore Mining Cost
WMCST	0	20	0.001	$/t	Waste Mining Cost
INCR	0	6	0.001	$/t	Bench Incremental Mining Cost
DEPTH	-50	40	1	# benches	Depth from Base Bench (pit exit)
OMCYC	0	100	0.01	min	Ore Mining cycle time
WMCYC	0	100	0.01	min	Waste mining Cycle time
PMTP2	0	1	0.0001		PMTOP original
ROYAL	0	5	1	code	Royalty code

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 catch bench widths are unique to each prescribed geotechnical domain.

Knight Piésold Ltd. (KP) has been retained for geotechnical assessments in support of the open pit slope design since 2021. KP completed a scoping level open pit geotechnical study in 2022 and a subsequent PFS-level open pit slope design in 2024. The KP scope of work for the FS-level geotechnical assessment includes a desktop review of previous open pit geotechnical studies, a supplementary geotechnical data collection program integrated into the 2025 exploration program, further rock mass characterization and geotechnical model refinement, slope stability analyses, and the development of recommended open pit slope design criteria in support of the FS.

The following geotechnical information was made available for the FS-level pit slope geotechnical design:

- 2021 open pit geotechnical drillhole logs, televiewer surveys, photogrammetric surveys, and laboratory testing data, collected by Golder USA Associates Ltd. (Golder)
- 2019 to 2023 exploration and geotechnical core hole logs and core photos prepared by Liberty Gold field geologists
- 2021 to 2023 televiewer surveys performed by International Directional Services (IDS) and DGI Geoscience Inc. (DGI)
- 2023 laboratory testing data performed by Intermountain GeoEnvironmental Services, Inc. (IGES)
- 2023 RC drillhole logs provided by Liberty Gold geologists
- 2024 laboratory testing data conducted by Call & Nicholas, Inc. (CNI)
- 2025 geotechnical core hole logs produced by Barr Engineering Co. (Barr)
- 2025 televiewer surveys performed by DGI
- 2025 laboratory testing data performed by Agapito Associates, LLC (Agapito).

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, 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 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 thin overburden and other minor lithologic units that are not correlated with the discrete stratigraphic units defined above are considered non-critical for the FS-level pit geotechnical study.

Intact rock strength of the sedimentary package ranges from Medium Strong to Strong, except in the Fault Zone unit. Overall rock mass quality ranges from Poor, to Fair, to Good reflecting different degrees of weathering and oxidization. Flatly dipping to sub horizontal bedding and steeply dipping fracture sets were identified in the deposit area. Localized

high-angle faults, lithologic contacts, and bedding were identified at irregular intervals and additional work is required to further verify and refine their extents. The groundwater table is expected to lie below the ultimate pit bottoms.

The site-specific geotechnical information and slope stability analysis results have been utilized to identify specific design sectors for the ultimate open pits. The geotechnical slope design criteria for the ultimate open pits were developed based on the results of kinematic and limit equilibrium slope stability analyses.

Maximum depths of the Discovery and Rangefront Pits are approximately 400 m, while the CD Pit is approximately 250 m deep. Design sectors were delineated for the proposed Discovery, 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 Figure 16-1 for all pits, including those 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 (°)	Bench Width (m)	Inter-ramp Angle (°)	Overall Slope Angle (°)
Discovery	Main West	420	20	70	10	**49**	**45**
	Main North	220	20	70	9	**51**	
	Main East	210	20	75	9	**54**	
	Southwest	120	20	70	9	**51**	
	Southeast	220	20	75	9	**54**	
CD	West	230	20	70	9	**51**	**44**
	East	110	20	75	9	**54**	
	South	100	20	70	11.5	**47**	
Rangefront	Southwest	380	20	70	13.5	**44**	**39**
	North	270	20	70	11.5	**47**	
	East	200	20	70	11.5	**47**	
	South	200	20	70	11.5	**47**	
Others	-	100-150	20	75	9	**54**	

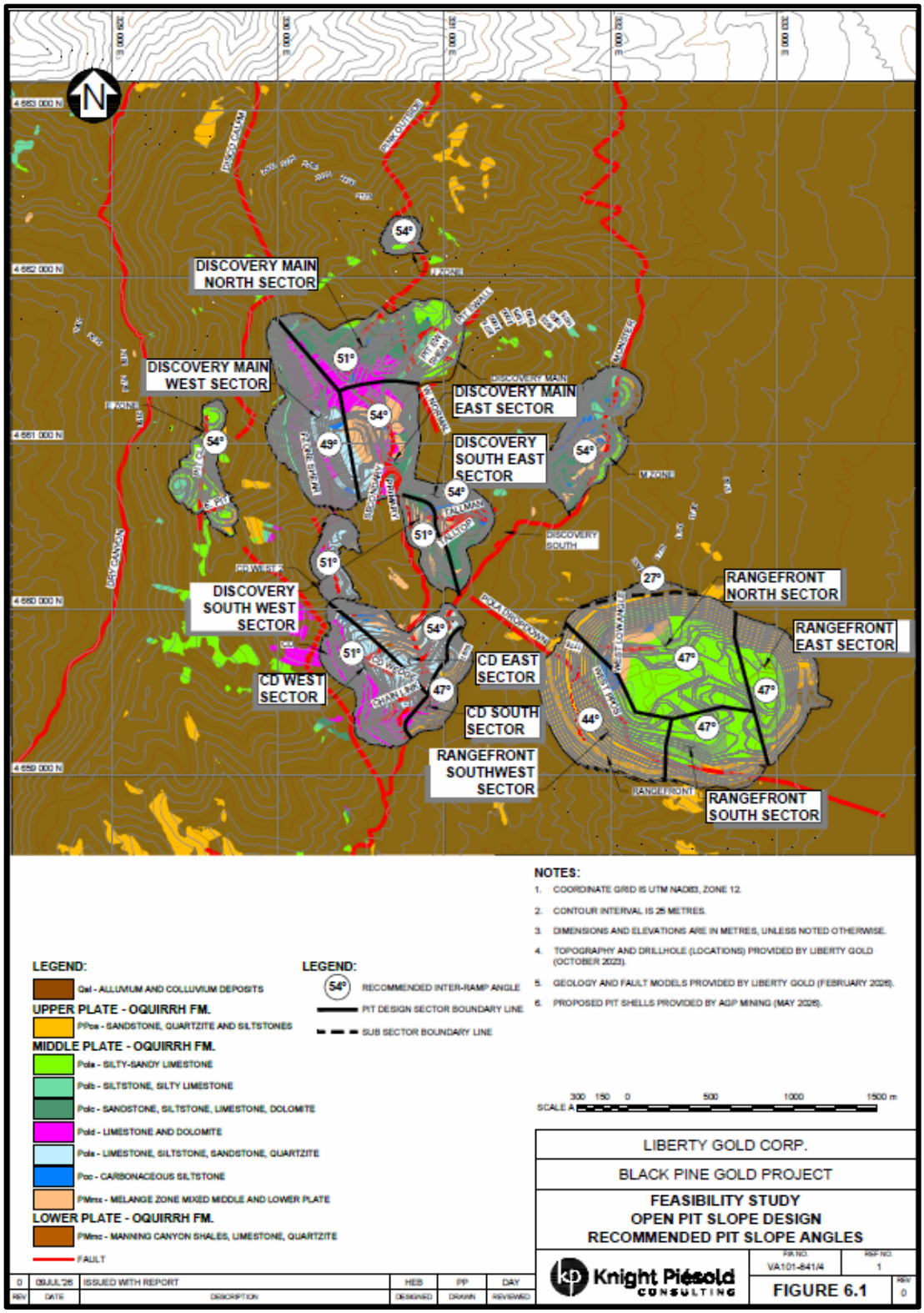
It is recommended that the maximum height of the inter-ramp slopes not exceed 160 m due to the uncertainty of faults and contacts and the variability in rock mass competency. A geotechnical step out (wider bench) or haul ramp should be placed on inter-ramp slopes exceeding 160 m in height to effectively flatten the overall slope angle (OSAs) and to provide additional capacity for rockfall containment and access for debris cleanout. The minimum width of a geotechnical step out should be twice the width of a standard catch berm.

The maximum OSAs represent the highest slopes in Discovery, CD, and Rangefront Pits. The actual OSAs may vary based on the slope height and step out/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 adopted IRAs.

Overburden slopes are not included in the design criteria table due to negligible exposure in the deposit area. Thick overburden, up to approximately 10 m, may be expected on the east side of the Rangefront Pit. The overburden primarily consists of non-plastic to low plasticity alluvial sand and gravel deposits. A 2H:1V slope (27°) is recommended for cut slopes in overburden material to provide adequate long-term stability under drained conditions.

The E, F, J, and M Pits are significantly smaller than the three major pits. The slope geometry is expected to be governed primarily by rock mass structural controls rather than rock mass strength. Steep BFA's and IRA's are recommended for these smaller pits, given structurally controlled instability is managed through appropriate bench configuration and slope orientation.

The implementation of the slope design in accordance with the mine plan not only includes specifications for slope geometry, but also requires low-damage, well-controlled blasting and excavation practices, and systematic slope monitoring throughout pit operations.



Source: Knight Piésold Ltd, 2026

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 long-term metal price assumptions. 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 Liberty Gold and M3 based on the anticipated throughput assumptions and preliminary costing estimates. Metal pricing and selling costs 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 240 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 pre-feasibility study haulage estimates performed by AGP in 2025.

Table 16-5: Pit Shell Cost Assumptions and Parameters

Description		Units	Black Pine
Conversions	Grams per Troy oz	g/oz	31.1035
Metal Prices	Gold Price	USD/oz	$2,500*
	Royalty	%	0.25%
Dore	Payable	%	100
	Selling Cost	USD/oz	$10.00
Metallurgical	Gold Recovery	%	Varies AURCV [1]
Mining	Base	$/t mined	$1.62
	Haulage	$/t mined	varies by pit
	Sustaining Capital	$/t mined	$0.26
Processing	Leaching Cost	$/t ore	$1.88
	Incremental Haul	$/t ore	OMCST[2]-WMCST[3]
G&A	General & Administrative	$/t ore	$0.77
Production Assumptions	Leach Feed Rate	t/year	18.25 Mt
	Mining Dilution	%	0.00%
	Mining Loss	%	0.00%
Planning Model	Block classification used		M+I
	Block Model Height	m	5 m
	Mining Bench Height	m	10 m

*The $2,250/oz optimised pit shell was selected for the ultimate pit limit.
[1]AURCV - Gold Recovery assigned in Block Model
[2]OMCST - Haulage cost from pit area to leach pad (from PFS)
[3]WMCST - Haulage cost from pit area to waste destination (from PFS)

16.4.1 Mining Cost Determination

Mining costs were assigned into the planning block model based on the 2025 pre-feasibility study. The haulage cost component was removed from the mining cost and a base mining cost without haulage was determined to be $1.62/t which includes a 5% escalation from the previous study. The smaller satellite pits base mining cost is $2.11/t as they will be mined by a smaller truck fleet and have an additional assumed 30% increase in cost.

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 $0.0231/t above the pit exit elevation and $0.0258/t below the pit exit. Incremental bench haulage costs were

built from first principles using a cost per hour for the haul trucks and 10% ramp cycle times. The pit exit base elevations were selected based on previous pit designs.

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	0.38	1.22	1850
F	2	9.3	26.9	0.38	1.22	1990
E	3	23.2	69.7	0.95	2.85	2300
Disco	4	15.3	34.4	0.53	1.19	1990
J	5	8.7	29.3	0.36	1.20	2000
Rangefront	6	29.6	19.4	1.03	0.67	1630
M	7	7.0	17.4	0.28	0.71	1790
Default	9	22.4	26.9	0.92	1.10	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 net revenue generated is greater than the incremental costs for that block, excluding the base mining cost. 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 $/g
- Selling Cost is the cost of selling gold (refining and royalties) in $/g
- Recovery is the metallurgical recovery (%)

The minimum 0.05 g/t recoverable gold cutoff was used for mine planning optimization and for final ore determination.

The cutoff grade parameters for ore-waste determination in the final mine plan and reserves differ from the pit optimization parameters. The main change is a higher gold price of $3,100/oz along with higher process and G&A costs. The parameters used for the reserve cut-off are in Table 16-7. The cutoff for each model block is determined using the actual estimated recovery from that block. Along with the variable recovery by block the incremental haulage cost is also variable leading to different cutoffs by pit area and block location. Table 16-8 shows the calculated recoverable gold cutoff by pit area as well as the applied cutoff for the pit. A minimum of 0.05 g/t Au recoverable was used in all areas which was applied to all areas except for E zone.

Table 16-7: Reserve Cutoff Parameters

Parameter	Unit	Item
Gold Price	US$/oz	$3,100
Refining	$/oz	$10.00
Royalty	%	0%
Payable	%	100%
Processing Cost	$/t ore	$2.00
G&A Cost	$/t ore	$1.25

Table 16-8: Incremental Cut-Off Grade by Pit

Pit Area	Incremental Ore Haul $/t	Calculated Recoverable Au Cutoff g/t	Applied Recoverable Au Cutoff g/t
CD	$0.84	0.041	0.050
F Zone	$0.84	0.041	0.050
E Zone	$1.90	0.052	0.052
Discovery	$0.66	0.039	0.050
J Zone	$0.84	0.041	0.050
Rangefront	($0.35)	0.029	0.050
M	$0.43	0.037	0.050

16.4.3 Topography

The topographic information for Black Pine used for mine planning was received along with topography coded into the resource 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 $1.88/t leach cost, $0.77/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.

$$\text{Incremental Ore Haul Cost} = \text{Ore Haul Cost} - \text{Waste Haul Cost}$$

The incremental ore haul cost will impact the cutoff grade of material sent to the leach pad. In cases such as Rangefront, where the waste haul cost is greater than the ore haul cost, the incremental ore haul cost is applied as a credit to the process cost.

16.4.5 Gold Recovery

Gold recovery assumptions are included in the resource block model received from Liberty Gold and are used to determine the recoverable metal. The recovery curves and equations used to input gold recovery into the block model are available in Section 13.3.3.

16.5 MINE PROCESS SCHEDULE

Forte Dynamics Inc. (Forte) utilized a dynamic heap leach model for the Black Pine heap leach facility (HLF) for forecasting metal production which in turn were used in financial and NPV analysis by Liberty Gold including flow

information based on ramp-up of stacking operations and mining production. A stacking plan for the selected mine plan was developed, and recovery modeling of gold was completed.

The model Forte created can evaluate various stacking configurations, mine plans, application rates, barren flow rates, leach cycles, and lift heights.

AGP and Liberty provided Forte with the ROM mine plan for loading the model with planned ore tonnes, and recoverable ounces by ore type as specified by Gary Simmons Consulting. The recoverable gold was calculated within the mine plan by AGP, with metallurgical input from Gary Simmons Consulting.

As part of the feasibility study, Forte reviewed the recovery rates provided by Gary Simmons Consulting to develop the extraction rate inputs for the model. Recovery rates for each ore type stacked were determined by analyzing the solution to ore recovery profiles for each ore type at the predicted ROM P_{80}. A K-means cluster analysis was performed on the solution to ore models in order to bin ore types by rates. The lithologies binned by rate formed four ore profiles which can be seen in Table 16-9. These solution to ore rate curve bins were subsequently analyzed by Forte and curve fit to produce kinetic extraction curves versus time as seen in Equation 2. The curves were fit to match both short-term and long-term extraction to predict metal recovery based on ore placement rates and time under irrigation. The recovery through time is important to capture as ore placement changes through time and the ultimate recovery utilized in the block model is reached at different times according to planned operations. The parameters of these kinetic extraction curves were combined with recoverable gold, as provided in the mine plan from AGP. This was then input to the recovery model to generate the gold recovery profiles over the life of the HLF.

Table 16-9: Identified Ore Profiles

	Red	Orange	Green	Blue
Lithologic types	DZ&RF-Pola + RF-Ppos	DZ-Polb+BRPolb	DZ-Polc+DZPols+PmMx	DZ&CDI-Pold+CDI-Pols+MET2+MET3
P_{80}	125 mm	125 mm	200 mm	250 mm

Equation 2: Extraction Kinetic Formula

$$E(t) = Ult * (A_1(1-(1-R_1)^t) + A_2(1-(1-R_2)^t) + A_3(1-(1-R_3)^t))$$

Where in Equation 2 E(t) is the calculated percent gold extracted as a function of time, t is time in days, and A_1-A_3 and R_1-R_3 are constants that are used to fit extraction to the provided solution to ore extraction rates by ore type as provided by Gary Simmons Consulting. Table 16-10 contains the kinetic parameters broken down by ore type as shown in Table 16-9. Figure 16-2 shows the corresponding curves in a graphical format that is used in the model to predict recovery based on time under irrigation based on stacking, primary irrigation, and residual leaching from placement of ore above or secondary leaching at the end of mine life. Modeled recovery achieves 100% of recoverable placed ounces value as defined by the mine plan provided by AGP. Discounts to recoverable ounces were applied during the modeling of recovery, the details of which can be seen in Section 13 of this report.

Table 16-10: Recovery Rate Parameters

	Red	Orange	Green	Blue
A1	65.0%	68.0%	55.0%	60.0%
A2	23.0%	25.0%	25.0%	25.0%
A3	12.0%	7.0%	19.0%	15.0%
R1	0.50000	0.50000	0.50000	0.50000
R2	0.01295	0.01307	0.01197	0.01284
R3	0.05000	0.05000	0.05000	0.05000

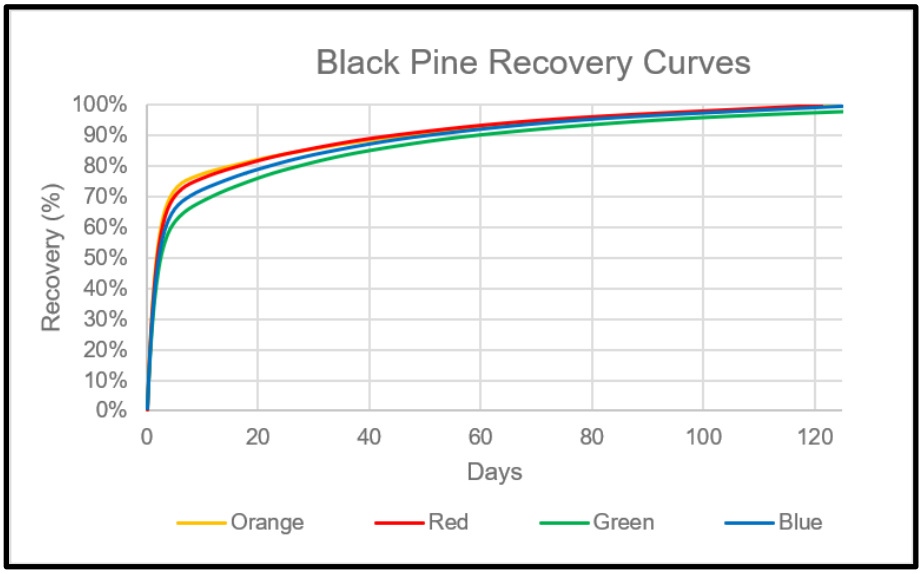


Figure 16-2: Extraction Curves

Forte was provided with HLF specific ore properties utilized in the FS. This data was gathered and compiled by Forte from the Project Team. Ore physical properties were provided by Newfields and operational parameters were provided by M3. Listed in Table 16-11 are the key input parameters used for the HLF recovery model.

Table 16-11: Ore Properties

	ROM
Initial Gravimetric Moisture Content*	4.5%
Residual Moisture Content*	9.5%
Tonnage Factor	17.3 ft^3/ton
Ore Density	115.6 lbs/ft^3
Specific Gravity	2.47
Saturated Hydraulic Conductivity	0.388 cm/sec
Leaching Application Rate	0.004 gpm/ft^2
Target Initial Leach Cycle	60 day plus Breakthrough
Average Available Leach Cycle	105 days
Lift Height	50 ft

*Gravimetric Moisture Content

Forte generated stacking plans, utilizing Hexagon MinePlan, for the mine plan provided, working within the defined boundaries for the HLF.

While generating the stacking plan, the following constraints and parameters were assessed including input from the Liberty Gold and NewFields project teams:

- Utilizing the designed HLF footprint and staying under the 300-ft height limitation above liner limit for Phases 1 and 2, and under a conceptual 500-ft height above liner limit for Phase 3 with additional stacking over Phase 1 and Phase 2 (which would require additional permitting) that would align Phase 3 construction with the Phase 1 and Phase 2 footprints being designed to accommodate stacking to 500-ft.
- Phase limits and topography grading provided by NewFields
- 20-ft initial lift height for liner protection, 50-ft lift height thereafter
- 36 degree angle of repose
- 3:1 reclamation angle
- Met 2, 3 and LHLF can only be stacked above Met 1 (i.e. no Met 1 material can be stacked above Met 2, 3, and LHLF) to reduce the risk from potentially preg-robbing material.
- Operational access points and ramp variations
- All ramps for placement were designed at 100 ft width at 10% grade

The stacking plan presented in Figure 16-3 uses warm colors (red, pink, yellow) for Met 1, and cool colors (blue, green, gray) for Met 2, 3, and HLF. Only Met 1 material is assumed to be used on the initial 20-ft lift for all phases. Phases 1 and 2 have a maximum height over liner of 300 ft unless additional regulatory review and permitting approval is obtained to allow for stacking to 500 ft. This would be projected to occur after Phase 3 stacking begins. There are three permanent access points onto the HLF pad, however intermediate access points will be required to access the lower lifts.

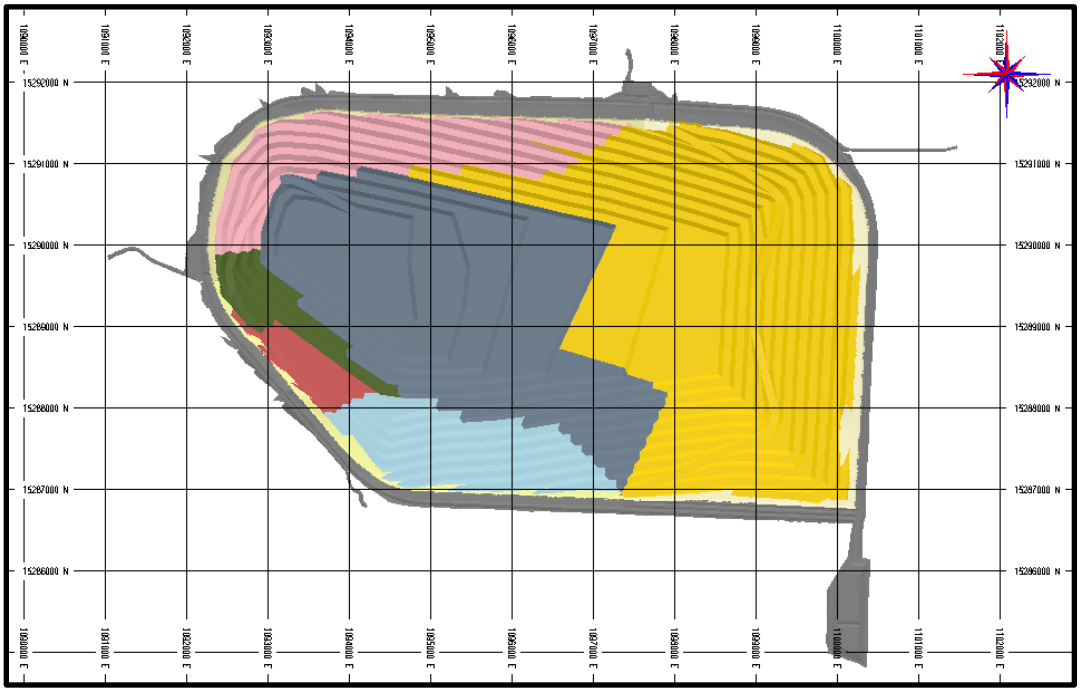


Figure 16-3: ROM Final Stacking Design

The recovery model utilizes first principles of hydrodynamics and kinetics to simulate recovery through time for the Black Pine HLF. The model utilizes discretized blocks to track tons and recoverable ounces placed, flow rate, leach cycle, application rate, extracted ounces, moisture content, solution tenor, and recovery from the HLF.

The model was developed to allow for flexibility in scenario analysis to change various input parameters to understand the overall impacts on recovery of gold. The specified flowrate reflects the targeted flowrate to the HLF and is limited based on the available leach area and only reaches the targeted barren solution flow when adequate area is available based on the associated application rate.

The model utilizes the Brooks-Corey methodology, Equation 3, for representing the flow through the pad based on leaching application rate and ore properties:

Equation 3: Brooks-Corey

$$q = k_{sat}\left[\frac{\Theta_t - \Theta_r}{\Theta_{sat} - \Theta_r}\right]^{\varepsilon}$$

This methodology captures micropore and macropore flow within heap leach facilities and can be calibrated to existing conditions as the Black Pine project moves into operations, as well as additional metallurgical and geotechnical testing. Equation 4 shows the degree of saturation, associated with macropore flow, that occurs within heap leach facilities that is accounted for within the model.

Equation 4: Specific Surface Area

$$SSA = \frac{Total\ Surface\ Area}{Mass\ of\ Sample}$$

The results of the recovery model for the ROM mine plan resulted in 2,829,106 recovered gold ounces by the end of stacking. With residual leaching operations post-stacking, accounting for approximately four (4) years of residual leaching operations, the final gold recovery was 2,840,895 oz. Figure 16-4 shows the recoverable placed gold ounces by year and the poured gold ounces by year. Figure 16-5 shows the recoverable Au inventory through time. Figure 16-6 shows the modeled rich and barren solution flows for the project. The modeled inventory is reflective of both Au complexed into solution as well as remaining extractable gold in the ore. There are periods where inventory grows due to changes in throughput as well as overall depth to liner. Inventory is also reduced when conceptual solution flow increases from 15,000 gpm to 18,000 gpm and at the end of mine life during residual leaching operations.

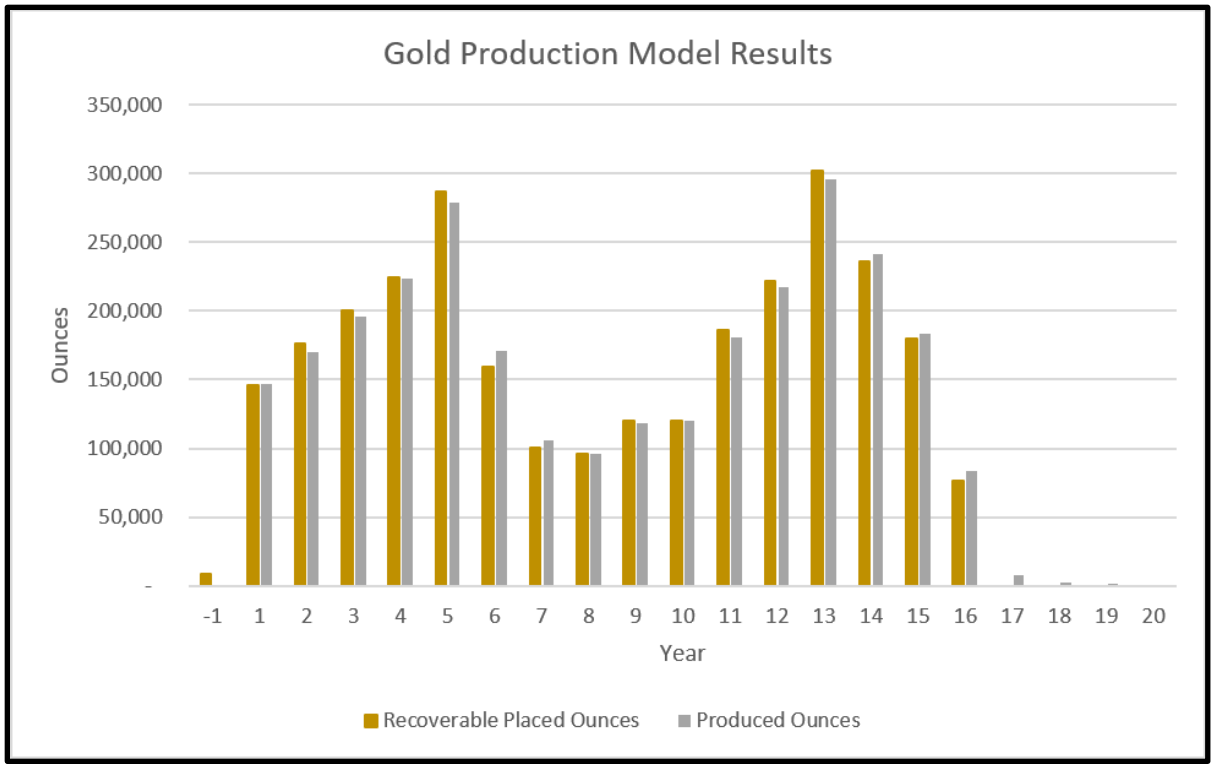


Figure 16-4: Recovered Gold Ounces by Year

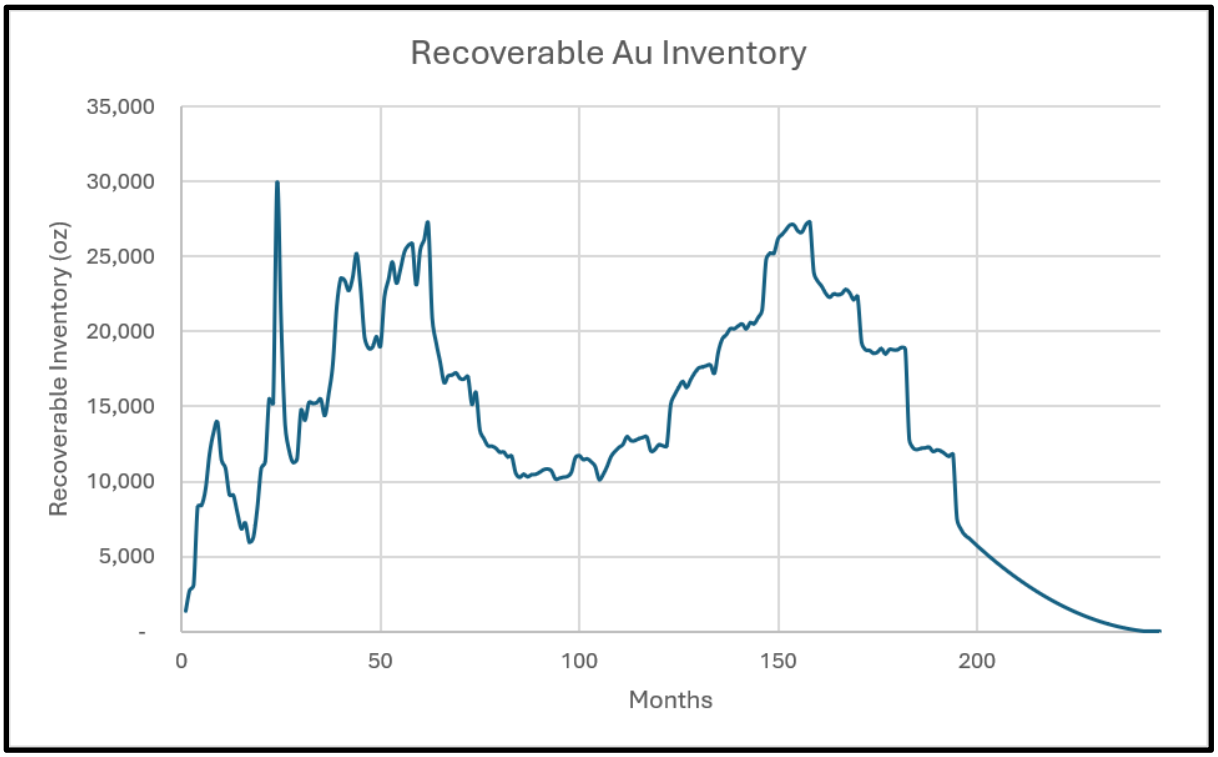


Figure 16-5: Recoverable Au Inventory through Time

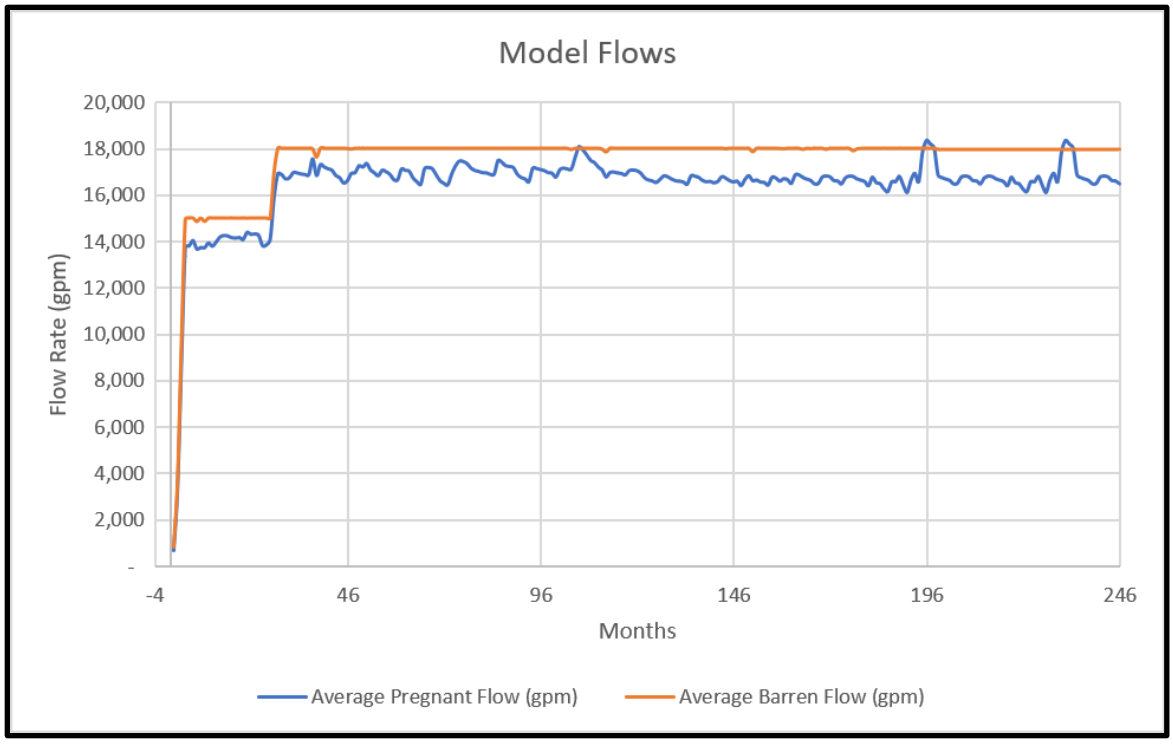


Figure 16-6: Modeled Rich and Barren Solution Flows

Table 16-12 shows the production through time of the recoverable ounces by year for the ROM mine plan. The row labeled “K Au Rec” shows the thousands of recoverable ounces of gold placed on the HLF, and the row labeled “K Au Prod” are the thousands of ounces of gold produced from the HLF. Forte has put together the resulting estimated gold production plan for ounces recovered onto carbon. Included in the production is the impact of solution grade on CIC efficiency with higher solution grades yielding higher CIC efficiencies and lower solution grades yielding lower CIC efficiencies.

Table 16-12: Ore and Metal Production Schedule

	Ore Tonnes	LHLF Tonnes	Total Tonnes	Grams Au/t Rec	K Au Rec	K Au Prod
Year -1	1,250		1,250	0.221	8.9	0.2
Year 1	27,144		27,144	0.168	146.2	147
Year 2	17,258		17,258	0.317	175.9	170
Year 3	27,098		27,098	0.23	200.2	196.2
Year 4	26,484		26,484	0.264	224.6	223.4
Year 5	28,311		28,311	0.315	286.6	278.5
Year 6	29,379		29,379	0.169	159.7	171
Year 7	16,036		16,036	0.195	100.3	105.8
Year 8	19,022	3,955	22,977	0.13	96.2	96.1
Year 9	19,725		19,725	0.189	120.1	118.4
Year 10	27,938		27,938	0.134	120	119.9
Year 11	37,000		37,000	0.156	186	181.1
Year 12	37,000		37,000	0.186	221.7	217.5
Year 13	37,000		37,000	0.254	301.9	296.1
Year 14	32,592		32,592	0.225	236.1	241
Year 15	32,506		32,506	0.172	179.7	183.2
Year 16	17,594	28,073	45,667	0.052	76.8	83.8
Year 17			0	0	0	7.4
Year 18			0	0	0	2.2
Year 19			0	0	0	1.5
Year 20			0	0	0	0.7
Total	433,338	32,028	465,366	-	2,841	2,841

16.6 ORE LOSS AND DILUTION CONSIDERATION

The geologic resource model received from Liberty Gold includes 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.

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. A dilution-bench height study was completed that combined model blocks based on 5 m, 10 m, 12.5 m, and 15 m bench heights. When a mineralized block was combined with Met 4 or Poc material, that entire block was considered waste due to the potential preg-robbing characteristics of those material types. Table 16-13 shows mining on 10 m benches has a minimal impact on gold ounces, reducing the overall recoverable ounces by 1.9% in the Discovery pit and a 0.3% reduction at Rangefront. A 10 m bench was chosen for the pit design with no mining dilution applied during mine planning based on the study results. Mining 5 m flitches as determined by material specifications will minimize ore loss and dilution in certain mining areas and will be utilized as needed.

Table 16-13: Bench Height Dilution Estimates

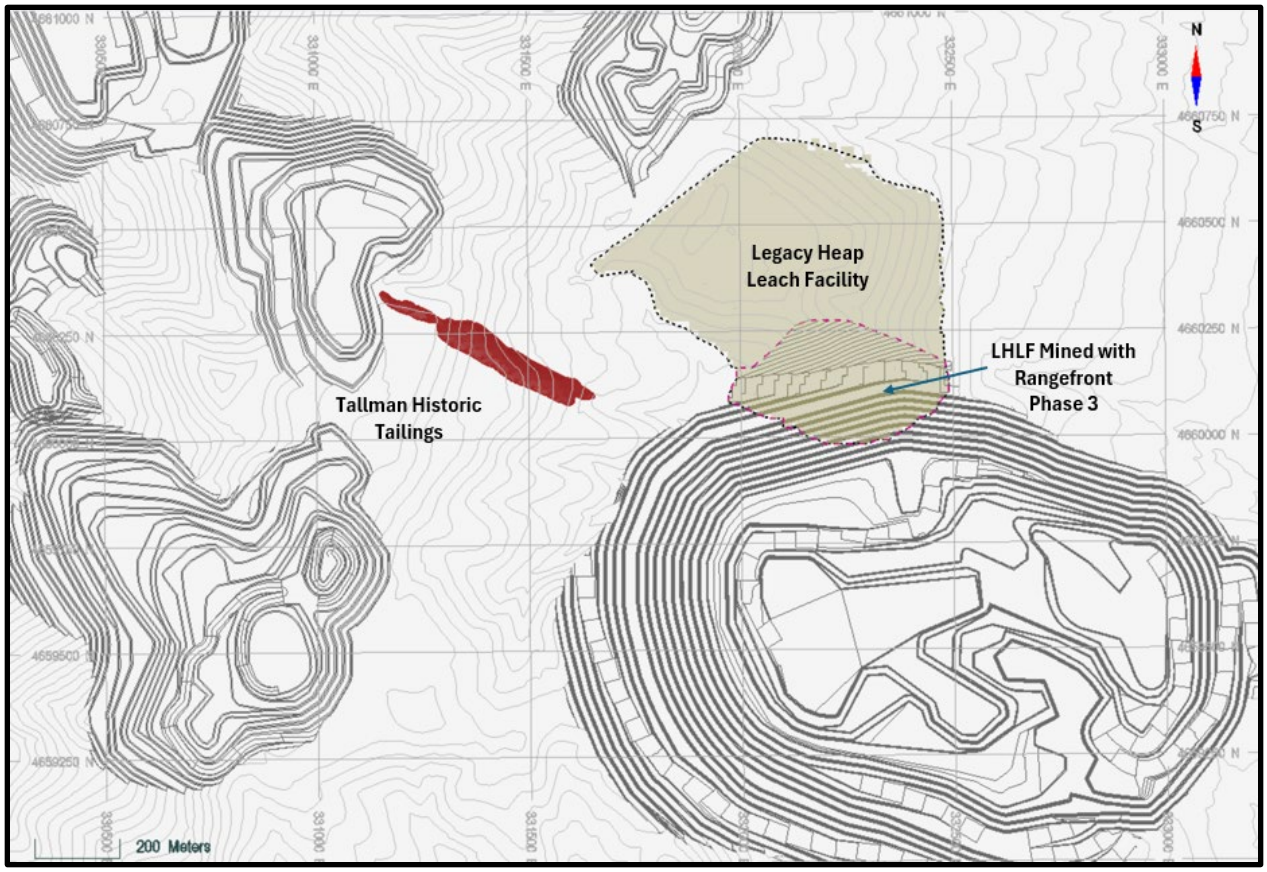
Bench Height	Tonnes and Grade					Dilution (%)	
	Leach Ore (Mt)	Au (g/t)	Au Recov. (g/t)	Au (kozs)	Au Recov. (kozs)	Au (ozs)	Au Recov. (ozs)
Discovery Pit							
5 m-BASE	163.3	0.34	0.23	1,784.6	1,189.5	0.0%	0.0%
10 m	168.6	0.32	0.22	1,754.0	1,167.3	-1.7%	-1.9%
12.5 m	167.9	0.31	0.21	1,663.4	1,110.8	-6.8%	-6.6%
15 m	168.8	0.30	0.20	1,650.0	1,101.6	-7.5%	-7.4%
Rangefront Pit							
5 m-BASE	138.8	0.25	0.19	1,111.6	835.7	0.0%	0.0%
10 m	144.6	0.24	0.18	1,112.4	833.1	0.1%	-0.3%
12.5 m	149.3	0.23	0.17	1,114.0	831.7	0.2%	-0.5%
15 m	150.5	0.23	0.17	1,114.9	831.8	0.3%	-0.5%

No ore loss is anticipated, and mining recovery is assumed to be 100% for all ore material.

16.6.1 Pit Restrictions

Pit restrictions include an elevation limit in the Rangefront and CD pits. The CD limit is set at 1760 masl and Rangefront is limited to 1470 masl. The elevation limit provides at least a 15 m (50 ft) offset above the water table in both pits.

The pit restriction of 50 m from Legacy Heap Leach Facility (HLF) was not used after an economic evaluation of the Rangefront pit determined that the additional revenue generated by expanding the Rangefront pit into the LHLF was greater than the cost of relocating the entire legacy heap leach material onto a lined facility. The expansion of the Rangefront pit into the LHLF is shown in Figure 16-7. This expansion is planned as part of Rangefront Phase 3 and is pending a proposed alternative action as part of the current MPO. The reclaimed Tallman tailings are not disturbed by projected mining and remain in place.

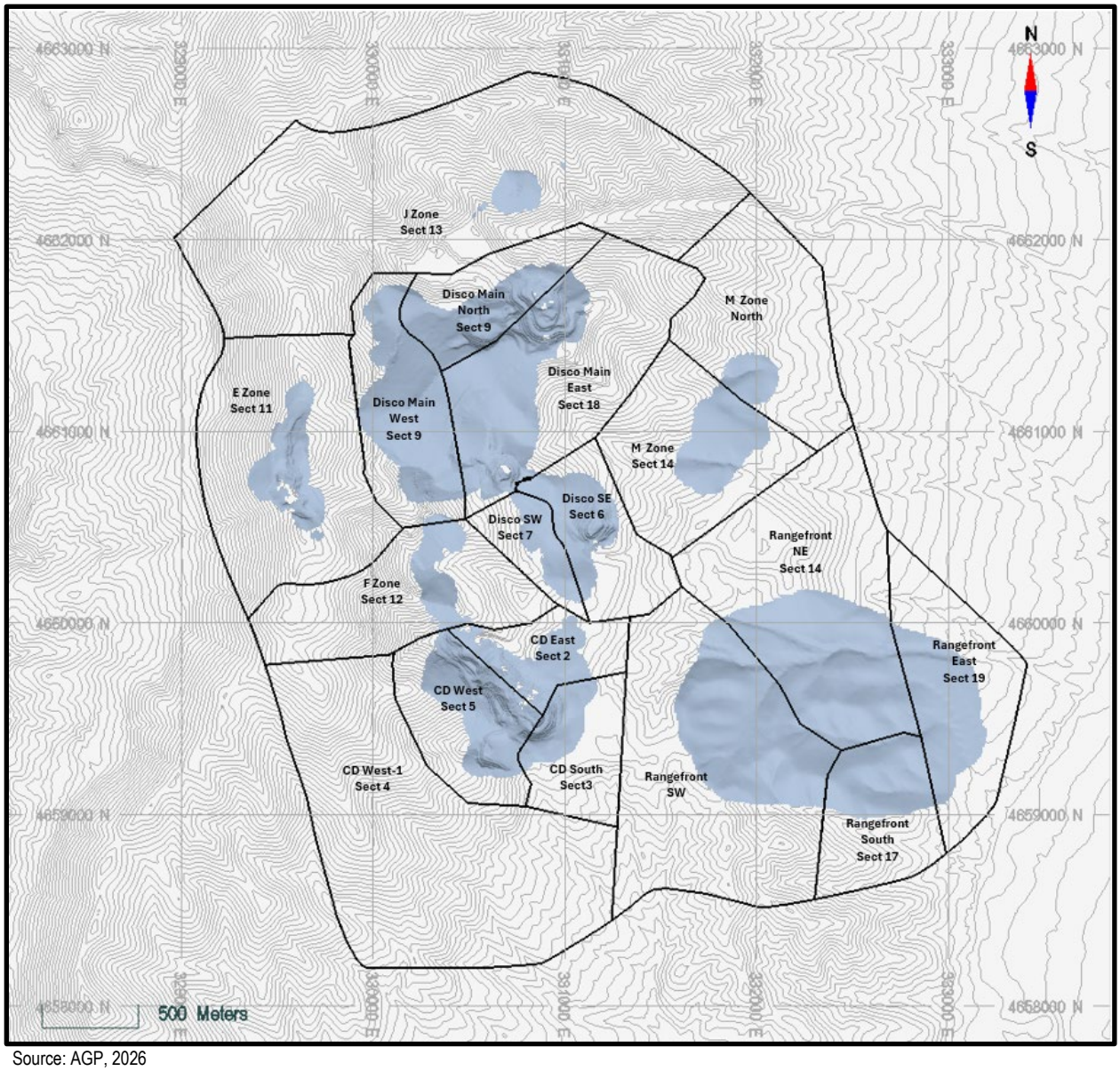


Source: AGP, 2026

Figure 16-7: Legacy Heap Leach Mining

16.6.2 Slopes Angle Recommendations

Wall slope angles for pit optimization were based on guidance from Knight Piésold Consulting with the final slope parameters report issued July 17, 2026. A design sector map was created which was defined by the pre-feasibility study pit designs and dominant geotechnical units, these sectors were not adjusted as part of the Feasibility Study Geotechnical Study. Slope sector areas were used to code the model SLPCD item using the area polygons in Figure 16-8.



Source: AGP, 2026

Figure 16-8: 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 in vertical height not intersected by a haul ramp. A 20 m geotechnical berm was added to the Discovery Main west and 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 pit slope development at Black Pine. Calculated OSA angles by sector are given in Table 16-14.

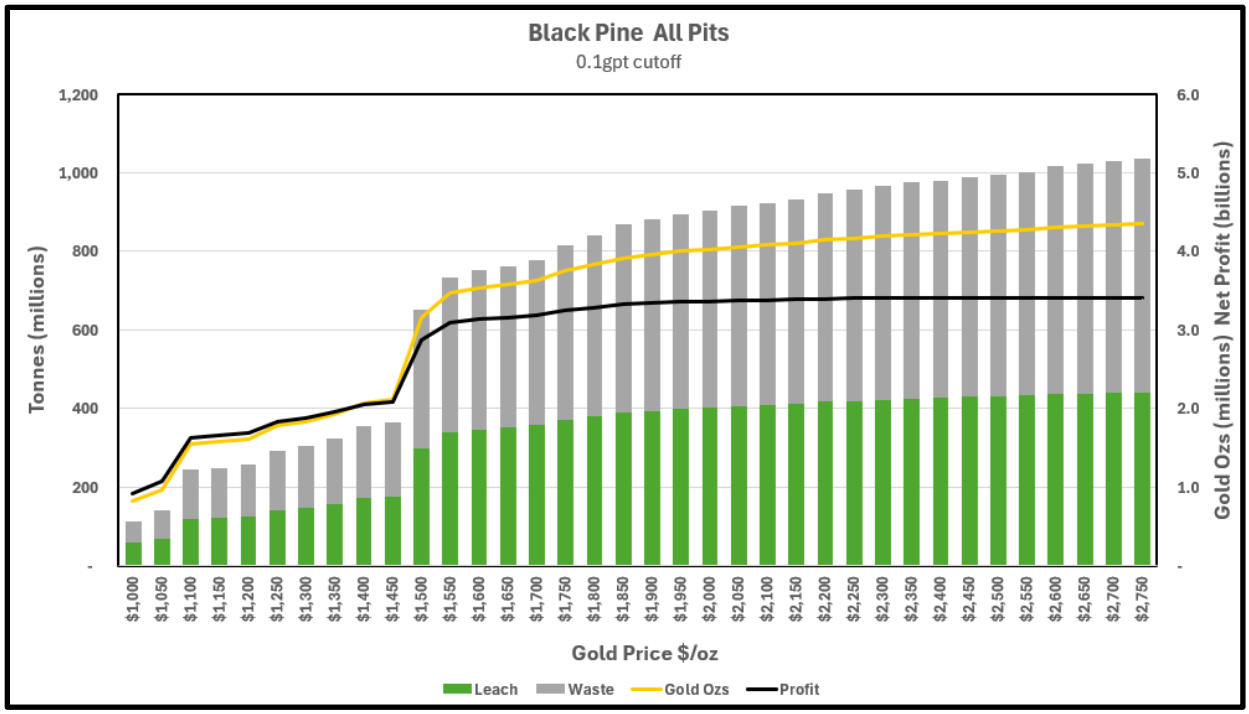
Table 16-14: Overall Slope Angles and Slope Parameters by Sector

Sector	Osa (°)	Ira (°)	Face (°)	Berm (m)	Sector Code	Bench Height (m)	Stacked Height (m)
CD East	54.0	54.3	75.0	9.0	2	10	20
CD South	35.0	47.0	70.0	11.4	3	10	20
CD West	54.0	54.3	75.0	9.0	4	10	20
CD West 1	44.0	50.9	70.0	9.0	5	10	20
Disco Southeast	48.0	54.3	75.0	9.0	6	10	20
Disco Southwest	45.0	50.9	70.0	9.0	7	10	20
Disco Main East	48.0	54.3	75.0	9.0	8	10	20
Disco Main North	45.0	50.9	70.0	9.0	9	10	20
Disco Main West	44.0	49.2	70.0	10.0	10	10	20
E Zone	47.0	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	46.0	54.3	75.0	9.0	14	10	20
M Zone North	50.0	50.9	70.0	9.0	15	10	20
Rangefront Northeast	42.0	47.0	70.0	11.4	16	10	20
Rangefront South	39.0	47.0	70.0	11.4	17	10	20
Rangefront Southwest	37.0	44.0	70.0	13.4	18	10	20
Rangefront East	37.0	47.0	70.0	11.4	19	10	20
Default	42.0	54.3	75.0	9.0	20	10	20

16.6.3 Price Shells

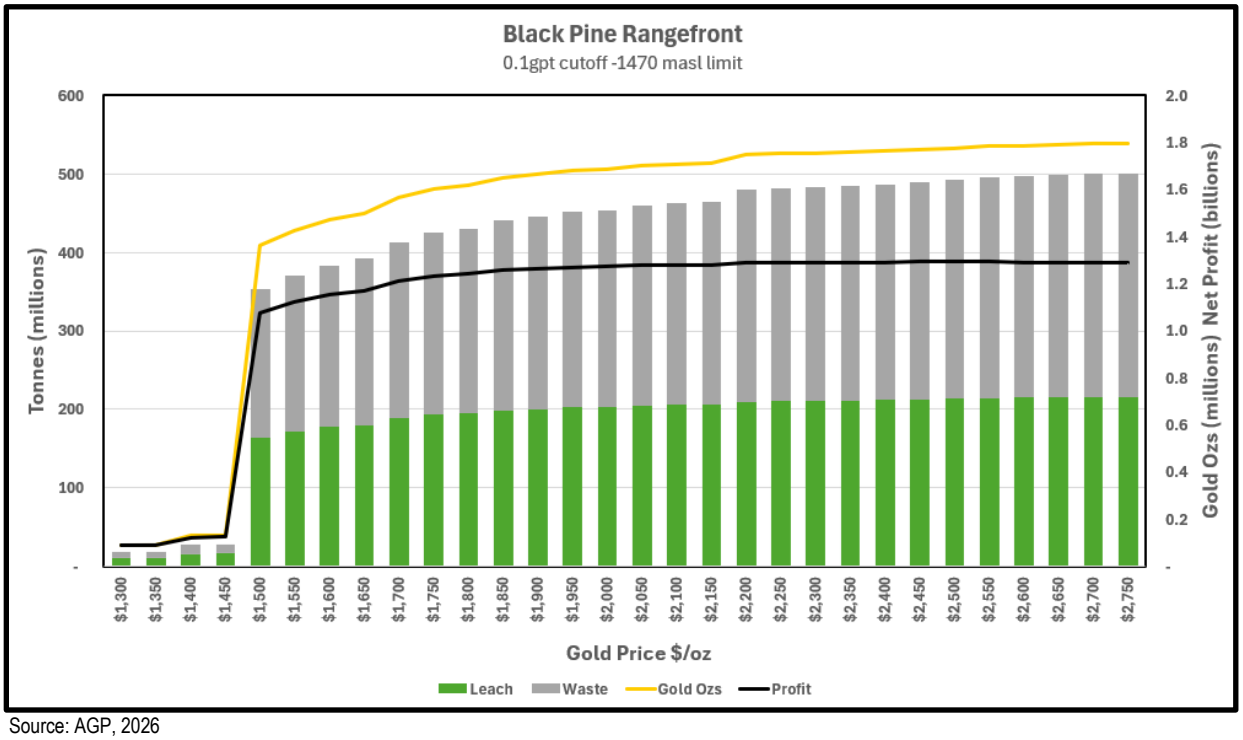
Pit optimization was run using incremental gold prices at $50 per ounce increments to generate a set of nested pit shells to US$3,000/oz gold. The incremental price shells run at 0.1 g/t cutoff, guide the selection for final pit extents along with pushbacks leading to the final pit design. The $2,250/oz pit shell was chosen as the ultimate limit for pit design. A $2,500/oz gold price was the selected target gold price used in the analysis and the $2,250/oz pit shell contains 98% of the gold ounces and 100% revenue that are contained in the $2,500/oz shell. 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 rock tonnages, contained gold, and potential net profit at $50/oz gold price increments for the site wide optimization shells are shown in Figure 16-9. Figure 16-10 and Figure 16-11 show the results of nested pit analyses for the Discovery and Rangefront pits. Figure 16-12 is a plan view of the $2,250/oz optimization shells.



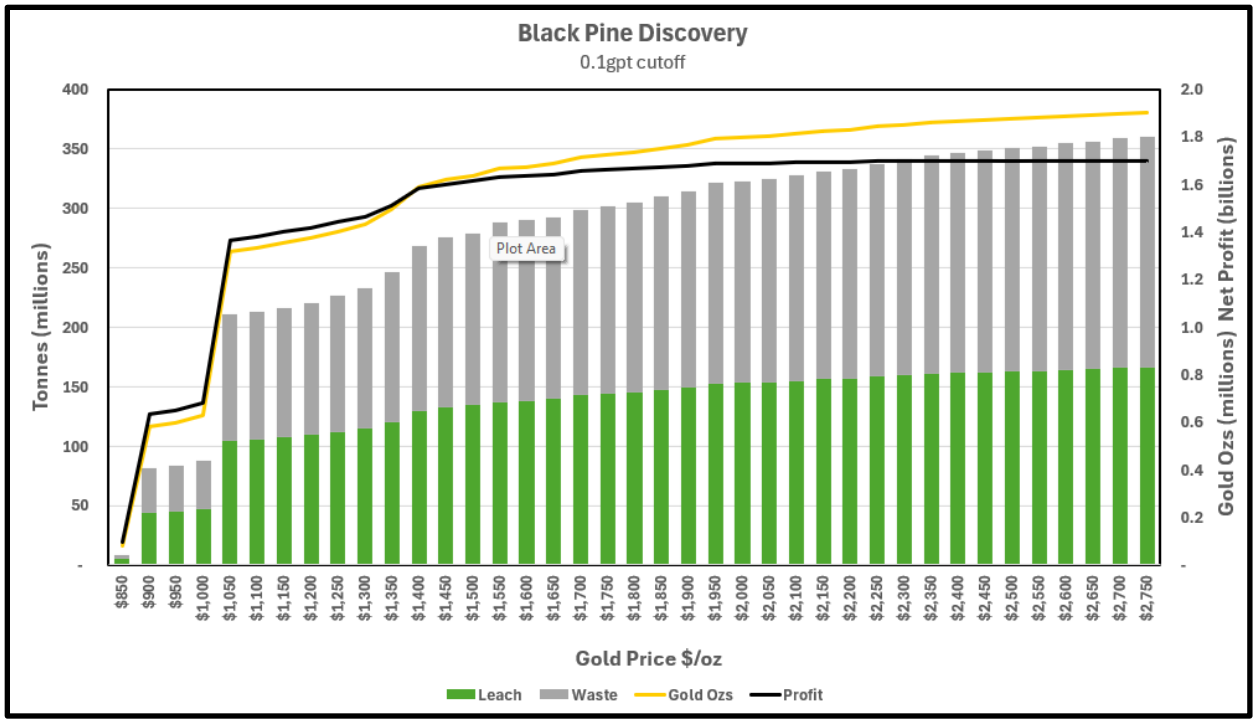
Source: AGP, 2026

Figure 16-9: Gold Price Optimization of All Areas



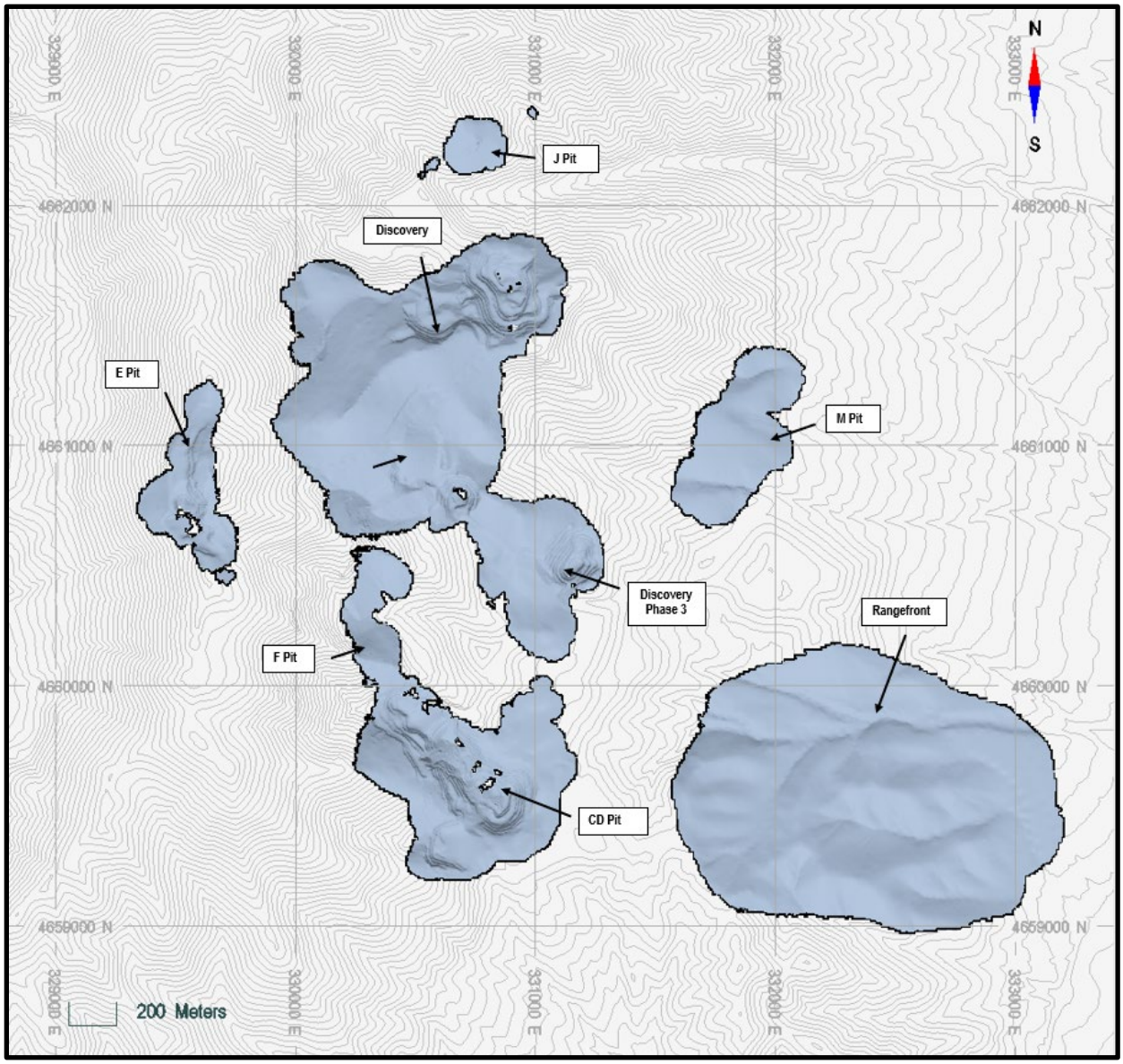
Source: AGP, 2026

Figure 16-10: Gold Price Optimization - Rangefront



Source: AGP, 2026

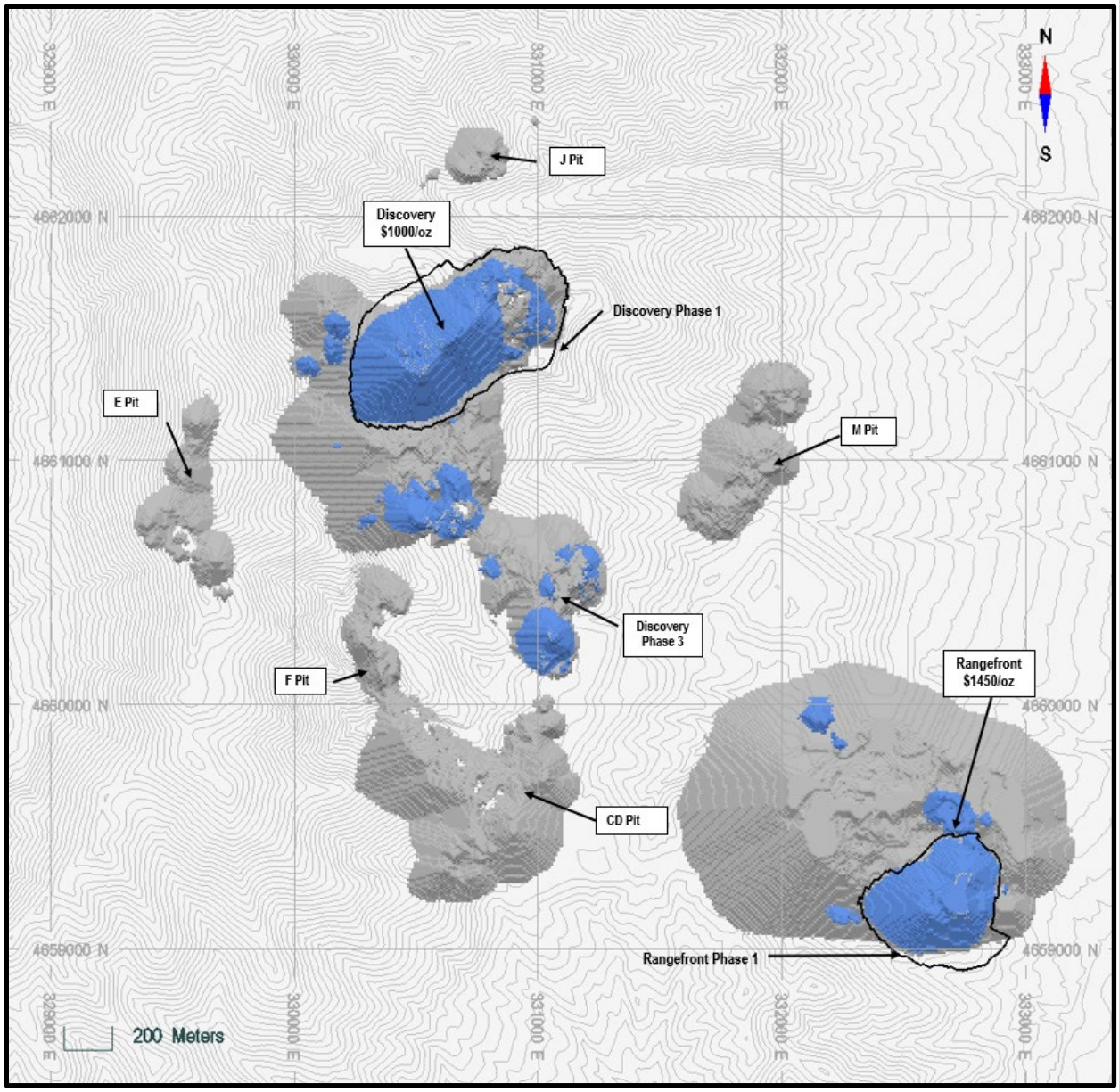
Figure 16-11: Gold Price Optimization - Discovery



Source: AGP, 2026

Figure 16-12: View of the Pit Optimization Solids

The optimization shells that constrain the Discovery and Rangefront areas were run at $50/oz increments and a 0.05 g/t Au recoverable grade cutoff. These constrained runs further identified areas of higher grade and higher value for targeting early mining and phase development. The $1,000/oz shell at Discovery was used for Phase 1 design. The $1,450/oz pit shell was used for the starting phase at Rangefront. The Phase 1 pit design outlines and corresponding pit shells at Discovery and Rangefront are shown in Figure 16-13.



Source: AGP, 2026

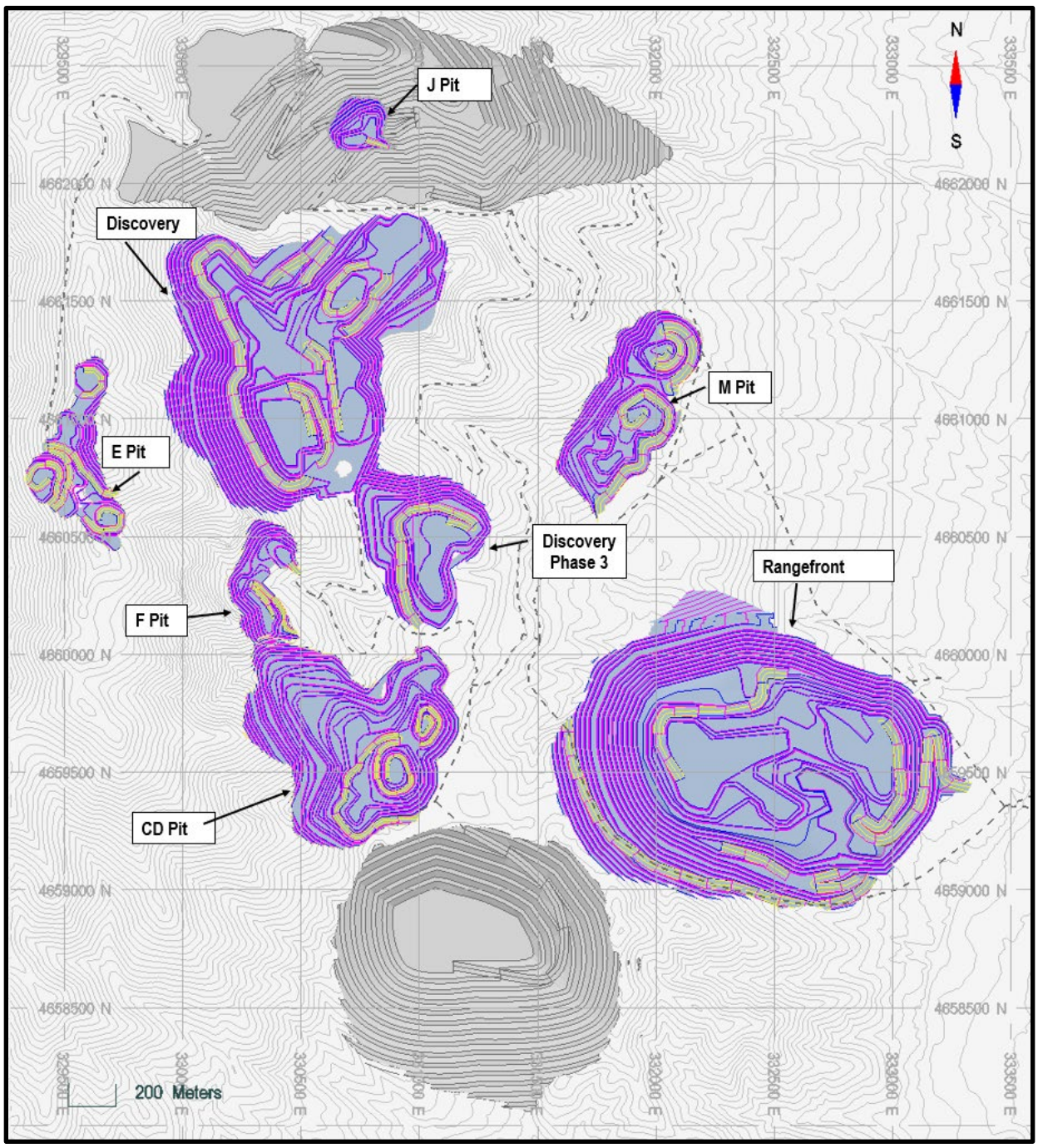
Figure 16-13: Pit Optimization Shells for Early Pushbacks

16.7 PIT DESIGNS

A conventional open-pit, truck-and-shovel mining method will be used for the Black Pine Project. Haul truck sizing includes 240 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 corresponding decreased ramp width will 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 seven areas: Discovery, Rangefront, M, J, F, E, and CD. Discovery and Rangefront will be developed in four phases each and the remaining pits will be developed as single phases.

The ultimate pit designs at Black Pine are shown in Figure 16-14. The pit and phase designs consist of double benching with an operating bench height of 10 m in all areas. The dashed lines in the figure show location of the main ex-pit haulage routes.



Source: AGP, 2026

Figure 16-14: Ultimate Pit Designs

Tonnes and grade for the designed pit phases are reported in Table 16-15.

Table 16-15: Pit Phase Tonnages and Grades

PIT	Phase	Leach Ore (Mt)	Au (g/t)	AU Recovery (%)	Waste Rock (Mt)	Total (Mt)	Contained Gold koz
Discovery	DSC_Ph01	50.4	0.37	65.7	52.0	102.5	601.5
	DSC_Ph02	30.1	0.33	66.3	41.5	71.6	322.3
	DSC_Ph03	13.4	0.33	64.7	23.7	37.1	143.1
	DSC_Ph04	60.2	0.35	69.9	75.5	135.7	683.8
	Total	**154.1**	**0.35**	**67.4**	**192.7**	**346.9**	**1750.8**
Rangefront	RF_Ph01	19.2	0.23	73.9	14.4	33.6	141.8
	RF_Ph02	105.4	0.23	74.5	122.9	228.3	773.7
	RF_Ph03	66.5	0.28	75.8	102.6	169.1	604.4
	RF_Ph04	41.1	0.22	73.7	36.6	77.7	295.5
	Total	**232.1**	**0.24**	**74.8**	**276.5**	**508.6**	**1815.5**
Satellite	CD	28.9	0.28	58.4	39.7	68.7	259.6
	EZ	4.6	0.30	68.9	6.7	11.3	44.0
	F	5.2	0.33	63.1	3.2	8.5	56.0
	JZ	2.2	0.36	74.6	2.8	5.0	25.6
	M	6.2	0.48	71.2	28.7	34.9	96.5
	Total	**47.1**	**0.32**	**63.3**	**81.2**	**128.3**	**481.6**
	HHL				32.0	32.0	
Grand Total		**433.4**	**0.29**	**70.2**	**582.4**	**983.8**	**4047.8**

The larger pits (Discovery, Rangefront, CD, M, and Tallman) will primarily use 240 tonne trucks, while the smaller pits including E, F, and J that have smaller operating benches and are accessed by steep mountain terrain will utilize 64 tonne trucks. CD and M pits use the smaller fleet in the pit bottom and when the operating area is small. The road width is set to 37.5 m for the 240 tonne trucks while the 64 tonne trucks have a 22.8 m width. The haul road width allows for dual lane haulage and is three times the operating width of the haul truck plus a drainage ditch and a safety berm. Ramp gradients are 10% for the in-pit roads, stockpiles and waste dumps. The haul road design criteria is in Table 16-16.

Table 16-16: Haul Road Design Criteria

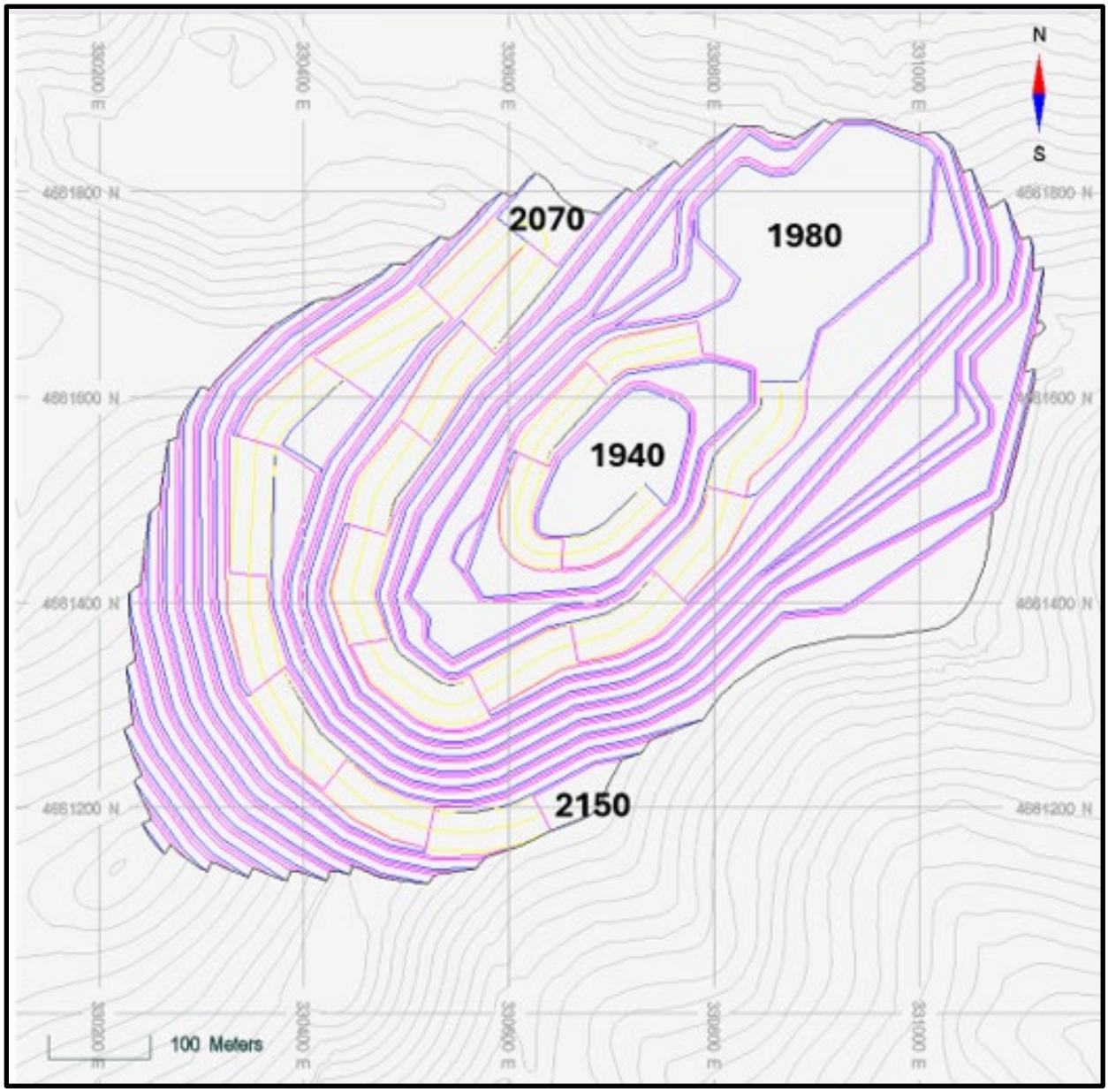
Ramp Attribute	240 t trucks	64 t trucks
Double Ramp width (m)	37.5	22.8
Single Ramp width (m)	28.5	17.1
Ramp Gradient (%)	10%	10%
Switchback design	Flat	Flat

16.7.1 Discovery

The Discovery pit is developed in four phases including Discovery Phase 3 which is developed as a standalone phase. Discovery is the main ore feed source for the first five years of production at Black Pine.

Phase 1 is located on the northeast side of the Discovery deposit and will start mining during the pre-production period and continuing through Year 4. Bench elevations range from 2,300 m to 1,940 m. Waste rock will be routed to the

Discovery Rock Storage Facility and the primary exits are located at the 2,150 and 2,070 elevations. Ore and waste rock can also be hauled directly from the 1980 bench through a small ramp to the surface haul network to reduce haulage requirements. Ore mined during pre-production will be sent to the stockpile for processing Year 1 with the Met 3 material stockpiled until the end of the mine life. See Figure 16-15 for the Discovery Phase 1 design.

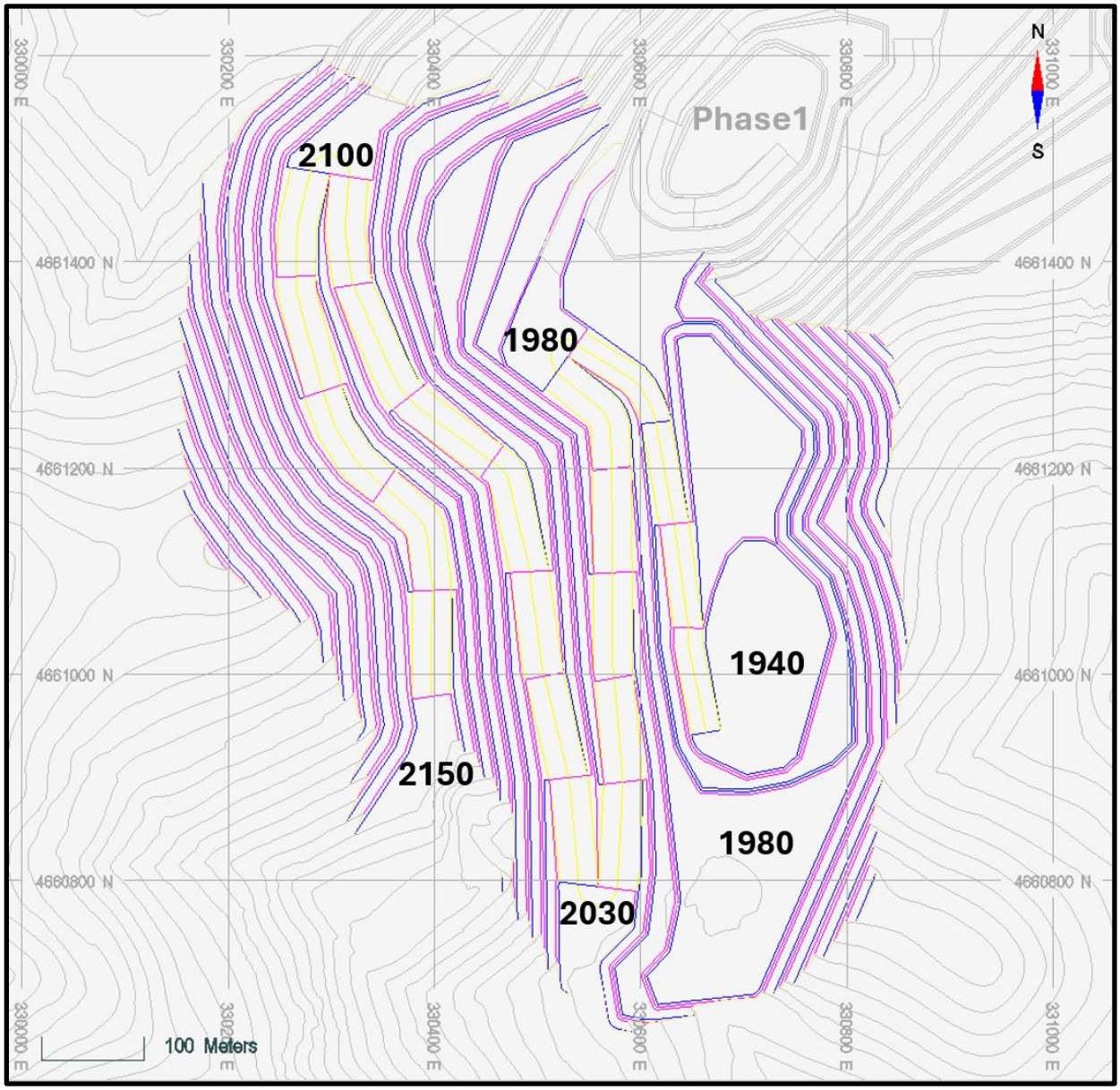


Source: AGP, 2026

Figure 16-15: Discovery Phase 1

Discovery Phase 2 is mined from pre-production to Year 5 and is located southwest of Phase 1. Bench elevations range from 2,300 m to 1,940 m. The main waste rock exit is adjacent to the Discovery Rock Storage Facility at the 2,100 elevation. Waste rock in the upper benches will be directly hauled to the rock storage facility. Once established, ore will be routed using the south pit exits located near the Discovery Phase 3 Pit at the 2150 and 2030 elevations.

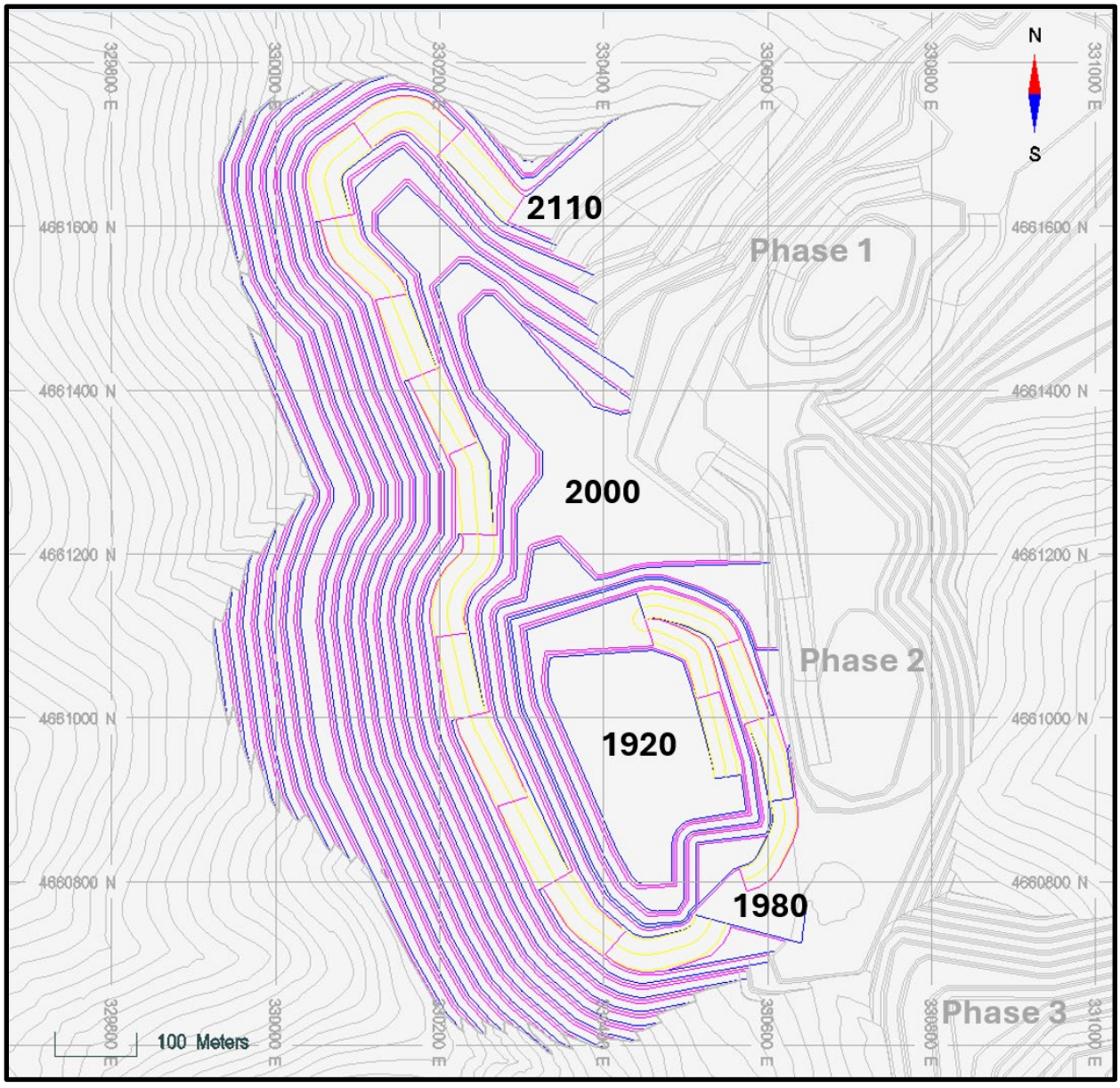
The existing ramp system established in Phase 1 will be used when it is more practical. The Discovery Phase 2 design is shown in Figure 16-16.



Source: AGP, 2026

Figure 16-16: Discovery Phase 2

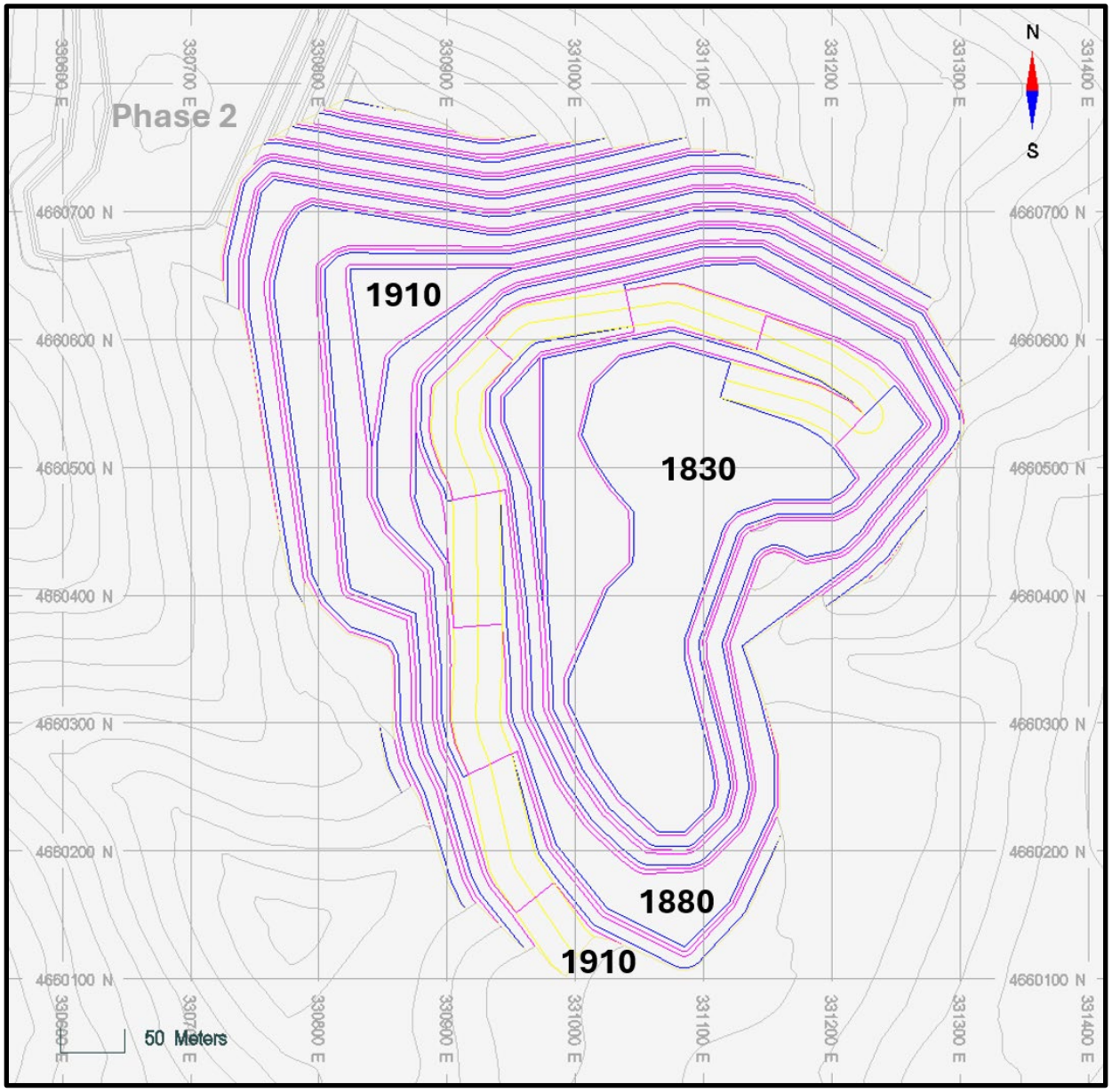
The final phase in the Discovery Main pit is Phase 4. Phase 4 bench elevations range from 2,320 m to1,920 m. A 20 m geotechnical berm is located on the west wall at the 2,200 elevation as shown in Figure 16-17. Phase 4 is mined from Year 2 through Year 7 and is located directly west of Discovery Phase 2. The pit exit to the north is at the 2,110 elevation while the south ramps exit at the 1,980 elevation.



Source: AGP, 2026

Figure 16-17: Discovery Phase 4

Discovery Phase 3 is a standalone phase south of the other Discovery phases (Discovery 2 and 4). The Discovery Phase 3 bench elevations range from 2,060 m to 1,830 m. Phase 3 is initially established in the pre-production period and subsequently mined in Year 2 to Year 4. Access to Discovery Phase 3 is to the south at the 1,910 elevation. The Discovery Phase 3 design is shown in Figure 16-18. The Discovery Phase 3 pit is backfilled starting in Year 5 from the adjacent Discovery Phase 4 and M Pit and reaches a height of 2,020 m.

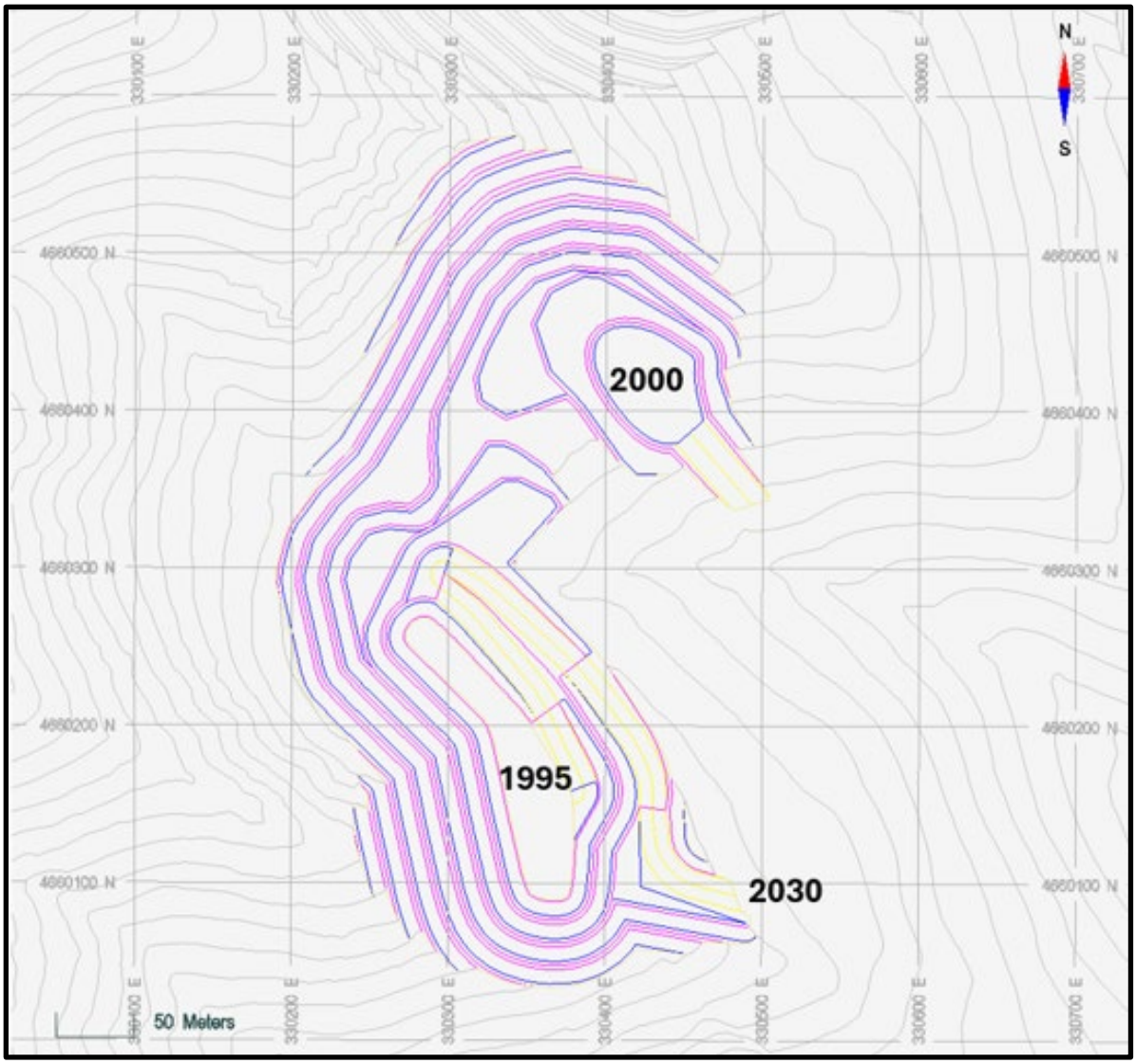


Source: AGP, 2026

Figure 16-18: Discovery Phase 3 (Tallman)

16.7.2 F Pit

F pit is located directly south of the Discovery pit, in an area that was not previously mined. F pit bench elevations are from 2,150 m to 1,995 m. F pit is mined during Years 9 and 10 with the pit exits at the 2,030 and 2,010 elevations. The F pit design is shown in Figure 16-19 and is designed for mining entirely with smaller fleet equipment.

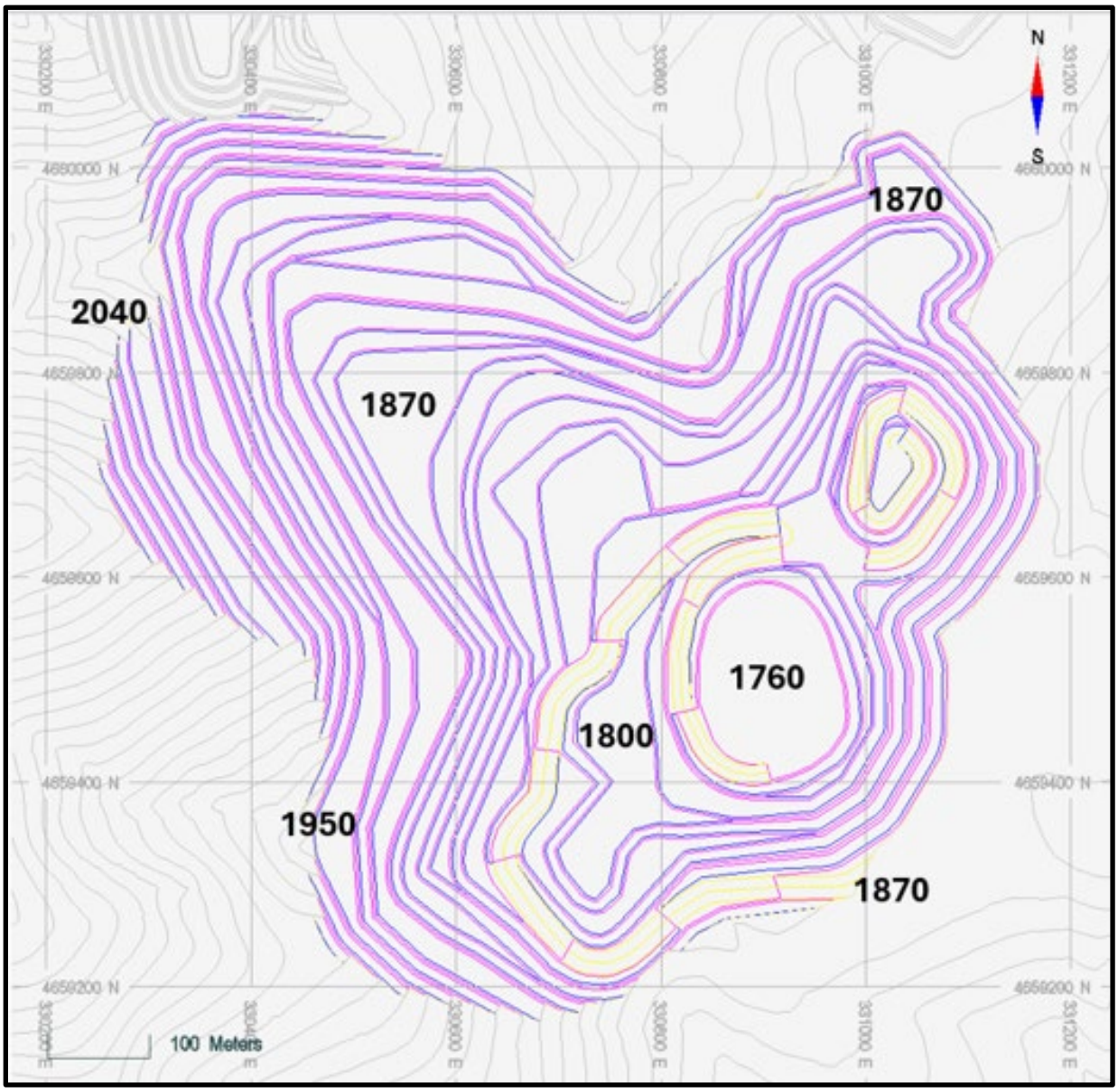


Source: AGP, 2026

Figure 16-19: F Pit

16.7.3 CD Pit

The CD pit is located directly south of F pit and is mined with a single phase starting in Year 5 and is completed in Year 8. The CD pit bench elevations range from 2,060 m to 1,760 m. CD mines west of and up the steep terrain located west of the previously mined CD pit. A geotechnical catch bench with a 20 m width is in the west wall of the CD pit at the 2040 elevation. The upper benches in the CD pit will require dozing and mining with the smaller equipment to establish narrow initial ore bearing benches in the steep west wall before transitioning to mining by the larger equipment. The CD pit design is shown in Figure 16-20. The CD pit is backfilled to the 2,070 m elevation and starts receiving backfill at the end of Year 8 through Year 15 mostly from the Rangefront Pit Phases.

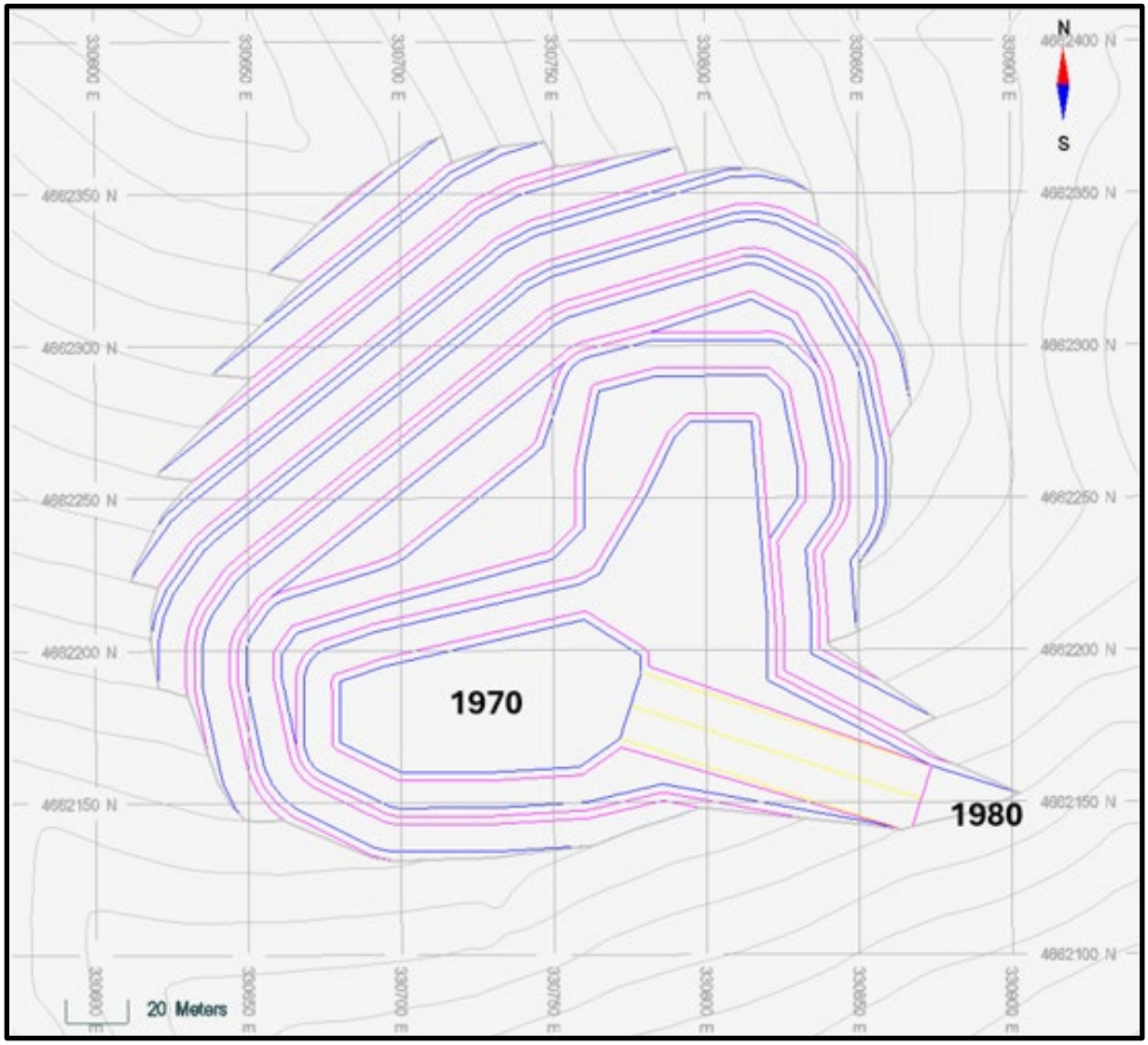


Source: AGP, 2026

Figure 16-20: CD Pit

16.7.4 J Pit

J pit is a small pit located north of the Discovery pit. J pit will be mined in pre-production and Year 1 after which it is backfilled and covered by the Discovery Rock Storage Facility. J pit bench elevations range from 2,130 m to 1,970 m. The phase design for J pit is shown in Figure 16-21 and is designed to be mined entirely by the small fleet equipment.

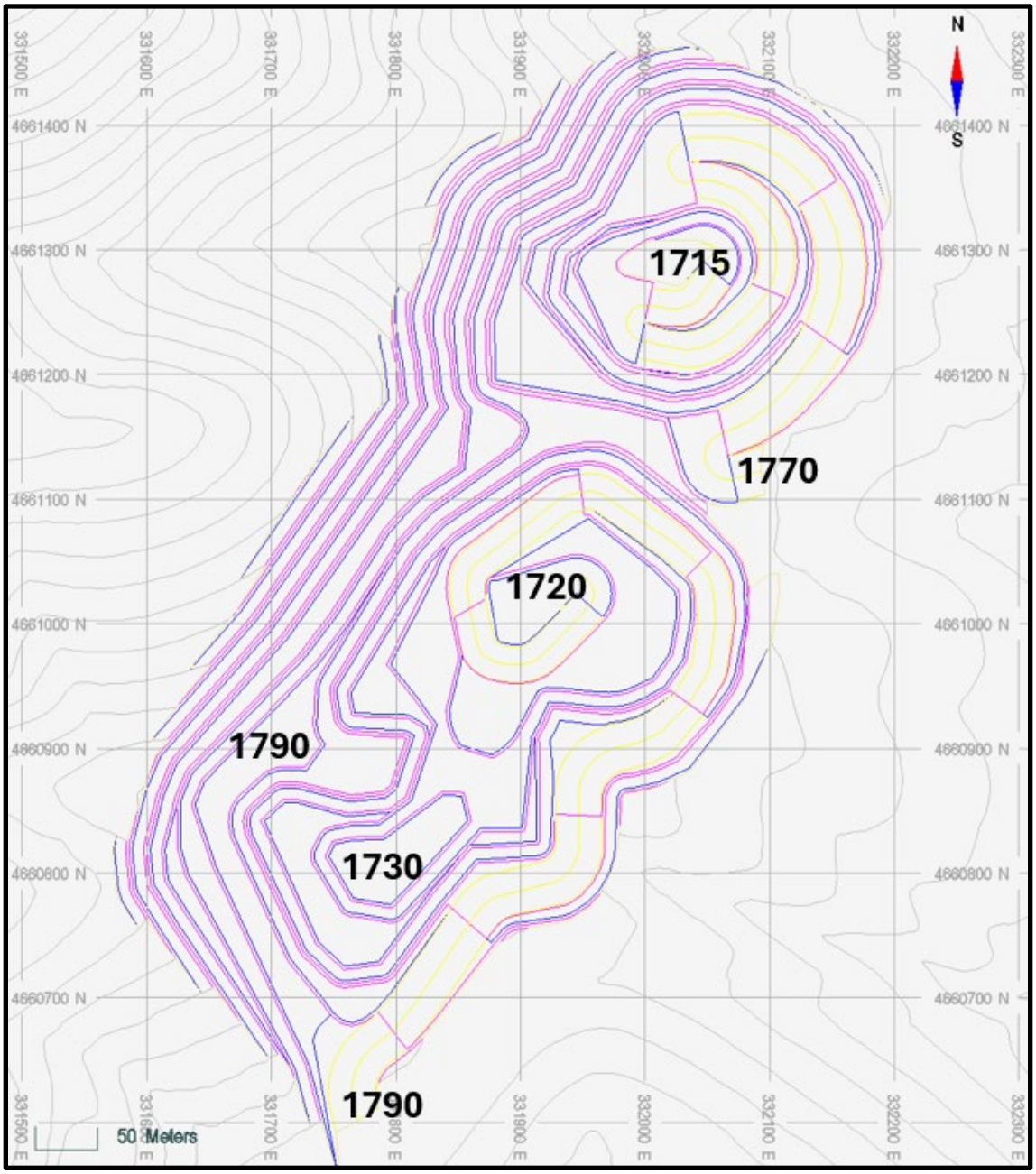


Source: AGP, 2026

Figure 16-21: J Pit

16.7.5 M Pit

The M pit is located east of the Discovery pit and is near the stockpiles and haul road leading from Discovery pit to the heap leach pad. M pit is mined in a single phase in Years 7 through 9. M pit bench elevations range from 1,900 m to 1,715 m. The M pit design is shown in Figure 16-22. M pit will be mined with the larger mining equipment and has a 37.5 m ramp width to accommodate the 240 t truck fleet. M pit is backfilled after mining is completed in Year 9 through Year 15 with waste mostly from Rangefront. The final height of the M pit backfill is 1,970 m.

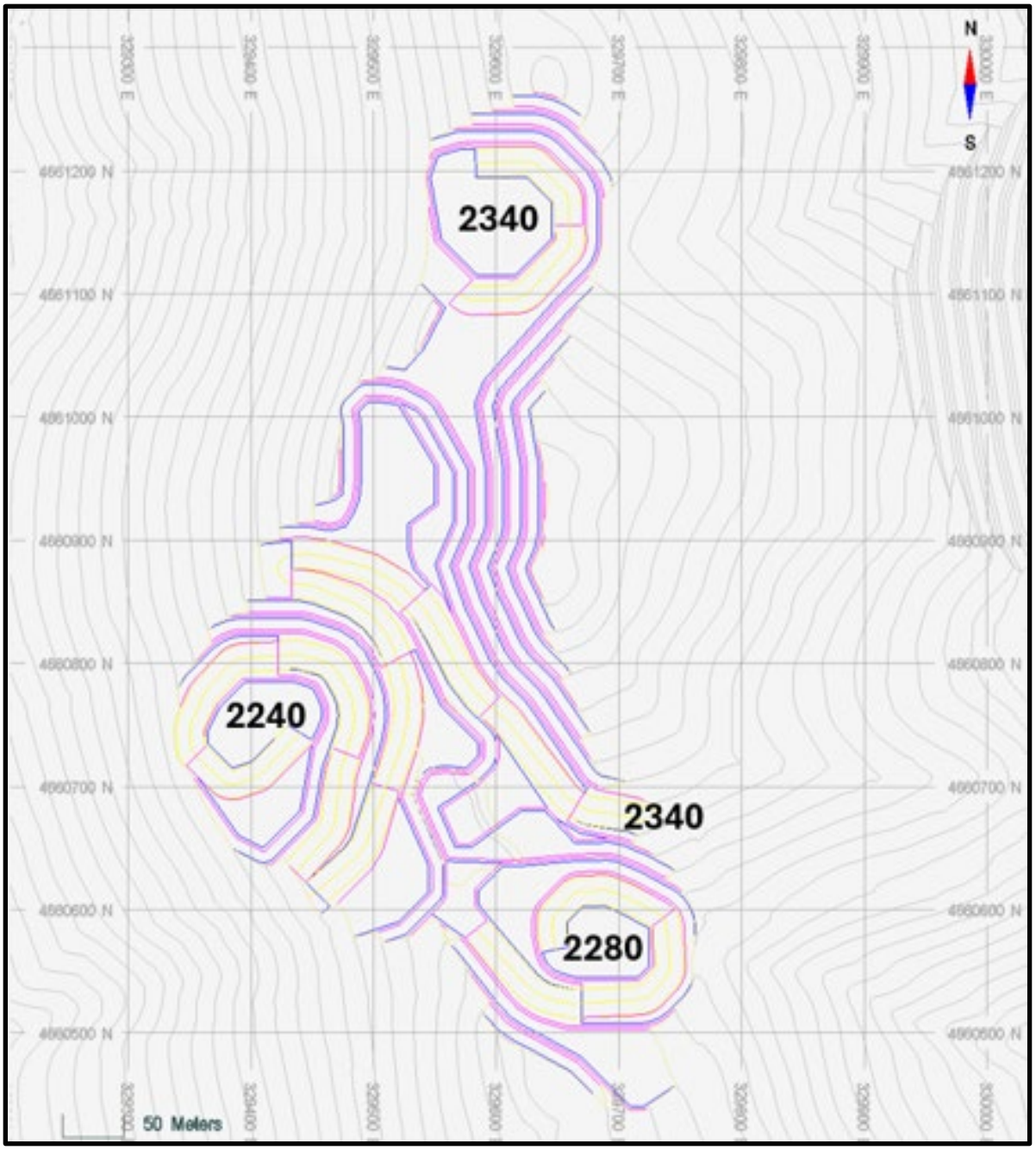


Source: AGP, 2026

Figure 16-22: M Pit

16.7.6 E Pit

E Pit is on the ridgeline west of the Discovery pit, at an elevation of 2,440 m to 2,240 m. There are three separate pit bottoms as shown in Figure 16-23. Mining occurs in a single phase using entirely the small fleet during Years 12 and 13. Access to E pit is via a surface road that runs from the top of Discovery Rock Storage Facility along the top of the ridgeline.



Source: AGP, 2026

Figure 16-23: E Pit

16.7.7 Rangefront

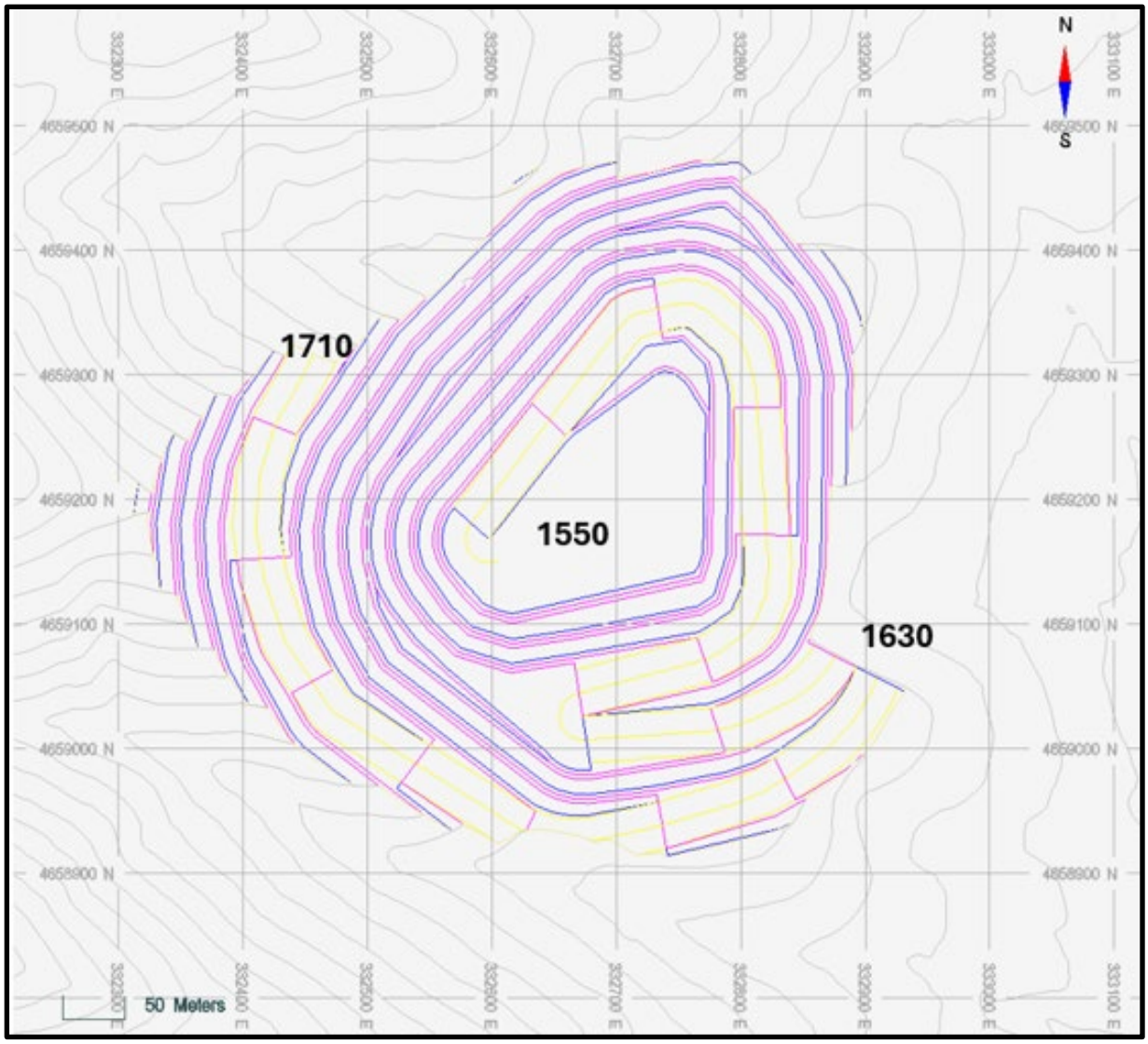
Rangefront is a large pit developed in four phases and is located south of both the legacy and proposed leach pad areas. All phases of the Rangefront pit utilize the large equipment for all material apart from some pioneering work to establish initial phase benches.

Phase 1 is a higher value phase that is mined in Years 5 through 8 with bench elevations that range from 1,770 m to 1,550 m. The Rangefront Phase 1 design is shown in Figure 16-24. Pit exits are established at the 1,710 and 1,630 elevations. The 1710 exit is utilized primarily for waste rock movement to and RSF or pit backfill, and the 1630 exit for ore haulage to the leach pad.

Rangefront Phase 2 extends Phase 1 to the north and west and is a large pushback that is mined continuously from Year 5 through Year 14. Bench elevations range from 1,840 m to 1,480 m. A 20 m wide geotechnical berm is placed at the 1,650 elevation due to an uninterrupted vertical wall height greater than 160 m, as shown in Figure 16-25. A haul ramp established on the West wall to the 1,710 elevation provides access to the Range Front Rock Storage Facility. Leach ore is routed though the Phase 1 ramps.

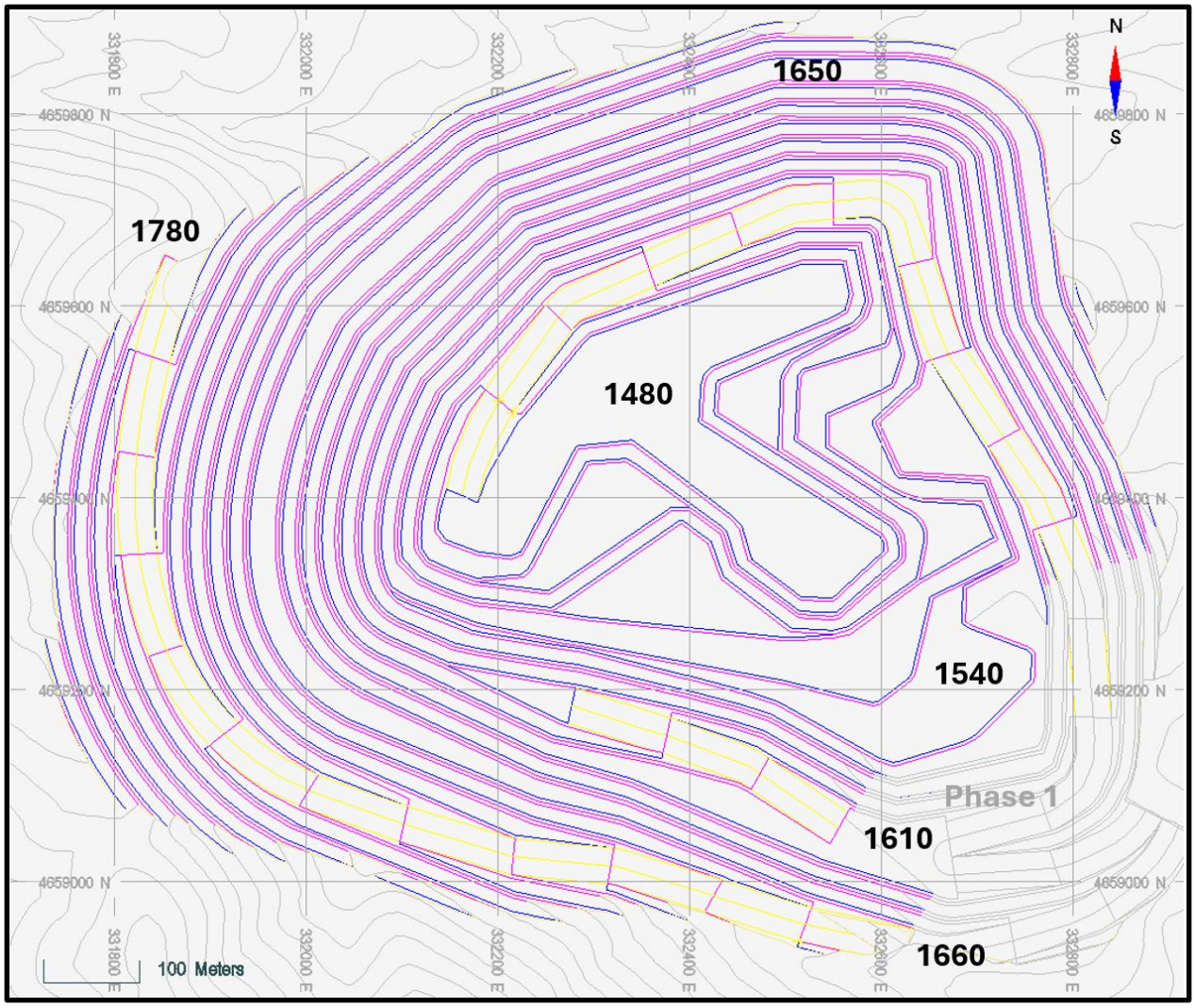
Rangefront Phase 3 is another large phase that mines the northern part of the Rangefront deposit in Year 8 through Year 14. Bench elevations range from 1,850 m to 1,470 m, as shown in Figure 16-26. Rangefront Phase 3 is currently pending a proposed alternative action as part of the current MPO. The phase design is truncated at the 1,470 elevation to allow adequate stand-off distance from the expected water table elevation. If allowable based on permitting status, Phase 3 removes a portion (4 Mt) of the LHLF which is mined to a 2 to 1 slope and includes a 50 m offset from the pit edge. See Figure 16-26 for the Rangefront Phase 3 design.

Rangefront Phase 4 is the eastern-most phase and is mined from Year 12 through Year 15. It is the last phase mined from Rangefront and as part of the mine plan prior to placement of stockpiled material on the leach pad in Year 16. Phase 4 bench elevations range from 1,660 to the 1,470 elevation. As with Phase 3, Phase 4 is truncated at the 1,470 elevation due to groundwater. The ramp exits Phase 3 at the 1,610 elevation as shown in Figure 16-27.



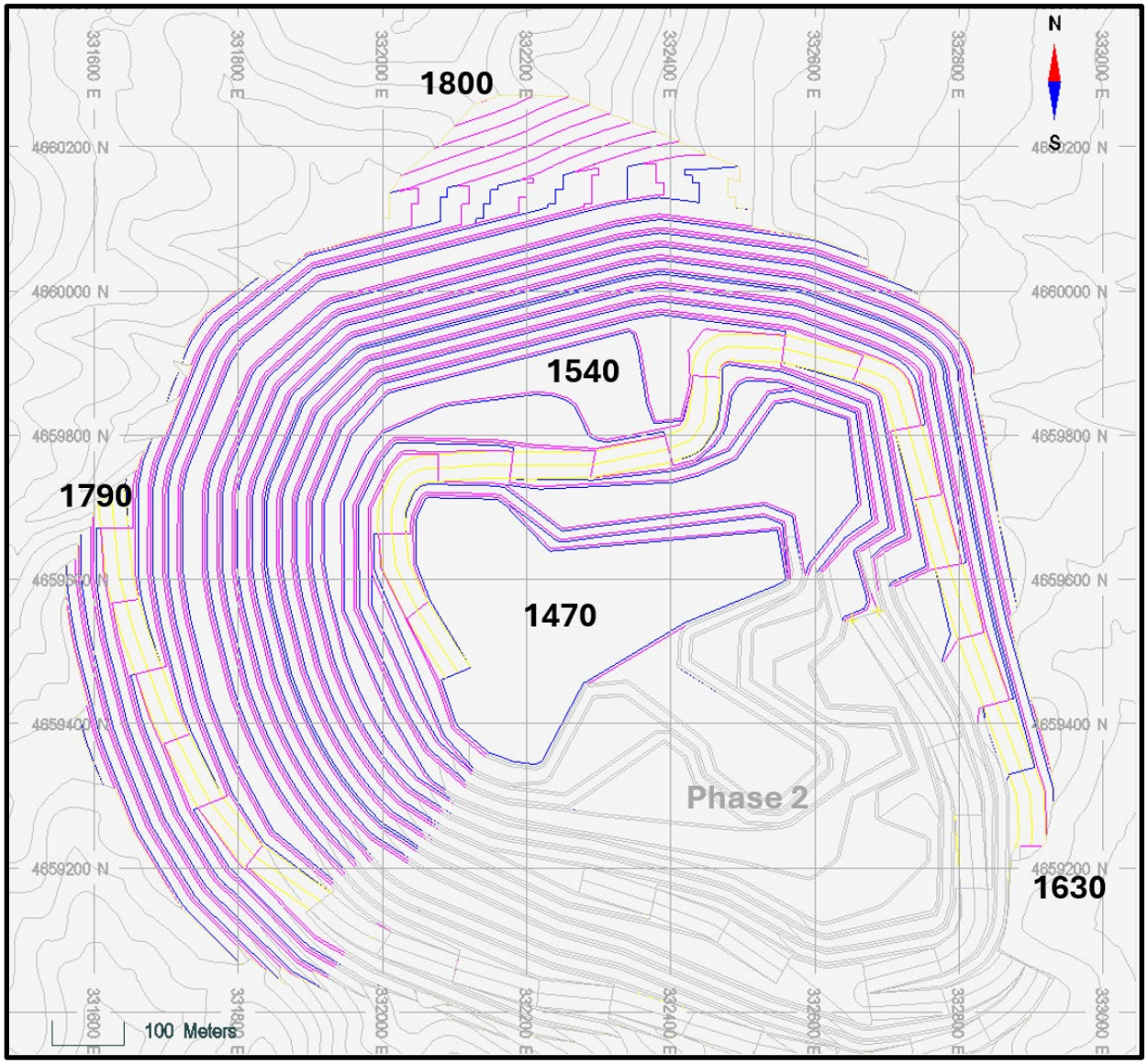
Source: AGP, 2026

Figure 16-24: Rangefront Phase 1



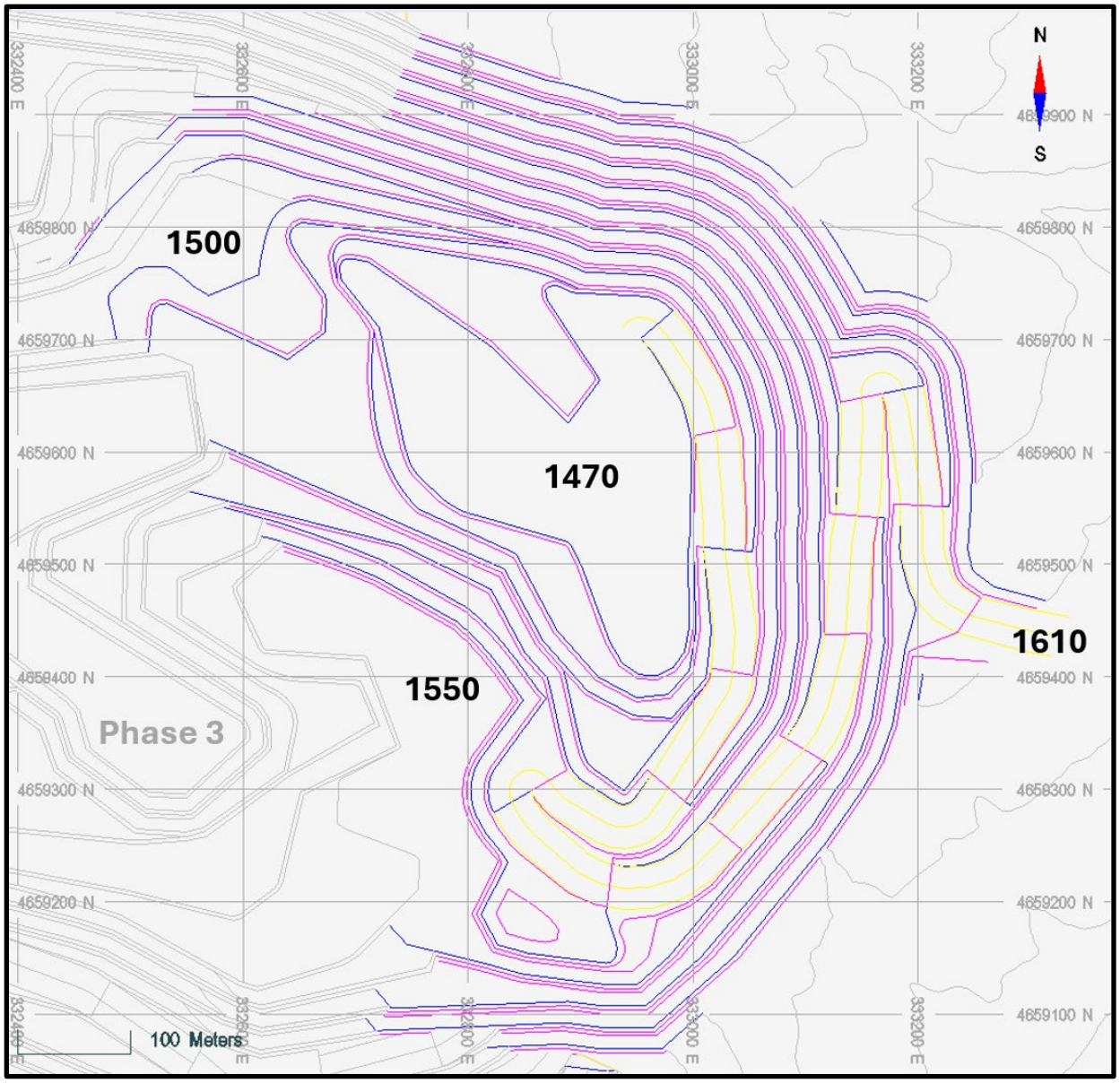
Source: AGP, 2024

Figure 16-25: Rangefront Phase 2



Source: AGP, 2026

Figure 16-26: Rangefront Phase 3

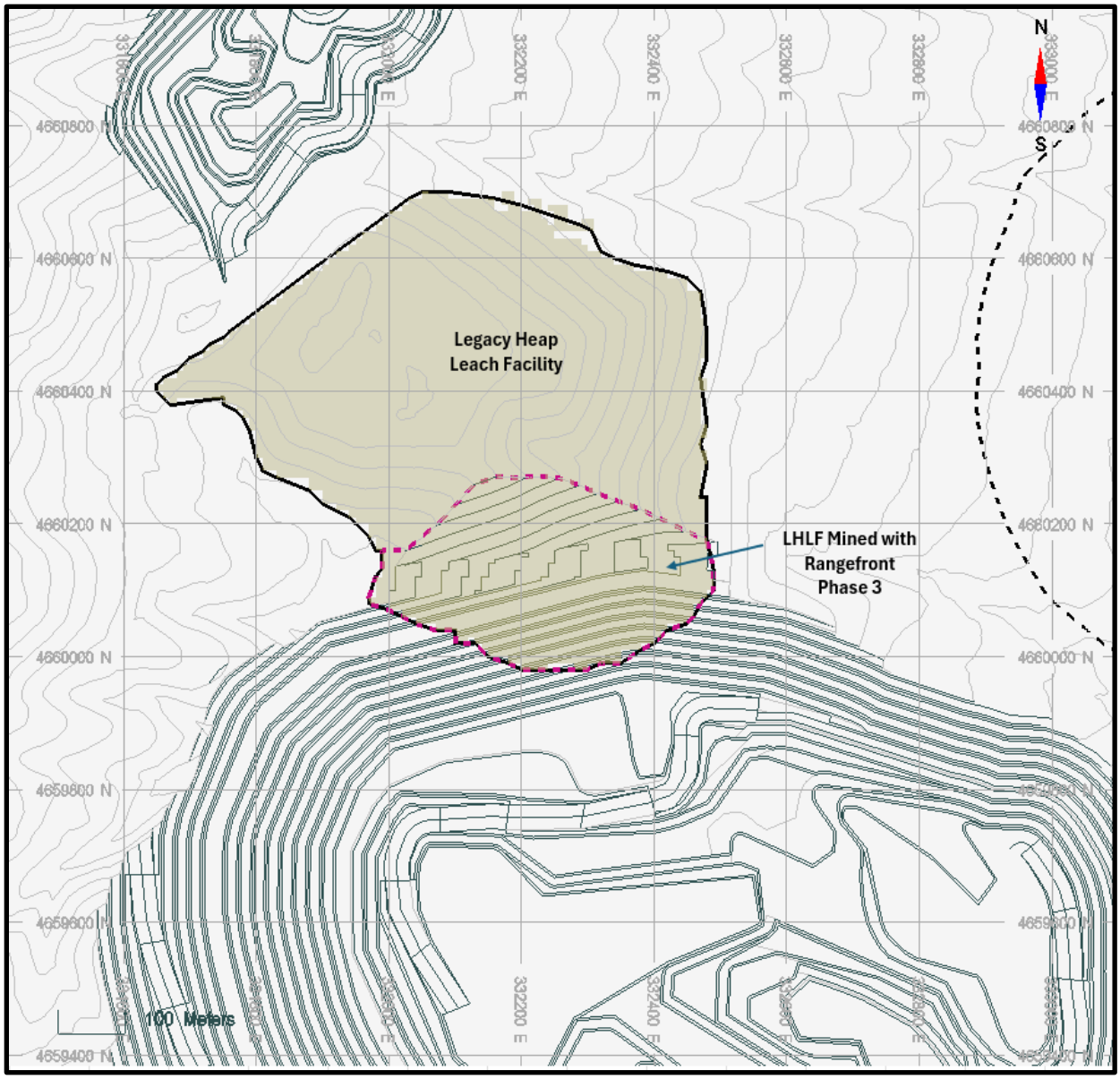


Source: AGP, 2026

Figure 16-27: Rangefront Phase 4

16.7.8 Legacy Heap Leach Relocation

If allowable based on permitting status, the Legacy Heap leach is mined in two phases. In Year 8, 4 Mt of the heap leach will be stacked on the new heap leach pad while mining Rangefront Phase 3. The remaining 28 Mt of material will be removed and placed on the new heap leach pad in Year 16 at the end of the mine-life (Figure 16-28).



Source: AGP, 2026

Figure 16-28: Legacy Heap Leach Mining

16.8 ROCK STORAGE FACILITIES

There are two main Rock Storage Facilities (RSF). The Discovery RSF will be located in a valley north of the Discovery pit and the Rangefront RSF will be located southwest of the Rangefront pit near the CD pit. The Discovery Phase 3, CD and M pits will be backfilled with waste rock as shown in Figure 16-29. The projected RSF and pit backfill capacities are given in Table 16-17. LHLF material totaling 32 million tonnes is stockpiled on the heap leach facility pending the outcome of ongoing metallurgical test work and permitting.

The design of the RSFs considers a final density of 1.96 t/m^3 or a swell factor of approximately 1.22 considering compaction will occur in the dumps over time. All RSFs 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 RSFs is designed at 37.5 m to

accommodate the larger trucks. Knight Piésold completed a stability assessment of the Rock Storage Facilities, "Feasibility Study -Waste Rock Dump Stability Assessment" dated June, 29 2026 that confirm that the proposed Rangefront and Discovery RSF configurations meet the minimum stability criteria for long-term static and seismic conditions.

Table 16-17: Rock Storage Facility Capacity

Rock Storage Facility	Volume (M^3)	Placed (Mt)	Capacity (Mt)
Discovery RSF	73.2	143.5	172.3
Rangefront RSF	104.3	204.5	208.7
Disco 3 Backfill	18.2	35.6	38.4
M Backfill	23.1	45.2	59.6
CD Backfill	62.3	122.1	129.0
Total Capacity	**281.0**	**550.8**	**608.1**

16.9 STOCKPILE FACILITIES

Leach feed will be stockpiled during the pre-production period and throughout the mine life. Leach ore is sent to different stockpiles based on the gold cyanide solubility codes in the resource model. The Stockpiles will be located northeast of the M pit (Figure 16-29).

- Met 1: gold cyanide solubility > 65%.
- Met 2: gold cyanide solubility > 50%, <65%
- Met 3: gold cyanide solubility > 25%, <50%

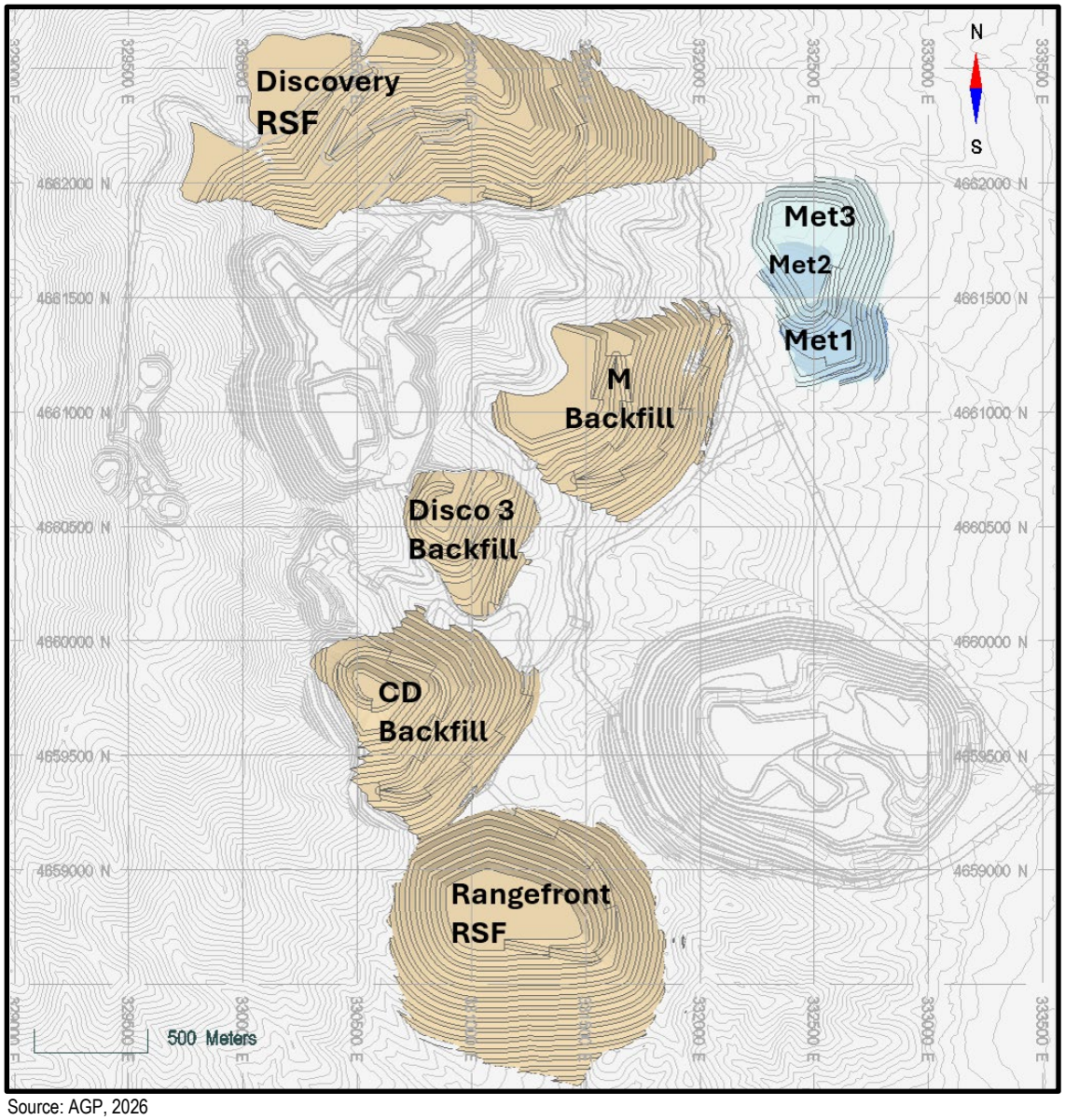
The Met 1 and Met 2 stockpiles built during pre-production will be sent to the heap leach facility in Year 1. The maximum size the Met 1 and Met 2 stockpiles reach is early in Year 1 at 7.2 Mt and 0.8 Mt, respectively. The Met 1 and Met 2 stockpiles are split into high grade and low-grade bins providing the opportunity to process higher grade material early in Year 1.

The Met 3 stockpile will be developed over the life of mine, reaching a maximum of 27.7 Mt in Year 15. Met 3 stockpile material will be placed on the heap leach facility in Years 15 and 16 after in-pit mining is completed.

The maximum stockpile sizes realized in the LOM plan are shown in Table 16-18 and stockpile locations are shown in Figure 16-29. The Met 3 stockpile expands into the Met 1 and Met 2 stockpile locations after they are reclaimed in Year 1.

Table 16-18: Stockpile Maximum Size

Stockpile Capacity	Volume (M^3)	Tonnes (Mt)
MET 1	3.7	7.2
MET 2	0.4	0.8
MET 3	14.1	27.7
Total Capacity	**18.2**	**35.7**



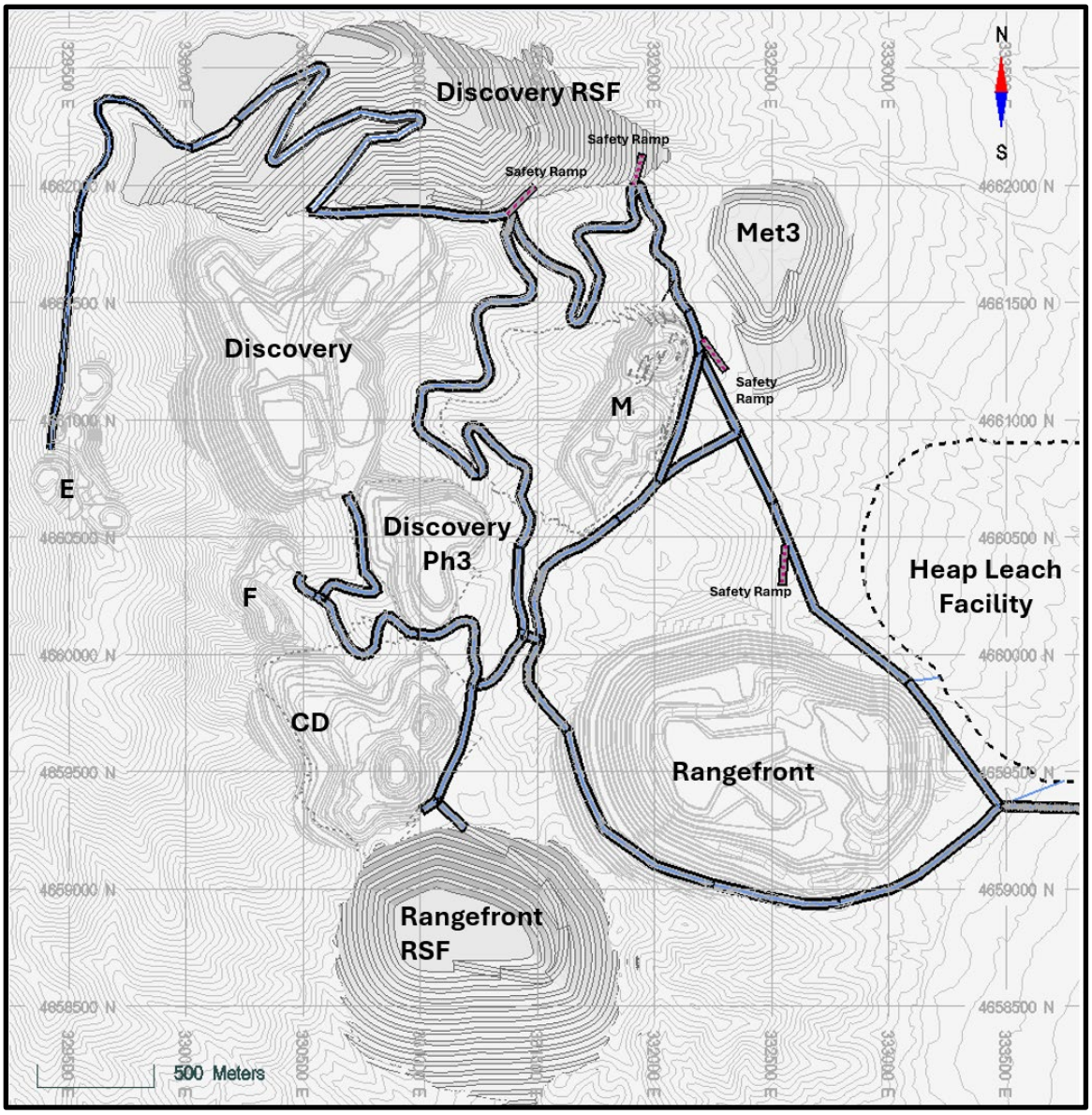
Source: AGP, 2026

Figure 16-29: Rock Storage Facilities and Stockpiles

16.9.1 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 16-30 shows the mine surface haul roads at Black Pine external to the pits.

There are over 24 km of surface haul roads planned that range from an elevation of 2,400 masl at the ridgeline near E pit to 1,510 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 discussed 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.



Source: AGP, 2026

Figure 16-30: Projected Mine Surface Haul Roads

To mitigate a safety hazard due to the long downhill loaded runs to the heap leach facility, runaway truck ramps are incorporated as part of the surface road design. In case of brake or retarder failure, the truck operator maneuvers the truck into a runaway or escape ramp. The runaway lane has a reverse grade (up to 15%) and is covered in a bed of loose gravel or coarse sand. This reverse grade design is utilized on the two ramps heading north into the Discovery RSF. The other two ramps heading to the south are designed to be longer in length due to not being able to add a reverse grade on these sections. Where space is limited, the use of collision berms or piles of loose granular material placed along the center line of the road can be used. In this case the truck straddles the collision berm and is brought to a stop.

16.10 MINE PLAN

The mine plan schedule delivers 433 Mt of ore to the leach pad grading 0.35 g/t Au in the first 5 years, 0.30 g/t Au in the first 10 years, and 0.29 g/t Au over the 16 year mine life, including one year of placing only stockpiled ore after in-pit mining has ended. Waste rock tonnage from all pits (including the stockpiling of the LHLF material on the new leach pad) totals 582 Mt and is placed into rock storage facilities including the backfilling of the Discovery Phase 3, CD, and M pits. The overall open pit strip ratio is 1.3:1. The total projected mine production from all pits and the LHLF is stated in Table 16-19.

Table 16-19: Mine Production totals

	Leach Ore Mt	Au g/t	Au Recovery %	Waste Mt	Total Tonnes Mt	Contained Gold Moz
All Pits	433.3	0.29	70.2	582.4	1,015.7	4.05

The mine plan includes one year of pre-production stripping followed by 15 years of mining, and 1 year of stockpile processing and is shown in Table 16-20. The mining rate reaches 60 M tpa of material mined in Year 2 and following additional permitting increases to a 72 M tpa rate through Year 11. The mining rate decreases to 68 M tpa, 60 M tpa, and 45 M tpa in Years 12 to 14. Production in Year 15 includes 10 Mt of Met 3 stockpile movement with the remaining 17.6 Mt of stockpiled material moved in Year 16. Total tonnes moved are limited to 60 M tpa through Year 2 and 72 M tpa thereafter.

The leach production schedule has a feed rate of 27.1 M tpa (74 ktpd) starting in Year 1, and averages 24.1 M tpa from Year 2 through Year 10, 37.0 M tpa from Year 10 to Year 13, and 32.5 M tpa in Years 14 and 15. Met 1 and Met 2 leach ore stockpiled in pre-production is sent to the leach pad in Year 1. Maximum stockpile sizes are shown in Table 16-18.

COMET was utilized as an integrated mine-to-processing optimization platform to evaluate a broad range of strategic planning alternatives for the Black Pine Project. The analysis considered alternative phase sequencing strategies, stockpile management policies, material routing priorities, cut-off grade methodologies, dilution assumptions, satellite pit development timing, and mining and stacking rate trade-offs. By simultaneously optimizing mining, processing, and stockpiling decisions, COMET generated and evaluated a large matrix of development scenarios to quantify the impact of operating policies on project value and long-term operational performance.

The optimization work identified several key value drivers and helped define preferred operating strategies for further consideration during the feasibility study process. The results highlighted the importance of phase sequencing, stacking and processing priorities, material handling strategies, and dilution management in maximizing project value. The analysis also quantified the trade-offs associated with alternate development pathways and provided a framework for comparing potential mine plans across a consistent set of technical and economic assumptions. The findings of the COMET scenario work served as the basis for the key assumptions and mine scheduling decisions that were utilized when developing the Feasibility Study Mine Plan

There are three types of clay that are coded into the Black Mine resource model. Blending the blocky and clay types with the durable type is important for leach pad permeability and performance. The ratio of clay types shown in Table 16-21 indicates that there is sufficient durable material available for blending with clay and blocky material. The blending requirements for blocky and clay material used for mine scheduling are as follows:

- No more than 40% clay material in the blend for any planning period without blending with durable or blocky material
- No more than 70% blocky material in the blend for any planning period without blending with clay or blocky material

16.11 MINE PLAN SCHEDULE

Prior to mining, proper roads from the open pits to the leach pad, waste destinations, and other facilities will be constructed to include stormwater BMPs to control surface water and sediment runoff. The initial mining fleet will be utilized for some pre-mining construction activities. A monthly mine plan was run for the first three years including the pre-production period, quarterly plans for two years (Years 3 and 4) and annually starting in Year 5. Annual plans are described below, tabulated in Table 16-20 and shown in the end of period maps (Figure 16-31 – Figure 16-40).

The large mine fleet consisting of 34 m^3 excavators, a 29 m^3 loader and 240 t trucks is anticipated to be used in the large pits of Discovery, Rangefront, and M Pit. A smaller fleet with 7.6 m^3 excavators and 64 t trucks will be used in satellite pits including J Pit, F Pit, E Pit, and for pioneering work in all mining areas. CD pit will use the large fleet on benches with sufficient mining space and the small equipment fleet in the pit bottom where mining space is limited. A mining contractor will provide and manage the smaller mining fleet as a measure to reduce initial mining execution risk as equipment operator training is ongoing and to eliminate the need to purchase a separate small equipment fleet with varying use throughout the plan.

Pre-production mining starts in Discovery Phase 1, Discovery Phase 2, J pit, and opens Discovery Phase 3. Leach ore mined during the pre-production year is sent to the appropriate stockpile. Waste rock is routed to the Discovery Rock Storage Facility, and suitable waste rock may be used for construction purposes but is not captured in the mine plan. During pre-production, a total of 8.5 Mt of leach ore will be sent to the ore stockpiles. The Discovery RSF will be established during the pre-production phase (Figure 16-31).

Mining in the first five years of production (Years 1-5) occurs predominantly in the Discovery Phases (Phase 1 – Phase 4) except for the 3.1 Mt mined from J Pit in Year 1 and initial mining in Rangefront Phases 1 and 2 in Year 5 (2.2 Mt). J Pit is required to be mined out in Year 1 of the plan to allow for the continued progression of the Discovery RSF which backfills and covers the J Pit design. Discovery Phase 1 is the predominant mining area in Years 1 and 2 but continues in Year 3 before completion in Year 4. Discovery Phase 2 is mined continuously in Years 1-4 before completion in early Year 5 (300 kt in Year 5). Discovery Phase 3 is mined in Years 2 and 3 before being completed in Year 4 with Phase 1 completion. Discovery Phase 4 is mined in Year 2 before becoming the predominate mining phase in Years 3-5 and continues until its completion in early Year 7 (300 kt in Year 7). In addition to the ore contributions from these phases, Year 1 also includes 8 Mt at a 0.19 g/t Au of MET 1 and MET 2 material being mined from stockpiles, depleting the MET 1 and MET 2 stockpile balances built in pre-production and early Year 1. The MET 3 stockpile continues to be built over this 5-year period with a closing balance of 24.4 Mt at the end of Year 5.

The overall mining rate for the first five years of production climbs from 52 Mt in Year 1 to 60 Mt in Year 2 and then remains consistent at 72 M tpa for Years 3-5 of the plan. Overall leach pad placement and grade vary during this period but is generally around 27.5 M tpa except Year 2 which is 17.3Mt. Feed grade generally increases during this period from 0.25 g/t Au in Year 1 to 0.43 g/t Au in Year 5 as mining progresses in the Discovery phases and uncovers higher grade ore. The exception being Year 2 of the plan at 0.44 g/t Au to partially offset the lower feed tonnes available. Additional details around the mine schedule for these periods can be found in Table 16-20, Table 16-21, and Table 16-22 below. End of Period maps for these years can be found in Figure 16-32 to Figure 16-36.

The remaining mine plan (Years 6-15) is predominantly driven by mining in the Rangefront Phases (Phase 1 – Phase 4) with contributions from the remaining four satellite pits (F Pit, M Pit, CD Pit, E Pit). Rangefront Phases 1 and 2 both have pioneering work occur in Year 5, but initial large-scale mining of both (20 Mt each) occurs in Year 6. Since Rangefront Phase 1 is a smaller high-value phase mining is completed in early Year 8 (100 kt in Year 8). Rangefront 2 is much larger and remains a large material source through Year 13 before being completed in Year 14. Rangefront Phase 3 starts in Year 8 with initial mining being the relocation of the legacy heap leach (4 Mt) and continues to along with Rangefront Phase 3 make up the majority of the material movement until completion in Year 14. Rangefront Phase 4 starts in Year 12 and is completed in Year 15 as the last in-pit ore mined prior to the Met 3 stockpile

being reclaimed and placed on the Leach Pad along with the balance of the LHLF (28 Mt). The MET 3 stockpile continues to be built through this period until the completion of Rangefront Phase 4 in Year 15 with a maximum balance of 27.7 Mt which is placed in Year 15 (10.1 Mt) and Year 16 (17.6 Mt).

In addition to the mining in the Rangefront Phases that make up the majority of the material movement in Years 6-15 the four remaining satellite pits are also mined to supplement the mining in Rangefront. The first of these to begin mining is the CD Pit in Year 5 and mining continues through Year 8. In Years 6-8 this makes up about 1/3rd of the material movement. The next satellite pit to be mined is M Pit which begins in Year 7 and continues through Year 9. Both CD and M will utilize a combination of smaller fleet and large fleet equipment which will need to be balanced as part of the scheduling interactions of these phases. F Pit begins in Year 9 after M Pit is completed and continues into Year 10 with an exclusively small fleet being used. The last satellite phase to be mined is E Pit in Years 12 and 13 and will also use an exclusively small fleet to be mined.

The overall mining rate for Years 6-15 remains at 72 M tpa until Year 12 when it begins to decrease in response to mining progression and lower stripping requirements. In Years 12 to Y15 the mining rate decreases from 68 Mt down to 29.9 Mt progressively. The leach pad placement rate in this period starts at 29.4 Mt in Year 6 before a decrease to between 16 Mt and 19.7 Mt in Years 7-9 and then increasing back up to 37 M tpa in Years 11-13. Years 14-16 see the placement rate decrease in response to the phase progression of the time and then the transition to stockpile and Legacy Heap Leach placement. The grade during this period is generally decreased from the Discovery mining in the first 5 years of mining and remains consistent at 0.20 – 0.30 g/t Au with most years average out close to 0.25 g/t. Even though this period is of lower gold grades it is partially offset by higher gold recovery rates associated with the material. Additional details around the mine schedule for these periods can be found in Table 16-20, Table 16-21, and Table 16-22 below. End of Period maps for these years can be found in Figure 16-37 to Figure 16-40.

Table 16-20: 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	Total
Mining Summary	Mined Waste (Mt)	15.2	27.8	35.9	39.9	40.8	41.3	42.0	55.7	52.7	51.7	44.0	34.8	30.5	22.4	12.3	7.5	28.1	582.4
	Leach Ore (Mt)	9.7	24.1	24.1	32.1	31.2	30.7	30.0	16.3	19.2	20.3	28.0	37.2	37.5	37.6	32.7	22.4	0.0	433.3
	Au (g/t)	0.22	0.27	0.40	0.32	0.38	0.42	0.26	0.27	0.24	0.27	0.19	0.21	0.25	0.33	0.30	0.25	-	0.29
	Au Recovery (%)	62.4	63.4	66.3	66.6	67.0	71.4	64.2	70.1	64.6	69.3	70.8	73.1	74.6	76.9	76.4	74.6	-	70.2
	Total Mined (Mt)	**24.9**	**52.0**	**60.0**	**72.0**	**72.0**	**72.0**	**72.0**	**72.0**	**71.9**	**72.0**	**72.0**	**72.0**	**68.0**	**60.0**	**45.0**	**29.9**	**28.1**	**1,015.7**
Stockpile Summary	Met 1	6.9	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7.2
	Met 2	0.6	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.8
	Met 3	1.0	4.5	6.9	5.0	4.7	2.4	0.7	0.3	0.2	0.6	0.1	0.2	0.5	0.6	0.1	0.0	-	27.7
	Total Stockpiled (Mt)	**8.4**	**5.0**	**6.9**	**5.0**	**4.7**	**2.4**	**0.7**	**0.3**	**0.2**	**0.6**	**0.1**	**0.2**	**0.5**	**0.6**	**0.1**	**0.0**	-	35.7
	Reclaim (Mt)	-	8.0	-	-	-	-	-	-	-	-	-	-	-	-	-	10.1	17.6	35.7
	Stockpile Balance (Mt)	8.4	5.4	12.3	17.3	22.0	24.4	25.1	25.4	25.6	26.2	26.3	26.5	27.0	27.6	27.7	17.6	-	
	Total Moved (Mt)	24.9	60.0	60.0	72.0	72.0	72.0	72.0	72.0	71.9	72.0	72.0	72.0	68.0	60.0	45.0	40.0	45.7	**1,051.4**
Processed Material	**Leach Feed (Mt)**	1.3	27.1	17.3	27.1	26.5	28.3	29.4	16.0	19.0	19.7	27.9	37.0	37.0	37.0	32.6	32.5	17.6	**433.3**
	Au (g/t)	0.32	0.25	0.44	0.33	0.37	0.43	0.26	0.28	0.24	0.27	0.19	0.21	0.25	0.33	0.29	0.27	0.31	0.29
	Au Recovery (%)	70.2	66.9	72.4	70.3	71.6	73.5	64.8	70.5	64.9	70.5	71.0	73.2	75.1	77.4	76.5	63.5	43.4	70.2
	Met 1 (Mt)	**1.3**	**22.8**	**12.0**	**21.4**	**22.0**	**25.3**	**26.0**	**14.6**	**16.8**	**18.8**	**27.5**	**36.3**	**36.5**	**35.7**	**31.9**	**22.2**	-	**371.1**
	Au (g/t)	0.32	0.25	0.50	0.34	0.38	0.44	0.26	0.28	0.24	0.27	0.19	0.21	0.25	0.33	0.29	0.25		0.29
	Au Recovery (%)	70.2	68.6	76.1	72.9	73.8	74.8	65.7	71.7	65.8	71.2	71.2	73.5	75.3	78.1	76.9	74.9		73.4
	Met 2 (Mt)	-	**4.3**	**5.3**	**5.7**	**4.5**	**3.0**	**3.4**	**1.4**	**2.2**	**0.9**	**0.4**	**0.7**	**0.5**	**1.3**	**0.7**	**0.2**		**34.5**
	Au (g/t)	-	0.23	0.30	0.28	0.31	0.35	0.25	0.26	0.25	0.34	0.19	0.16	0.20	0.29	0.32	0.22		0.28
	Au Recovery (%)		56.9	58.6	58.1	58.7	59.3	57.4	57.8	57.5	59.2	55.6	53.9	55.8	58.4	58.9	56.7		58.2
	Met 3 (Mt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**10.1**	**17.6**	**27.7**
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.31	0.31	0.31
	Au Recovery (%)																43.4	43.4	43.4

Table 16-21: Clay and Blocky Ratio %

Clay	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Durable	88.9	86.9	72.6	85.3	84.6	79.0	85.7	98.2	93.8	96.5	99.6	99.9	99.6	98.5	99.1	92.5	76.7
Clay	11.1	10.2	10.6	6.2	2.8	1.1	0.3	0.6	1.1	0.5	0.2	0.1	0.4	1.5	0.3	4.9	15.0
Blocky	0.0	2.9	16.8	8.5	12.6	20.0	14.0	1.2	5.1	3.0	0.2	0.0	0.1	0.0	0.6	2.6	8.3

Table 16-22: Tonnes Mined by Pit/Phase (Mt)

	Discovery				Rangefront				F	M	J	CD	E	HLF
	Ph1	Ph2	Ph3	Ph4	Ph1	Ph2	Ph3	Ph4	Zone	Zone	Zone		Zone	
PP-1	15.6	2.9	4.5	-	-	-	-	-	-	-	1.9	-	-	-
Y1	39.8	9.1	-	-	-	-	-	-	-	-	3.1	-	-	-
Y2	35.4	10.9	8.3	5.4	-	-	-	-	-	-	-	-	-	-
Y3	7.3	27.2	10.6	26.9	-	-	-	-	-	-	-	-	-	-
Y4	4.4	21.2	13.8	32.7	-	-	-	-	-	-	-	-	-	-
Y5	-	0.3	-	62.5	0.9	1.3	-	-	-	-	-	6.9	-	-
Y6	-	-	-	7.9	20.1	20.0	-	-	-	-	-	24.0	-	-
Y7	-	-	-	0.3	12.4	29.2	-	-	-	10.0	-	20.0	-	-
Y8	-	-	-	-	0.1	28.0	12.0	-	-	10.1	-	17.6	-	4.0
Y9	-	-	-	-	-	36.2	15.0	-	6.0	14.8	-	-	-	-
Y10	-	-	-	-	-	33.0	36.6	-	2.5	-	-	-	-	-
Y11	-	-	-	-	-	30.0	42.0	-	-	-	-	-	-	-
Y12	-	-	-	-	-	21.6	26.3	15.1	-	-	-	-	5.0	-
Y13	-	-	-	-	-	24.6	20.0	9.1	-	-	-	-	6.3	-
Y14	-	-	-	-	-	4.3	17.2	23.5	-	-	-	-	-	-
Y15	-	-	-	-	-	-	-	29.9	-	-	-	-	-	-
Y16	-	-	-	-	-	-	-	-	-	-	-	-	-	28.1
Total	102.5	71.6	37.2	135.7	33.5	228.2	169.1	77.6	8.5	34.9	5	68.5	11.3	32.1

16.12 MINE EQUIPMENT SELECTION

The mine equipment fleet has been sized to achieve an annual production rate of 60 M tpa from pre-production through Year 2 and 72 M tpa from Year 3 onward. The major mining equipment required by period is shown in Table 16-23. The selected primary drills provide the capability to drill 10 m bench heights in a single pass. The haulage equipment is planned to include fatigue monitoring systems and utilize nitrogen filled tires for added haulage and operator safety.

Table 16-23: Primary Mining Equipment Requirements

Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Primary Drill (171 mm)	3	6	7	8	8	8	8	8	8	8	8	8	7	6	6	6	
Secondary Drill (127 mm)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Hydraulic Shovel (34 m^3)	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1
Production Loader (29 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck (240 t)	15	18	18	22	22	22	22	22	22	22	22	22	22	22	22	15	15
Hydraulic Shovel (7.6 m^3)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Production Loader (8.0 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Haul Truck (64 t)	5	5	5	10	10	10	13	13	13	13	13	13	13	8	7	7	
Track Dozer (Leach)	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Track Dozer	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Grader	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Wheel Dozer	3	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3	3
Water Truck	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2

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. Aggregate required for blasthole stemming and road surfaces will be provided by a contactor.

The detailed equipment requirements for the LOM plan are included in Section 21.

16.13 BLASTING AND EXPLOSIVES

Blast Dynamics, Inc. was contracted to assist with the drilling and blasting analysis for the Black Pine feasibility study. The study targeted ore ROM P_{80} size ranges from 100 to 200 mm depending on lithology and a targeted waste P_{80} of 250 mm for small pits, 300 mm for large pits. The study recommended that a 171 mm blasthole diameter be used for production and waste blasting.

Wall control blast designs were included in the blasting study. Angled presplitting with smaller diameter (127 mm) blastholes is recommended to help define the bench face. However, in highly fractured rock the effectiveness of the presplit will be minimal and will not justify the cost. In these cases, it is recommended that a vertical 7 m production hole (171 mm) be drilled 1 m in from the designed crest of the upper bench to reduce the potential for overbreak.

The report from Blast Dynamics, Inc, Drilling and Blasting Feasibility Study Version 1.2 dated May 2026 provides greater detail on the blasting recommendations.

Typical blast patterns are different for ore and waste material and vary by rock strength. The weighted average blasthole hole pattern for ore is 4.8 m x 4.2 m (spacing x burden) and 6.6 m x 5.7 m (spacing x burden) in waste. Holes will be 10-m deep with an additional 1.0-m sub-drill for a total of 11.0 m depth.

The powder factor with these pattern sizes is 0.30 kg/t for ore and 0.16 kg/t in waste. ANFO is the primary explosive as the pits are expected to be dry and the majority of the rock well fractured. However, to achieve the relatively small fragmentation target for the Polb ore, and any highly competent rock, the use of a 30/70 emulsion/ANFO blend may be required.

A local vendor will deliver explosives to the hole for a monthly fee. The monthly fee includes prill silos, ANFO truck, explosives and accessories magazines, labor costs, pickup trucks, and miscellaneous equipment and tools required for delivering the explosive product to the hole. The owner is responsible for the loading process including boosters, stemming, and ignition of the shot. Further blasting and explosives details are available in Section 21.

16.14 GRADE CONTROL

Grade control will be done using the blast hole sampling method. The standard sample length is 10 m. To allow for flitch mining, 50% of the ore holes are allocated to be sampled on 5 m intervals. 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 mine achieves its targets for metal production.

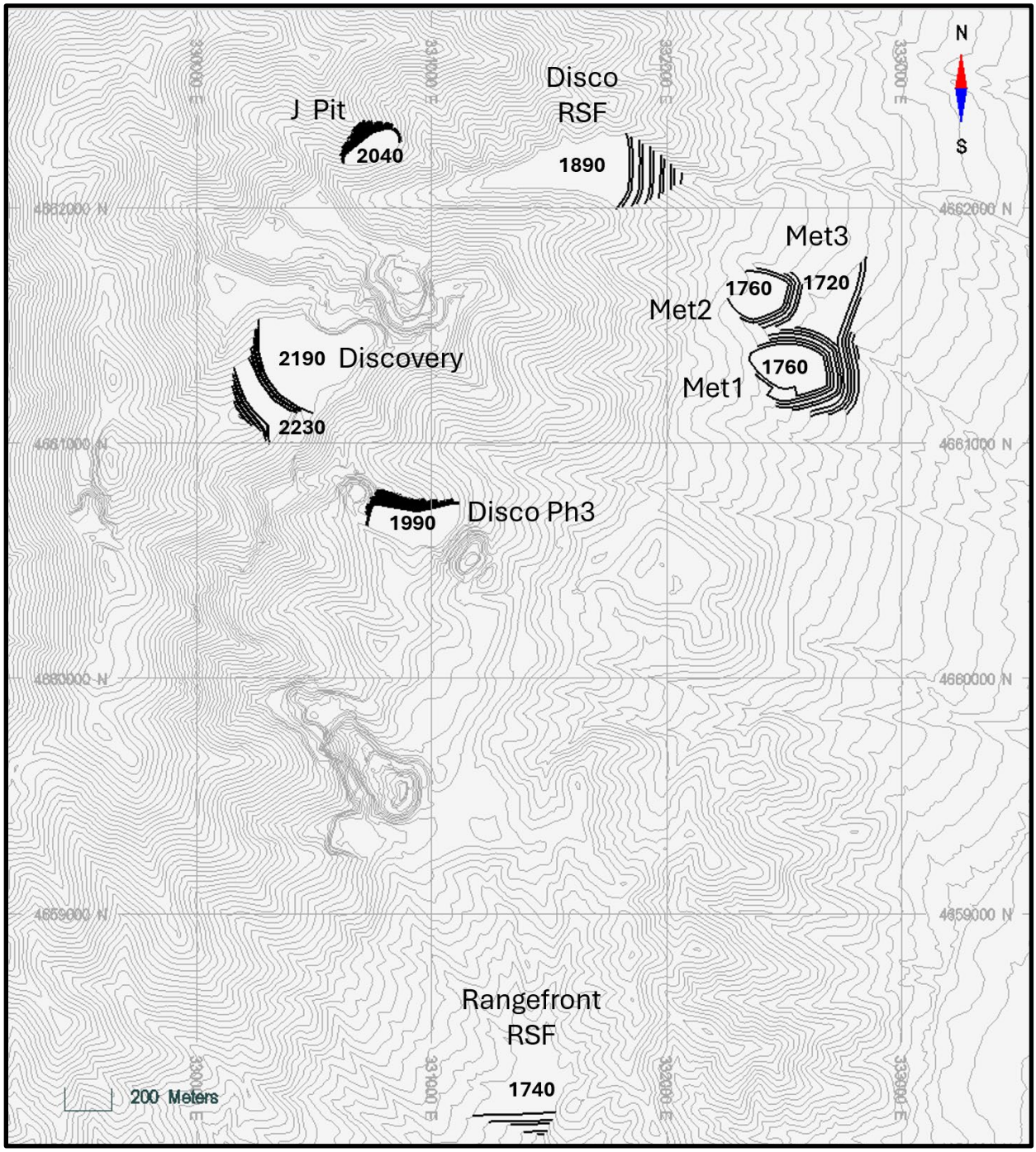
RC drilling is not included for grade control but should be assessed as the project advances to see if it improves grade control results.

16.15 PIT DEWATERING

All pits are considered dry and will not require dewatering. At Rangefront and CD, the pit depth is restricted, and the lowest mining bench is a minimum of 15 m above the estimated water table.

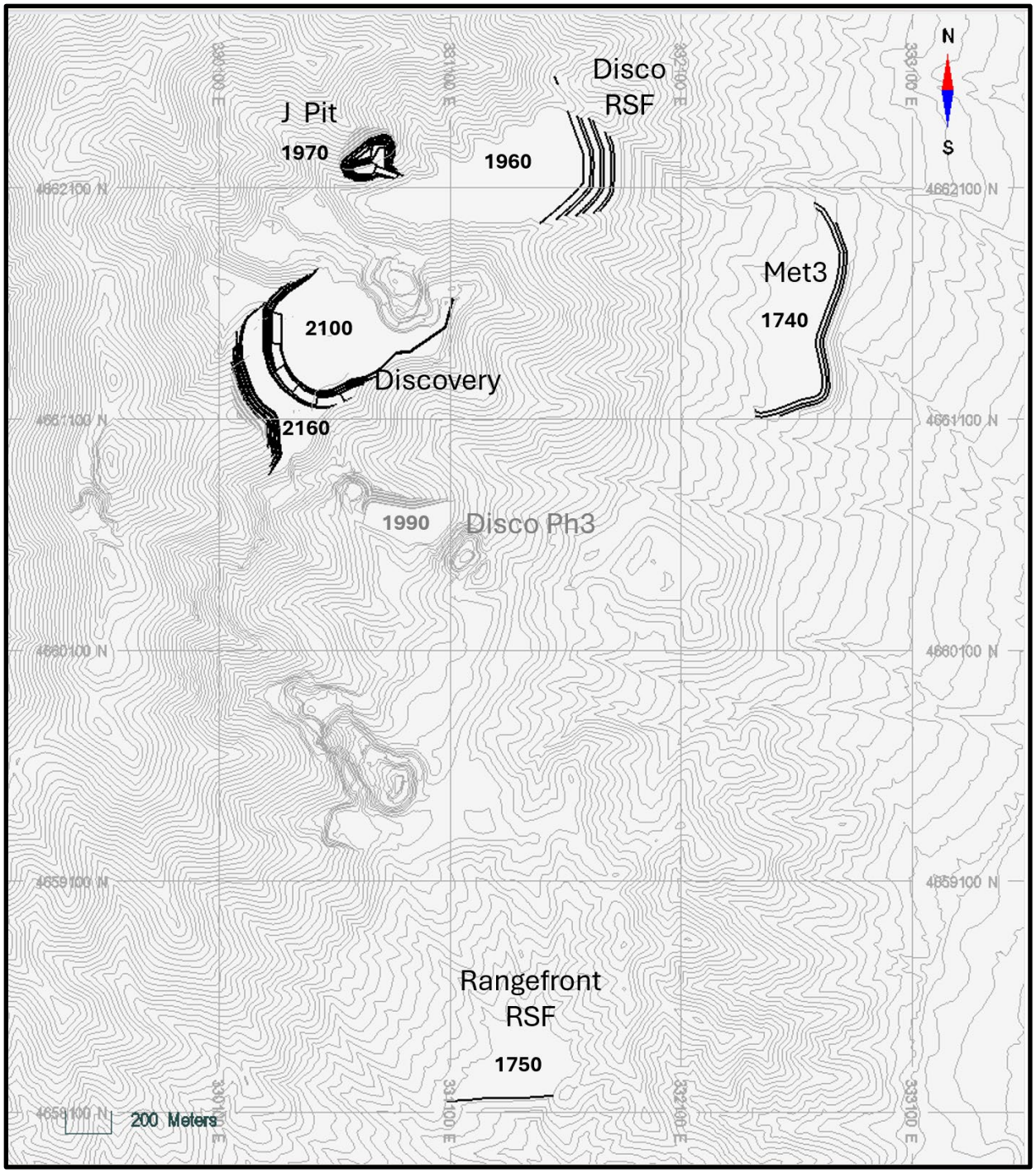
A small allowance has been included for the accumulation of surface water due to storm events and snow melting. If required, surface water will be pumped by mobile pumps placed in sumps on the mining level.

16.16 END OF PERIOD MAPS



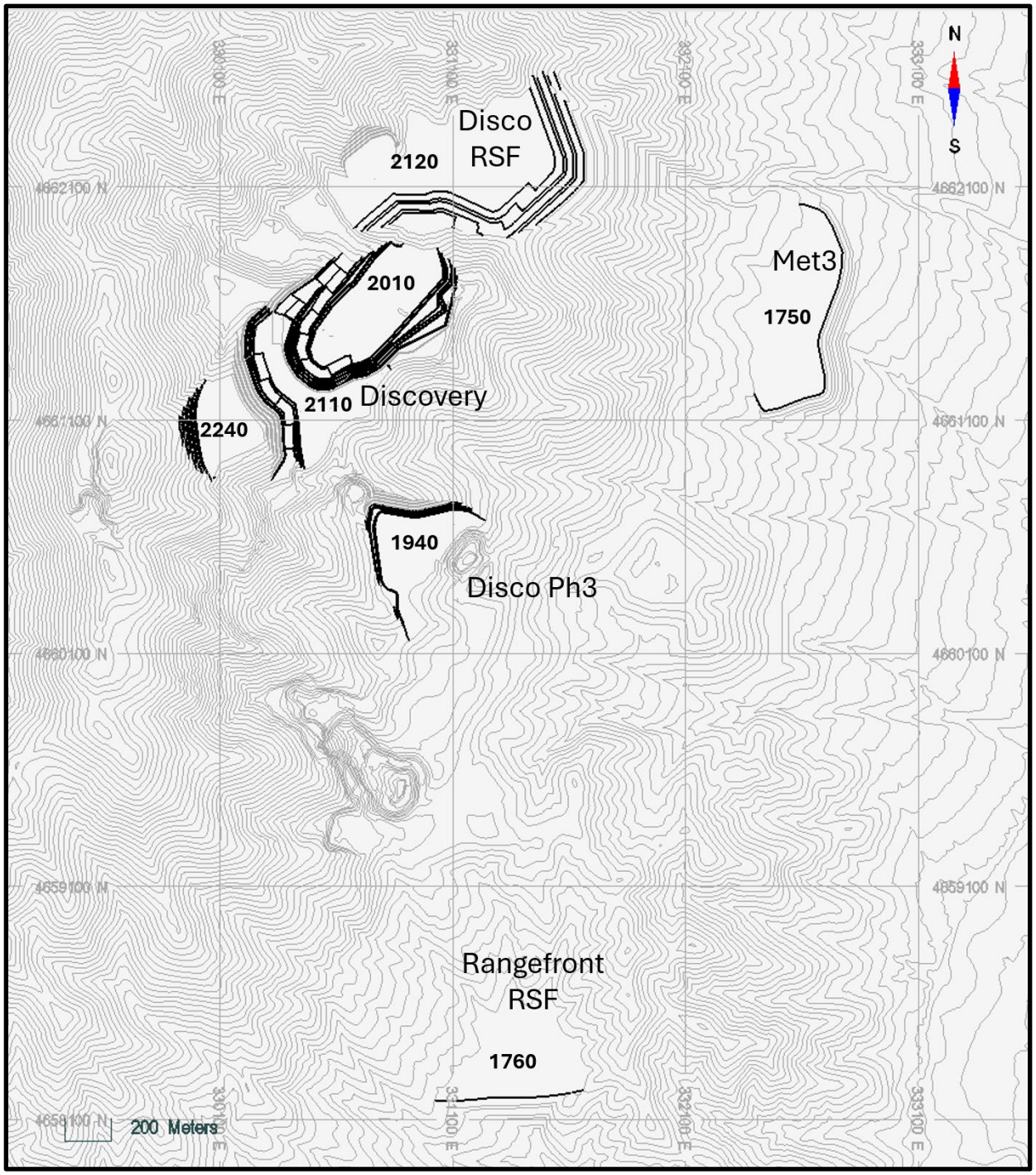
Source: AGP, 2026

Figure 16-31: End of Year -1 (Pre-Production)



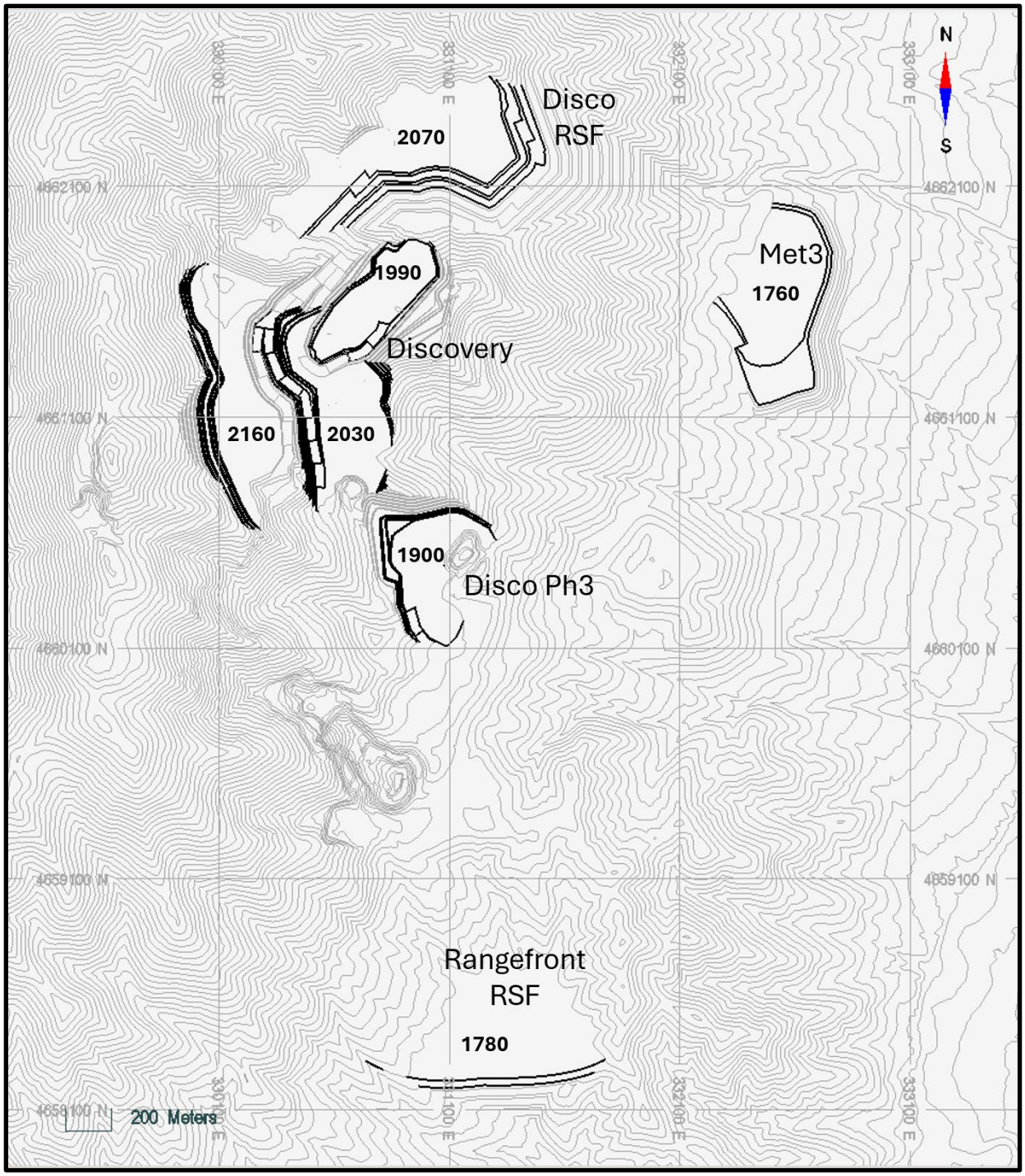
Source: AGP, 2026

Figure 16-32: End of Year 1



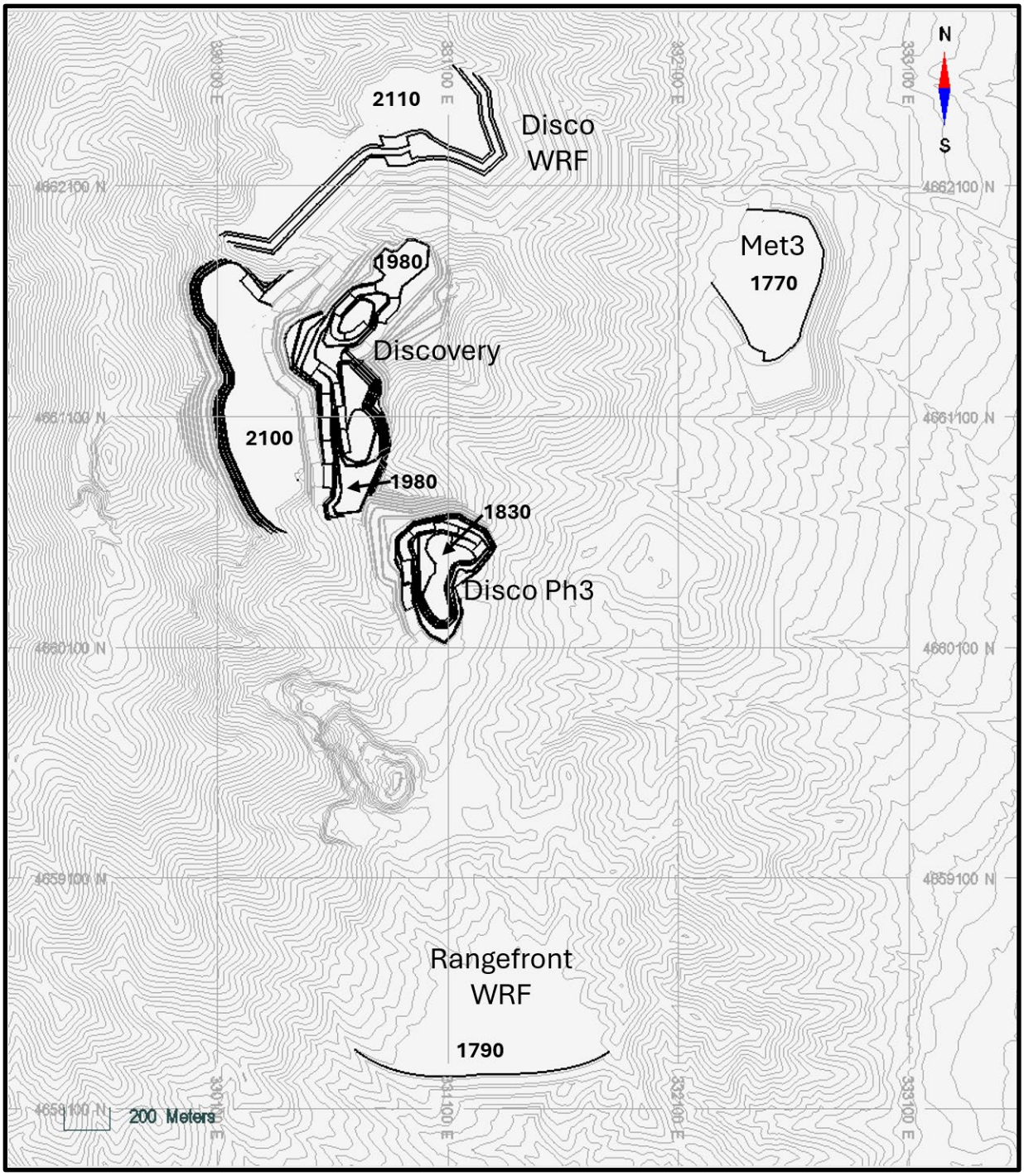
Source: AGP, 2026

Figure 16-33: End of Year 2



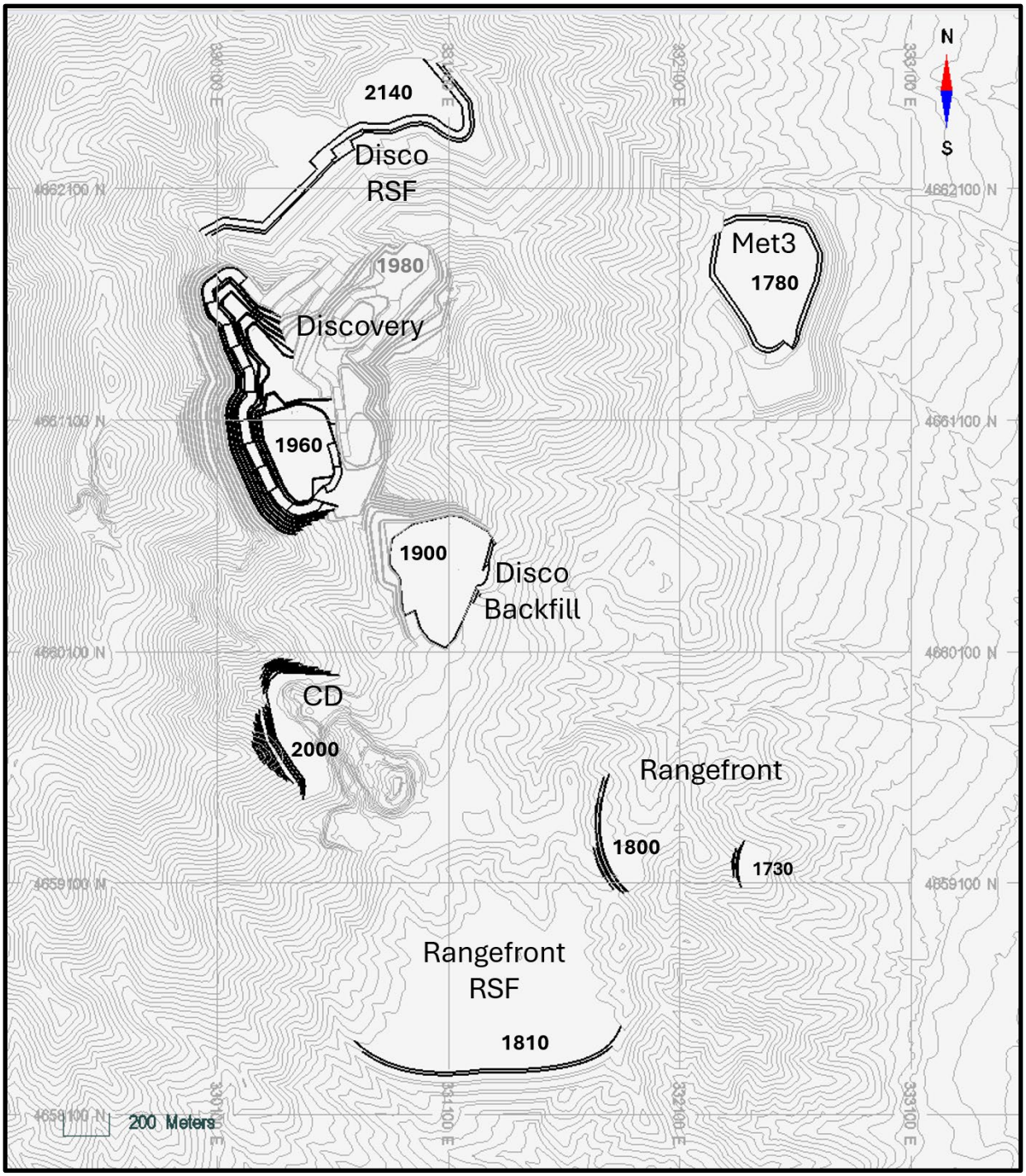
Source: AGP, 2026

Figure 16-34: End of Year 3



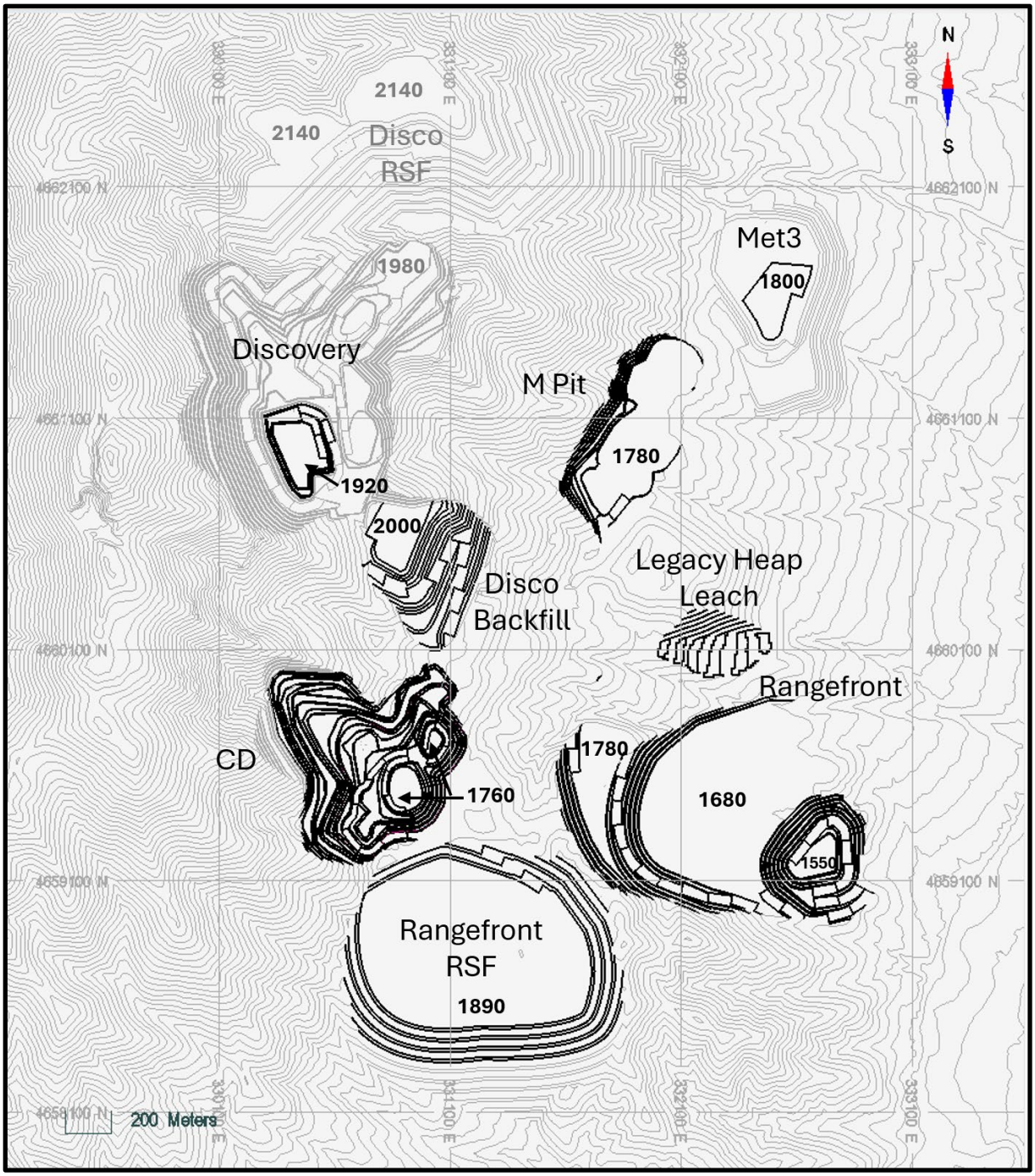
Source: AGP, 2026

Figure 16-35: End of Year 4



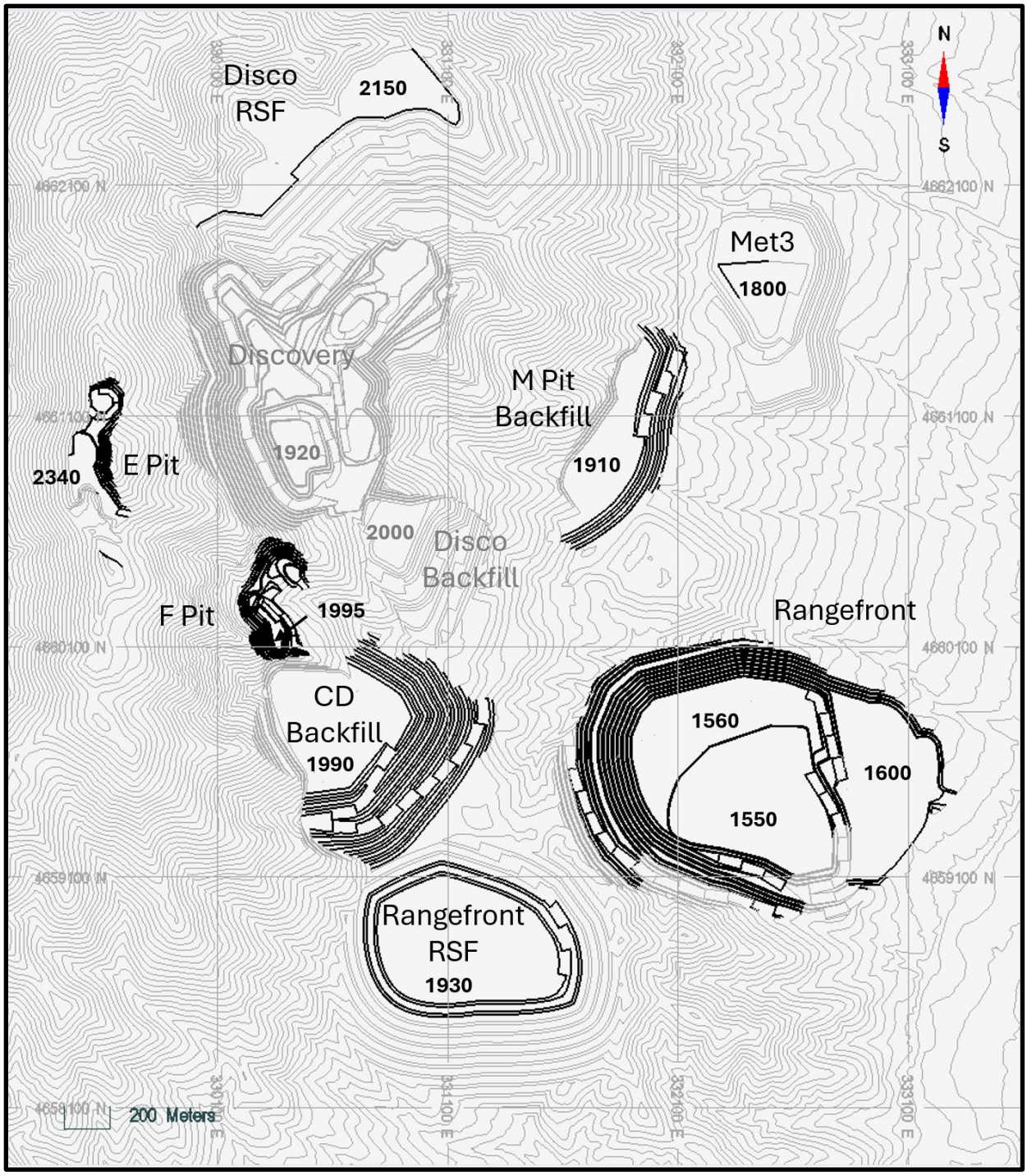
Source: AGP, 2026

Figure 16-36: End of Year 5



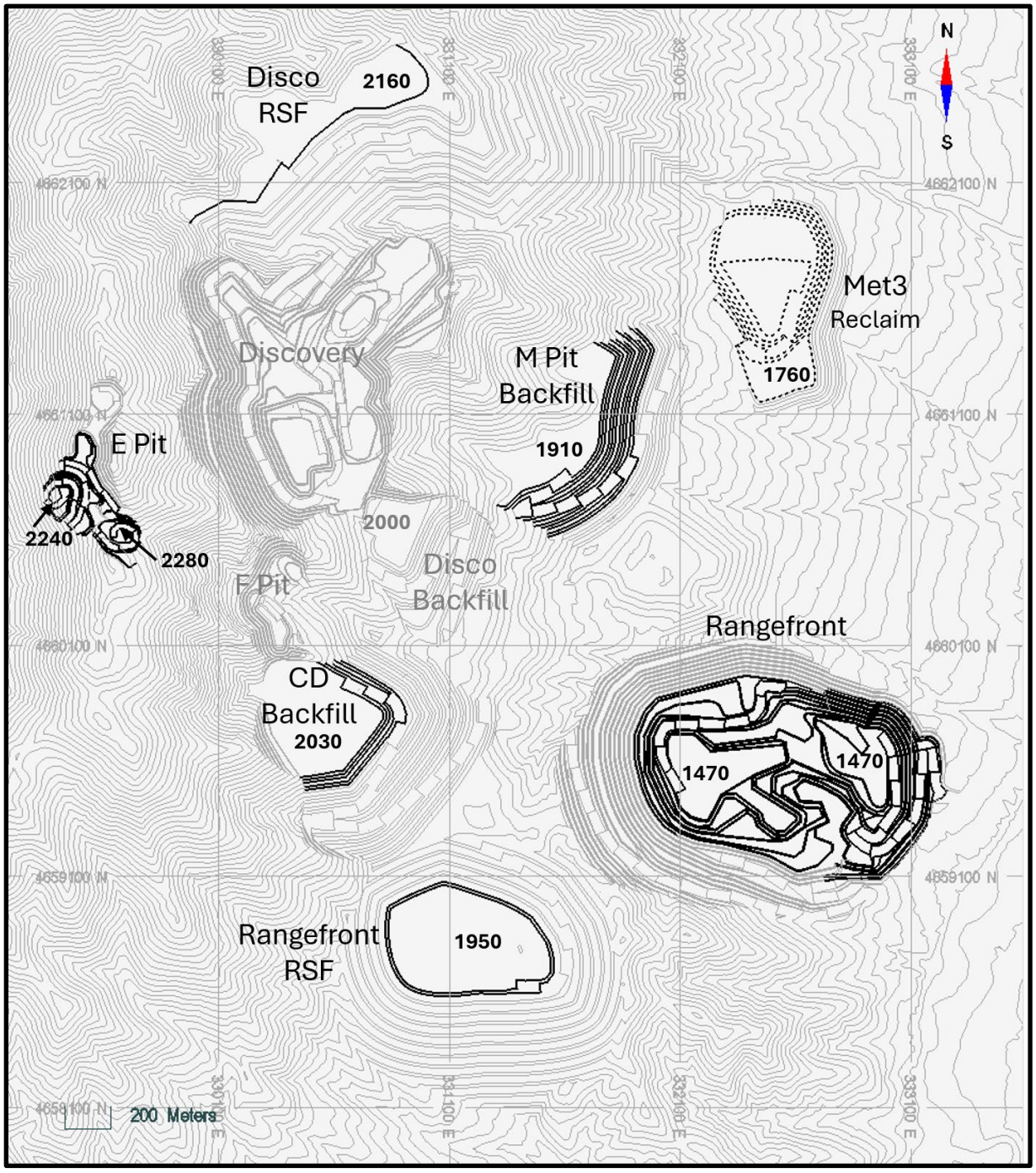
Source: AGP, 2026

Figure 16-37: End of Year 8



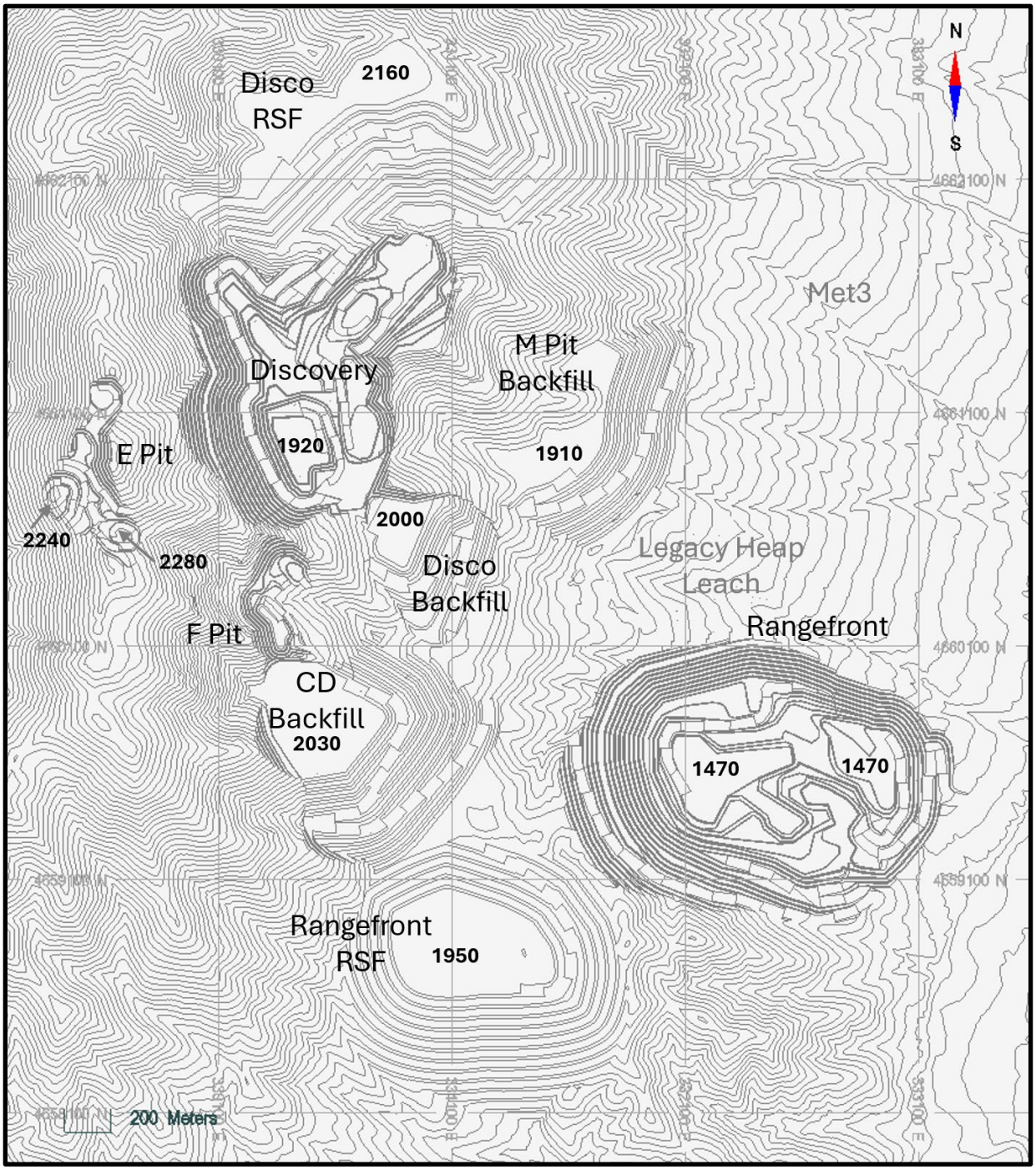
Source: AGP, 2026

Figure 16-38: End of Year 12



Source: AGP, 2026

Figure 16-39: End of Year 15 (Mining Complete) - 1 year of stockpile reclaim remaining



Source: AGP, 2026

Figure 16-40: End of Year 16 – End of Mine Life Stockpile Reclaimed

17 RECOVERY METHODS

The ADR plant and refinery designs presented in this Feasibility Study have been developed to a level appropriate to support the specified process flow rates, recovery assumptions, equipment capacities and cost estimates. These feasibility-level designs remain subject to normal refinement through detailed engineering and permitting and are not intended to replace, supersede, amend or otherwise modify the Mine Plan of Operations or any active permitting submission or regulatory review.

The process selected for recovery of gold and silver from the Black Pine deposit is a conventional ROM 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.

Note that the terms pregnant and rich solution will be used interchangeably throughout this section to refer to process solution which contains dissolved precious metals which can be economically processed through the ADR.

There are three main classifications of ore at Black Pine, each associated with a ratio of cyanide solubility to fire-assay (AuCN:AuFA). Mineralized material with an AuCN:AuFA of over 65% is identified as “Met 1”. Mineralized material with an AuCN:AuFA of less than 65%, but greater than 50%, is identified as “Met 2”. Mineralized material with an AuCN:AuFA of less than 50%, but greater than 25%, is identified as “Met 3”. Mineralized material with an AuCN:AuFA of less than 25% is identified as “Met 4” and is always classified as waste. The heap leach pad is designed such that Met 1 ore will occupy the majority of the pad, Met 2 will be placed on the outside edges, and Met 3 ores will be stockpiles and placed on top of the heap at the end of the life of mine. Barren solution or lean solution (low grade solution which will bypass the ADR) will be applied to the Met 1 areas of the pad before being routed to the ADR facility for processing. Only barren solution will be applied to the Met 2 and Met 3 areas. Under this operating philosophy, lean solution will never be exposed to the risk of potential preg-robbing from Met 2 or Met 3 materials.

A dilute cyanide solution (barren solution) will be applied to the pad for both Met 1 and Met 2/3 sections of ore. Process solution from the heap leach pad will report to either a rich solution tank or a lean solution tank, depending on gold grade. 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 433.3 million tonnes. The total estimated mine life is 16 years; solution application on the heap leach pad continues for an additional 4 years after mining operations have ceased to recover additional solubilized metal ounces. The nominal ore placement rate on the pad is an average of 27.1 million tonnes per annum, equivalent to 74,200 tonnes per day.

A back-up diesel generator system is included in the design to allow for pumping of a portion of the barren solution to the pad in the event of a power outage. Two of the barren solution pumps, as well as one of the rich solution pumps, will be electrically connected to the bus that sustains the critical loads (IE: the bus that can be supplied by line power or generated power).

17.1 GOLD AND SILVER RECOVERIES

The gold recoveries for heap leaching of the Black Pine ore are sourced from the recommendations detailed in Section 13 of this Technical Report.

The overall life-of-mine average gold recovery is estimated at 70.2 percent.

17.2 REAGENTS AND CONSUMPTIONS

The major reagent consumptions for heap leaching of Black Pine ore have been sourced from metallurgical test results from column leach tests on crushed material. Scaling factors for reagent consumption were used as per industry standards in order to account for the difference between crushed and ROM material particle sizes. It should be noted that all reagent additions are tied to ore placement rate, metal production rate, or solution application rates to accurately represent the expected variability in the consumption and associated cost.

17.2.1 Sodium Cyanide

Sodium cyanide (NaCN) will be used in the leaching process and will be delivered in either tanker trucks as a liquid at 30% concentration by weight (1.16 SG) or in briquettes in Iso-Containers. When delivered in dry briquette form, solution will be circulated through the Iso-Containers to dissolve the cyanide to 30% strength. The dissolution of cyanide will be conducted with a mixing tank and pump. Following dissolution of the cyanide briquettes, the sodium cyanide solution will be transferred to a 30,000-gallon steel tank at the ADR area within a 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 have double-containment, either by “pipe-within-pipe” or “pipe-overliner” containment systems. Cyanide consumption is detailed in Section 13.14.

17.2.2 Lime

Pebble quicklime (CaO) will be used to treat the ROM ore prior to cyanide leaching to maintain an alkaline pH. Lime is anticipated to be delivered in bulk by 20-ton trucks and-loaded pneumatically into two 200-ton storage silos. Variable speed feeders will meter lime directly onto the ore being carried by haul trucks to the heap leach pad. It will be added in proportion to the tonnage of ore in each truck. Lime consumption is detailed in Section 13.14.

17.2.3 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.

17.2.4 Sodium Hydroxide (Caustic)

Sodium hydroxide (caustic) will be delivered to site as a liquid at 50% caustic by weight (1.53 SG) and diluted upon arrival to 20% strength to minimize the risk of freezing. Liquid caustic will be stored in a 15,000-gal steel tank and metered to the strip solution tank, acid wash circuits, lean and barren circuits as needed by a distribution pump with individual control valve stations at each point of use.

17.2.5 Hydrochloric Acid

Hydrochloric acid will be used in the acid wash section of the elution circuit prior to desorption. Hydrochloric acid will be delivered to site as a liquid at 30% solution strength and diluted to 3% in a dilute acid tank. Acid washing consists of circulating a dilute acid solution through the bed of carbon to dissolve and remove inorganic scale from the carbon. Carbon acid washing will be done before each desorption cycle, or as required, to maintain carbon activity levels.

17.2.6 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 kg per kg of gold and silver produced.

17.2.7 Antiscalant

Antiscalant will be used to prevent the build-up of scale on carbon, in the process piping, and on pumps. 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-gal (1 m^3) totes or as preferred by the selected vendor.

Antiscalant will be added directly from the supplier tote bins into the rich, 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.

17.3 PROCESS FLOWSHEET

An overall process flowsheet for the Project is presented in Figure 17-1.

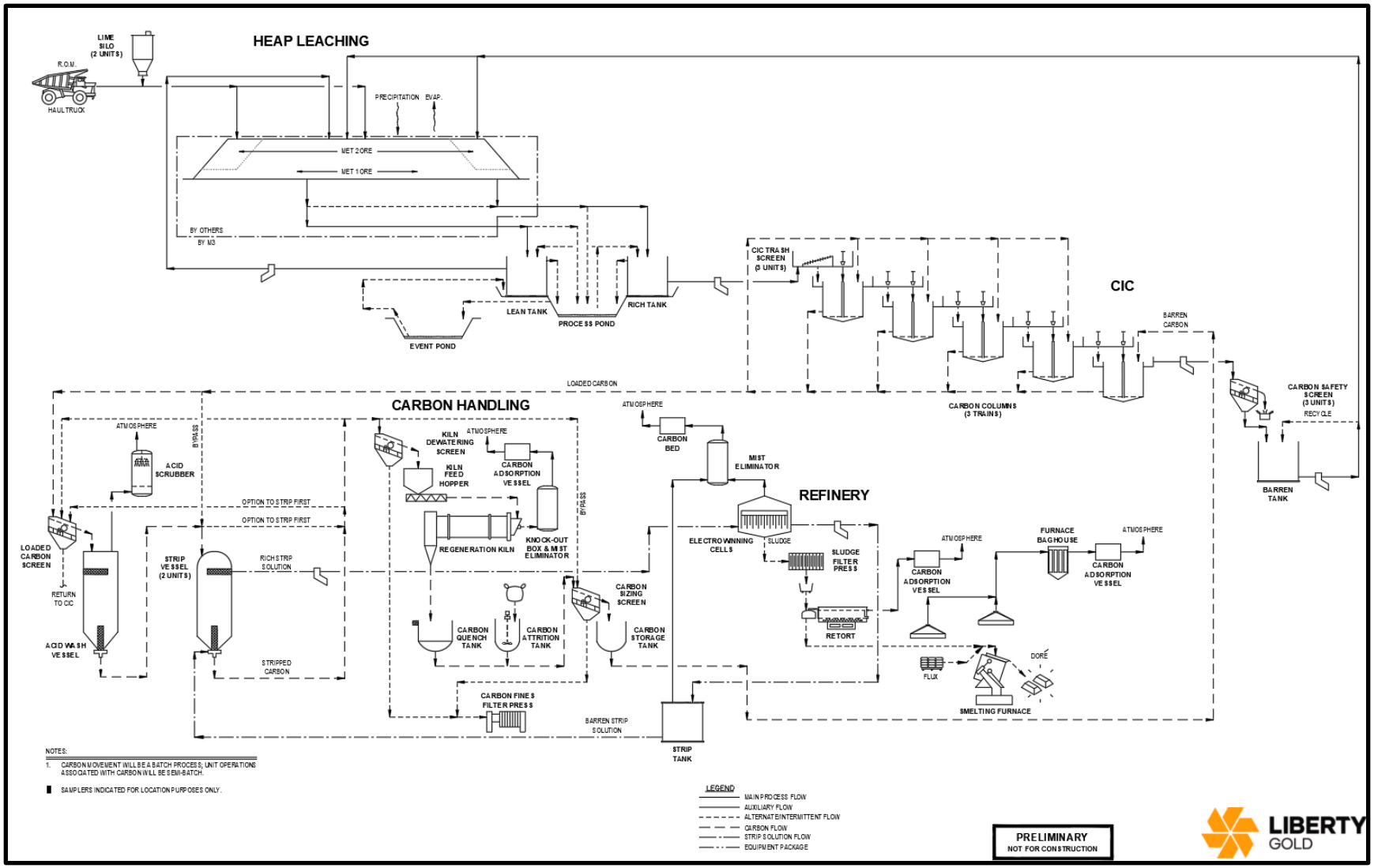


Figure 17-1: Process Flow Diagram for the Black Pine Project

17.4 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 approximately 74,200 tonnes/day. ROM production and stacking rates will vary based on ore availability from the pits.

Quicklime (CaO) will be used for pH control within the heap leach facility. Pebble quicklime will be stored in two 200-ton silos which will be equipped with variable speed feed system that will feed a clamshell gate for lime addition to the trucks. The lime silos will be installed athaul road locations along the route to the heap leach facility. Lime will be added in proportion to the tonnage of ore being hauled.

The ore haul trucks will operate on top of the current 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 a dozer prior to leaching. 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 compacted sections.

Following stacking, the ore will be drip irrigated with dilute cyanide solution and the resulting gold-bearing solutions will be collected in either the rich or lean solution tanks. The leach pad will be a multiple-lift, single-use pad.

Lifts on the leach pad will generally be 50 foot tall with the exception of the first lift for each new phase which will be 20 ft tall to protect liner integrity, and occasional “leveling lifts” as required.

17.5 LEACHING AND SOLUTION HANDLING

After each leach cell has been stacked and ripped, the irrigation system will be installed. Primarily dripline emitters will be used to apply a dilute cyanide solution at an application rate of 0.003 – 0.005 gpm/ft^2 for ROM ore. 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 and trim addition of NaOH to the barren solution as required. Barren solution will be delivered from the barren tank located at the recovery plant by high-flow high-head pumps at a nominal flow rate of 15,000 gpm (expected range: 15,000-18,000 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 applied to the material by drip emitters. Barren solution may occasionally be applied using misters, sprinklers, or wobblers as needed to leach sides slopes or control water balance.

Solution passing through the heap will dissolve the soluble metals contained therein, travel through the heap leach pad, and be collected in a network of perforated solution collection pipes which feed to a common discharge point within each drainage zone at the base of the heap. The solution will then be carried by gravity to either the rich or lean solution tanks. The determination of rich versus lean will be made via periodic, manual sampling of the solution grade from each solution collection section of the pad. The rich solution will be routed to the process plant for gold recovery and reagent reconditioning, primarily consisting of increasing the cyanide concentration. The lean solution will be reconditioned with cyanide as required, as then routed back to the pad for further leaching. Solution that reports to the lean solution tank can be re-applied to the heap leach pad, but only directly above Met 1 areas. No lean solution can be applied to the Met 2/3 ore. Once an area has been leached for the target time or targeted metal recovery, the next lift will be placed on top of the already leached ore and the process repeated.

In the event that excess solution is produced from the heap, either tank can overflow to the operations pond. Flows in excess of the operations pond capacity will overflow into the event pond via a spillway between the two ponds. Once the overflow condition is resolved, the event pond solution can be pumped back to the operations pond; the operations pond solution may be pumped back to the either the rich solution tank or the lean solution tank.

17.6 LEACH PAD PHASING AND CONSTRUCTION

It is planned that the leach pad will be constructed in three phases. The capacities and construction methods are further detailed in Section 18.

17.7 ADR PLANT

The recovery plant at Black Pine has been designed to recover gold and silver using an adsorption-desorption-recovery (ADR) process. Rich solution from the heap leach pad will be pumped to the CIC and adsorbed onto activated carbon (adsorption). The carbon adsorption circuit consists of three trains of cascade-style columns. 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). Prior to elution, each batch of carbon may be acid washed to remove any scale and other inorganic contaminants that might inhibit gold adsorption on carbon when that carbon is put back into the CIC circuit. Note that acid washing is not a requirement for 100% of the carbon and operations may elect to acid wash as required. All or a portion of the carbon will be thermally reactivated using a rotary kiln to increase the adsorption activity of the carbon in CIC.

The ADR plant General Arrangement is presented in Figure 17-2.

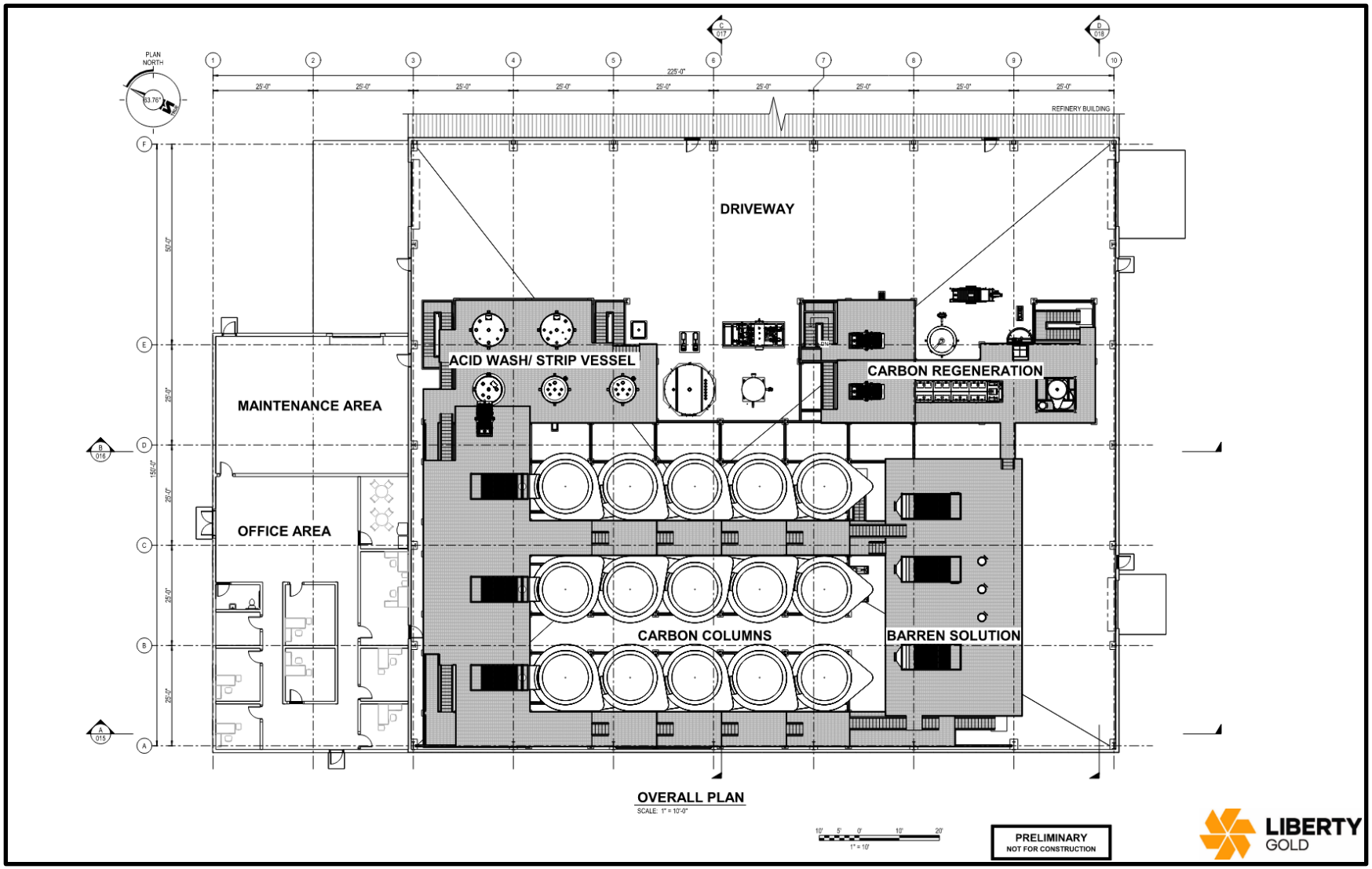


Figure 17-2: ADR Recovery Plant General Arrangement

17.7.1 Adsorption

Adsorption of gold onto carbon will take place 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, for a total of 15 columns. Each of the carbon columns are nominally 15 ft in diameter and are sized to hold 8 tonnes (8.8 tons) of activated carbon. As part of the study, alternative column technologies, namely vertical carbon in column (VCIC), were evaluated as an opportunity to improve the process as well as the project economics. The results of the tradeoff study concluded that cascading CICs were preferable due to being proven at the optimal carbon capacities.

Three trains of carbon columns are included in the design, to allow for operational flexibility with the rich solution flow rate range. Essentially one train will have the ability to process between 3,000 to 6,000 gpm of solution with suitable carbon fluidization. The flow rate range will be from 3,000 gpm (via a single train) to 18,000 gpm (with all three trains). As a secondary benefit, the 15 ft diameter of the columns will be within the transportation shipping envelope. Barren solution exiting the last carbon adsorption column in the train will flow through a vibrating screen to separate any fugitive carbon from the solution. The screened solution will flow by gravity into the barren tank, where make up cyanide is added; caustic solution may also be added to the barren tank to adjust pH, if required. Though not expected, any overflow from the barren tank will report to the ADR building sump; if the ADR building sump experiences an upset condition, the excess solution will be pumped to the process pond by the ADR building sump pumping system.

Magnetic flowmeters equipped with totalizers will measure solution flow to the adsorption circuit. Rich solution will flow by gravity through each set of five columns in series, exiting the lowest column as barren solution. Rich and barren solution continuous samplers will be installed at the feed and discharge end of each carbon column train, respectively. Composite and an instantaneous solution samples will be utilized to measure rich and barren solution gold and silver concentrations, as well as obtain other information, such as solution chemistry, alkalinity, pH, and cyanide concentration.

Adsorption of gold and silver from rich leach solution 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.

17.7.2 Carbon Acid Wash

Loaded carbon transferred from the CIC circuit will pass through a 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 dilute acid tank, and circulation will be established between the acid wash vessel and the dilute acid tank. Completion of the cycle is confirmed when the pH stabilizes between 1.0 and 2.0 without acid addition for a minimum of thirty minutes of circulation.

Dilute acid in the acid wash column will be returned to the dilute acid tank to the greatest practical extent. The carbon will then be rinsed with raw water followed by rinsing with dilute caustic solution to remove or neutralize residual acid on the carbon. A pH probe will be included in the circulation line from the acid wash column on the overflow side of the column. The sustained pH reading on this probe is representative of the pH inside the acid wash column. As part of the design, the control system requires a sustained high pH permissive, 10 or higher, prior to transferring the acid-washed carbon. This will significantly reduce the potential of inadvertent transfer of acidic carbon outside of the acid wash circuit.

The acid wash system has been designed for 100% acid washing of the carbon; however, in operation, acid washing can be bypassed and the circuit can be operated as required. The total time required for acid washing a batch of carbon will be approximately six hours.

17.7.3 Desorption

A pressure Zadra hot caustic desorption circuit for the stripping of gold and silver from carbon has been selected for Black Pine, which requires 12 hours or less to complete a cycle. During the elution cycle, metals loaded on the carbon, including gold, silver, and other metals that dissolved in the heap leach pad, will be continuously extracted by electrowinning from the rich eluate concurrently with desorption. The stripping of carbon will occur once per day, on average. If required by the operation, multiple strips can occur per day.

Two elution columns are included in the design; though, only one is required to sustain the expected production. The second column can be utilized to expedite the operation time by pre-loading carbon or, alternatively, the columns can also be operated in series. This will allow additional metal ounces to be stripped during periods of elevated metal production. The process descriptions that follow are based on a single elution column; nonetheless, the second elution column could be incorporated as indicated.

The desorption circuit is sized to strip loaded metals from carbon in 8 tonne (8.8 ton) batches. The circuit will be equipped with a strip solution tank, strip solution pump, primary (heat up), secondary (heat recovery), tertiary (cooling), and trim (upset condition) heat exchangers, boiler, two elution columns, elution column drain pumps, and a water softener. After carbon has been transferred to the elution column, barren strip solution (eluant) containing sodium hydroxide and sodium cyanide (if required) 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 rich eluate leaving the elution column. The barren eluant solution then passes through the primary heat exchanger to raise the temperature up to 300°F. An electric boiler provides the heating requirements for the desorption process. The boiler will heat the hot-side solution of the primary heat exchanger to approximately 330°F. The hot-side heating solution is maintained under pressure at all times to prevent boiling.

The elution columns will contain internal stainless-steel inlet screens, commonly known as Johnson screens, to hold carbon in the column and to distribute incoming stripping solution evenly in the column. Rich eluate leaving the elution column will pass through two external stainless-steel screens before passing through the heat recovery exchanger and the cooling (trim) heat exchanger to reduce the temperature to about 175°F (to prevent boiling), as the cooled rich eluate solution will flow to the electrowinning cell, which is not pressurized. Barren solution from the CIC circuit will be utilized to supply the cold-side solution of the trim heat exchanger. Comparatively, the barren solution in the CIC circuit is several orders of magnitude higher in volumetric capacity versus the potential requirement of the trim heat exchanger. Heat transferred from the rich eluate in the trim heat exchanger will be transferred to the cooling solution, which reports back to the CIC barren solution tank. A temperature monitoring system will be included with the heat exchanger barren solution outlet to provide indication if the heat load is outside expected ranges. In the design, space will be allocated for a future chiller system for this duty, if it is required.

After desorption is complete, the stripped carbon will have the option to be transferred to the carbon regeneration circuit or to the carbon storage tank ahead of CIC. A carbon transfer pumping system will be utilized for this carbon movement.

17.7.4 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. Dissolved metals 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 return 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 discharged to the barren tank that is part of the CIC circuit or, alternatively, discharged to the head of the CIC circuit to allow soluble metals an opportunity to re-load onto the carbon and remain in the circuit. A small fraction (approximately 10%) 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 strip solution tank during fresh strip solution make-up.

The precious metal-laden cathodes in the electrolytic cells will be washed approximately once per week and processed to produce the final doré product. The cathodes will be washed in place with a pressure washer, where precipitated precious metals will form a sludge at the bottom of the EW cells. 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.

17.7.5 Carbon Handling & Thermal Regeneration

The carbon preparation and storage system will include a 1-tonne agitated carbon attrition tank, a 10-tonne regenerated carbon storage tank (oversized to accommodate minor fluctuations in operation), carbon sizing 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 make-up carbon. Carbon being transferred to the carbon storage tank will pass to a carbon fines/dewatering screen 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. The carbon fines from the filter press will be stored in bulk bags for removal from the system. The carbon fines bags will be transported to a third-party refiner.

Fresh carbon being added to the system will first be conditioned in the carbon attrition tank before being pumped to the carbon sizing screen to remove carbon fines and then transferred to the regenerated carbon storage tank. The conditioning process is to remove weak outer edges or fragments off the carbon prior to the carbon being introduced in the circuit.

Thermal regeneration will consist of drying the carbon thoroughly and heating it to approximately 1,300ºF for ten minutes to maintain elevated carbon activity levels, which are reduced by organic contaminants in the process. The carbon regeneration circuit has been designed to regenerate 100% of the carbon; however, regeneration of carbon could be limited to once every three strips.

Carbon from the elution circuit to be thermally reactivated will be dewatered on a vibrating 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 sizing screen and regenerated carbon storage tank.

17.7.6 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 sludge trays (also known as boats) and heated in the retort oven. After closing the retort door, temperature gradually is increased to and sustained at discrete levels, eventually up to and sustained at 1,100°F for a minimum of 6 hours. This promotes water removal via drying and mercury removal via volatilization. The temperature profile follows a similar gradual cool down period until the retort oven temperature allows for safe opening of the door for boat removal.

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 periodically transferred to and stored in flasks. Cooled, mercury-depleted vapor leaving the trap will pass through a sulfur-impregnated carbon bed to further remove any fugitive mercury. A portable, high sensitivity mercury vapor analyzer (commonly referred to as a Jerome) will be utilized in the refinery to detect if there are any fugitive mercury vapors in the plant.

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 2,200°F. When the furnace charge is fully molten, it separates into two distinct layers: 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. The slag will be crushed and reprocessed to keep as much of the precious metal in the refinery as possible and placed back in the furnace. Processed slag and spent refractory material from the furnace will be transported offsite to a third-party refiner.

17.7.7 Mercury Abatement System

In addition to the mercury retort, which has a dedicated sulfur impregnated carbon bed and other mercury abatement components described earlier in this section, the ADR & refinery facility will be fitted with an exhaust gas handling system to treat mercury emissions from the various thermal pieces of equipment. The exhaust system will be designed to combine mercury-containing exhaust streams and treat them in three separate sulfur-impregnated carbon beds prior to discharge to the atmosphere.

The first carbon bed will be dedicated to treating exhaust from the smelting furnace. The smelting furnace will be fitted with a hood which will collect the exhaust and direct it to a high temperature baghouse, which will remove suspended particles from the gas and cool the gas before passing through the carbon bed. The carbon bed will collect residual entrained mercury vapor before exhausting the gas to atmosphere.

The second carbon bed will treat the exhaust gas streams from the electrowinning cells and the eluant solution storage tank. The gas will pass through a mist eliminator and duct heater (nominally 140°F) to preclude condensation from impacting the carbon bed, which will collect residual entrained mercury vapor before exhausting the gas to atmosphere.

The third carbon bed will treat the exhaust gas streams from the regeneration kiln. The gas will pass through a steam condenser, a mist eliminator, and a duct heater (nominally 140°F) to preclude condensation from impacting the carbon bed, which will collect residual entrained mercury vapor before exhausting the gas to atmosphere.

17.8 ADR REAGENTS AND UTILITIES

Recovery plant reagents will include cyanide, caustic, hydrochloric acid, antiscalant, activated carbon, and various furnace fluxes including silica sand, borax, and sodium carbonate (soda ash).

Well water (raw) is utilized for hose stations, though usage of raw water will need to be monitored as to reduce potential of sustaining the plant water balance in a surplus condition. Raw water will be the source water for the treated water system. The treated water system will be a water softener, which primarily replaces scale forming calcium ions with sodium ions. Treated water is utilized in the strip/elution circuit.

Potable water will be utilized for eyewash and safety showers. The water for the eyewash and safety showers will circulate through a tepid water loop. This will ensure that the temperature of the safety systems is not on either extreme where it could create a scalding or freezing hazard.

Barren solution from the CIC circuit will be utilized to support carbon transfers in the circuit. Usage of barren solution for carbon transfer does not add load to the circuit water balance as the solution is internal to the process; that is, it is not adding to the circuit as is the usage of raw water.

17.9 LABORATORY FACILITIES

Analytical support for ore control and process operations including cyanide soluble assays, fire assays, preg rob assays, full carbon and sulfur speciation by LECO, XRF, and multi-element by ICP-OES will be conducted on site using a dedicated laboratory. Metallurgical testing required to support the Project operations including bottle rolls, column testing, screen analysis, and other tests will also be conducted on-site.

A Build-Own-Operate-Transfer ("BOOT") delivery model has been selected for the assay laboratory facility to support early availability of reliable assay services for mine development and grade control. Under this model, a qualified laboratory service provider will design, construct, commission, own, and operate the facility for an agreed term before transferring ownership to the Company in accordance with defined contractual requirements. This approach is expected to accelerate delivery, leverage specialized laboratory expertise, reduce initial capital requirements and hiring demands on the project team, and provide clear accountability for laboratory performance. The BOOT agreement should establish appropriate requirements for analytical quality, turnaround times, capacity, accreditation, data security, operating costs, maintenance, and the condition of the facility at transfer.

18 PROJECT INFRASTRUCTURE

The infrastructure planned for the Black Pine Project has been developed to support the construction, operation, and maintenance of the proposed mining and heap-leach facilities and to provide the utilities, transportation access, water-management systems, and ancillary services required for safe and efficient operation. Major infrastructure components include the Project access road, electrical power supply and distribution systems, communications systems, heap-leach pad, process plant, and ancillary buildings supporting administration, security, warehousing, equipment maintenance, and personnel services. Site water infrastructure, including water-supply systems, storage tanks, pipelines, ponds, drainage controls, and surface-water diversions, is described in greater detail later in this section. Haul roads within the mining area and the mine rock storage facilities are described separately in Section 16. The overall arrangement of the proposed Project infrastructure is shown in Figure 18-1.

18.1 ACCESS ROAD

The primary site access for the Black Pine Project will utilize existing Interstate Highway-84 (I-84), with traffic from the south utilizing the existing interchange and overpass near the UT/ID state line for Hwy 30. The existing interchange at Juniper is anticipated to be utilized for most traffic coming from the north. From Highway-84 the route will continue due South along BLM Road 587 for approximately 6 miles to the security building located near the intersection of BLM road 601 (Black Pine Road) and BLM road 587.

The route from the Juniper interchange will be improved to a standard two-way road section consisting of a 12-ft-wide lane and 6-ft shoulder in each direction. The shoulders will provide an area for any safety and drainage structures that will be needed along the route. All traffic regardless of access direction will be required to check in at the security office before being allowed onto the site.

18.2 POWER & COMMUNICATIONS SUPPLY

Electrical utility service at the site is currently available. Electrical power will be supplied by Raft River Rural Electric Co-op, Inc. and transmitted to the Project via existing transmission and power lines capable of delivering up to 10.58 MW. Minor improvements include substation and transmission line upgrades from the Curlew substation on Utah State Highway 30, located southeast of the Project, 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.

Fiber-optic connectivity will be extended to the Project site to provide a reliable, high-capacity communications backbone for the information technology (IT), operational technology (OT), and site-wide communications networks. The infrastructure will support business systems, process control and monitoring, laboratory data transfer, security, voice and video communications, and other critical operational applications. Appropriate network redundancy, cybersecurity controls, and backup communications systems will be incorporated to enhance resilience and maintain continuity of operations.

18.3 PROJECT BUILDINGS

Project buildings will include site built and prefabricated modular buildings required to support mine operations, equipment maintenance, administration, security, warehousing, and personnel services. The facilities will be located within designated operational areas to provide efficient access to the open-pit mining and site-support infrastructure while maintaining appropriate separation between heavy-equipment traffic, light vehicles, and pedestrian activities. Building designs will reflect their intended function, anticipated occupancy, equipment requirements, and the environmental conditions at the Project site. The principal facilities will include the gold recovery facility, laboratory, mine maintenance complex, warehouse and material-storage areas, security and access-control facilities, administrative and personnel-support spaces, and other ancillary structures necessary for safe and efficient operation of the operation.

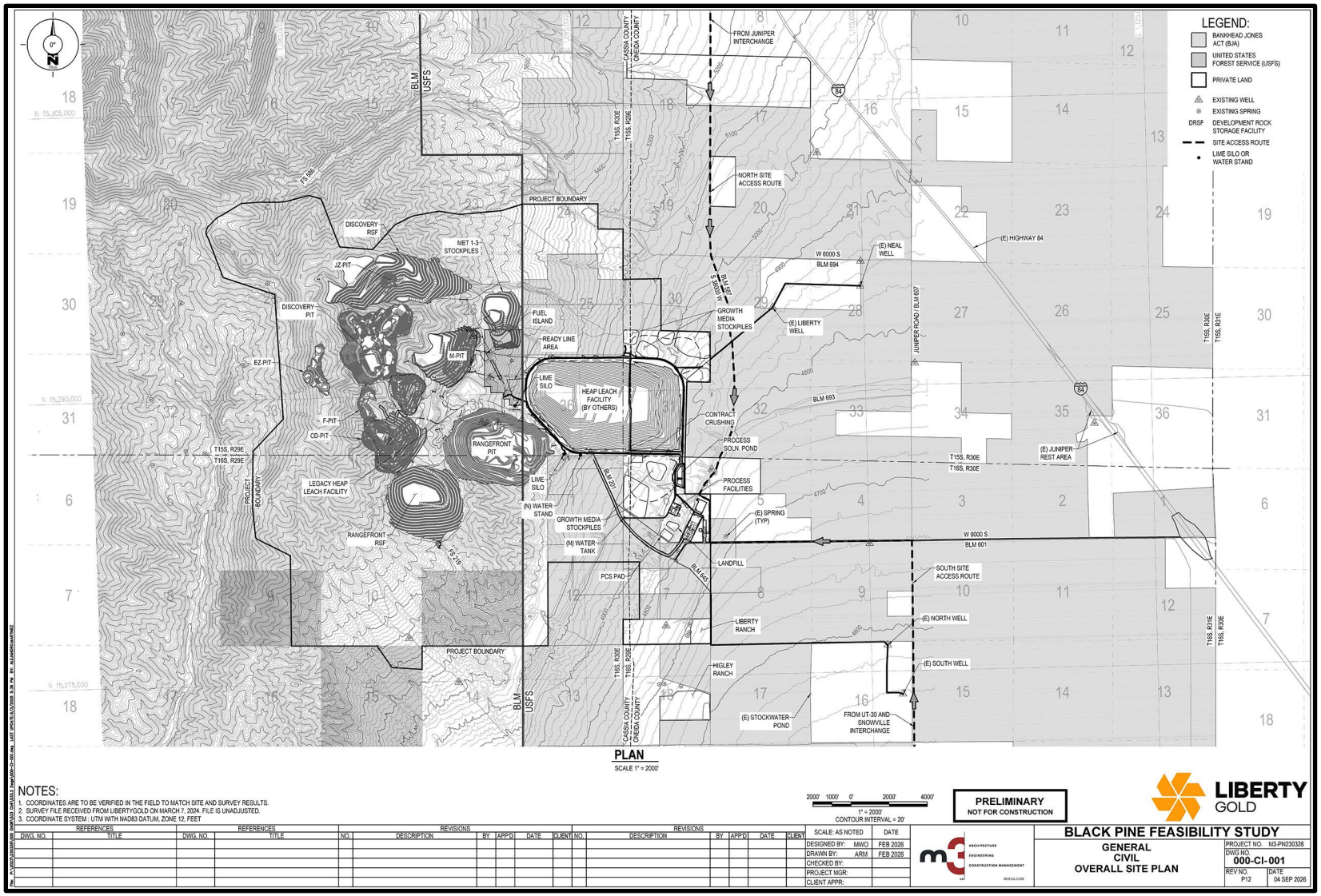


Figure 18-1: Site Plan Drawing

18.3.1 Security Building at Access Gate

The security building is located at the controlled entrance to the project site and is configured to support access control, safety coordination, and the monitoring of personnel and vehicle movements. The approximately 28 ft by 24 ft building provides about 670 ft² of enclosed space and includes a security and safety office, entrance lobby, storage room, and unisex restroom. The office is positioned to provide security personnel with visibility toward the adjoining entrance facilities and controlled traffic areas. Site access features include chain-link perimeter fencing, an entry and exit pedestrian turnstile security gate, and vehicle barrier gates for controlling inbound and outbound traffic. A truck scale is located adjacent to the security building for weighing commercial and mine-service vehicles entering or leaving the property. Exterior stairs and sloped access routes provide entry to the elevated building floor, while the compact arrangement centralizes security, visitor processing, safety support, and vehicle-control functions at the principal site entrance.

18.3.2 Administration Building

The administrative office building is an approximately 156 ft by 60 ft pre-engineered modular building providing about 9,400 ft² of space for management, engineering, technical, and general administrative personnel. The facility is organized around a central reception area and interconnected corridors, with approximately 25 enclosed offices distributed around the building perimeter and several open work areas located within the interior. Shared personnel and meeting facilities include a conference room, breakroom, women's and men's restrooms, and a mother's room. Technical and operational support spaces include an engineering records room, plotter room, server room, general storage room, dedicated blasting storage room, and janitorial room. Multiple exterior entrances and exits are provided along the building perimeter to support convenient personnel access and emergency egress. The overall arrangement consolidates the principal administrative and technical functions required to support operations while providing dedicated areas for meetings, document management, communications equipment, employee services, and routine office support.

18.3.3 Mine Operations Building

The mine operations facility is a combined mine-dry, office space, emergency-response, and personnel-support building with an overall length of approximately 138 ft providing 7,062 ft² of space. The facility includes separate men's and women's mine-dry areas arranged to provide controlled transitions through dirty vestibules, changing and locker areas, shower facilities, and clean vestibules, thereby supporting the separation of work clothing and clean personal areas. Office accommodations include four enclosed offices, a server room, two restrooms, and a janitorial room, with an internal corridor providing access between the office and additional workspaces. A large muster room is provided for shift assembly, safety meetings, crew briefings, and emergency accountability, with an adjacent storage room for operational or emergency-response supplies. A portion of the facility incorporates a dedicated ambulance and fire-truck bay with separate overhead vehicle doors and direct exterior access, allowing emergency-response vehicles to be housed near mine personnel and dispatched promptly when required. Multiple exterior entrances and exits provide circulation to the administrative, mine-dry, muster, and emergency-response areas and support efficient personnel movement during shift changes and emergency events.

18.3.4 Truck Shop Building

The proposed mine maintenance facility is a combined heavy-equipment truck shop, warehouse, maintenance-support, and personnel-services building measuring approximately 332 ft long by 102 ft wide (33,900 ft^2). The truck-shop portion is configured to accommodate 240 tonne-class haul trucks, including a maintenance position allowing the truck bed to be raised, together with a separate tracked equipment work position and a welding bay equipped with removable partitions. Heavy-maintenance features include overhead cranes, embedded floor rails, hose-reel stations, bollards, and adjacent mechanics' tool and crew line-out areas. A dedicated light-vehicle maintenance bay is provided between

the heavy-equipment shop and the warehouse. The warehouse includes internal storage racks, a warehouse technician area, direct access to a covered exterior laydown area, and an adjoining maintenance-fluid storage area with bulk storage tanks and a distribution cable tray. Personnel and administrative accommodations include offices, a conference and training room, breakroom, storage and janitorial spaces, men's and women's restrooms, and change rooms with lockers. The general arrangement also reserves an area adjacent to the truck shop for future expansion, allowing the facility's maintenance capacity to be increased as future operating requirements warrant.

18.3.5 Truck Wash Building

The truck wash facility is designed to accommodate mine haulage equipment up to the size of a 240-tonne haul truck within a dedicated, drive-through wash bay measuring approximately 50 ft by 80 ft (4,000 ft^2). The bay is accessed through 30-ft-wide by 30-ft-high roll-up doors and incorporates water cannons, sloped wash-down surfaces, and drainage features to collect wash water. An adjoining service area houses the truck-wash water tank, overflow weir, skimmer, and associated water-handling equipment to support solids separation and water recovery.

18.3.6 ADR Plant

The ADR (adsorption, desorption and recovery) Plant will be located directly to the northwest of the Administrative Facilities. Rich leach solution from the Heap Leach Pad will be processed in the 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 Plant includes an open CIC circuit consisting of three carbon column trains operated in parallel inside a 17,000 ft^2 insulated, engineered steel walled building with an overall height of 45 ft. The building will contain the desorption, acid wash, carbon handling and regeneration circuits, as well as an office, break/lunchroom, and men's and women's locker/bathroom facilities. The ADR Plant also includes an attached 7,500 ft^2 insulated, engineered steel walled refinery building with an overall height of 25 ft 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 the 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.

18.3.7 Laboratory

The Project laboratory is an approximately 72 ft by 60 ft (4,300 ft^2) facility designed to support metallurgical testing, sample preparation, and analytical activities required for mine and process operations. The building includes a sample-receiving area with direct exterior access, a dedicated sample-preparation room, metallurgical laboratory, wet-assay laboratory, and exterior test-column area. Laboratory support systems include a dust collector serving the sample-preparation area and a wet scrubber and acid-holding tank associated with the wet-assay functions. Additional support spaces include a mechanical equipment room, server room, reagent and supplies storage room, offices for the laboratory supervisor, metallurgist, and other staff, a breakroom, and separate restrooms. The arrangement provides separation between sample receipt and preparation, metallurgical testing, wet-chemistry activities, equipment and chemical-storage areas, and personnel accommodations, supporting safe and efficient movement of samples through the laboratory workflow.

A separate assay laboratory will be located adjacent to the metallurgical laboratory and will be provided by a qualified commercial laboratory vendor under a BOOT arrangement. Under this delivery model, the vendor will be responsible for equipment installation, owning, staffing, operating, and maintaining the assay facility and its specialized analytical equipment throughout the agreed service period. The Owner will supply the building space to house the vendor supplied equipment. The facility is anticipated to provide routine analytical services for mine-grade control, process monitoring, metallurgical accounting, and other operational sampling requirements, with procedures for sample custody, preparation, analysis, quality assurance and quality control, and reporting established in accordance with the

applicable service agreement and recognized laboratory practices. Locating the vendor-operated assay laboratory adjacent to the Project laboratory will facilitate sample transfer, communication between laboratory and operating personnel, and timely delivery of analytical results while reducing the Project's requirements for specialized assay-laboratory staffing, equipment ownership, and maintenance. The assay lab under a BOOT model is anticipated to be available early in the project period to support early operations.

18.4 SITE-WIDE STORM WATER MANAGEMENT

The storm water management strategy for project infrastructure entails minimizing contact water volumes by separating non-contact water surface flows from process solution sources (contact water). In addition, sediment-laden water due to runoff from exposed stockpiles or other sources will be prevented from discharging directly to downstream drainages. The storm water management strategy includes the following:

- Contact water: 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.
- Non-contact water: Direct precipitation from areas outside these process solution sources will be directed away from and downstream of contact areas after appropriate sediment controls.

18.4.1 Storm Water Design Methodology

Design storms were determined using NOAA Atlas 14 Precipitation Frequency Estimates. Design storm depths and freeboard were specified according to Table 18-1, and were designed to meet or exceed Idaho regulatory requirements for all process water components.

Table 18-1: Design Storms and Freeboard Requirements for Storm Water Controls

Infrastructure	Design Storm	Design Storm Depth (in)	Freeboard (ft)
Non-Contact Water Channels - Pits*	100-year, 24-hour	2.62	1
Non-Contact Water Channels - Mine Development Rock	100-year, 24-hour	2.62	1
Non-Contact Water Channels - Stockpiles	100-year, 24-hour	2.62	1
Culverts	100-year, 24-hour	2.62	N/A
Contact Water Channels	100-year, 24-hour	2.62	2
Sediment Ponds	Storage: 2-year, 24-hour Spillway Conveyance: 100-year, 24-hour	1.27 2.62	2 2
	Sediment: 2-year, 24-hour		
Contact Water Channels	100-year, 24-hour plus snowmelt	3.55	2
HLF Pond (Process Component)	100-year, 24-hour plus snowmelt	3.55	2

Note:
*Pits were originally anticipated to be sized for the 2-year, 24-hour design storm; however, several pits will be backfilled with development rock and all pits were conservatively sized for the larger design storm.

18.4.2 Non-Contact Storm Water

Non-contact storm water management infrastructure includes the following:

- Culverts where haul roads cross natural drainages;
- V-ditches along haul roads in specific locations where water can be directed along haul roads;

- Storm water diversions around the upgradient sides of the open pits and stockpiles to direct upland surface water runoff away from mining activities;
- Sediment ponds downstream of development rock storage facilities and ore stockpiles to prevent discharge of sediment-laden water off-site;
- Storm water diversions around the HLF to direct non-contact storm water runoff away from the containment area.

Sediment controls were not developed for topsoil stockpiles, assuming these features will be revegetated and any short-term sediment runoff will be managed with best management practices and standard operating procedures developed at a later time.

Where diversions are not planned around the entire pit perimeter, water is expected to infiltrate into the pit rock or evaporate within a reasonable timeframe according to observations of the existing historic pits. This limited storm water runoff reporting to pits during operation should not cause appreciable delays in pit operations. The pit operations Ground Control Management Plan will define practices and equipment necessary to remove storm water from the pits prior to and during operations, as necessary.

Figure 18-2 shows the conceptual non-contact water storm water management plan at the end of the operating mine life. The storm water controls presented will be developed systematically throughout operations to minimize effects to mine property and the surrounding environment.

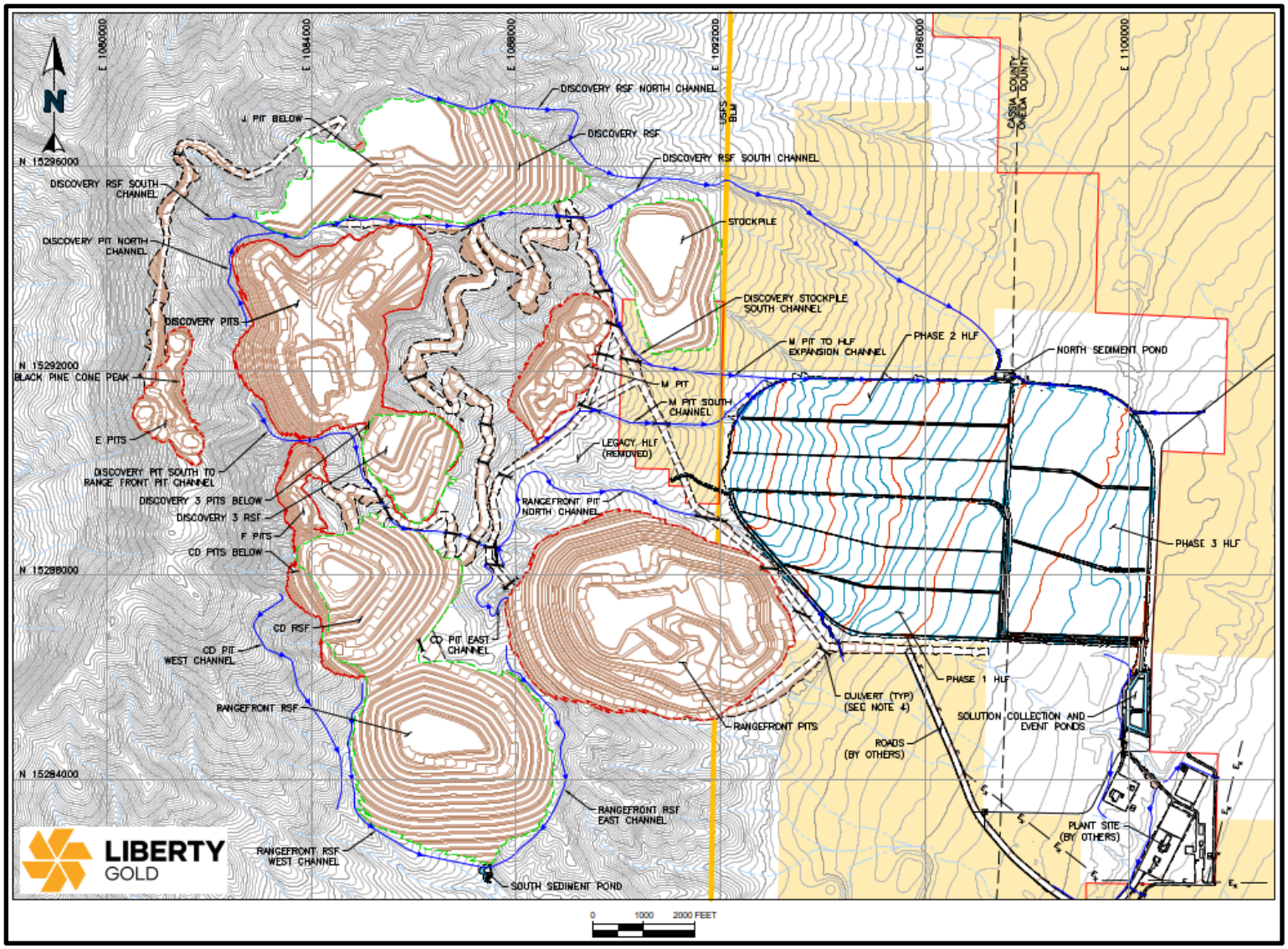


Figure 18-2: Storm Water Management Plan

18.4.3 Contact Storm Water

Contact storm water management infrastructure includes any component that may come in contact with process solution, including the following:

- Internal channels capturing stormwater runoff from the heap and conveying it to the process pond;
- The Process pond and associated process solution draindown tank system;
- The Spillway and event pond for excess storm water overflowing from the process pond;
- Any storm water infiltrating through the closure cover on the LHLF.

To meet Idaho State regulations, all components containing process solutions were sized to contain a 100-year, 24-hour design storm event plus snowmelt, while maintaining 2 ft of freeboard.

18.5 HEAP LEACH FACILITY

The HLF will be constructed in three phases to contain up to 510 million dry tonnes (approximately 560 million dry tons) of leachable material, an estimated 45 million tonnes more than the total estimated 465 million tonnes needed, consisting of 433 million tonnes of leach ore and 32 million tonnes of legacy HLF material. Within each phase, the base of the HLF will be divided into separate cells to subdivide the HLF into smaller areas for solution collection pipe routing and sizing.

The HLF achieves the design requirements as stipulated by Idaho State regulations, including the following:

- An 80-mil double-sided textured HDPE geomembrane liner.
- 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.

Geotechnical investigations have confirmed that suitable low permeability materials exist in local borrow sources wholly owned by Liberty Gold.

Within each phased area, ore will generally be stacked from downhill to uphill and barren solution applied for the leaching process. Rich leachate will drain into the solution collection system installed immediately above the geomembrane liner. The solution collection system consists of a network of perforated pipes surrounded by free-draining gravel, currently expected to be sourced from legacy mine waste materials or pit development rock readily available on site. The perforated pipes will transition to solid-wall HDPE pipes that will convey process solution within the HLF to the process solution tanks located within the double-lined pond (with leak detection). Most process solution will be pumped from the rich tank to the ADR plant; process solution that does not meet minimum requirements to be processed in the ADR plant will be directed to the lean tank and pumped back to the heap for additional leaching.

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 500 ft above the geomembrane liner. Leachable material will be truck-stacked in nominal 50-ft lifts with the exception of the first lift of each new phase which will be 20-ft high. Existing infrastructure such as the land application area for the LHLF, existing powerlines, and Black Pine Mine Well (BPMW), will need to be removed or relocated prior to constructing the HLF.

18.5.1 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 design components:

- HLF containment provided by 80-mil double-sided textured HDPE geomembrane liner underlain by 12 inches of low-permeability compacted soil.
- Divider berms and solution collection channels within the HLF pad to subdivide the pad into three cells per phase, with a single primary solid-wall pipe conveying the flows from each individual cell.
- A solution collection and conveyance system designed to limit hydraulic head on the liner system to 12 inches, consisting of the following components:
    - A network of perforated 4-in diameter dual-wall corrugated polyethylene (CPe) secondary solution collection pipes at a spacing of 35 to 40 ft.
    - Primary perforated dual-wall CPe pipes, up to 36 inches in diameter, installed in engineered channels to protect against pipe crushing.
    - A 2-ft-thick layer of drainage material surrounding the primary solution collection pipes;
    - A 2-ft-thick layer of drainage material (likely sand and gravel) covering the surface of the geomembrane liner and solution collection pipe network.
    - Solid-wall HDPE solution collection header pipes, up to 36 inches in diameter, to convey flows to flow measurement flumes that will also function as sampling ports.
    - A Solution collection manifold collecting flows from each cell's main solid-wall pipe, reporting to the rich tank or lean tank.
- Solution tanks located on a tank shelf within the geomembrane-lined containment system of the process pond, connected to the solution collection manifold. From the rich solution tank, solution will be pumped to the ADR plant for processing. From the lean solution tank, solution will be pumped back to the heap for additional leaching. In the event of upset conditions, the tanks will overflow into the process pond.
- Process pond that was conservatively sized for 12 hours of power loss, at a maximum solution inflow rate of 18,000 gpm, corresponding to the maximum anticipated barren solution flowrate to the heap. During normal operations, solution from the heap will drain to the tanks and the process pond is not anticipated to have a normal operating inventory. The process pond will have a divider berm that will contain the minimum 12-hr power loss volume; excess flows will spill over the geomembrane-lined berm into the event pond. The divider berm between the two portions of the pond is designed at a lower elevation than the tank shelf, allowing flows to spill into the event pond before inundating the tank foundation.
- Event pond that was sized for an additional 12-hour solution draindown volume plus runoff from the heap resulting from the 100-year 24-hour design storm (plus snowmelt) with two ft of freeboard.

The pond system 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.
- A leak detection sump and reclaim pump for each side of the pond system (process and event ponds) to return potential seepage from the primary liner back to the process pond.
- Protective layers and a structural concrete foundation where the tanks sit on a shelf within the geomembrane-lined pond.
- Bird balls within the process pond and a wildlife fence surrounding the pond system.

Potential borrow sources with suitable engineering properties for earthworks construction, namely the low-permeability soil and drainage materials, were identified as part of project development. Low permeability soils were identified on nearby private land controlled by Liberty Gold and drainage materials will be taken from legacy mine waste materials or pit development rock. Based on the investigations and laboratory testing completed to date, borrow 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-in-thick secondary containment layer with a permeability

equivalent to 24-inches of $1x10^{-6}$ cm/s material. Additional studies may be required during detailed design to demonstrate chemical compatibility and other parameters necessary to satisfy Idaho State regulatory requirements.

18.5.2 HLF Geotechnical Evaluations

Geotechnical investigations and evaluations supporting project development have not identified fatal flaws at the selected facility locations. Based on the current data, subsurface conditions within the HLF footprint 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 phased expansions and the ultimate HLF configuration.

Data and observations from the geotechnical investigations indicate that approximately 18 inches of topsoil removal will be required in the HLF area. The remaining soils within the HLF and pond area that will be regraded to accommodate the planned infrastructure were determined to be suitable for structural fill.

Stability analyses were completed using the Slide2 software program by Rocscience. The Morgenstern-Price and Spencer methodologies for conducting limit equilibrium analysis were considered. The HLF model geometry was developed using the maximum pad grades per phase and an idealized heap stacking surface, with a maximum heap height of 500 ft and an overall slope of 3H:1V. The pond geometry was developed using the FS designs. Material properties were adapted from laboratory testing, although some testing is ongoing and results may change slightly in the future. Table 18-2 summarizes the geotechnical parameters used for the FS evaluation and Table 18-3 summarizes the slope stability model results.

Table 18-2: Slope Stability Modeling Parameters

Material	Effective Friction Angle	Effective Cohesion (psf)	Source
Ore	Shear/Normal Function, low-loose material		Cylwik et al. (2025)
Clay/Geomembrane Interface	Shear/Normal function (power curve)		Laboratory Testing
Low-Permeability Soil	31	0	Laboratory Testing
Native Ground/Subsurface	35	0	Engineer's Selection

Table 18-3: Slope Stability Model Results

Section/Case	Target Static Factor of Safety	Calculated Static Factor of Safety	Target Pseudostatic Factor of Safety	Calculated Pseudostatic* Factor of Safety
Phase 1 (Starter)	1.5	1.6	1.1	1.3
Phase 2	1.5	1.5	1.1	1.1
Phase 3	1.5	1.5	1.1	1.1

*Phase 1 and Phase 2 used the 1:500 annual exceedance probability (AEP) operational seismic event applying a peak ground acceleration (PGA) equal to 0.120 g. Phase 3 used the 1:1,000 AEP closure seismic event applying a PGA equal to 0.196 g.

NewFields estimated potential for settlement within the native subgrade, under the loading from the assumed 500-ft heap height. Drainage from the upgradient areas to the rich tank and process pond will not be impacted by settlement and the containment system can accommodate the expected strains.

18.5.3 HLF Design and Operational Safety

The HLF and associated structures have been designed to meet regulatory requirements and industry-accepted standards and practices. Designs presented are suitable for a FS. Additional investigations, evaluations, and analyses will be required during the detailed design phase to confirm assumptions and reduce the risk of encountering unforeseen conditions during construction.

The heap leach facility design presented in this Feasibility Study has been developed to a level appropriate to support the specified stacking rates, solution application rates, capital costs and operating assumptions. Phase 1 of the facility remains within the disturbance footprint identified in the Mine Plan of Operations, and the design is not intended to replace, supersede, amend or otherwise modify any active permitting submission or regulatory review.

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 through modeling or other analyses, evaluations, and calculations, that the deviations will not negatively impact facility performance.

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 protocols regarding 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 outcomes. The CAPEX includes preliminary instrumentation and monitoring systems which may include piezometers, water level indicators, laser scanners, and tiltmeters.

In addition, 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. An additional review will be completed as part of the concluding activities for the FS-design.

18.5.4 HLF Phasing

Table 18-4 summarizes the HLF phasing and estimated capacities by phase.

Table 18-4: HLF Phasing

Phase	Estimated Construction Year	Years of Operation	Contained Capacity (million tonnes)	Designed Capacity (million tonnes)
Phase 1 (Starter)	Year 0	4	79	81
Phase 2	Year 3	9	129	130
Phase 3	Year 9	16	257	278
Total			465	489

As reported in Table 18-4, the HLF has a total capacity of approximately 489 million dry tonnes (approximately 539 million dry tons) at an assumed average placed dry density of 115.6 pounds per cubic foot (pcf) and the detailed stacking plans developed for this FS. This reflects approximately 24 million dry tonnes of excess capacity, compared to the 465 million dry tonnes of material to be trucked and stacked at the HLF. An additional 21 million dry tonnes of capacity may be available within the planned liner limits, for a total excess capacity of approximately 45 million dry tonnes.

Figure 18-3 shows the Phase 1 HLF configuration with the planned delineation of Met 1 (Red) and Met 2 (Blue). Figure 18-4 shows the Ultimate (Phase 3) HLF configuration including delineation of Met 1 and Met 2. Labeling for material placement is reflective of the timing of placement related to phase, per Table 18-4, not physical placement above the liner phases.

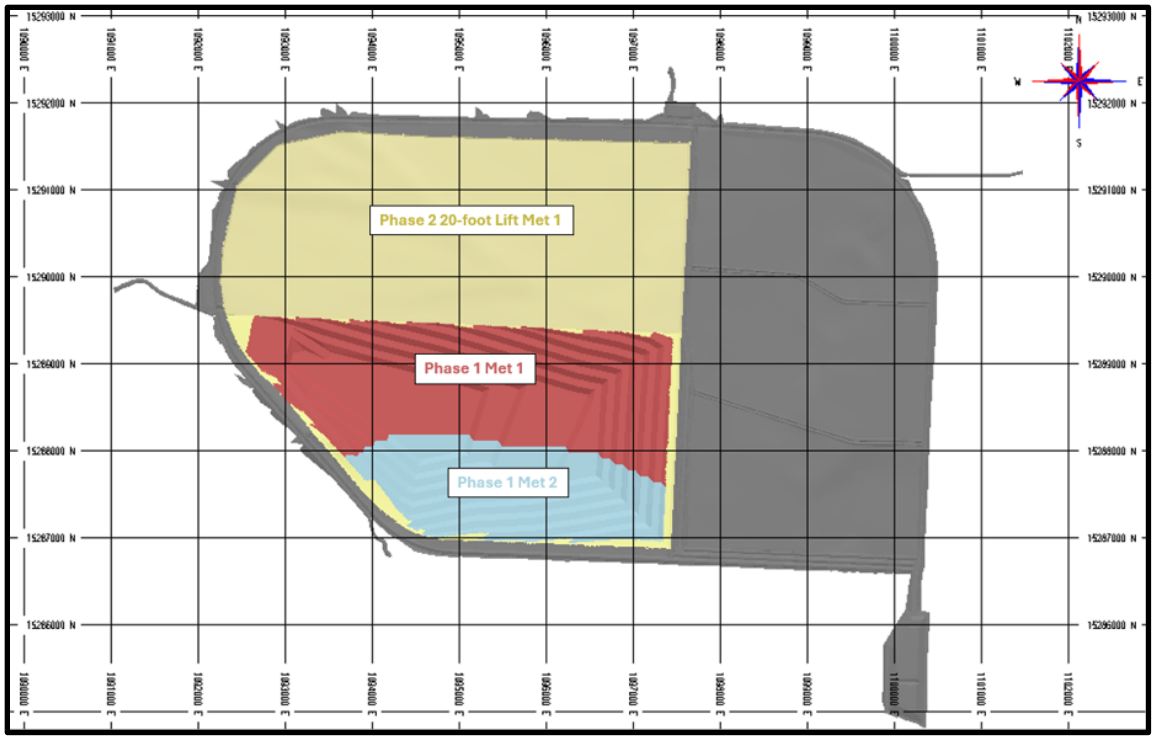


Figure 18-3: Phase 1 HLF Configuration with Phase 2 and 3 Pad Grading

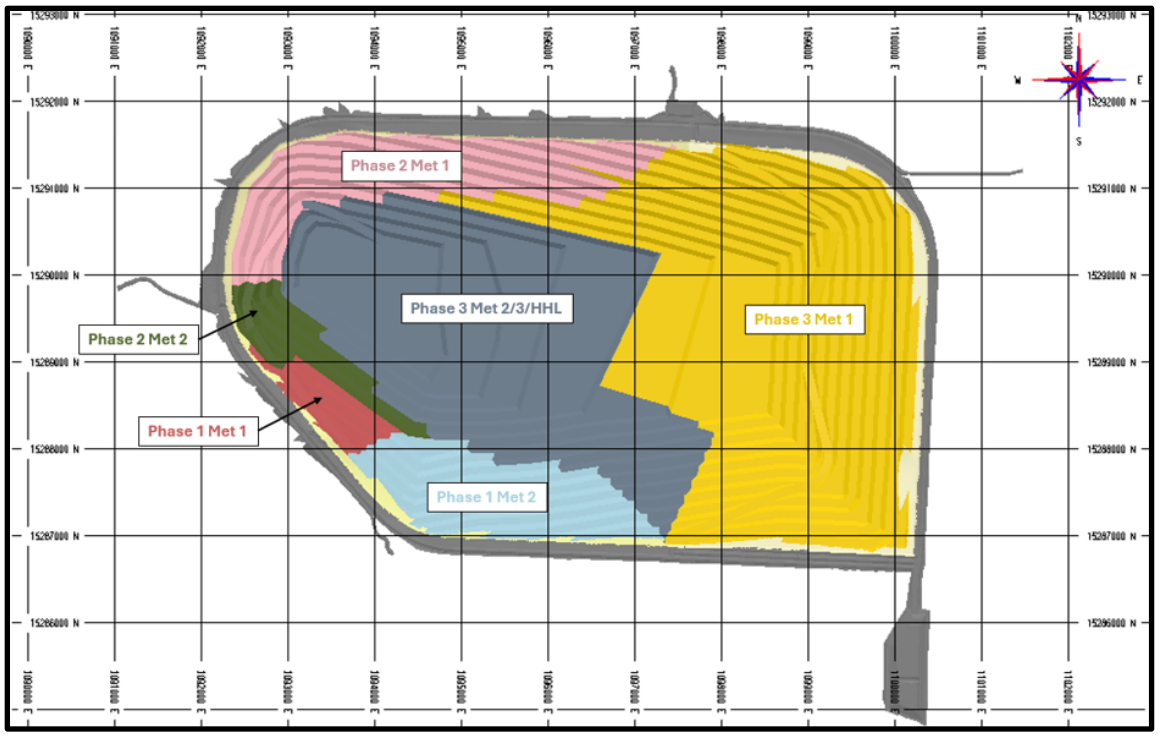


Figure 18-4: Phase 3 (Ultimate) HLF Configuration

18.6 HLF STACKING

Stacking plans were developed utilizing Hexagon MinePlan. Detailed stacking plans were developed to ensure that the planned phasing and ore placements, by ore type, allows for adequate solution application to achieve the desired leach cycles based on depth to liner. Figure 18-5 and Figure 18-6 show a few period end designs of the stacking plan during the life of the HLF. The stacking information was then input to the metal recovery model to develop the metal plan for the project including consideration for desired leach cycles for the project.

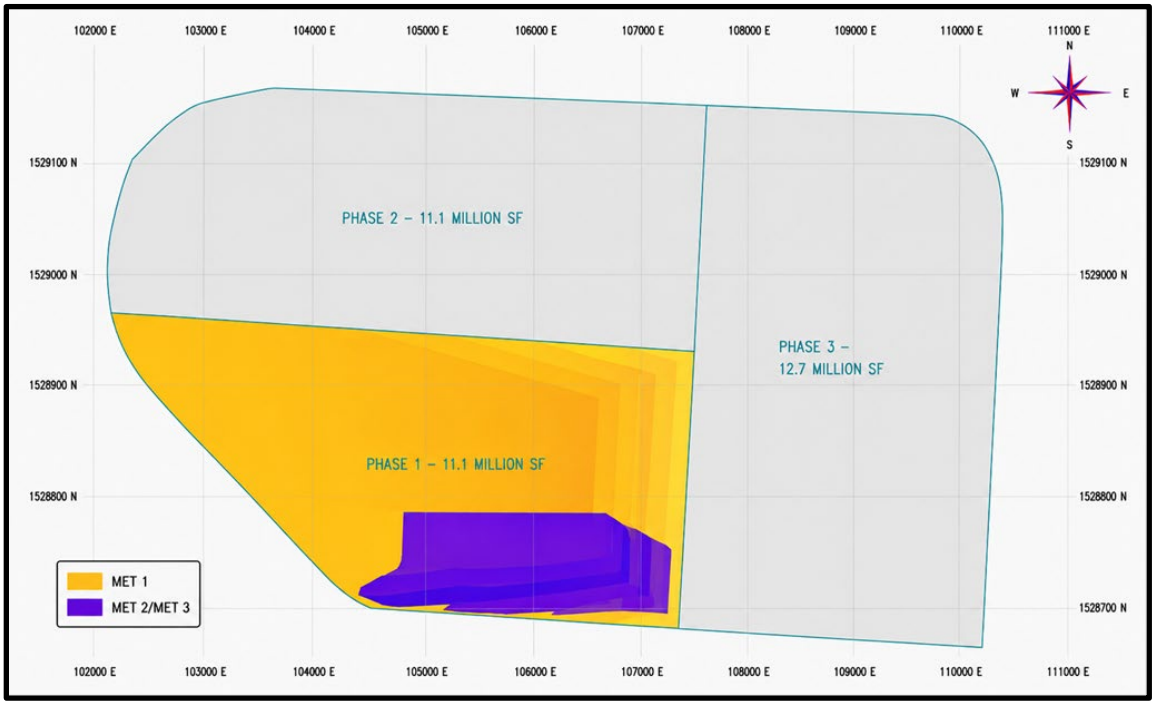


Figure 18-5: HLF Stacking Year 1 Month 7

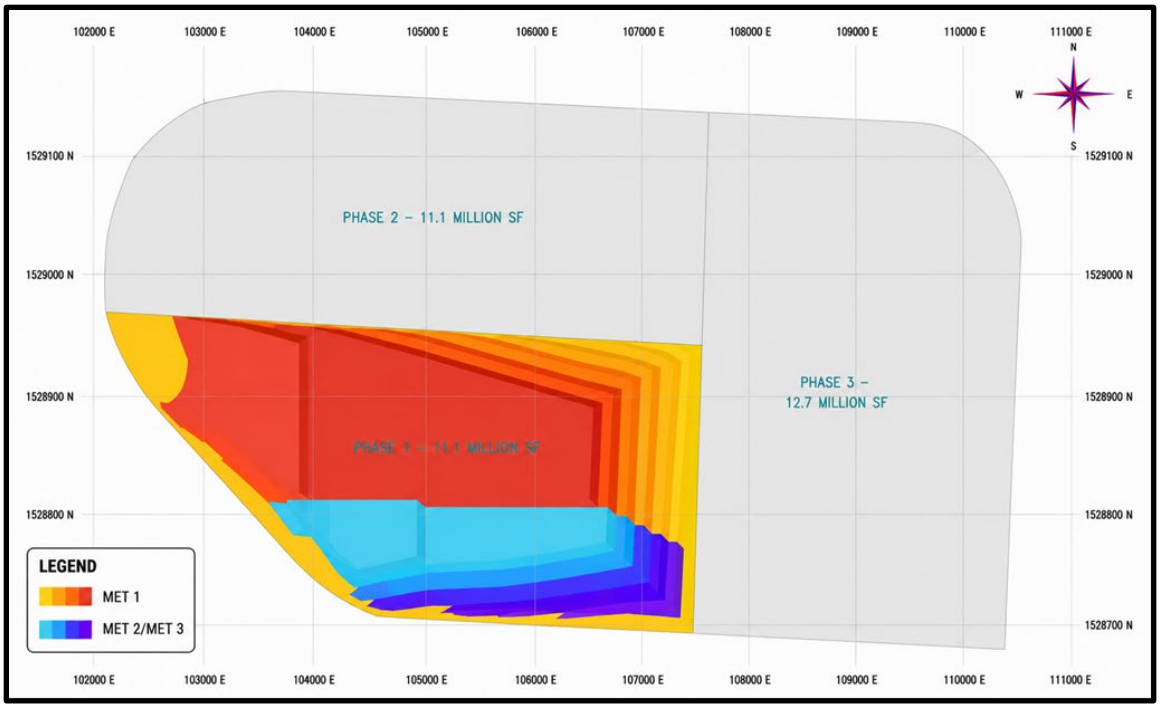


Figure 18-6: HLF Stacking Year 2 Month 11

18.7 LHLF SOLUTION DRAINDOWN MANAGEMENT AND OTHER INFRASTRUCTURE

Solution draindown from the LHLF 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. As part of the Phase 1 HLF construction, the solution draindown from the LHLF 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 process pond. Due to uncertainties in the flowrate from the LHLF draindown, the pipe is oversized; future studies will need to refine the flowrate estimate and evaluate the water chemistry to optimize the pipe sizing.

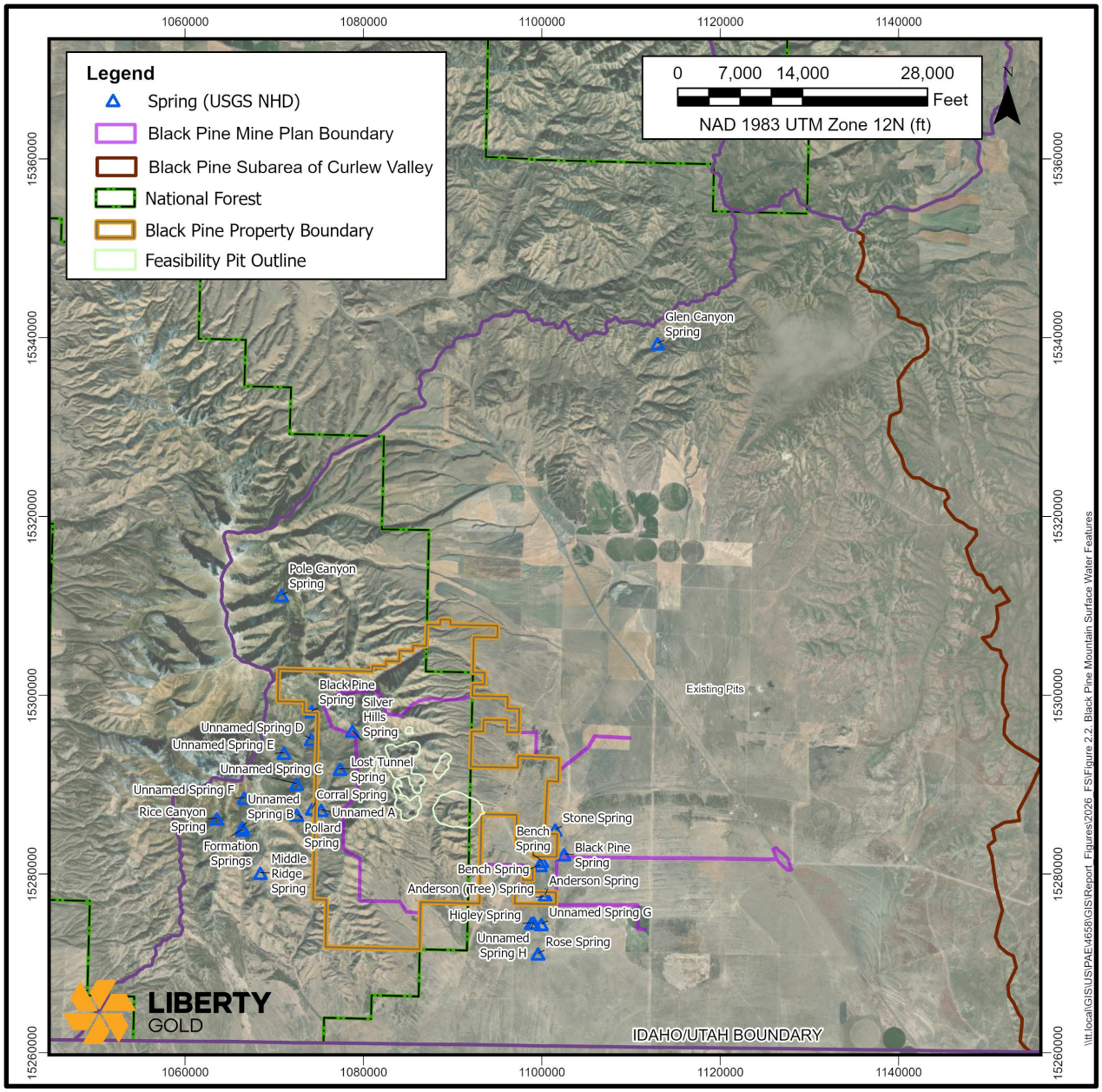
The 18 inches of topsoil within the LHLF land application area is anticipated to be stored in a separate stockpile from the topsoil removed from the HLF, pits, and other site infrastructure as part of clearing and grubbing activities. No special treatment or removal for remaining soils are considered. The existing powerline and wells within the HLF footprint will also be decommissioned and relocated during Phase 1 HLF construction.

18.8 SURFACE WATER

There are no perennial streams within the Project area, nor Waters of the US, with all drainages considered ephemeral in the National Hydrology Dataset (NHD) (USGS, 2026). 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 the majority do not meet the criteria for classification as a stream (Stantec, 2020). Chapman and Young (1972) also indicate that all intermittent streams infiltrate into the ground prior to reaching the valley.

There are 24 identified seeps and springs found within the Black Pine Subarea of Curlew Valley (USGS, 2026) (Figure 18-7). 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. There is a cluster of small bedrock

springs in the Black Pine Mountains at the western margin of the Black Pine Subarea west of the project (Figure 18-7). The absence of springs elsewhere in the area suggests a geologic control on spring presence, with less permeable rocks to the west causing localized discharge to surface, while more permeable rocks in the east in the Project area contributing to deep infiltration and recharge across the balance of the Black Pine Subarea.



Source: Piteau, 2026

Figure 18-7: Black Pine Mountain Surface Water Features

There are only two surface water features near the Feasibility Study open pits, referred to as Silver Hills Spring and Mineral Gulch Spring. However, both “springs” are presumed discharge points for old mine workings. 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 18-7). The discharge point elevations for these springs range between about 4,632 and 4,780 ft (1,410 and 1,430 m) amsl. Estimated discharge rates are low, ranging between about 2 and 9.5 gpm (Stantec, 2026). These sites have been developed for domestic use and stock water (Torrealday, 2018) including piping and earth works.

18.9 GEOLOGIC CONDITIONS

A detailed description of the bedrock geology at Black Pine can be found in Section 7, and its relationship to the hydrology of the area is detailed below in Section 18.10. The following description of the Quaternary age siliciclastic rocks provides additional details about geology not found in Section 7 and helps provide context as it pertains to the hydrology of the area. 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 on the lakebed floor are present throughout the valley. Younger and shallower Holocene-age alluvium, colluvium, and lake-shore deposits overlie the Lake Bonneville deposits and are thickest along the higher elevation margins of the valley, including along the rangefront in the project area. In many places, the younger deposits are located above the deeper, confined Lake Bonneville deposit groundwater system. However, permeable portions of the shallower Quaternary lacustrine and alluvial 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 sedimentary rocks. They often act as aquitards, but in some cases, they can provide groundwater to wells (Baker, 1974).

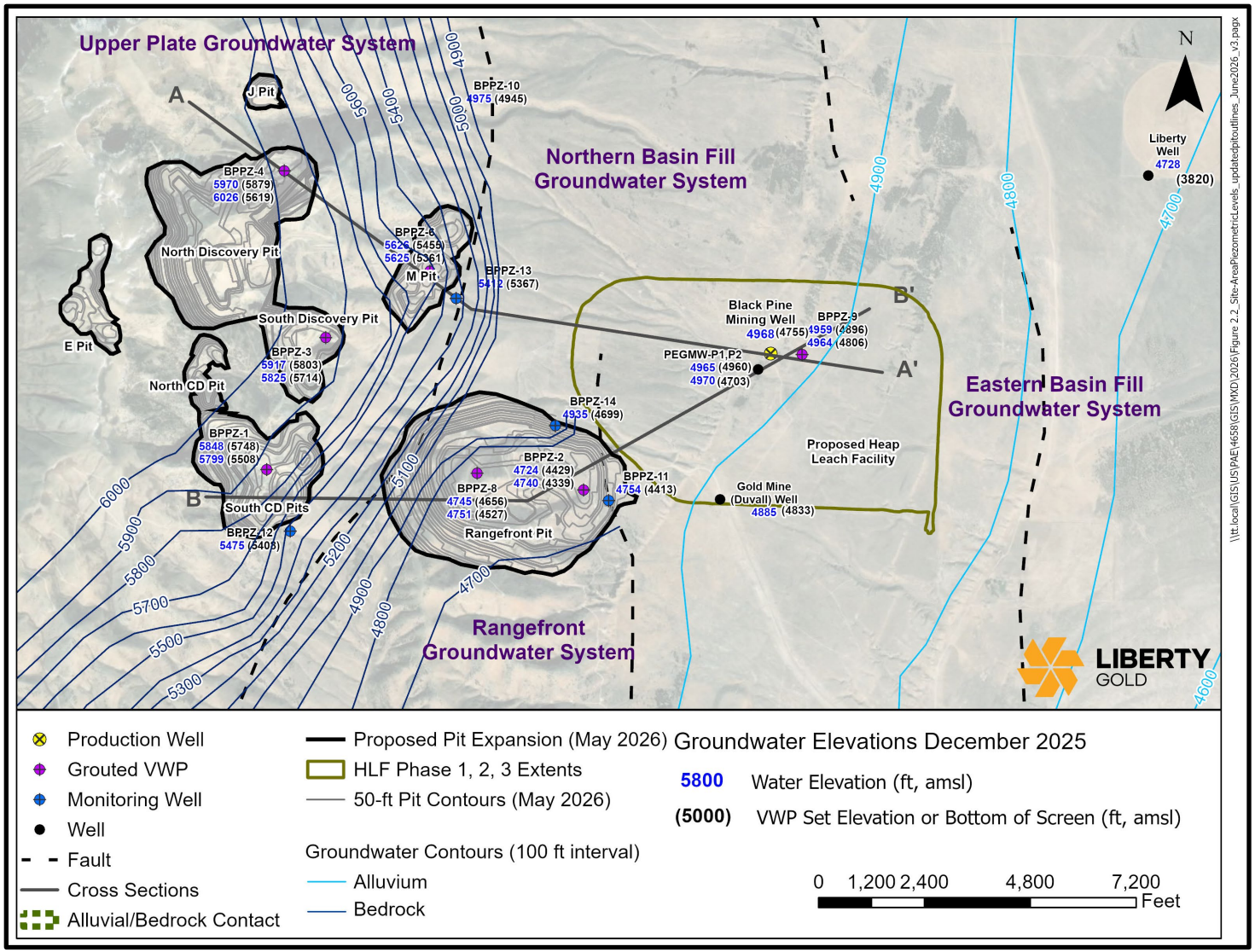
18.10 HYDROGEOLOGIC CONCEPTUAL MODEL

The following hydrogeologic conceptual model focuses on the site hydrogeology in the Black Pine Subarea (Figure 18-8). The Black Pine basin-fill groundwater system has been well-characterized by Bolke and Price (1969), Chapman and Young (1972), Baker (1974), Brandixen (1994), and Hurlow and Burk (2008). Hydrogeologic characterization in the pit and HLF area was completed between 2024 and 2026 by Piteau (2025, 2026). 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 basin-fill groundwater system.

18.10.1 Hydrogeologic Units and Characteristics

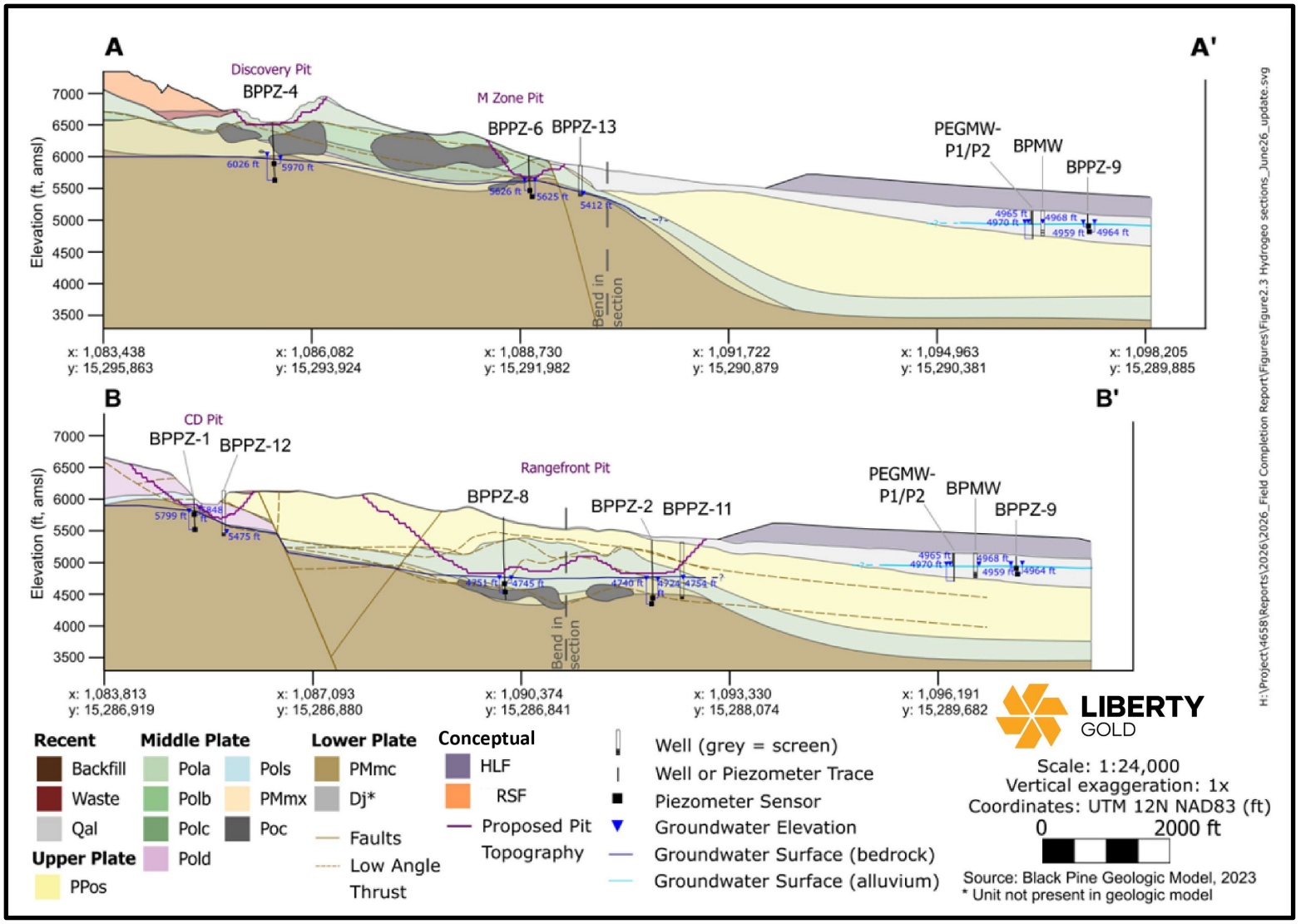
Project-area groundwater occurs in two primary systems (Figure 18-8 and Figure 18-9):

- **Bedrock Groundwater System:** Groundwater occurs primarily in fractured Devonian-age siliciclastic rocks of the Lower Plate, beneath several hundred ft of unsaturated Upper and Middle plate units of the Oquirrh group, which include highly-fractures limestone, dolomite, siltstones and sandstones. The available monitoring data (Figure 18-8 and Figure 18-9 and Plan 1) confirm steep lateral and vertical gradients and compartmentalization within the Lower Plate bedrock system. Airlift testing from five boreholes across the project area, including four holes in the Lower Plate and one in the Middle Plate, indicate hydraulic conductivities range between 0.01 to 6 ft/d (Piteau 2025, 2026). The lower end of that range is representative of less fractured bedrock, while the upper end is representative of secondary permeability in structures. Drainable groundwater storage in bedrock is expected to be below 1%.
- **Basin-fill Groundwater System:** The basin-fill groundwater system east of the open pits is hosted in lacustrine and alluvial fan deposits. The well log in BPMW indicates substantial layers of permeable rock, gravel, and sands, some of which are likely buried lakeshore deposits (Piteau, 2026a). Shallow clay units and high seasonal groundwater recharge along the rangefront lead to variable confinement, with most of the deeper system east of the site being partially to fully confined. Most basin-fill groundwater occurs in permeable, high-storage sand and gravel layers and have substantial storage capacity.



Source: Piteau, 2026

Figure 18-8: Project Area Groundwater Levels



Source: Piteau, 2026

Figure 18-9: Hydrogeologic Cross Sections A-A' and B-B'

A 6-day pumping test was completed on the BPMW in 2025 returned 660 gpm. The hydraulic conductivity estimated by the test was between 70 and 1,810 ft/d, indicating the system is highly permeable (Piteau, 2026a). This is consistent with the Black Pine Subarea basin-fill hydraulic conductivity estimates by Hurlow and Burk (2008), although the BPMW values are in the upper range of the estimates. The same test showed no response in any of the nearby bedrock wells, indicating the two systems are hydraulically disconnected. Drainable storage for the basin fill sediments is on the order of 10 to 20% (Piteau, 2026).

18.10.2 Observed Piezometric Levels and Gradients

Approximately one year of continuous monitoring data is available for bedrock Vibrating Wire Piezometer (VWP) installations BPPZ-1, BPPZ-8, and BPPZ-4. Additional VWPs installed in 2025 have shorter records, ranging from approximately two months to one week prior to the start of the BPMW pumping test at the end of 2025. Hydrographs for the Black Pine Monitoring Network are shown on Plan 1.

Observations include:

- **BPPZ-1 (CD Pit Area):** Piezometric levels are approximately 5,848 ft amsl (shallower VWP) and 5,799 ft amsl (deeper VWP) (Figure 18-9). The deeper VWP, completed in Lower Plate bedrock has remained stable, with only about 1 ft of decline over the monitoring record. In contrast, the upper VWP, also completed in Lower Plate rocks, recorded a significant decline of approximately 46 ft, which is hypothesized to be from overpressurization of the hole from drilling fluid following by equilibration, indicative of low-permeability bedrock. A strong downward hydraulic gradient (0.20 ft/ft) indicates the two fracture systems intercepted by the respective instruments are most likely hydraulically disconnected and is also at least partially attributable to the overpressurized upper VWP.
- **BPPZ-4 (Discovery Pit Area):** Piezometric levels are approximately 5,970 ft amsl (shallower VWP) and 6,026 ft amsl (deeper VWP). Both VWPs were completed in Lower Plate bedrock and have exhibited a steady decline since installation, with approximately 9 ft of decline in the deeper VWP and 7 ft in the shallower VWP. The rate of decline was greater during early monitoring and has decreased over time, likely due to extended equilibration with the surrounding low-permeability bedrock. A pronounced upward gradient (-0.22 ft/ft) combined with gradual piezometric decline indicates the two fracture systems intercepted by the respective instruments are most likely hydraulically disconnected, but they are being influenced by the same factors that are affecting the hydraulic equilibration process.
- **BPPZ-8 (Rangefront Pit Area):** Piezometric levels are approximately 4,745 ft amsl (shallower VWP) and 4,751 ft amsl (deeper VWP). The groundwater elevation in these two VWPs are over 1,000 ft lower than BPPZ-1 and BPPZ-8, demonstrating the steep groundwater gradients across the pit area, and underscoring the low permeability of the Lower Plate beneath the western pits. Piezometric levels equilibrated quickly following installation and have remained relatively stable, which is consistent with BPPZ-8 being drilled into a more permeable block of Middle Plate bedrock. Figure 18-8 shows that the groundwater system beneath the Rangefront Pit has a much flatter gradient than the Upper Plate bedrock northwest of BPPZ-8. The upper VWP, installed in Middle Plate rock, recorded approximately 1 ft of decline, while the deeper VWP in the Lower Plate showed no measurable change. A moderate upward vertical hydraulic gradient (0.05 ft/ft) was recorded, indicating the Middle Plate and Upper Plate are not well-connected hydraulically.

Interpretation of the site-area groundwater levels (Piteau, 2026a, and Figure 18-8 and Figure 18-9) have led to the delineation of four main groundwater systems beneath and downgradient of the open pits (Figure 18-8), as follows:

- **Lower Plate Groundwater System:** this system underlies all the Project open pits except for Rangefront. Groundwater levels range between about 5,000 and 6,100 ft amsl, with the prevailing hydraulic gradient towards the east. Lateral and vertical gradients are steep, owing to the very low hydraulic conductivity and overall lack of good hydraulic connection between structures at the pit scale. Groundwater is compartmentalized, and depths to water are at a minimum a few hundred ft from undisturbed land surface.

- **Rangefront Groundwater System:** The system corresponds to a several hundred ft thick layer of permeable Middle Plate rocks between the two major range front faults shown in Figure 18-8. Groundwater levels are much flatter than in the Upper Plate, and average about 4,740 ft amsl, with a prevailing lateral hydraulic gradient to the south. It has a higher hydraulic conductivity than the Lower Plate system, with good lateral interconnectedness across the Middle Plate in the area (Figure 18-8).
- **Northern Basin Fill Groundwater System:** This system is located north of the Rangefront Pit Groundwater System and east of the Lower Plate Groundwater System. The basin fill is thin, with only about 150 ft of saturated thickness at BPMW. Groundwater levels average about 4,965 ft, with a prevailing hydraulic gradient to the east. It is hypothesized there is a permeable Upper Plate Bedrock System underlying the basin fill but only the shallowest portion of the Upper Plate has been penetrated by monitoring boreholes to date (Piteau, 2026) and therefore the characteristics of this system are not well understood.
- **Eastern Basin Fill Groundwater System:** This system is the main basin-fill groundwater flow system that drains the Black Pine Subarea. Levels east of the mine in this system are in the range of 4,700 to 4,750 ft amsl (e.g., Liberty Well 4,728 ft amsl, Figure 18-8). The prevailing lateral gradient in this system is to the south. The Eastern Basin Fill Groundwater System will be the mine's primary water supply.

18.10.3 Groundwater Balance

The groundwater balance for the Black Pine Subarea of Curlew Valley was evaluated by Chapman and Young (1972). The results can be summarized as follows:

Recharge

- Owing to a relatively high average precipitation rate of 15.2 in/yr (190,000 AFA), deep groundwater levels, and permeable surface soils and rocks, the Black Pine Mountains receive substantial groundwater recharge.
- This translates to a water yield of approximately 14,250 gpm for the subarea.

Discharge

- Groundwater pumping is the major discharge component in the Black Pine Subarea, estimated to be approximately 5,300 gpm in 2026.
- There are no major areas of phyreatophytic vegetation.
- Minor evapotranspiration occurs near springs, such as those shown in Figure 18-7, but taken as a whole, the volume is on the order of a few hundred gpm.

The balance of groundwater that is not pumped discharges to the south across the Utah/Idaho State Line into the lower reaches of Curlew Valley.

18.11 GROUNDWATER QUALITY

Groundwater quality sampling has been 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:

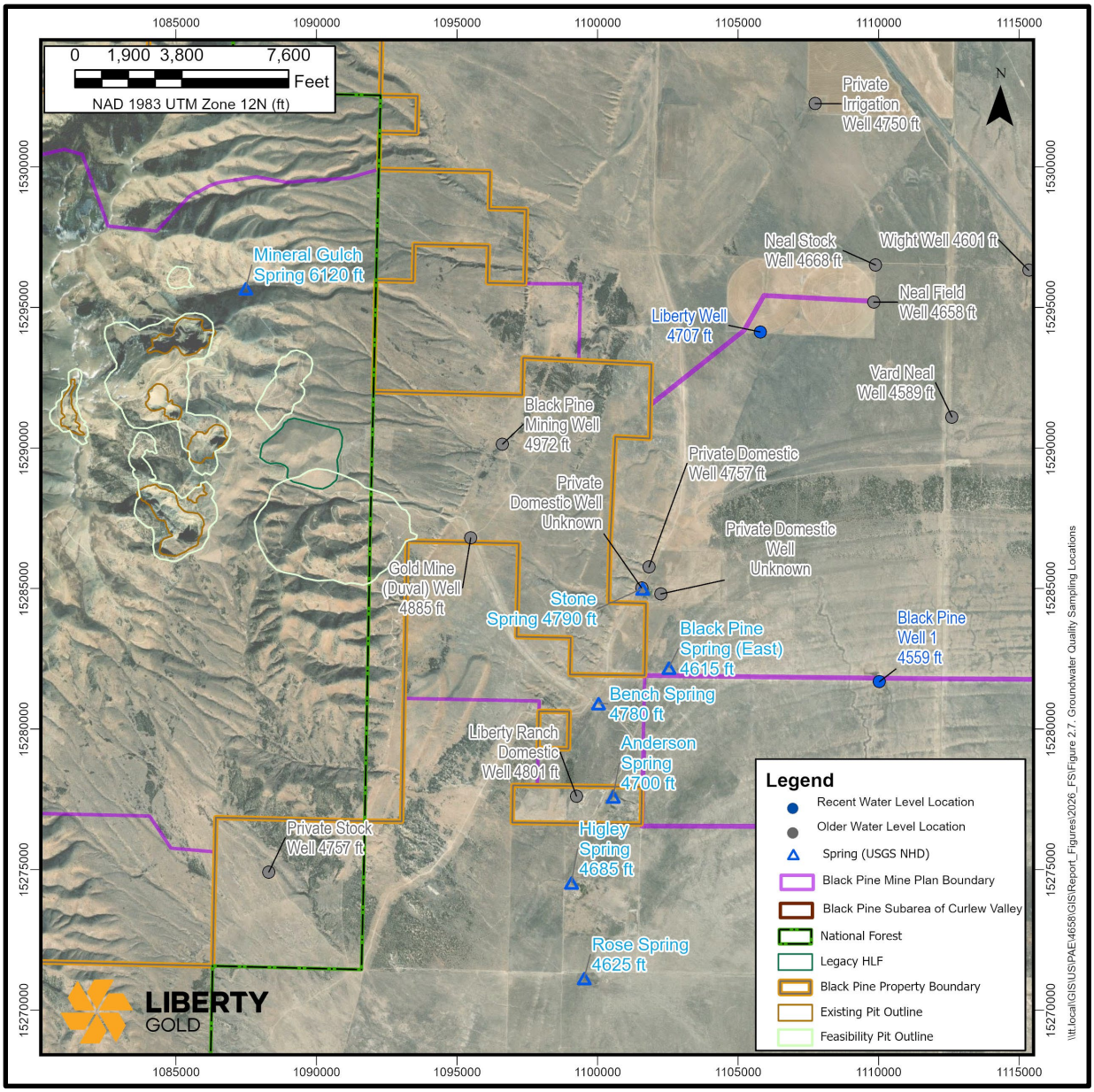
- The natural springs and wells have good water quality characteristics, with neutral to slightly alkaline pH, low to moderate TDS, moderate alkalinity, and low natural sulfate.
- There does not appear to be any influence of the higher-salinity deep groundwater system located to the south on the fringes 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 an exceedance for iron and detectable concentrations of copper, iron, lead, manganese, selenium, and zinc.
- Higley Spring has also 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.



Source: Piteau, 2026

Figure 18-10: Groundwater Quality Sampling Locations

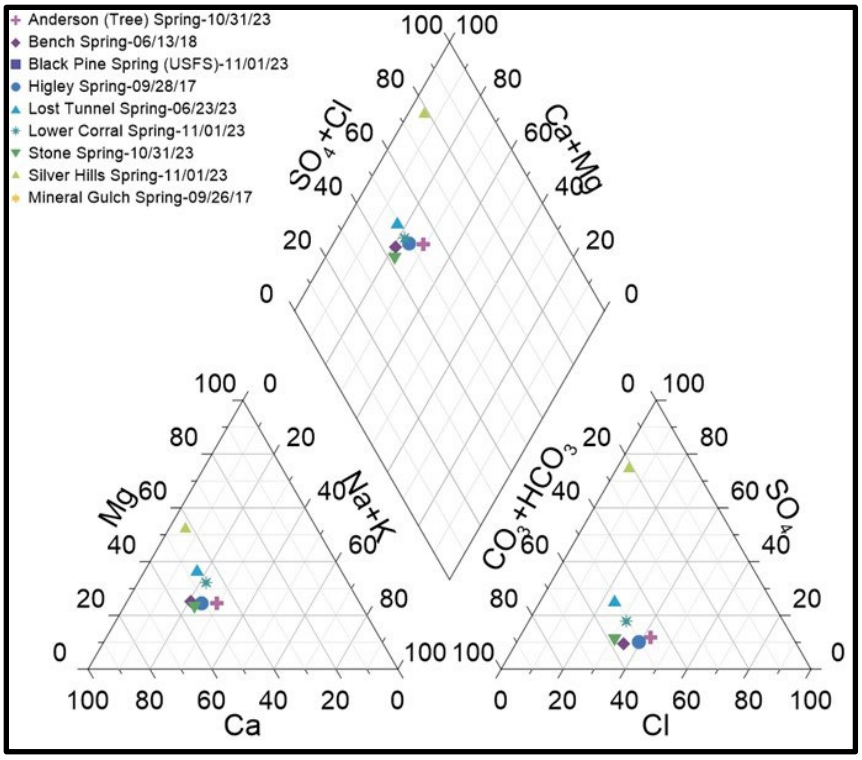


Figure 18-11: Spring Water Quality Trilinear Plot

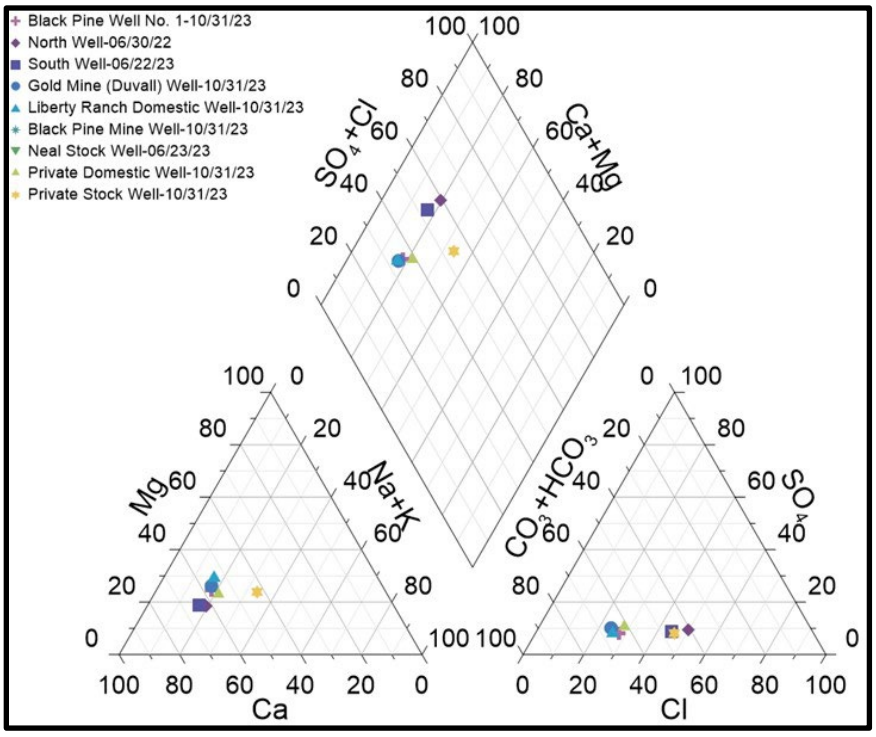


Figure 18-12: Groundwater Quality Trilinear Plot

18.12 HEAP LEACH FACILITY FREEBOARD ASSESSMENT

The hydrogeology beneath the HLF can be described as follows (Piteau, 2026a):

- The basin-fill groundwater system is hosted primarily in Lake Bonneville sediments with less prevalent overlying alluvial fan deposits.
- The well log from BPMW beneath the future HLF (location shown in Figure 18-13) indicates substantial layers of permeable rock, gravel, and sands, some of which may be buried stream channels or lake strand lines.
- Shallow clay units and strong recharge along the range front lead to variable degrees of confinement, with most of the deeper system being partially to fully confined.
- Most of the groundwater in the basin fill system occurs in sand and gravel layers which are highly permeable and have substantial storage capacity.

All of the available water strike data indicates that at least 100 ft of freeboard exists between the HLF and the bottom of the confining layer / top of the saturated groundwater system. The dataset includes 14 boreholes, as follows:

- Nine Reverse Circulation (RC) and geotechnical boreholes (BH- and LBP-series).
- Four standpipe completions, including BPMW and Gold Mine (Duvall) Well (pumping wells) and PEGMW-P1, -P2 (PVC standpipe piezometers).
- One Vibrating Wire Piezometer (VWP) completion BPPZ-9.

Geotechnical boreholes south of the proposed HLF indicate a shallower groundwater level than inside the footprint of the HLF (BH23-06, -04, and -07, Figure 18-13). This is likely due to the proximity of upwelling bedrock groundwater that includes a series of springs east of the Anderson Domestic Well (Piteau, 2026a). However, the data still confirm there is approximately 100 ft of freeboard to groundwater in this area.

18.13 OPEN PIT FREEBOARD ASSESSMENT

The open pit freeboard assessment is summarized in Table 18-5. Pit floor elevations are for the low point of each respective pit. Estimated groundwater elevations are taken from the contour map shown in Figure 18-8. All of the pits are projected to have a minimum of 50 ft of freeboard.

Table 18-5: Projected Open Pit Freeboard

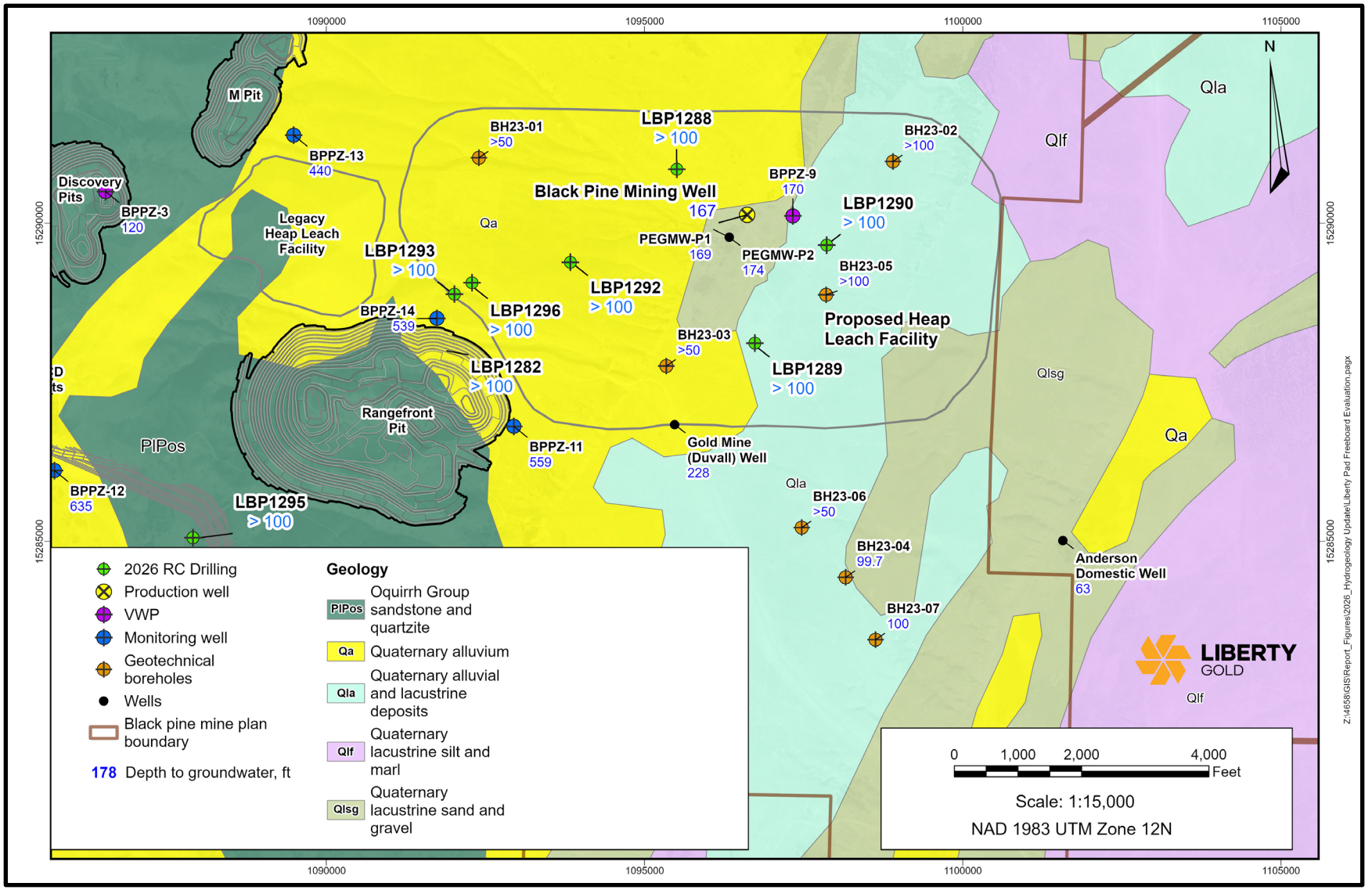
Pit	Pit Floor Elevation (ft amsl)	Estimated Groundwater Level Beneath Pit Floor (ft amsl)	Estimated Freeboard (ft)
J Pit	6,463	6,000	463
E Pit	7,349	7,000	349
North Discovery Pit	6,299	6,026	273
South Discovery Pit	6,004	5,917	87
North CD Pit	6,545	6,000	545
South CD Pit	5,775	5,725	50
M Pit	5,627	5,400	227
Rangefront Pit	4,823	4,750	73

18.14 GROUNDWATER SUPPLY INFRASTRUCTURE REQUIREMENTS

Liberty Gold has secured five production wells with approximately 4,050 gpm peak capacity (Liberty Gold, 2026). That is sufficient to meet the peak rate of 3,500 gpm. Based on proximity to process facilities, the proposed water supply will be produced as follows (Figure 18-14):

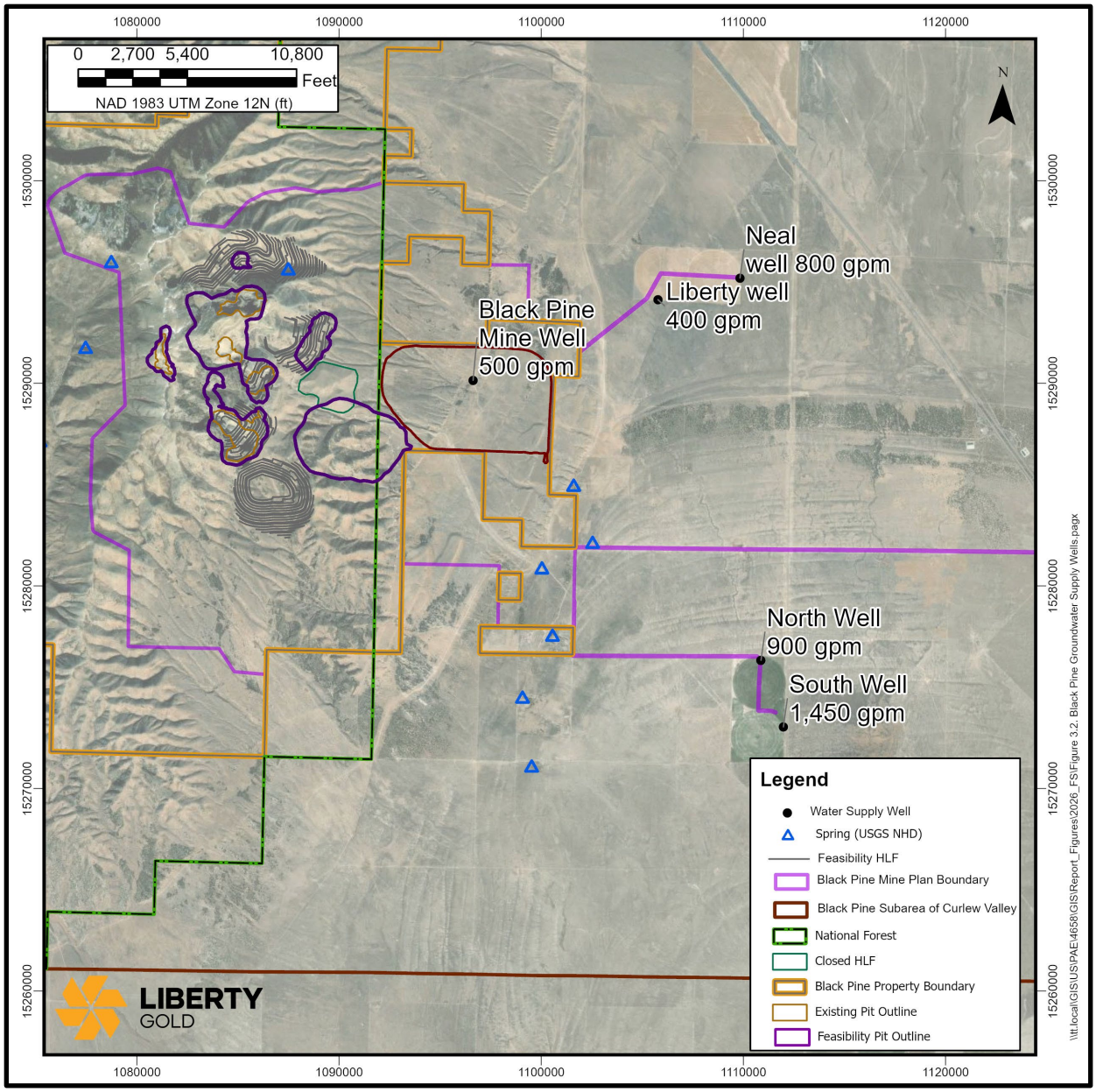
- Sufficient water supply is available from the northernmost three wells (Black Pine Mine Well, Liberty Well, and Neal Field Well) (Figure 18-14).
- The existing Black Pine Mine Well is located within the footprint of the proposed HLF and will be abandoned prior to construction of Phase 2 of the HLF. A replacement well will be needed for water supply.
- Potential locations for the replacement well BPMW-R near mine operations are being evaluated. 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.



Source: Piteau, 2026

Figure 18-13: HLF Depth to Water Analysis



Source: Piteau, 2026

Figure 18-14: Black Pine Groundwater Supply Wells

18.15 POTENTIAL FOR WATER QUANTITY AND QUALITY IMPACT

The Black Pine Project proposed by Liberty is considered to have low potential for environmental impacts for the following reasons:

Water Quantity:

- The 2024/2025 field program determined the bedrock groundwater level coincides approximately with the top of the lower plate and that at least 50 ft (15 m) of freeboard will be maintained beneath the anticipated 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 the Basin Fill Groundwater System. 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 drawdowns will be offset, reducing the likelihood of impacts considerably. A MODFLOW-USG groundwater model is being developed by Liberty to confirm the potential for groundwater level drawdowns through permitting studies in coordination with the United States Forest Service (USFS) and Idaho Department of Water Resources (IDWR) (Piteau, 2026b).

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 due to the nature of the mineralization and host rock characteristics, and can be managed in closure, through 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 to 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

This Section of the Feasibility Study presents information on the permitting and regulatory requirements of the Project, environmental, social and community considerations, general management plans and monitoring, and mine closure and reclamation planning.

20.1 ENVIRONMENTAL PERMITTING STATUS

The Black Pine Project (Project) is located in Cassia and Oneida counties, Idaho. Surface mineral estates associated with the Project are located on public lands administered by the United States Forest Service (USFS), Sawtooth National Forest, Minidoka Ranger District and the Bureau of Land Management (BLM), Pocatello Field Office and controlled by Federal unpatented lode and placer claims owned by Liberty Gold, and on private land owned by Liberty Gold.

Information on permitting associated with exploration programs are described in Section 4.5.

Liberty Gold prepared a Mine Plan of Operations (MPO; Liberty Gold, 2026) based on the mine plan presented in the Pre-Feasibility Study (M3 Engineering, 2024) in accordance with USFS Code of Federal Regulations (CFR) 228 Subpart A – Locatable Minerals (36 CFR 228A.4[c]), BLM Surface Management Regulations under 43 CFR 3809, as amended (43 CFR 3809.401[b]), Surface Occupancy regulations under 43 CFR 3715, and Idaho Department of Lands (IDL) Reclamation Plan Title 47, Chapter 15, Idaho Code and Idaho Administrative Procedures Act (IDAPA) 20.03.02.070 of the Rules Governing Mined Land Reclamation. The MPO was deemed administratively complete by the USFS, Sawtooth National Forest, Minidoka Ranger District, and the BLM, Pocatello Field Office on November 24, 2025. The USFS subsequently filed a NOI to prepare an Environmental Impact Statement (EIS) in the Federal Register on April 3, 2026, to initiate the National Environmental Policy Act (NEPA) review process.

In January 2026, the Project was accepted into the U.S. Federal Permitting Improvement Steering Council (FPISC) FAST-41 process as a Covered Project. FAST-41 refers to Title 41 of the Fixing America's Surface Transportation (FAST) Act, a U.S. government program established in 2015 intended to improve the timeliness, predictability, and transparency of environmental reviews and permitting for large, complex infrastructure projects that meet national economic and security goals and qualify for coverage. Acceptance as a Covered Project provides access to the full range of FAST-41 benefits with a coordinated permitting timetable and project advisor. The schedule has been posted to the government permitting dashboard, pursuant to the FAST-41 Federal permitting framework which provides transparency on permitting milestones and timelines, and accountability as Federal and State agencies advance environmental review and permitting activities for the Project. FAST-41 does not alter any applicable statutory or regulatory requirement, environmental law, regulation, review process, or public involvement procedure, and does not predetermine the outcome of any federal decision-making process for the Project. The Federal Permitting Dashboard currently identifies the Draft EIS as in progress.

The formal NEPA process commenced on April 3, 2026, with publication of a NOI to prepare an EIS. The NOI initiated public scoping and requested information and comments concerning the proposed action described in the 2026 MPO, potential alternatives, environmental effects, and relevant studies or analyses. The initial scoping period was intended to assist the agencies in identifying the issues and resources requiring detailed evaluation and in developing a reasonable range of alternatives for the EIS.

The USFS, in coordination with the BLM and participating agencies, is in the process of preparing a Draft EIS. Based on the FAST-41 permitting timetable, the Draft EIS should be made available for public review and comments on February 5, 2027. The agencies will consider substantive comments and may revise the impact analysis, alternatives, mitigation measures, or proposed conditions of approval in preparing the Final EIS. The current target date for the NOA for the Final EIS is November 12, 2027.

Following completion of the Final EIS, the USFS and BLM will issue separate decisions for portions of the Project under their respective authorities. The decisions may approve the Project as proposed, approve a modified alternative, impose mitigation and monitoring requirements, or decline to authorize some or all of the proposed activities. The decisions would be documented in a Record of Decision (ROD) currently scheduled for January 5, 2028. Following issuance of the ROD, Liberty Gold will prepare a final MPO that incorporates the selected alternative and any required mitigation or monitoring requirements. In accordance with 43 U.S. Code 1761, BLM can issue a ROW grant to allow use of a tract of public land for a road or utilities. A ROW grant authorizes rights and privileges for a specific use of the land for a specific length of time.

As part of the November 24, 2025 Completeness Determination for the 2026 MPO (Liberty Gold, 2026), BLM stated that Liberty Gold must also submit Right-of-Way (ROW) applications under 43 CFR 2800 for access routes and pipelines associated with the Project. BLM also established that Liberty Gold must apply for a lease under 43 CFR 2920 for operations and facilities proposed to occur on BLM surface with non-federal mineral estate and BLM-Bankhead Jones land within the “barbed wire fencing area”, including the new HLF, HLF event pond, process plant, explosives storage area, growth media stockpile, mine services area and ancillary disturbance as depicted in the MPO since those lands were removed from operation under the mining laws.

On March 24, 2026, Liberty Gold submitted a ROW application to use portions of the existing Black Pine-Stone Road (BLM Road 601) and Juniper Road (BLM Road 587) situated outside the barbed wire fence area to serve as the main access roads to the Project area during construction and/or operations. Liberty Gold also submitted a separate ROW application for the North and South water pipelines and associated access routes located outside the barbed wire fence area. According to the FAST-41 permitting timetable, final decision on the BLM ROW applications is currently scheduled for March 15, 2028.

On that same date (March 24, 2026), Liberty Gold also submitted an Application for a Lease under 43 CRF 2920, including a Plan of Development describing details such as associated facilities, access needs, operation and maintenance activities, mitigation measures, and reclamation plans for the proposed land use. According to the FAST-41 permitting timetable, final decision on the lease is also scheduled for March 15, 2028.

At the time of this report, no material environmental issues have been identified that are expected to preclude securing the required permits and authorizations to develop the Project as presented in the 2026 MPO (Liberty Gold, 2026).

20.2 REGULATORY, LEGAL, AND POLICY FRAMEWORK

Liberty Gold will need to secure modified or additional permits and authorizations from several Federal, State, and local agencies to construct, operate, and close/reclaim the Black Pine Project as presented in this Feasibility Report. The forthcoming permitting path associated with the revised mine plan presented in this report assumes that the 2026 MPO (Liberty Gold, 2026) has been issued a ROD and that Liberty Gold has secured required State permits. The subsequent subsections provide a description of the critical path permits and authorizations associated with development of a revised mine plan as presented in this Feasibility Study.

20.2.1 Federal Permitting

20.2.1.1 Mine Plan of Operations / Record of Decision

Liberty Gold will need to formally submit a revised MPO to address changes to the existing and approved mine plan as part of this Feasibility-level study, including any modifications to the open pit limits, rock storage facility (RSF) areas, the HLF (as presented in this Feasibility Study), process facilities footprint, haul roads, backfill plan, and the management and relocation of the LHLF. The revised MPO will need to provide sufficient engineering, environmental, reclamation, and operational detail for the USFS and BLM to evaluate the changes relative to the 2026 MPO and determine whether the revisions would result in new or different environmental effects.

For lands administered by the USFS, the revised MPO will be submitted under the USFS locatable minerals regulations at 36 CFR Part 228, Subpart A. Under 36 CFR 228.4, an operator must submit a plan of operations when proposed operations would likely cause significant disturbance of surface resources, and the plan must be sufficient for the authorized officer to evaluate the proposed activities and measures to minimize adverse environmental effects. The USFS regulations also include environmental protection requirements addressing water quality, solid waste, scenic values, fisheries and wildlife habitat, roads, reclamation, and other surface resources. For lands administered by the BLM, the revised MPO will also need to meet the BLM Surface Management regulations at 43 CFR Subpart 3809. Under 43 CFR 3809.11, an operator must submit and obtain approval of a plan of operations before conducting operations greater than casual use, and 43 CFR 3809.401 identifies required plan content. Importantly, BLM reviews modifications to an approved plan of operations under 43 CFR 3809.432, which states that BLM will review and approve a modification in the same manner as the original plan, unless the change qualifies for more limited treatment.

The revised MPO may also need to address BLM Surface Occupancy regulations under 43 CFR Subpart 3715, to the extent the revised mine plan includes occupancy or use of public lands that is reasonably incident to mining operations, such as security facilities, equipment yards, storage areas, or other support facilities on mining claims or mill sites. The purpose of 43 CFR 3715 is to manage use and occupancy under the mining laws and to limit such occupancy to uses reasonably incident to locatable mineral development.

The revised MPO must clearly identify and describe the differences between the 2026 MPO and the Feasibility-level mine plan. The revised MPO must also include updated design drawings, material balance, water balance, geochemical characterization, contact-water management plans, reclamation and closure plans, and an updated reclamation cost estimate.

Liberty Gold will also need to modify the lease under 43 CFR 2920 for operations and facilities located on BLM surface with non-federal mineral estate and BLM-Bankhead Jones land within the “barbed wire fencing area”, including the modified HLF, event pond, process plant, growth media stockpile, mine services area and ancillary disturbance as presented in this Feasibility Study.

Following submittal, the USFS and BLM will determine whether the revised MPO is complete and whether the changes require additional environmental review under NEPA. That determination will depend on the nature and extent of the changes relative to the 2026 MPO (Liberty Gold, 2026) and the environmental analysis already completed. If the Feasibility-level modifications are considered minor and remain within the range of effects previously analyzed, the agencies may be able to rely on the existing NEPA analysis with supporting documentation. However, if the revised MPO would result in new disturbance areas, greater or different impacts, a material change in water management, changes to the LHLF strategy, new infrastructure, or different reclamation outcomes, the agencies may require supplemental NEPA analysis or a new/revised NEPA document.

20.2.2 State Permitting

The subsections below describe the most comprehensive State operational permits that will likely be required for the project.

20.2.2.1 Reclamation Plan

Liberty Gold will need to update the IDL reclamation plan under Title 47, Chapter 15, Idaho Code, and IDAPA 20.03.02.070, Rules Governing Mined Land Reclamation. The IDL rules require an operator to submit a complete application package before a reclamation plan is approved, and the rules address mined land reclamation, cyanidation facilities, closure requirements, and financial assurance. Since the Feasibility-level mine plan changes the disturbance footprint, reclamation sequence, closure configuration, cyanidation facility design, and financial assurance amount, those changes will need to be reflected in the revised reclamation plan and associated bond estimate.

20.2.2.2 Cyanidation Permit

The Cyanidation Permit is issued by the Idaho Department of Environmental Quality (IDEQ) under IDAPA 58.01.13 and IDL IDAPA 20.03.02 and applies to facilities that use cyanide as the primary reagent to process mineralized material. The Project's HLF, based on the mine plan presented in the Pre-Feasibility Study, requires Liberty Gold to secure a Cyanidation Permit since the facility will receive process solution. Idaho Code Section 39-118A specifically identifies design controls for cyanidation facilities, including containment for process equipment, tanks, and piping, and primary and secondary liners associated with tailings storage facilities and process ponds.

In addition to submitting a revised MPO and Reclamation Plan, Liberty Gold will need to evaluate and, as necessary, update State cyanidation facility approvals for the Feasibility-level mine plan. Cyanidation facilities in Idaho are regulated under the Idaho Mined Land Reclamation Act, Title 47, Chapter 15, Idaho Code, and IDAPA 20.03.02, Rules Governing Mined Land Reclamation, which establish application, permanent closure, reclamation, financial assurance, and inspection requirements for cyanidation facilities. Since the Feasibility-level mine plan modifies the footprint, capacity, design, operations, water balance, and process solution management system, along with the removal and placement of material from the LHLF on the new HLF, Liberty Gold will likely need to submit a cyanidation permit modification and/or an amendment to the approved permanent closure plan to IDL.

Also, because the modified HLF as described in the Feasibility-level mine plan would require additional IDEQ cyanidation permitting under IDAPA 58.01.13, and the existing LHLF is currently under IDEQ's regulatory oversight, Liberty Gold will need to coordinate with IDEQ to determine how excavation, reprocessing, reuse, or placement of LHLF material would affect the modified or amended cyanidation permit application and any existing IDEQ obligations for the legacy facility and its existing authorizations. The remaining bond for the LHLF would also presumably be transferred to Liberty Gold given the relocation and incorporation of the LHLF material within the new HLF design.

The submittal will include updated design drawings, process and stormwater management controls, liner and leak detection details, water balance, draindown and detoxification assumptions, closure and reclamation measures, monitoring requirements, contingency procedures, and an updated financial assurance estimate. The cyanidation permit modification or amendment will need to be coordinated with the revised MPO and any associated NEPA review to ensure that the State permit, Federal mine plan, reclamation plan, and closure assumptions are consistent.

20.2.2.3 Point of Compliance Determination

Liberty Gold will also need to evaluate whether the Feasibility-level mine plan changes require revision of the Point of Compliance (POC) application or an amended POC determination associated with the cyanidation permit and groundwater protection requirements. The POC is an important component of the groundwater compliance framework because it establishes the location or locations where groundwater quality standards, background concentration limits, or permit-specific compliance criteria would be evaluated. Liberty Gold may need to submit a revised POC application or request an amended POC determination from IDEQ. The revised submittal must demonstrate that any proposed POC locations remain appropriate for detecting potential releases from the modified cyanidation facility and protecting downgradient groundwater and surface-water receptors.

20.2.2.4 Air Permit to Construct and Operate

Because the Feasibility-level mine plan will increase the rate of material placement on the HLF, Liberty Gold will need to evaluate whether a new or modified IDEQ air quality Permit to Construct is required under IDAPA 58.01.01, Rules for the Control of Air Pollution in Idaho. The increased HLF placement rate may increase fugitive particulate emissions from ore handling, haul roads, truck dumping, dozer spreading, grading, conveyor transfers, stockpiling, and wind erosion from active heap surfaces. Liberty Gold will therefore need to update the Project air emissions inventory to reflect the revised hourly, daily, seasonal, and annual material handling rates; haul distances; equipment fleet; operating schedule; active disturbance areas; and proposed dust-control measures. If the revised emissions inventory

exceeds the assumptions, limits, or source descriptions in the existing or pending air permit application, Liberty Gold will likely need to submit a permit modification or revised Permit to Construct application to IDEQ before constructing or operating the modified HLF placement scenario.

20.3 ENVIRONMENTAL CONSIDERATIONS

Historical permitting records indicate that closure of the LHLF was previously evaluated through Federal and State regulatory processes, including the 1998 BLM/USFS Environmental Assessment and Finding of No Significant Impact (FONSI; BLM and USFS, 1998) for land application of treated wastewater generated during HLF closure. The 1998 EA/FONSI identified dissolved metals, nitrates, dissolved solids, process water, and weak acid dissociable cyanide as constituents associated with closure water management, and described treatment, land application, monitoring, and compliance with applicable cyanidation and reclamation requirements as part of the approved closure approach.

Removal and relocation of the LHLF material could provide environmental and operational benefits by addressing a previously reclaimed legacy facility, allowing additional characterization of the underlying subgrade, and potentially enabling reuse of suitable spent ore material in construction of the new HLF. Incorporating removal of the LHLF into the mine plan could also provide an opportunity to reduce long-term legacy-site risk through a coordinated remediation, material management, and reclamation approach. However, these activities could also create environmental and regulatory liabilities that are not fully defined at this stage. Potential liabilities include characterization, excavation, handling, transport, reuse or disposal of process-affected material; management of residual cyanide-related constituents, nitrates, metals, and contact water; evaluation of liner, underdrain, solution collection, and subgrade conditions; stormwater; and remediation of impacted soils or water, if encountered.

The nature and spatial extent of potential environmental residual liability associated with the LHLF are currently unknown. Liberty Gold could inherit an obligation to delineate, excavate, treat, or otherwise manage impacted subgrade soils or contact water beneath or adjacent to the LHLF. This represents one of the larger uncertainties associated with removal because the condition of the underlying subgrade may not be fully known until the LHLF material is removed. As a result, pre-removal geochemical and geotechnical characterization, contact-water management planning, and confirmation sampling would be important components of the removal strategy.

Because the legacy facility remains subject to prior closure authorizations and regulatory oversight, and the modified HLF would require additional IDEQ cyanidation permitting, Liberty Gold would need to coordinate with IDEQ, IDL, BLM, and USFS, as applicable, to determine how excavation, reprocessing, reuse, or placement of LHLF material could affect the modified or amended HLF permit application and existing regulatory obligations, closure requirements, monitoring requirements, and financial assurance for the legacy facility.

If LHLF material is incorporated into the new HLF, the remaining financial assurance associated with the LHLF would also need to be addressed. This could include transfer, revision, replacement, or integration of the existing bond to reflect Liberty Gold's effective ownership and management responsibility for the LHLF material within the new facility. The financial assurance should be sufficient to cover characterization, excavation, handling, transport, reuse or disposal of material, management of residual metals and contact water, subgrade investigation and remediation, stormwater controls, contingency disposal, and reclamation if the Project were interrupted before removal and relocation activities were completed.

20.4 ENVIRONMENTAL STUDIES

Liberty Gold has completed multiple baseline studies, including field data collection surveys, to characterize existing environmental conditions within the Project area in support of prior exploration permitting activities. Since the issuance of the Pre-Feasibility Study Report (M3 Engineering, 2024), additional studies continue to be commissioned to support the ongoing permitting process associated with the current mine plan (Liberty Gold, 2026) and future mine plan development. Reporting and agency review for the studies listed in Table 20-1 are forthcoming.

Liberty Gold may need to complete additional surveys and studies associated with a future revised MPO. The breadth and depth of the surveys would be determined following engagement with regulatory agencies.

Table 20-1: Latest Environmental Studies

Category	Study / Survey	Year(s) / Status
Wildlife Surveys	Bat Acoustical and Infrared Camera Monitoring	2025/2026
	Amphibian Surveys	2025
	Aerial Golden Eagle and Raptor Nest Surveys and Nest Monitoring	2025/2026
	Burrowing Owl Surveys	2025
	Short-eared Owl Surveys	2025
	Pinyon Jay Surveys (completed by Intermountain Bird Observatory)	2025
	Bee Surveys (habitat assessment conducted during Sage-grouse Habitat Assessment Framework [HAF] surveys)	2025/2026
Greater Sage-Grouse Studies	Grouse Lek Counts	2025, 2026
	Greater Sage-Grouse HAF/Habitat Assessment Transects	2020, 2025, 2026
	Lek Noise Study	2025
Habitat & Vegetation Surveys	Overall Habitat Evaluation/Pygmy Rabbit Habitat Surveys	2025/2026
	Sensitive Plant (Cacti) Surveys	2023–2025
Water Resources	Water Quality Sampling Memos	Fall and Spring sampling results memos since 2017
	Seep and Spring Survey	2025/2026
	Hydrology Baseline Studies	2024–2026
Soils	Order 2 Soil Survey	2025
Cultural & Socioeconomic	Cultural Surveys	2026
	Socioeconomic Studies	2025

As requested by USFS and BLM, Liberty Gold also prepared an Environmental Condition of Federal Property report (Stantec, 2025) to further characterize existing environmental conditions at the site.

The subsections below summarize the existing conditions based on available information.

20.4.1 Meteorology and Air

The climate in the Black Pine area is characterized as continental intermontane (Piteau, 2026). Temperatures and precipitation rates vary widely with elevation, with significantly more precipitation and cooler temperatures in the Black Pine Range than in Curlew Valley. Climate data for Snowville, Utah from Ashcroft et al. (1992) indicate annual precipitation of 12.8 inches per year (in/yr) and 46.2 in/yr of potential evapotranspiration. Mean monthly temperature for Snowville was 22°F in January and 69°F in July/August (Bolke and Price, 1969).

Liberty installed two weather stations in 2021 to collect climate data at the mine site. Annual precipitation was estimated by WSP (2026) at the 5,260-ft elevation to average 10.4 in/year, with a 1 in 100-year event of 17.7 inches. At 7,490 ft elevation, the average estimated annual rate is 18.0 inches, with a 1 in 100-year event of 27.4 inches. Hurlow and Burk (2008) indicate that average-annual precipitation rates are estimated to be as high as 35 in/yr at the highest elevation of the Black Pine Mountains.

20.4.2 Surface Water

There are no perennial streams or jurisdictional Waters of the US within the Black Pine Mine Project Area; all drainages are considered ephemeral in the National Hydrology Dataset (NHD) (USGS, 2026). Stantec (2020a) 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). Chapman and Young (1972) also indicate that all intermittent streams infiltrate into the ground prior to reaching the valley.

There are 24 identified seeps and springs found within the Black Pine Subarea of Curlew Valley (USGS, 2026). Spring discharge rates are very low ranging between 0.1 and 18 gpm (Stantec, 2026). Some of the springs are dry seasonally or during drought periods. There is a cluster of small bedrock springs in the Black Pine Mountains at the western margin of the Black Pine Subarea west of the Project area. The absence of springs elsewhere in the area suggests a geologic control on the spring presence, with less permeable rocks in the west causing localized discharge to surface, while more permeable rocks in the east contributing to deep infiltration and recharge across the balance of the Black Pine Subarea (Piteau, 2026c).

There are only two surface water features near the Feasibility Study open pits, referred to as Silver Hills Spring and Mineral Gulch Spring. However, both “springs” are presumed discharge points for old mine workings. The discharge point elevations for the Mineral Gulch and Silver Hills Springs are at approximately 6,119 and 7,054 ft amsl, respectively. During recent sampling events the Mineral Gulch spring has been dry (Piteau, 2026c).

There are also six alluvial groundwater seep/spring complexes located southeast of the site. The discharge point elevations for these springs range between approximately 4,632 and 4,780 ft amsl. Estimated discharge rates are low, ranging between about 2 and 9.5 gpm (Stantec, 2026). These sites have been developed for domestic use and stock water including piping and earthworks (Piteau, 2026c).

20.4.3 Groundwater

Groundwater is described in detail in Section 18.10.

20.4.4 Wetlands and Waters of the U.S.

Stantec (2021) identified a total of seven wetland features within the wetlands and waterbody delineation survey area. Three of these were wetlands too small to be delineated. The four wetlands delineated ranged in size from 0.02 acres to 1.44 acres, with a total of 1.82 acres of wetlands delineated. Of the delineated wetlands, all were palustrine (nonflowing, non-tidal) emergent wetlands (Stantec, 2021). One wetland is located within the Project area, at the Black Pine Ranch (associated with the Anderson Spring). Wetlands were observed as moderately disturbed seeps, springs, or livestock watering troughs (typically created by pumped water to a trough location) and associated plant communities impacted by livestock use and watering.

The U.S. Army Corps of Engineers (USACE) issued an Approved Jurisdictional Determination (AJD) on May 12, 2023 (USACE, 2023), establishing that there are no Waters of the United States (WOTUS), including wetlands, within the Eastern Slope Assessment Area of the Project area where Liberty Gold will disturb land to support the mining operations.

The USACE also issued an AJD on January 25, 2024 (USACE, 2024), indicating there are WOTUS, including wetlands, within the Western Slope Assessment Area (west of E Pit) that includes the Black Pine Canyon. This area is located outside the Project area. Liberty Gold will not create disturbance on the west side of the ridgeline area.

20.4.5 Geochemical Characterization

Liberty Gold initiated a geochemistry characterization program in 2023 to assess the potential to generate ARD and leach Constituents of Potential Concern (COPCs) from waste rock material, legacy waste rock material, ore, and spent ore (i.e., material that will remain in the HLF at closure). The program will generate a comprehensive Project-specific static and kinetic dataset meeting the geochemistry testing requirements under Idaho's Ground Water Quality Rule (IDAPA 58.01.11). Liberty Gold will continue the geochemical testing program throughout the operation phase of the Project to assess the potential for mine materials to generate acid and mobilize deleterious elements during and after operations.

Through kinetic testing, the geochemistry characterization program will also support predictions on the quality of water in contact with mine materials (leachate) during long-term surface storage conditions and provide data for modeling the quality of downgradient waters to support IDEQ's POC determination (IDAPA 58.01.11) and the requirements for permanent closure of cyanidation facilities (IDAPA 58.01.13). Liberty Gold will coordinate the geochemistry characterization program and modeling requirements with IDEQ.

The program consists of two phases:

- Phase 1: Phase 1 testing will assess the potential for ARD and metalliferous leachate from mine materials and inform additional Phase 2 studies necessary to estimate changes in leachate water quality over time (kinetic leach testing of mine materials).
- Phase 2: Phase 2 testing (kinetic testing) of selected waste rock material and ore will quantify COPCs in leachate over time, and evaluate relationships between elemental composition, ARD, mineralogy, and leachate water quality.

The static and kinetic test methods selected for this Project were designed to address the bulk geochemical characteristics of the samples, including the total acid generating and acid neutralizing potential of the materials and the concentration of chemical constituents in leachates derived from the materials in the open pits. The results of the study will guide the plan for material management and closure for the facilities along with management of contact and non-contact water.

20.4.5.1 Phase 1 Results

The Phase 1 Geochemical Characterization of Mine Materials (GSI Environmental Inc. [GSI], 2025) presents the results to assess the potential for mine materials to generate acid and mobilize deleterious elements during and after operations. Liberty Gold tested rock from seven development zones containing five primary and three secondary stratigraphic units (strat-units) consisting of sedimentary and brecciated materials dominated by quartz, carbonates, or clay.

Based on the results of Phase 1 static testing, Acid Base Accounting (ABA) testing classifies 99 percent of the samples as not Potentially Acid-Generating (non-PAG).

Phase 1 leach testing consisted of Synthetic Precipitation Leaching Procedure (SPLP), Meteoric Water Mobility Testing (MWMT) and Net Acid Generating Extraction (NAGpH Extract). Based on the Phase 1 testing results, there were limited COPCs in a small number of samples and the majority of Constituents of Interest (COIs) do not exceed IDEQ groundwater standards regulated under Idaho's Ground Water Quality Rule (IDAPA 58.01.11-Appendix F).

20.4.5.2 Phase 2 Testing Program

Phase 2 of the geochemistry characterization program (ongoing) examines changes in mine material leachate water quality over time during and after mining. Phase 2 testing will determine the following:

- Potential for COPCs to mobilize from each zone and strat-unit under the range of weathering conditions anticipated at the site.
- Relationship (if any) between Net Neutralization Potential (NNP) and leachate water quality.
- Relationship (if any) between petrology/mineralogy, elemental composition, and leachate water quality.

Liberty Gold will complete waste rock material benchtop kinetic tests to determine potential COPC release from samples using known water/rock interaction ratios. Results will enable quantitative estimates of COPC release rates (source terms) which could be applied to future modeling efforts to estimate downgradient water quality during and after mining.

To address potential differences in waste rock material leaching by zone and strat-unit, Liberty Gold will conduct approximately half the sequential MWMT tests on mixed strat-units from each zone, and the other half on mixed-zone samples of each main strat-unit (defined as units comprising more than 10 percent of site tonnage). Additionally, all kinetic test sample composites will undergo Phase 1 testing to assist in comparing ABA, elemental composition, petrography and mineralogy with leachate water quality.

20.4.6 Soil

The general soil types in the Project area, as summarized below, are described in Stewart (2023).

Argic Calciorthids are deep, well drained soils formed in colluvium and alluvium from marine limestone. Topsoil has a gravelly loam texture and is from 0 to 6 inches thick. Depth to bedrock is generally greater than 40 inches. Permeability is moderate and the potential for erosion from surface flow is low to moderate.

Calcixerollic Duriargids are deep, well-drained soil formed in colluvium and alluvium from marine limestone. Topsoil depth is from 0 to 5 inches deep and has a clay loam texture. Permeability is moderate and the potential for erosion from surface flow is low to moderate.

Argis Cyroborolls are a fine loamy mixed soil that is deep and well drained. They are formed in colluvium and residuum derived dominantly from marine limestone. Topsoil is generally about 14 inches thick. Permeability is moderate and the potential for erosion from surface flow is moderate.

Lithic Cryoborolls are a gravelly loam-textured soil that is shallow and well drained. They are formed in colluvium and residuum derived dominantly from marine limestone. Topsoil ranges between 0 and 8 inches thick. Permeability is moderate and the potential for erosion from surface flow is moderate.

Pachic Argiborolls are deep, well drained soils formed in colluvium and residuum from marine limestone. Topsoil varies in depth from 0 to 10 inches and has a loam texture. Permeability is moderate and the potential for erosion from surface flow is moderate.

Argic Pachic Cryoborolls are a deep, well-drained soil. They are characterized by a surface layer of organic matter about eight inches thick. Permeability is moderate and the potential for erosion from surface flow is moderate.

Lithic Argixerolls are shallow and well drained. The parent material is colluvium and residuum derived from basalt mixed with loess. Topsoil is 0 to 5 inches and has a cobbly silt loam texture. Permeability is moderate and the potential for erosion from surface flow is moderate.

Ardic Argixerolls are shallow and well drained, and the parent material is sedimentary. Topsoil is 0 to 4 inches thick and has a loamy texture. Permeability is moderate and the potential for erosion from surface flow is high.

Pachic Argixerolls are a moderately deep, well-drained soil formed in colluvium and residuum derived from basalt mixed with loess. Topsoil varies in depth from 0 to 11 inches and has a fine-loamy texture. Permeability is moderate and the potential for erosion from surface flow is moderate.

20.4.7 Vegetation

The Project area lies in a very sparsely populated portion of the sagebrush-grass region of Southern Idaho in the northeastern portion of the Basin and Range Province. The area supports a semi-desert type of vegetation known as shrub-steppe. The location of the various vegetation communities found in any one area are influenced by soils, aspect, previous wildland fire impacts, 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 is diverse (M3 Engineering, 2024).

As described in Stantec (2018; 2020b), most of the undisturbed vegetation/wildlife habitat found within the USFS-administered land of the Project area consists of sagebrush, grasses, and juniper in the lower elevations on generally south and west exposures. Mixed conifer and aspen communities generally occur on the north and east exposures at higher elevations.

Surveys within the BLM-administered land of the Project area included a characterization of vegetation types (Stantec, 2022). Vegetation is primarily composed of three basic vegetation types: areas dominated by live juniper (Juniperus sp.) trees (Unburnt Juniper Forest), areas of shrubland and grassland matrix that recently burned with post-fire vegetation dominated by cheatgrass (Bromus tectorum), sagebrush (Artemisia tridentata.), and rubber rabbitbrush (Ericameria nauseosa) (Shrubland and Grassland), and stands of burnt juniper trees with an understory of rubber rabbitbrush, and mixed native/non-native grasses (Burnt Juniper Forest). Recently burned shrubland and grassland matrix composes most of the area surveyed. No vegetation associated with surface water, seeps, or springs was noted on BLM-administered land within the Project area (BLM, 2022).

Past wildfires have greatly affected the habitat structure and composition both within and adjacent to the Project area. Reclamation and seeding of native vegetation have taken place in areas disturbed from past mining, which encompasses a large portion of the Project area. These areas of past mining activities along with past wildfires have greatly altered the habitat for all species within the Project area. In addition, the small number of springs or other surface water sources is a limiting factor for many wildlife species (Stantec, 2024).

There is no documentation on the existence of a population of threatened, endangered, rare, or sensitive plants in the Project area (M3 Engineering, 2024).

Canada thistle, cheatgrass, diffuse knapweed, spotted knapweed, black henbane, dyer's woad, Scotch thistle, and bur buttercup are known to sporadically occur in the Project area, particularly along main travel ways and in areas of high activity (Stewart, 2023).

20.4.8 Wildlife

Most of the undisturbed vegetation and wildlife habitat found within the Project area consists of sagebrush, grass, and juniper in the lower elevations on generally south and west exposures. Mixed conifer and aspen communities generally occur on north and east exposures at higher elevations. Past wildfires have greatly affected the habitat structure and composition both within and adjacent to the Project area (USFS, 2024; BLM, 2022).

There are no known occurrences of threatened, endangered, or candidate species or designated habitat for any Endangered Species Act (ESA) listed species within the Project area (M3 Engineering, 2024).

20.4.8.1 Columbian Sharp-Tailed Grouse and Greater Sage-Grouse

There are no documented Columbian Sharp-Tailed Grouse (CSTG) leks and Greater Sage-Grouse (GRSG) leks within the Project area. There are three CSTG leks within 4 miles of the Project area that have had activity within the last three years (leks 4CT047, 012, and 1O065) and three GRSG leks (1O065, 1O083, 4C140) that have had activity in the last three years. The closest recently active GRSG lek (1O065) is located by a stem (corridor) of the Project area and the closest recently active CSTG lek (4CT047) is located approximately 1 mile from the northeastern portion of the main Project area.

Habitat in the Project area is known to have been affected by habitat fragmentation from roads, invasive species, mining, and fire (Stantec, 2018; 2020b). Habitat within the general Project area is primarily summer/late brood-rearing habitat, although locations with denser stands of sagebrush, for example in the lower elevations on the east side (i.e., closer to the known leks) may also be used as nesting/early brood-rearing and winter habitat (USFS, 2024). On USFS land, habitat is mapped as General Habitat Management Area (GHMA). The BLM-administered land east and immediately adjacent to the Project area is mapped as Important Habitat Management Area (IHMA). However, due to Idaho reaching a hard population trigger in the greater curlew fine scale area under its Approved Resource Management Plan Amendments (ARMPA; BLM, 2015), the IHMA must adhere to stipulations and management practices of a Priority Habitat Management Area (PHMA).

Based on the Idaho's Sage-Grouse Habitat Quantification Tool (HQT) evaluation completed in 2020 (Stantec, 2022), general suitability of undisturbed GRSG habitat in the Project area is low (the most suitable habitat is east of the Project area). In addition, the Project area was rated as having low suitability as late brood-rearing habitat. These results are supported by the site-scale habitat data that determined the Project area is mostly unsuitable as nesting/early brood-rearing habitat, mostly marginal or unsuitable as summer/late brood-rearing habitat, and marginal or unsuitable as winter habitat (USFS, 2024).

20.4.8.2 Golden Eagles

Within the Project area, golden eagles likely forage within sagebrush and grassland habitats on the eastern side of the Project area. In addition, golden eagles have nested in the Project area within an existing and inactive historic pit in multiple years, including in 2019, 2021, 2023, and 2024. However, in 2023, the nest appeared to have failed. A potential alternate eagle nest site is known to exist to the west of the Project area, and it is possible that the eagles observed nesting in 2019, 2021, and 2023 have used the alternate site in the other years (USFS, 2024).

20.4.8.3 Big Game Species

Game species that occur in the Project Area include moose (Alces alces), mule deer (Odocoileus hemionus), elk (Cervus elaphus canadensis), and pronghorn antelope (Antilocapra americana) (BLM, 2022). Mule deer and elk have been observed frequently throughout the Project area. Because moose more typically use the northwest portion of the Project area (Stantec 2018, 2020), they have been observed less frequently.

Winter aerial surveys for big game have been conducted within the Project area over multiple years since 2008. Mule deer occur throughout the Project area and surrounding lands; however, few occurrences have been recorded within the Project Area (Stantec, 2022). Mule deer are known and expected to utilize the area based on previous observations and available resources; however, they appear to prefer, or more heavily utilize land west of the Project area (BLM, 2022).

A multi-year telemetry study on mule deer movement and habitat use within and adjacent to the Project area, funded by Liberty Gold has been conducted since 2019 using Global Positioning System (GPS) collared mule deer. The aerial survey data showed a pronounced decline in mule deer numbers in the winter of 2022/2023 (Stantec, 2024). Although the exact cause is uncertain, it is likely that the abnormally deep snowpack present in 2022 and 2023 (which

exceeded the highest recorded amount from November-January at many nearby stations) was a contributing factor as forage was unavailable and deer likely moved out of the area (USFS, 2024). Due to confounding weather patterns, it is difficult to ascertain if mule deer are avoiding the existing exploration areas; however, the ongoing telemetry study should continue to inform managers on habitat movements and use. The telemetry data shows that summer mule deer use is concentrated on the west side of the Project area (i.e., in Black Pine Canyon). Winter use also includes Black Pine Canyon but use also expands south and east into the Project area and onto adjacent agricultural fields.

20.4.8.4 Migratory Birds and Raptors

Numerous surveys for migratory birds and raptors have been conducted within the Project area as part of previous baseline surveys, pre-construction surveys, and monitoring. A total of 82 species of migratory birds, including raptors, have been identified within or adjacent to the Project area. Due to the diverse nature of the habitat surrounding the Project area, a diverse population of migratory birds is expected to occur, and existing habitats provide foraging and nesting opportunities (BLM, 2022). The following active raptor nests have been noted (USFS, 2024):

- American kestrel (Falco sparverius) – two separate nests observed in 2023.
- Golden eagle – nesting observed in 2019, 2021, and 2023, as described above.
- Great-horned owl (Bubo virginianus) – two separate nests, one in 2019 and one in 2023.
- Long-eared owl (Asio otus) – 2022.
- Prairie falcon (Falco mexicanus) – 2023.
- Red-tailed hawk (Buteo jamaicensis) – multiple nests observed in 2019, 2021, 2022, and 2023.

20.4.9 Cultural Resources

Liberty Gold completed intensive Cultural Resources inventories over 9,936 acres in and around the Project area between 2017 and 2023 (Commonwealth Heritage Group, Inc. [CHG], 2017, 2020a, 2020b, 2021a, 2021b, 2022a, 2022b, 2022c; PaleoWest, LLC, 2023). The pedestrian surveys identified eligible sites within the Project area. These eligible sites are historic in character. Reports of these surveys have been submitted to relevant Federal and State agencies.

20.4.10 Visual Resources

The Project area is located within the Snake River Plain landscape character type and is visually dominated by the Black Pine Mountains, which include moderate-to-steep upper slopes, rolling lower hills, minor intermittent drainages, mixed mountain shrub and sagebrush communities, and conifer stands in northern drainages (USFS, 2024). Although above-ground facilities from prior mining operations have been removed, the area remains visibly affected by historic mining, including large open pits, former haul-route road cuts, a reclaimed heap leach pad, and two rock storage facilities.

The USFS portion of the Project area is mapped primarily with Visual Quality Objectives of Partial Retention and Modification, with a small area of Maximum Modification; however, the USFS has previously recognized that the area was already mining-disturbed before the VQOs were established and that achieving the mapped objectives would require extensive reclamation. BLM-administered lands within the Project area are designated as VRM Class IV (BLM, 2022), which allows major landscape modifications while still requiring impacts to be minimized through careful siting, limiting disturbance where practicable, and using natural form, line, color, and texture where possible.

20.4.11 Land Use

The Project area is primarily undeveloped land that is managed for recreation, wildlife habitat, and livestock grazing. 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 (Liberty Gold, 2026).

20.5 WASTE ROCK MANAGEMENT

The Feasibility Study mine plan is not expected to require material changes to the waste rock management approach presented in the 2026 MPO. Liberty Gold initiated a geochemistry characterization program in 2023 to assess the potential to generate ARD and leach COPCs (GSI, 2025) from waste rock material, legacy waste rock material, ore, and spent ore (i.e., material that will remain in the HLF at closure). The program will generate a comprehensive Project-specific static and kinetic dataset meeting the geochemistry testing requirements under Idaho's Ground Water Quality Rule (IDAPA 58.01.11).

Based on the geochemical characterization data available to date (GSI, 2025), waste rock material at the Project exhibits very low to no potential for ARD, and limited potential for metal leaching. Based on these geochemical characterization results, special management of waste rock material is not indicated, and Liberty Gold will place the material within the RSFs or in mined-out pits without constraint during operations. Furthermore, Liberty Gold will regrade and revegetate the final RSF surfaces at closure, reducing the potential for ponded water and infiltration through the RSFs based on the closure configuration.

Liberty Gold is completing further geochemical testing (kinetic testing) of selected waste rock material and ore to quantify COPCs in leachate over time, and is evaluating relationships between elemental composition, ARD, mineralogy, and leachate water quality.

20.6 WATER MANAGEMENT

The Feasibility Study mine plan is not anticipated to require material revisions to the water management plan presented in the 2026 MPO. Liberty Gold intends to manage contact water and non-contact water in accordance with applicable permits, approvals, and regulatory requirements. Liberty Gold designed the Project's stormwater management infrastructure to meet the following objectives:

- Minimize run-on (i.e., non-contact water) volumes by diverting water surface flows away from disturbed areas and mine facilities such as process solution sources, selected mine pits, and temporary ore stockpiles.
- Capture runoff water (i.e., contact water) downstream of mine infrastructure (e.g., RSFs) in diversion channels and sediment ponds to prevent discharge of sediment-laden water off-site.
- Ensure direct precipitation (i.e., meteoric water) on disturbed areas remains within the primary and secondary containment systems or is captured in diversion channels and sediment ponds.

Liberty Gold designed the diversion channels upstream from the temporary ore stockpiles to sustain flows from the 100-year, 24-hour design storm event plus 1 ft of freeboard. Liberty Gold will also construct surface water diversions around the HLF to direct surface water runoff away from the containment area. The north and south stormwater diversion channels along the HLF will be permanent infrastructure and will remain post-closure to keep water from eroding the heap. These non-contact waters will be routed to natural draws/drainages for natural infiltration, with standard BMPs such as straw bales used as necessary for minor sediment or flow control.

Contact water stormwater infrastructure for the RSFs (i.e., downstream diversion channels and sediment ponds) were designed to contain flows from the 100-year, 24-hour design storm event and 2 ft of freeboard. Liberty Gold will not discharge water captured in the sediment ponds as water will infiltrate and evaporate.

Liberty Gold designed the diversions around the open pits to prevent nuisance flows into the pits during operations and to protect the stockpiles from erosion. Pit diversions were designed to convey flows from the 2-year, 24-hour design storm event. During larger rainfall events, the diversions could overtop, and flows would report directly into the pits.

Water is expected to infiltrate into the pits or evaporate within a reasonable timeframe according to observations of the existing pits and should not cause appreciable delays to mining operations.

20.7 FLUID MANAGEMENT

Liberty Gold intends to manage waste and process-related fluids as required by the construction and operating permits listed above. Liberty Gold will develop and maintain a Process Fluid Management Plan for the Project. The Process Fluid Management Plan will be developed to support IDEQ Cyanidation Permit under IDAPA 58.01.13. It will describe the containment and monitoring of process solutions, and will address components associated with process solutions, including the HLF, process plant, and HLF event pond. The plan will also include response and reporting requirements specific to process solutions. Liberty Gold will update this plan periodically to incorporate changes and improvements identified during operations.

20.8 SITE MONITORING

Environmental monitoring requirements for the Project will be established in accordance with the conditions of applicable Federal and State permits and the findings of the NEPA review as part of the ROD, based on a revised MPO reflecting this Feasibility-level mine plan. Site monitoring for the Project will be developed to:

- Demonstrate compliance with the approved MPO and other Federal or State permits/authorizations for the Project, and other Federal or State environmental regulations.
- Provide data to validate baseline characterization and calibrate predictive modeling efforts.
- Provide data for early detection of potential issues.
- Provide data to evaluate the effectiveness of the Project's environmental protection measures and engineering controls during mine planning.
- Supply information that would assist in directing corrective actions/adaptive management plans should they become necessary.

The monitoring program will be refined in coordination with the USFS, BLM, IDEQ, IDL, and other applicable permitting processes such as the MPO, NEPA analysis, State permit applications, and closure plans are developed.

The environmental monitoring program is expected to include, as applicable, surface water and groundwater quantity and quality (including springs), meteorology and ambient air quality, geological hazards, wildlife mortality, and noxious weeds. Operational monitoring will include slope stability monitoring, monitoring the release of leaching solutions from the HLF, and visual observation of the integrity of the stormwater management infrastructure.

Monitoring locations, parameters, frequency, quality assurance procedures, reporting requirements, and response thresholds will be defined through permit conditions/requirements included in permits such as POC Determination, Cyanidation Permit, Idaho Pollution Discharge Elimination System (IPDES) Permit (if required), and Air Quality Permit including associated environmental management plans and closure and reclamation plans.

Post-closure monitoring will entail general maintenance of safety features and monitoring of water resources, revegetation success, invasive and noxious weeds, and slope stability. In general, post-reclamation monitoring and maintenance will include the following:

- Conduct berm, fencing, and sign inspection and maintenance.
- Monitor HLF draindown quantity and quality, as well as performance of the E-Cell in accordance with IDEQ's Cyanidation Permit.
- Conduct post-mining groundwater quality monitoring in accordance with IDEQ requirements and the POC Determination.

- Conduct revegetation monitoring for a minimum of three years following completion of revegetation activities or until revegetation success has been achieved.
- Monitor and control noxious weeds for three years following closure.
- Visually inspect slopes to detect any crest deformations and signs of failure (e.g., slope bulging), evidence of seepage, formation of surface cracks, and identification and documentation of erosional features (e.g., vegetation loss, rills, gullies, etc.).

20.9 SOCIAL CONSIDERATIONS

Liberty Gold has implemented an active stakeholder and community outreach program since June 2023. Engagements have included regular project updates and introductory meetings with municipal, county, and state officials in Idaho and Utah, including the Malta and Snowville city councils; Cassia, Oneida, and Box Elder county commissioners; and members of the Idaho Legislature. The Project team has also met with regulatory agencies, industry representatives, utilities, emergency-response organizations, local businesses, and community service groups to provide information on Project planning and development.

Outreach activities have been conducted through council and commission meetings, community presentations, site visits, public events, and direct discussions with interested stakeholders. The Project has maintained a recurring presence at fairs, rodeos, Independence Day celebrations, and other regional events in Cassia, Minidoka, and Box Elder counties. These events have provided opportunities to introduce the Project, geology, and mining process; respond to questions; invite community members to participate in site visits; and receive feedback regarding potential Project effects and community concerns.

Liberty Gold has also undertaken targeted engagement on matters relevant to future development and environmental review. Discussions with local emergency service providers have addressed response capacity, interstate coordination, and potential mutual-aid arrangements, while meetings with the Raft River Electric Cooperative have addressed anticipated power requirements during different stages of the Project. Project representatives have participated in agency and public scoping activities, including virtual and in-person public scoping meetings administered by the USFS in April 2026, and have supported agency presentations to local county commissioners.

Community investment and educational initiatives have formed an additional component of the outreach program. These activities have included scholarships for local high school students, winter clothing donations through the Coats for Kids program, sponsorship of community events, classroom presentations, geology and earth-science field experiences, and volunteer activities. The documented outreach record indicates sustained engagement across the Project area and generally positive public interest, while providing ongoing opportunities for stakeholders to obtain information, raise concerns, and participate in discussions such as Project planning, permitting, and environmental review progress.

At the time of this report, Liberty Gold has not entered into any agreements with the communities.

20.10 CLOSURE AND RECLAMATION

Disturbed areas associated with the Project are expected to be reclaimed in accordance with 36 CFR 228.8(g), BLM regulations 43 CFR 3809.401, and the Idaho Mined Land Reclamation Act, and IDAPA 20.03.02, Rules Governing Mined Land Reclamation.

Liberty Gold will complete the reclamation of the site to:

- Comply with applicable State and Federal environmental rules and regulations.
- Stabilize and protect surficial soil materials from previously disturbed areas from wind and water erosion.

- Stabilize slopes through grading and recontouring to create a natural post-mining landscape and provide rounded landforms similar to adjacent undisturbed land.
- Establish long-term, self-sustaining vegetation communities to support post-mining land use.
- Protect surface and groundwater quality including compliance with all applicable water quality standards at closure and during the post-closure period.
- Protect public health by reducing potential hazards within the Project area.
- Limit and/or eliminate long-term closure requirements associated with ongoing care and maintenance, to the extent practicable.

Liberty Gold will design the closure and reclamation of the Project to achieve post-mining land uses consistent with: the Sawtooth National Forest's land use management plans for the Black Pine Management Area, which are outlined in the Sawtooth National Forest Land and Resource Management Plan (RMP; USFS, 2012); the BLM land use management plans, which are outlined in the Pocatello Approved Resource Management Plan (ARMP; BLM, 2012) and the 2015 Idaho and Southwestern Montana Greater Sage-Grouse Approved Resource Management Plan Amendment (ARMPA; BLM, 2015) or most current great-sage grouse plan. The closure and reclamation plan will focus on re-establishment of productive post-mining uses of the Project area that emphasize livestock grazing, wildlife habitat, hunting, and dispersed recreation.

20.10.1 Reclamation of Mine Facilities

General measures associated with closure and reclamation will not significantly differ from the 2026 MPO and will include concurrent reclamation, establishment of adequate stormwater drainage, earthwork, placement of salvaged growth media as a cover on the HLF and RSFs as needed, revegetation, noxious weed management, and waste removal. During construction, Liberty Gold plans to salvage and stockpile suitable and available growth media for use in future reclamation activities. To the extent feasible, Liberty Gold will perform concurrent reclamation of areas no longer required for mining and processing operations. The sections below describe the general closure methods for the main mine facilities.

20.10.1.1 Open Pits

Liberty Gold intends to backfill the CD Pit, M Pit, and portions of the Discovery Pit using haul trucks, dozers, and graders (or similar equipment). The J Pit will also be backfilled during the placement of waste rock material on the Discovery RSF (J Pit is located within the footprint of the RSF) throughout the mine life. Use of waste rock as pit backfill will reduce the overall surface disturbance footprint and visual effects of the Project, particularly where backfilling is undertaken concurrently with mining operations. Pit backfilling will generally be completed by end-dumping waste rock from designated safe access areas, with dozer or grader support as required to maintain safe and efficient access. The backfill material will not be mechanically compacted, thereby allowing infiltration of precipitation and snowmelt.

Liberty Gold does not intend to backfill the E Pit, F Pit, and Rangefront Pit as part of Project closure. Where required for public safety, access controls will be installed at principal access points to limit entry by the public and to discourage livestock and wildlife access. Over time, pit benches are expected to ravel, reducing the prominence of linear features and promoting the development of more natural-appearing scree and talus slopes.

20.10.1.2 Rock Storage Facilities

Liberty Gold intends to reclaim the RSFs to achieve stable landforms, limit erosion, reduce visual contrast with the surrounding landscape, and support the establishment of self-sustaining vegetation. Reclamation activities are expected to include regrading and recontouring of the outer slopes and upper surfaces of the RSFs using dozers or similar equipment to improve long-term geotechnical stability, reduce erosion potential, and facilitate revegetation. Shallow ripping of RSF slopes may be completed, as required, prior to seeding. Final reclaimed slopes are expected to be graded to an overall configuration of 3.0H:1.0V or flatter. The upper surfaces of the RSFs will be graded to

promote positive drainage, avoid ponding, and provide a topography consistent with long-term stability and reclamation objectives. Liberty Gold will place a layer of growth media and seed the RSFs using the approved reclamation seed mix to promote revegetation.

20.10.1.3 Ore Stockpiles

Once ore stockpiles are depleted and the material has been placed on the HLF, Liberty Gold will grade and, if necessary, recontour the areas to tie into the surrounding topography and reestablish drainage patterns. Liberty Gold will rip and scarify the ore stockpile areas to alleviate compaction and promote root growth and seed the areas with the approved seed mix (following placement of a growth media layer).

20.10.1.4 Heap Leach Facility

The general concept for closure and reclamation of the HLF following placement of material from the temporary ore stockpiles and LHLF will include the following steps:

- Recovery of economic grade solution (estimated four years).
- Placement of 18-24 inches of growth media over the graded HLF to limit meteoric water infiltration and promote transpiration through vegetative cover.
- Active draindown management (recirculation of draindown on the HLF and active evaporation during non-freezing months) to draw down the fluid inventory in the heap (estimated 10 to 15 years).
- Once residual draindown levels have reached a point where they can be managed within the lined HLF and event pond without active evaporation, Liberty Gold will convert the event pond to an Evaporation Cell (E-Cell) to be used as a zero-discharge long-term facility.
- Post-closure monitoring following active draindown (monitoring estimated for five years).

Liberty Gold will decommission the draindown pipe from the LHLF to the new HLF once the LHLF is fully removed. Any geomembrane, pipes, or other remaining infrastructure from the LHLF will be buried in place, covered with growth media, and revegetated. The combination of limiting infiltration into the new HLF and removing fluid inventory through evaporation and managing the remaining fluid with the E-Cell will provide for effective long-term, contained closure of the LHLF.

Reclamation activities are expected to include regrading and recontouring of the outer slopes and upper surfaces of the HLF using dozers or similar equipment to improve long-term geotechnical stability, reduce erosion potential, and facilitate growth media placement and seed application. The crest of the HLF will similarly be regraded to promote runoff and control erosion. Following regrading a growth media material cover consisting of native soil (18-24 inches of growth media over the heap material with mulch and seed on all exposed surfaces) will be placed on the exposed slopes.

20.10.1.5 Process Plant and Mine Buildings and Facilities

Closure of the process plant will include the decontamination, decommissioning, demolition, or disposal of the precipitation facilities, the cyanide mixing area, and the fluid management system. Liberty Gold will ensure that all rinse fluids are recycled back to the fluid management system. Following salvage of equipment (if applicable), decontamination, demolition, and disposal activities, Liberty Gold will rip and scarify the disturbed area of the process plant and grade the footprint, if necessary, to blend with the adjacent topography. Liberty Gold will rip and scarify the plant area to alleviate compaction and promote root vegetation, place growth media, and seed the areas with the approved seed mix.

Liberty Gold will remove all structures, equipment, and other facilities that have not been identified as needed during the post-closure phase (e.g., truck shop, truck wash, lab, warehouse, etc.). Liberty Gold will remove all buildings and infrastructure except for the substation and buried powerline that will remain post-closure.

Buildings and support structures necessary for the reclamation of the HLF and processing facilities will remain until these facilities are fully closed and reclaimed. Reclamation will generally include decommissioning, demolition, material salvage (if applicable), waste/debris removal, backfilling and regrading, scarifying, and seeding affected areas.

Building parts and waste/debris will be taken offsite to a licensed facility. Liberty Gold may need to break up in place concrete foundations, culverts, pipelines, and other non-reactive, non-combustive, non-corrosive, and non-hazardous demolition waste for most buildings and structures. Liberty Gold will cover the foundations, regrade, scarify, and seed the area.

20.10.1.6 Stormwater Management Infrastructure

The west, north and south diversion channels at the HLF are permanent infrastructure and will remain at closure to direct stormwater away from the heap to reduce the potential for erosion.

Liberty Gold will fill and regrade the diversion channels around the ore stockpiles to match the surrounding area contours once the stockpiles are depleted in the two years following the end of mining operations and backfill and regrade the diversion channels around the open pits. Liberty Gold will also backfill/regrade the diversion channels and sediment ponds associated with the RSFs at closure. The ponds will be backfilled and regraded to have positive drainage prior to revegetation activities.

20.10.1.7 Water Conveyance and Pipeline

Liberty Gold will decommission equipment associated with the Project water supply infrastructure including well pumps and booster pumps and associated piping connecting to both underground water pipelines. Liberty Gold will leave the underground water supply pipelines in place.

Liberty Gold will abandon and plug wells according to the State of Idaho Rules for Exploration Operations and Required Reclamation (IDAPA 20.03.02.06), Well Construction Standards Rules (IDAPA 37.03.09), and USFS and BLM policies.

20.10.1.8 Access and Haul Roads

Liberty Gold will reclaim access roads that are not required to support reclamation, monitoring, and maintenance during the post-closure period. Reclamation of road surfaces will include ripping and scarifying the road surface to reduce compaction and allow for root penetration, and grading of the road surfaces to tie into existing ground contours, limit erosion, and to generally reestablish the existing drainage patterns. Liberty Gold will use the material salvaged in the road berms (side-cast material) and road subgrade within the road footprints as growth media for restoration of native plant species in the disturbed areas.

20.10.1.9 Fencing

Liberty Gold will remove and salvage, if possible, fences surrounding the process plant, and HLF event pond once reclamation is complete, and the areas have been released from bonding requirements. Liberty Gold will remove all range and wildlife fences not required to support post-mining land uses after operations. Any removed fence parts would be taken offsite to a licensed facility.

20.10.2 Closure and Reclamation Cost Estimate

Costs were estimated using the Standardized Reclamation Cost Estimator (SRCE) Version 1.4.1. The closure and reclamation cost estimate for the Project is $160.1 million.

A comprehensive Reclamation Plan is expected to be prepared in accordance with applicable Idaho permitting requirements, in consultation with IDL, as part of the ongoing permitting process.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs were estimated for the Feasibility Study by AGP (mine development), M3 (process plant, site development, power transmission and distribution, and ancillaries), and NewFields (heap leach facility and stormwater management). Table 21-1 shows the estimated capital costs for the Project. This includes $411.4 million in pre-production capital and $414.1 million for sustaining capital. Total capital costs are estimated at $825.5 million, which are inclusive of reclamation and closure, presently estimated at $160.1 million and applied at the end of the mine life (starting in Year 20).

Table 21-1: Capital Cost Summary

Category	Units	Initial	Sustaining	Total
Site General (Earthworks)	K USD	$16,765	-	$16,765
Process Plant (ADR, Refinery, Reagents)	K USD	$76,123	$1,400	$77,523
Power Systems	K USD	$7,853	-	$7,853
ADR Bldg. & Ancil. (Warehouse, Maint, Admin, Fuel)	K USD	$33,062	$2,975	$36,037
Freight (Process Plant)	K USD	$5,421	-	$5,421
Sub-Total Direct Cost (Process Plant & Support)	K USD	**$139,225**	**$4,375**	**$143,600**
Construction Support (inc. Mobilization)	K USD	$4,177	-	$4,177
Engineering, Procurement, & Const. Mgmt.	K USD	$19,930	-	$19,930
Vendor Support	K USD	$2,578	-	$2,578
Spare Parts (Capital, Commissioning)	K USD	$2,609	-	$2,609
First Fills (Process Plant)	K USD	$595	-	$595
Contingency (Process Plant)	K USD	$25,367	-	$25,367
Owner's Cost	K USD	$24,367	$3,600	$27,967
Sub-Total Indirect Cost (Process Plant & Support)	K USD	**$79,624**	**$3,600**	**$83,224**
Heap Leach Facility Direct Cost (NewFields)	K USD	$69,257	$109,146	$178,403
Heap Leach Facility Indirect Cost (NewFields)	K USD	$3,905	$6,295	$10,200
Heap Leach Facility Contingency (NewFields)	K USD	$13,827	$24,795	$38,621
Sub-Total Heap Leach Facility	K USD	**$86,990**	**$140,235**	**$227,225**
Stormwater Controls Direct Cost (NewFields)	K USD	$2,036	$18,299	$20,335
Stormwater Controls Indirect Cost (NewFields)	K USD	$121	$1,131	$1,252
Stormwater Controls Contingency (NewFields)	K USD	$379	$4,326	$4,705
Sub-Total Stormwater Controls	K USD	**$2,535**	**$23,757**	**$26,292**
Mine Capital Equipment (AGP)	K USD	$37,004	$46,199	$83,203
Mine Preproduction Costs (AGP)	K USD	$59,382	$35,857	$95,239
Mining Capital Contingency (AGP)	K USD	$6,612	-	$6,612
Sub-Total Mine Capital	K USD	**$102,998**	**$82,056**	**$185,054**
Sub-Total Closure	K USD	**-**	**$160,100**	**$160,100**
TOTAL CAPITAL COST	K USD	**$411,370**	**$414,123**	**$825,493**

Table 21-2 shows the estimated operating costs for the LOM Project. Operating costs were estimated at $3.89 billion for the LOM. This is equivalent to $8.98 per tonne of ore or $1,371 per ounce of payable gold.

Table 21-2: Operating Cost Summary

Category	K USD	Operating Cost	
		$ / tonne processed	$ / Au oz
Mining Costs	$2,635,819	$6.08	$928.74
Process Plant	$898,622	$2.07	$316.63
G&A	$347,100	$0.80	$122.30
Refining	$10,653	$0.02	$3.75
TOTAL OPERATING COST	**$3,892,193**	**$8.98**	**$1,371.43**

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 process solution management systems, including the high-volume application solution pumping systems and rich/lean solution pumping systems, 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 stations.

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 15% 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 by 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 -10%/+15%.

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	8,681	2,305	2,259	1,010	2,510	16,765
Solution Transfer	8,798	10,060	6,717	393	1,433	27,401
ADR	9,086	17,478	6,333	835	1,256	34,989
Refinery	1,259	569	852	153	169	3,001
Water Systems	1,710	3,258	1,370	910	243	7,490
Power Systems	1,629	4,645	1,425	154	0	7,853
Reagents	1,191	1,156	680	62	152	3,241
Ancillary Facilities	6,062	4,454	4,209	17,434	903	33,062
Freight		3,514	1,907			5,421
Sub-Total Direct Cost (Process Plant)	**38,417**	**47,439**	**25,751**	**20,951**	**6,667**	**139,225**
Construction Support (inc. Mobilization)						4,177
Engineering, Procurement, & Const. Mgmt.						19,930
Vendor Support						2,578
Spare Parts (Capital, Commissioning)						2,609
First Fills (Process Plant)						595
Contingency (Process Plant)						25,367
Owner's Cost						24,367
Sub-Total Indirect Cost (Process Plant)						**79,624**
TOTAL CAPITAL COST (Process Plant)						**218,848**

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, Project Management and Administration, Project Services, Project Controls, and Construction Management are included per a proposal from M3 to LGC in June 2026. Included in the proposal are the EPCM Fee, as well as temporary construction facilities and temporary construction power.

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 by 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 are also allowances for external services, such as geotechnical investigations, as well as major execution readiness items such as IT and operator training.

There is an allowance of $3.6 million for a royalty percentage buydown that is envisioned to be executed by Liberty Gold. The royalty buydown would go into effect once the plant is in production and, as such, is included in Year 1 as a sustaining cost.

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 estimated 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 estimated using the available equipment rates in the Standardized Reclamation Cost Estimator (SRCE, 2026).
- Production rates for each unit rate 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 FS-level designs. The MTO was generated from AutoCAD Civil 3D design surfaces and recent survey data as well as typical layouts, sections, and details developed for this Project.

This estimate includes site preparation, earthworks, geosynthetics, piping, and associated storm water management infrastructure such as diversions, culverts 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.

21.3.3 Closure Capital Costs

A detailed SRCE cost model was developed to inform the CAPEX for this FS to aid in estimating reclamation bonds. The SRCE model includes closure and reclamation estimates for all site infrastructure and reflects the closure strategy outlined in Section 20 of this report. The total estimate of $160.1M, including mobilization, de-mobilization, closure of all site facilities, conversion of the solution ponds to an evaporation cell (E-Cell), management of draindown fluid, as well as indirect costs such as construction management, monitoring and maintenance, and water sampling. Table 21-4 summarizes the breakdown of estimated closure and reclamation costs.

Table 21-4: Closure and Reclamation Cost Summary

Component	Total (USD 1,000)
Regrading and Revegetation	$75,995
HLF Solution Management	$37,550
Building Removal	$3,420
Monitoring and Miscellaneous	$7,420
Indirect Costs	$35,715
TOTAL CAPITAL COST (Closure/Reclamation)	**$160,100**

21.4 MINING CAPITAL COSTS

21.4.1 Pre-Production Mining

There are 9 months of pre-production mining planned at Black Pine, where the mine is planning to move 15.2 Mt of waste material, place 8.5 Mt of ore into stockpiles, and place 1.3 Mt on the leach pad. The operating costs associated with 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, mobilization and all maintenance costs. The portion of the pre-production cost that is associated with ore is considered inventory. These costs are included as part of the overall pre-production requirement; however, due to the costs being associated with eventual ore, these costs are included in year 1 of operation as part of the sustaining capital. Contractor mining and equipment finance costs have been added to the operating cost for the pre-production 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 are not included as a mining cost and are covered in other areas.

21.4.2 Mine Equipment

The primary mine fleet and select support equipment are financed. Major equipment vendors provided financing terms including interest rate, down payment, and term length. The down payment is treated as a capital cost with the principal and interest applied to operating costs. 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 21-5.

Table 21-5: Major Mine Equipment – Capital Cost and Full Finance Cost

Equipment	Capital Cost $	Initial Payment $	Total Finance Cost $
Primary Drill (171 mm)	3,235,779	485,400	3,714,519
Secondary Drill (127 mm)	1,272,068	190,800	1,460,095
Hydraulic Shovel (34 m^3)	13,100,000	2,620,000	15,520,795
Production Loader (29 m^3)	8,334,910	1,667,000	9,875,254
Haul Truck (240 t)	5,729,540	859,500	6,577,315
Track Dozer (Leach)	2,777,572	416,700	3,188,793
Track Dozer	1,889,718	283,500	2,169,481
Grader	900,497	180,000	1,066,314
Wheel Dozer	2,695,000	539,000	3,193,018
Water Truck	1,859,042	278,850	2,133,897

Various pieces of support mining equipment were not financed including items such as spare truck boxes or shovel buckets. Other excluded equipment includes the blasting truck, pump truck, ambulance, fire truck, and light plants.

The cost of spare truck trays, loader buckets, and shovel buckets are included in the capital cost for the major equipment cost estimate. Options around full equipment rebuilds and recertification of mining equipment were not considered, and no used equipment is included. All mobile mining equipment is diesel. Pit electrification is not planned for Black Pine currently.

The blasthole drills selected for Black Pine are diesel powered. Diesel drills have more mobility than electric drills and can be moved to different pits and to areas in 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 21-6: 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
Primary Drill (171 mm)	3	2		1					1	3	1						
Secondary Drill (127 mm)	1	1															
Hydraulic Shovel (34 m^3)	2	1															
Production Loader (29 m^3)	1									1							
Haul Truck (240 t)	15	3		4													
Track Dozer (Leach)	1		1							1	1						
Track Dozer	3	1							3	1							
Grader	3									2							
Wheel Dozer		2				1	1					2					
Water Truck	2		1									2					

Table 21-7: Major Equipment Fleet Size

Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Primary Drill (171 mm)	3	5	5	6	6	6	6	6	6	6	6	6	6	6	5	4	
Secondary Drill (127 mm)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Hydraulic Shovel (34 m^3)	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1
Production Loader (29 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck (240 t)	15	18	18	22	22	22	22	22	22	22	22	22	22	22	22	15	15
Track Dozer (Leach)	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Track Dozer	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Grader	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Wheel Dozer		2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Water Truck	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2

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 less than the available fleet hours. In this case, the required hours are distributed evenly across the available equipment fleet on site. Smaller support equipment is based on the number of units required for various locations around 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 support equipment are average values from AGP's experience.

Material for road construction, road maintenance, and blasthole stemming will be provided from suitable waste rock onsite but is planned to be crushed by a contractor. The total crushed rock is expected to average approximately 360 ktonnes/year with the majority (90%) being road rock. The rock crusher and screen plant for this material will be located on site.

The timing of equipment purchases for mining are shown in Table 21-6. Fleet size for the major equipment is shown in Table 21-7.

21.4.3 Miscellaneous Mine Capital

Miscellaneous mine capital covers the cost associated with the engineering office. This includes mining software, computers, plotters, GPS equipment, fleet management system (FMS), slope monitoring equipment etc. Dispatch costs for the equipment units are captured as well as hardware and subscription costs.

Pit dewatering pumps and piping costs are 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 $10.85 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 $500,000.

Total miscellaneous mine capital cost including dewatering infrastructure is estimated at $22.5 million.

21.4.4 Mine Capital Estimate

The mining cost capital, including financing is tabulated in Table 21-8. Inventory (ore) mined during pre-production is allocated in the financial model as a sustaining cost. This is reflected in the table below.

Table 21-8: Mine Capital Cost Estimate

Mine Capital Category	Initial Cost ($)	Sustaining Cost ($)	Total Cost ($)
Pre-Production Mining	59,381,685	35,856,992	95,238,676
Mining Equipment	27,449,250	32,790,700	60,239,950
Miscellaneous Mine Capital	9,554,815	12,994,278	22,549,093
Dewatering Infrastructure	-	414,000	414,000
Contingency	6,612,137	-	6,612,137
Total Mine Capital	**102,997,887**	**82,055,969**	**185,053,856**

21.5 PROCESS OPERATING COST SUMMARY

Process operating costs have been estimated by M3 from first principles. Labor costs were estimated using project specific staffing, salary, wage, and benefit requirements, informed by Liberty Gold. Unit consumption of materials, supplies, power, and delivered supply costs were also estimated. LOM overall average processing costs are estimated at an average cost of $2.07 per tonne of ore.

Operating costs were estimated based on 3rd quarter 2026 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 +/- 15%.

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 cost estimates are based on 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 does not exist, cost allowances are based on consumption and operating requirements from other similar properties for which reliable data exist.

21.5.1 Personnel and Staffing

Staffing requirements for process personnel were estimated by M3 and Liberty Gold based on experience with similar-sized operations in the region. Total process personnel requirements are estimated at 63 persons for the ROM operation; this includes 1 plant superintendent, 4 metallurgists, 6 supervisors, 1 administrative assistant, 30 plant & heap leach pad operators, 2 refinery operators, 4 technicians, plus the 15 maintenance personnel detailed in the maintenance section. Personnel requirements and costs are estimated at $9.1 million per year for the ROM operation. Process labor cost estimates were based on a May 2026 salary benchmarking study received from Liberty Gold. Shift schedules are based on 12-hour shifts with a 4 day on/ 4 day off schedule. Some management positions will be on a 4-day on and 3-day off basis.

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 21-9.

Table 21-9: Power Requirements Summary

	ROM Process		
Area Description	**Connected Power (kW)**	**Demand (kW)**	**Annual (kWh)**
Area 000 - Site General	35	10	90,780
Area 310 - Heap Leach Pad & Ponds	17	1	6,974
Area 350 - Solution Transfer	10,904	7,714	67,578,657
Area 400 - ADR	210	80	699,038
Area 500 - Refinery	243	71	617,595
Area 650 - Water Systems	525	220	1,923,968
Area 800 - Reagents	26	7	63,186
Area 900 - Ancillary Facilities	57	26	224,790
Area 960 - Fuel Station	5	1	10,116
Total	**12,024**	**8,130**	**71,215,103**

The values on the table above are based on Year 3, as that is when the heap leach pad application solution flow rates will be at full capacity. In Years 1 and 2, the annual power consumption will be approximately 62 million kWh per year, roughly 13 percent lower than in Year 3. Power is available via a transmission line connected to the Idaho Power grid; the power cost is estimated at $0.10/kWh, based on input from Liberty Gold.

21.5.3 Consumable Items

The cost of operating supplies was estimated based upon unit costs and consumption rates projected by metallurgical tests. Freight costs are included in all operating supply and reagent estimates. Other consumable items were estimated by M3 based on experience with other similar operations. Table 21-10 presents average consumptions for major consumables.

Table 21-10: Process Consumables Average Annual Consumptions

Item	Delivered Form	Average Annual Consumption
Sodium Cyanide	Liquid at 30% NaCN by Weight OR Dry Briquettes in Iso Container	6,122 tonnes
Lime	Bulk Delivery (20 tonnes)	27,084 tonnes
Antiscalant	Liquid Tote (IBC)	280 tonnes
Carbon	1000 lb Supersacks	140 tonnes
Hydrochloric Acid	Liquid at 30% Acid by weight	350 tonnes
Caustic	Liquid at 50% NaOH by Weight	875 tonnes
Refinery Fluxes	Dry Solid Bags	43 tonnes

Operating costs for consumable items have been distributed based on tonnage placed on the heap leach pad, application solution flow rate, and gold production or smelting batches, as appropriate.

21.5.4 Maintenance

Annual maintenance costs were calculated for the process facilities. The maintenance costs were 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, eight mechanics, and six electricians. These personnel are included as part of the overall process personnel quantity. An allowance for outside repairs is also included at 20% of the maintenance parts allowance. The total annual maintenance is estimated at $5.120 million.

21.5.5 Supplies and Services

Estimates for supplies and services were 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 budget for supplies and services is estimated at $0.928 million.

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 $19.5 million per year for the first sixteen years of operation, which are the full years of active mining and/or ore stacking on the pad. A G&A cost of $8.78 million per year is included for Years 17, 18, 19, and 20, during which the heap leach pad will be irrigated; however, no additional ore is expected to be placed on the pad. Costs have been estimated from first principles for G&A.

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 $3.60/gal ($0.95/L) is used in this estimate.

The mine will employ a contract mining company for the small satellite pits, pioneering work, and general mine support work. The contract mining company is scheduled to mine 92 million tonnes or 9% of the total material mined. The contractor cost is added to the overall mining cost on an annual basis and is estimated from the mine plan and contract mining company quotes.

A Maintenance and Repair Contract (MARC) for the major mine equipment is included in the operating costs. The MARC contract will be structured so that the owner will take on increasing maintenance responsibility as mining progresses. For the first five years (pre-production through Year 4), the MARC will cover 100% of the major mine equipment maintenance. Starting in Year 5, the owner will assume 25% of the maintenance for two years and then 50% for the following two years. In Year 9, the owner will have 75% of the maintenance with 25% allocated to the MARC contract. The 75% owner, 25% MARC rate will continue through the end of the mine life.

Staff cost estimates were based on a May 2026 salary benchmarking study received from Liberty Gold. Shift schedules are 12-hour shifts with a 4 day on/ 4 day off schedule. Some management positions will be on a 4-day on and 3-day off basis. A burden rate of 35% was applied to all rates, as well as expected Short-Term Bonus targets by job level for staff positions. The mine staff labor remains consistent for the mine life. Staff positions and salaries are shown in Table 21-11.

The Engineering department consists of a Chief Engineer, two Senior Engineers, three Planning Engineers, a Blasting Engineer, Surveyors, and a Geotechnical Engineer. The planning engineers will take the short-range planning role (including the leach pad stacking plan) with the Senior Engineers coordinating the long range mine plans and assisting with short range planning. The Geotechnical engineer will cover all aspects of the wall slopes and waste dumps and share the surveyors/mine technicians with the short-range team. The Blasting Engineer is included in the short-range planning group and would double as drill and blast foreman as required. Surveyors and mine technicians will assist in the field with staking, surveying, and sample collection.

In the Geology department, there will be a Chief Geologist, a Senior Geologist responsible for the long range and resources and a Grade Control Geologist working on short range modeling and grade control. There will be four grade control technicians (one per mine operations crew).

The mine will have four mine operations crews, each with a Senior Shift Foremen who has one Junior Shift Foreman reporting to them. For the mine life, there is also a Road Crew/Services Foreman responsible for roads, drainage, and pumping around the mine. This person will also be a backup Senior Mine Shift Foreman. The Mine Operations department will have its own administrator.

The Maintenance General Foreman will report to the Mine Maintenance Superintendent. Four Mine Maintenance Shift Foremen will report to the Maintenance General Foreman. In addition, the mine will employ two maintenance planners/mechanical engineers and an administrative assistant.

Table 21-11: Open Pit Mine Staffing Requirements and Annual Salaries

Staff Position	Employees	Full Load Annual Salary $/a
Mine Maintenance		
Maintenance Superintendent	1	286,011
Maintenance General Foreman	1	229,408
Maintenance Shift Foremen	4	190,728
Maintenance Planner/Contract Admin	2	142,524
Clerk/Secretary	1	119,947
Subtotal	**9**	
Mine Operations		
Mine Ops/Technical Superintendent	1	244,121
Mine General Foreman	1	237,968
Mine Shift Foreman	4	190,728
Junior Shift Foreman	4	149,934
Trainers	1	189,069
Road Crew/Services Foreman	1	190,728
Clerk/Secretary	1	119,947
Subtotal	**13**	
Mine Engineering		
Chief Engineer	1	282,480
Senior Engineer	2	220,848
Open Pit Planning Engineer	3	167,509
Geotech Engineer	1	220,848
Blasting Engineer	1	167,509
Blasting/Geotech Technician	1	142,043
Dispatch Technician	4	167,509
Surveyor/Mining Technician	3	126,260
Surveyor/Mine Tech helper	3	110,478
Subtotal	**19**	
Geology		
Chief Geologist	1	171,200
Senior Geologist	1	138,030
Grade Control Geologist/Modeler	1	108,070
Sampling/Geology Technician	4	96,300
Subtotal	**7**	
Total Mine Staff	**48**	

The number of loaders, trucks, and support equipment operators is estimated using the projected equipment operating hours. The 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 7 is shown in Table 21-12. The labor numbers do not include the contract mining company or MARC employees.

There are 16 shovel and loader operators throughout the mine life, or 4 per shift. Haul truck drivers generally vary from 60 to 80 in full production periods but peak at 92 in Year 3. There are 12 general equipment operators and 8 mine laborers in the general mine labor pool. Along with running support equipment, the general mine personnel will cover vacation and absenteeism in the production workforce.

Table 21-12: Hourly Manpower Requirements and Annual Salary (Year 7)

Hourly Position	Employees	Full Load Annual Salary $/a
Mine General		
General Equipment Operator	12	117,463
Road/Pump Crew	4	107,669
General Mine Laborer	8	97,209
Trainee	4	87,321
Subtotal	28	
Mine Operations		
Driller	28	133,315
Blaster	2	140,157
Blasters Helper	4	110,553
Loader Operator	4	127,447
Hydraulic Shovel/Excavator Operator	12	127,447
Haul Truck Driver	80	101,837
Dozer Operator	16	124,975
Grader Operator	7	124,975
Transfer Loader Operator	2	124,975
Snowplow/Water Truck	7	110,553
Subtotal	162	
Mine Maintenance		
Heavy Duty Mechanic	15	132,613
Light Duty Mechanic	2	138,889
Welder	3	138,889
Electrician	1	123,580
Apprentice	2	98,794
Tire Maintenance	2	119,428
Lube Truck Driver	4	109,476
Subtotal	29	
Total Mine Hourly	219	

Maintenance factors are used to determine the number of heavy-duty mechanics, welders and electricians required and are based on the number of drill operators, shovel operators, truck drivers, etc. Heavy duty mechanics work out to 0.20 mechanics required for each drill operator. Welders are 0.06 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 21-13.

Table 21-13: Maintenance Labor Factors (Maintenance per Operator)

Maintenance Job Class	Drilling	Loading	Hauling	Support
Heavy Duty Mechanic	0.20	0.20	0.20	0.20
Welder	0.06	0.07	0.07	0.06
Electrician	0.05	0.10	-	-
Apprentice	-	-	-	0.01

Major mine equipment hourly costs are listed in Table 21-14. The repair and maintenance cost shown in the table is based on direct owner repair and maintenance parts. The repair and maintenance cost including the level of MARC participation is shown in Table 21-15. Tire costs were also collected from various vendors for the sizes expected to be used. The estimates of 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 240 tonne haulage truck tires is set at 4,000 hours per tire with proper rotation from front to back. The lower-than-standard industry tire life estimate is attributed to the long, downhill, fully loaded ore hauls, which are expected to result in increased tire wear. Each 240 tonne truck tire costs $35,626 so the cost per hour for tires is $53.44/h for the truck using six tires in the calculation.

Table 21-14: Major Equipment Operating Costs – No Labor ($/h)

Maintenance Job Class	Fuel/ Power	Lube/ Oil	Tires/ Under-Carriage	Repair & Maintenance	GET/ Consumables	Total
Primary Drill (171 mm)	57.532	5.75	2.31	97.72	85.16	248.48
Secondary Drill (127 mm)	86.184	8.62	4.62	132.48	128.94	360.84
Hydraulic Shovel (34 m^3)	258.89	25.89	incl	240.00	45.00	569.78
Production Loader (29 m^3)	251.70	25.17	65.23	109.00	45.00	496.10
Haul Truck (240 t)	122.26	12.23	53.44	120.00	6.00	313.92
Track Dozer	57.53	5.75	8.00	128.30	4.00	203.59
Grader	17.66	1.77	3.00	12.00	3.00	37.42
Wheel Dozer	57.21	5.72	32.00	13.27		108.20
Water Truck	53.94	5.39	37.50	61.68	2.31	160.82

Table 21-15: MARC Participation – R&M Cost

Maintenance Job Class	R&M Base	MARC FULL	MARC 75%	MARC 50%	MARC 25%
Primary Drill (171 mm)	97.72	166.12	149.02	131.92	114.82
Secondary Drill (127 mm)	132.48	225.22	202.03	178.85	155.66
Hydraulic Shovel (34 m^3)	240.00	406.04	364.53	323.02	281.51
Production Loader (29 m^3)	109.00	192.18	171.39	150.59	129.80
Haul Truck (240 t)	119.56	148.61	141.35	134.09	126.82
Track Dozer	128.30	165.82	156.44	147.06	137.68
Grader	12.00	63.20	50.40	37.60	24.80
Wheel Dozer	38.00	93.19	79.39	65.60	51.80
Water Truck	61.68	90.29	83.14	75.99	68.83

Ground Engaging Tool (GET) cost 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 171 mm bits. The initial drill platforms and pioneering benches will be developed by the secondary drill. The pattern size for leach

material is smaller than waste to achieve the desired fragmentation. A fragmentation study was completed to model the drill and blast design parameters for various rock types. The parameters shown in Table 21-16 fall within the guidance of the fragmentation study.

Table 21-16: Drill Pattern Specification

Specification	Unit	Production/Presplit Drill		Wall Control	
		Ore	Waste	Buffer	Presplit
Bench Height	m	10	10	10	10
Sub-Drill	m	1.0	1.0	1.0	1.0
Blasthole Diameter	mm	171	171	171	127
Pattern Spacing – Staggered	m	4.80	6.60	5.20	1.50
Pattern Burden – Staggered	m	4.20	5.70	4.50	4.60
Hole Depth	m	11.0	11.0	11.0	11.0

Table 21-17: Drill Productivity Criteria

Drill Activity	Unit	Production Drill		Wall Control	
		Ore	Waste	Buffer	Presplit
Pure Penetration Rate	m/min	0.72	0.72	0.65	0.65
Hole Depth	m	11.0	11.0	11.0	11.0
Drill Time	min	15.3	15.3	16.9	16.9
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
Pull Drill Rods	min	1.0	1.0	1.5	1.5
Total Setup/Breakdown Time	min	4.5	4.5	5.5	5.5
Total Drill Time per Hole	min	19.8	19.8	22.5	22.5
Drill Productivity	m/h	33.4	33.4	29.4	29.4

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 33.4 m/h and the smaller pre-split/secondary production drill is 29.4 m/h. The parameters used to estimate drill productivity are shown in Table 21-17.

Quotations from local explosive vendors were obtained which included delivery to the blasthole. The explosives cost includes monthly fees from the explosive vendor including magazine storage 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 and when required, and with the dry conditions expected, ANFO will be the primary explosive and is applied to 95% of all blastholes. The powder factors used in the explosive calculation are shown in Table 21-18.

Table 21-18: Design Powder Factors

Description	Unit	Leach	Waste
Powder Factor	kg/m^3	0.753	0.393
Powder Factor	kg/t	0.305	0.164

Total monthly cost in the service of delivering the explosives down the hole is $75,000 which includes the vendor's ANFO trucks, prill silos, emulsion silo, accessories magazines, crew trucks, forklift, wash pad, office and 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 loading units. The contract miner with smaller shovels will be 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 21-19. This highlights the focus on the larger more efficient shovels over the contractor's smaller loading units.

Table 21-19: Loading Parameters

Description	Unit	Hydraulic Shovel (34 m^3)	Hydraulic Shovel (6.7 m^3)	Loader (29 m^3)
Bucket Capacity	m^3	34	6.7	29
Waste Tonnage Loaded	%	79	11	10
Leach Ore Tonnage Mined	%	80	12	8
Bucket Fill Factor	%	82	90	95
Cycle Time	sec	35	35	35
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 for material hauled from each phase (by bench), as well as from each stockpile to the respective destinations. These cycle times were then used to estimate 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 21-20.

Table 21-20: Haulage Cycle Speeds

Truck Size	Down (10%)	Down (8%)	Down (5%)	Down (3%)	Flat (0%)	Up (3%)	Up (5%)	Up (8%)	Up (10%)
Haul Truck (240 t)									
Loaded (km/h)	14	20	32	45	50	32	22	14	12
Empty (km/h)	22	25	35	50	50	50	45	28	23

Support equipment hours and costs were determined using the percentages shown in Table 21-21. These percentages resulted in the need for six track dozers including two dozers for the leach pad. Their tasks include cleanup of the mining faces, roads, dumps, blast patterns and ripping on the heap leach.

Three graders and three water trucks will be 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.

Equipment hours are used to estimate the operating costs for each piece of equipment. Several support equipment units have no labor cost directly allocated to them and rely on the general mine operators as needed.

Table 21-21: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Dozer	25%	Of haulage hours to a maximum of 8 dozers
Grader	15%	Of haulage hours to a maximum of 3 graders
Auxiliary Loader	10%	Of loading hours to maximum of 1 loader
Support Backhoe	10%	Of loading hours to maximum of 2 backhoes
Water Truck	10%	Of haulage hours to a maximum of 3 trucks
Lube/Fuel Truck	6	hours/day
Mechanic's Truck	14	hours/day
Welding Truck	8	hours/day
Blasting Loader	8	hours/day
Blaster's Truck	8	hours/day
Integrated Tool Carrier	4	hours/day
Compactor	2	hours/day
Lighting Plants	12	hours/day
Pickup Trucks	10	hours/day
Dump Truck – 20 ton	4	hours/day

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 majority of the mine equipment, and support equipment, where it was considered reasonable, are financed. If the equipment has a life greater than the finance period, then the years following repayment of the loan do not have a finance payment applied. In the case of the mine trucks, 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 $0.29/t to the mine operating cost over the life of the mine. On a cost per tonne of leach ore basis, it is $0.67/t.

The total life of mine operating costs per tonne of material moved and per tonne of leach ore processed are shown in Table 21-22 and Table 21-23. The contract miner cost is included as contractor services.

Please note the unit rates provided in Table 21-20 and Table 21-21 may not add exactly to the total due to rounding of the unit costs of all operating categories.

Table 21-22: Open Pit Mine Operating Costs – with Finance Cost ($/t Total Material)

Open Pit Operating Category		Year 1	Year 5	Year 10	LOM
General Mine and Engineering	$/t moved	0.27	0.21	0.22	0.24
Drilling	$/t moved	0.31	0.30	0.28	0.27
Blasting	$/t moved	0.30	0.29	0.29	0.29
Loading	$/t moved	0.26	0.21	0.20	0.21
Hauling	$/t moved	0.89	0.58	0.47	0.65
Support	$/t moved	0.37	0.25	0.22	0.27
Grade Control	$/t moved	0.02	0.02	0.02	0.02
Finance Costs	$/t moved	0.74	0.24	0.13	0.27
Dewatering	$/t moved	-	-	0.00	0.00
Contractor Services	$/t moved	0.41	0.45	0.36	0.39
Total	$/t moved	3.58	2.56	2.18	2.60

Table 21-23: Open Pit Mine Operating Costs – with Finance Cost ($/t Leach Ore)

Open Pit Operating Category		Year 1	Year 5	Year 10	LOM
General Mine and Engineering	$/t leach ore	0.52	0.54	0.55	0.55
Drilling	$/t leach ore	0.59	0.76	0.71	0.63
Blasting	$/t leach ore	0.57	0.75	0.75	0.67
Loading	$/t leach ore	0.50	0.53	0.51	0.50
Hauling	$/t leach ore	1.71	1.47	1.20	1.52
Support	$/t leach ore	0.72	0.64	0.57	0.64
Grade Control	$/t leach ore	0.05	0.06	0.05	0.05
Finance Costs	$/t leach ore	1.41	0.62	0.34	0.63
Dewatering	$/t leach ore	-	-	0.00	0.00
Contractor Services	$/t leach ore	0.79	1.13	0.92	0.90
Total	$/t leach ore	6.86	6.51	5.61	6.08

22 ECONOMIC ANALYSIS

The economic analysis in this study includes a 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 were developed specifically for this Project and were 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 produced 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. Produced 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 4 years 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

Mined	Units	PP	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	Total
Ore	K Tonnes	9,699	24,142	24,117	32,099	31,171	30,746	30,036	16,347	19,245	20,323	28,045	37,189	37,470	37,580	32,685	22,443	-	433,338
Ore	Au gpt	0.22	0.27	0.40	0.32	0.38	0.42	0.26	0.27	0.24	0.27	0.19	0.21	0.25	0.33	0.30	0.25	-	0.29
Ore	K oz Au	67	207	307	329	379	420	253	144	150	178	170	255	300	396	310	182	-	4,047
Waste	K Tonnes	15,151	27,814	35,883	39,901	40,829	41,254	41,964	55,653	52,670	51,677	43,955	34,811	30,530	22,420	12,315	7,466	28,073	582,366
Mined	K Tonnes	24,850	51,957	60,000	72,000	72,000	72,000	72,000	72,000	71,914	72,000	72,000	72,000	68,000	60,000	45,000	29,910	28,073	1,015,704
Strip Ratio	W:O	1.56	1.15	1.49	1.24	1.31	1.34	1.40	3.40	2.74	2.54	1.57	0.94	0.81	0.60	0.38	0.33	-	1.34
Processed	**Units**	**PP**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**Total**
Ore	K Tonnes	-	28,394	17,258	27,098	26,484	28,311	29,379	16,036	19,022	19,725	27,938	37,000	37,000	37,000	32,592	32,506	17,594	433,338
Ore	Au gpt	-	0.25	0.44	0.33	0.37	0.43	0.26	0.28	0.24	0.27	0.19	0.21	0.25	0.33	0.29	0.27	0.31	0.29
Placed	K oz Au	-	231	243	285	314	390	246	142	148	170	169	254	295	390	309	283	177	4,047
Recoverable	K oz Au	-	155	176	200	225	287	160	100	96	120	120	186	222	302	236	180	77	2,841
Produced	K oz Au	-	145	170	196	223	278	173	107	96	118	120	180	217	295	242	184	85	2,841
Recovery	%	-	67.0%	72.4%	70.3%	71.6%	73.5%	64.8%	70.5%	64.9%	70.5%	71.0%	73.2%	75.1%	77.4%	76.5%	63.5%	43.4%	70.2%

Table 22-2: Life of Mine Process Statistics

Total Ore (kt)	**433,338**
Gold Grade (g/t)	0.29
Contained Gold (kozs)	4,047
Gold Recovery %	70.2%
Produced Gold (kozs)	2,841
Payable Gold (kozs)	2,838

22.3 SMELTER RETURN FACTORS

No contractual payable metal rates have yet been negotiated with a refiner. M3 utilized typical rates based on industry experience or published guidelines. Payable rates for metals used were 99.90% for gold. A bullion refining, transportation and insurance charge of $3.75 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 ($000)

Capital Exp ($000)	PP	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	20	21	22	23	24	Total
Mine Pre-Production	64,144	35,857																				100,001
Mine Capital	38,854	11,862	3,956	6,982	1,450	2,411	1,339	1,059	3,439	5,846	2,433	2,098	1,750	775	400	400						85,053
Process	284,005	17,293	24,657	49,457	2,975	2,860		35,233	35,233		660											452,372
Owner's Cost	24,367	3,600																				27,967
Reclamation/Closure																	60,100	25,000	25,000	25,000	25,000	160,100
Total	411,370	68,612	28,613	56,439	4,425	5,271	1,339	36,291	38,672	5,846	3,093	2,098	1,750	775	400	400	60,100	25,000	25,000	25,000	25,000	825,493

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 $3,250 per troy ounce.

22.6 TOTAL PRODUCTION COST

The total production cost includes mine operations, process plant operations, general administration, 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	$2,635,819
Process Plant	$898,622
G&A	$347,100
Refining	$10,653
Total Operating Cost	**$3,892,193**
Royalty	$45,772
Salvage Value	$0
Total Production Cost	**$3,937,965**

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 one existing royalty prior to production. An Idaho state royalty of 5% NSR is applicable to certain ounces in the mine plan associated with Rangefront. 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 5.3% 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 $3.579 billion.

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 Mid-Year discounting method.

Table 22-5: Key Economic Results

Indicators	Before-Tax	After-Tax
LOM Cash Flow ($000)	$4,460,218	$3,578,943
NPV @ 5% ($000)	$2,928,582	$2,396,566
NPV @ 10% ($000)	$2,016,198	$1,673,650
IRR	63.7%	60.5%
Payback (years)	2.0	2.0

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	Alternate High Case	Alternate High Case	Alternate High Case	Alternate High Case	Alternate High Case	Base Case	Alternate Low Case	Alternate Low Case	Alternate Low Case
Gold Price (per troy oz)	$5,500	$5,000	$4,500	$4,000	$3,500	**$3,250**	$3,000	$2,500	$2,000
Pre-tax Cash Flow in $M	$10,814	$9,402	$7,990	$6,578	$5,166	**$4,460**	$3,754	$2,342	$930
Pre-tax NPV (5%) in $M	$7,325	$6,348	$5,371	$4,394	$3,417	**$2,929**	$2,440	$1,463	$486
Pre-tax IRR	145.7%	127.6%	109.4%	91.3%	72.9%	**63.7%**	54.3%	35.3%	15.7%
Pre-tax Payback (Years)	0.9	1.0	1.2	1.4	1.7	**2.0**	2.3	3.3	4.7
After-tax Cash Flow in $M	$8,554	$7,449	$6,343	$5,239	$4,133	**$3,579**	$3,024	$1,905	$775
After-tax NPV (5%) in $M	$5,888	$5,112	$4,336	$3,562	$2,786	**$2,397**	$2,006	$1,215	$397
After-tax IRR	139.4%	121.8%	104.3%	87.0%	69.4%	**60.5%**	51.6%	33.4%	14.3%
After-tax Payback (Years)	0.9	1.0	1.2	1.4	1.7	**2.0**	2.3	3.4	4.7

22.15 DETAILED FINANCIAL MODEL

The detailed financial model, shown in Table 22-7 below, was developed in compliance with the FS requirement. This model has captured all the parameters of the mine production volume, annual sales revenue, and all 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

	LOM	Pre-Prod	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	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Mine																											
Ore (tonnes)	433,337,539	9,699,344	24,142,470	24,116,877	32,098,659	31,170,837	30,746,368	30,035,873	16,346,950	19,244,835	20,322,602	28,044,933	37,189,459	37,469,695	37,580,299	32,685,297	22,443,042	-	-	-	-	-	-	-	-	-	-
Gold (gpt)	0.290	0.216	0.266	0.395	0.318	0.378	0.425	0.262	0.275	0.243	0.273	0.188	0.213	0.249	0.328	0.295	0.252			-	-	-	-	-	-	-	-
Contained Gold (kozs)	4,047	67	207	307	329	379	420	253	144	150	178	170	255	300	396	310	182	-	-	-	-	-	-	-	-	-	-
Waste (tonnes)	582,366,226	15,150,656	27,814,407	35,883,123	39,901,341	40,829,162	41,253,632	41,964,127	55,653,050	52,669,649	51,677,398	43,955,067	34,810,541	30,530,305	22,419,701	12,314,703	7,466,469	28,072,894	-	-	-	-	-	-	-	-	-
Total Material Mined (tonnes)	1,015,703,764	24,850,000	51,956,876	60,000,000	72,000,000	72,000,000	72,000,000	72,000,000	72,000,000	71,914,483	72,000,000	72,000,000	72,000,000	68,000,000	60,000,000	45,000,000	29,909,511	28,072,894	-	-	-	-	-	-	-	-	-
Process Plant																											
ROM Processing																											
Met_1 (tonnes)	371,105,374	1,250,000	22,804,319	11,960,001	21,370,286	21,983,588	25,314,316	25,993,227	14,607,145	16,834,231	18,836,811	27,538,738	36,329,286	36,459,262	35,711,025	31,910,206	22,202,932	-	-	-	-	-	-	-	-	-	-
Met_2 (tonnes)	34,527,744	-	4,339,823	5,298,067	5,727,964	4,500,708	2,996,717	3,385,320	1,429,294	2,187,607	888,500	399,492	670,714	540,738	1,288,975	681,380	192,444	-	-	-	-	-	-	-	-	-	-
Met_3 (tonnes)	27,704,420	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	10,110,226	17,594,194	-	-	-	-	-	-	-	-	-
Total ROM (tonnes)	433,337,538	1,250,000	27,144,143	17,258,069	27,098,250	26,484,296	28,311,033	29,378,547	16,036,439	19,021,838	19,725,311	27,938,231	37,000,000	37,000,000	37,000,000	32,591,586	32,505,601	17,594,194	-	-	-	-	-	-	-	-	-
Gold (gpt)	0.290	0.315	0.251	0.438	0.327	0.368	0.429	0.261	0.276	0.243	0.269	0.188	0.213	0.248	0.328	0.294	0.271	0.313		-	-	-	-	-	-	-	-
Contained Gold (kozs)	4,047	13	219	243	285	314	390	246	142	148	170	169	254	295	390	309	283	177	-	-	-	-	-	-	-	-	-
Gold Recovery %	70.2%	70.2%	66.9%	72.4%	70.3%	71.6%	73.5%	64.8%	70.5%	64.9%	70.5%	71.0%	73.2%	75.1%	77.4%	76.5%	63.5%	43.4%	-	-	-	-	-	-	-	-	-
Recoverable Gold (kozs)	2,841	9	146	176	200	225	287	160	100	96	120	120	186	222	302	236	180	77	-	-	-	-	-	-	-	-	-
Total Processing																											
Total Ore (tonnes)	433,337,538		28,394,143	17,258,069	27,098,250	26,484,296	28,311,033	29,378,547	16,036,439	19,021,838	19,725,311	27,938,231	37,000,000	37,000,000	37,000,000	32,591,586	32,505,601	17,594,194	-	-	-	-	-	-	-	-	-
Gold (gpt)	0.290		0.253	0.438	0.327	0.368	0.429	0.261	0.276	0.243	0.269	0.188	0.213	0.248	0.328	0.294	0.271	0.313	-	-	-	-	-	-	-	-	-
Contained Gold (kozs)	4,047		231	243	285	314	390	246	142	148	170	169	254	195	390	309	283	177	-	-	-	-	-	-	-	-	-
Gold Recovery %	70.2%		67.0%	72.4%	70.3%	71.6%	73.5%	64.8%	70.5%	64.9%	70.5%	71.0%	73.2%	75.1%	77.4%	76.5%	63.5%	43.4%									
Recoverable Gold (kozs)	2,841		155	176	200	225	287	160	100	96	120	120	186	222	302	236	180	77	-	-	-	-	-	-	-	-	-
Produced Gold (kozs)	2,841		145	170	196	223	278	173	107	96	118	120	180	217	295	242	184	85	9	2	1	1	-	-	-	-	-
Payable Metals																											
Gold (kozs)	2,838		145	169	196	223	277	172	107	96	118	120	180	217	295	242	184	85	9	2	1	1	-	-	-	-	-
Metal Prices																											
Gold ($/oz)	$3,250.00		$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$3,250.00	$0.00	$0.00	$0.00	$0.00	$0.00
Revenues ($000)																											
Gold	$9,223,677		$470,635	$550,685	$635,606	$723,957	$901,524	$560,372	$346,582	$312,410	$383,485	$389,216	$585,133	$704,394	$957,528	$785,200	$597,593	$276,999	$27,833	$7,315	$4,756	$2,454	$0	$0	$0	$0	$0
Total Revenues	$9,223,677		$470,635	$550,685	$635,606	$723,957	$901,524	$560,372	$346,582	$312,410	$383,485	$389,216	$585,133	$704,394	$957,528	$785,200	$597,593	$276,999	$27,833	$7,315	$4,756	$2,454	$0	$0	$0	$0	$0
Operating Cost ($000)																											
Mining	$2,635,819		$186,249	$182,953	$225,716	$208,315	$184,391	$189,056	$165,363	$178,602	$164,169	$156,807	$152,933	$190,956	$165,380	$122,734	$100,876	$61,320	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$898,622		$50,821	$47,466	$54,645	$54,644	$55,377	$53,904	$49,856	$50,754	$50,998	$53,547	$56,412	$56,458	$56,531	$55,195	$55,700	$51,265	$11,267	$11,261	$11,261	$11,260	$0	$0	$0	$0	$0
G&A	$347,100		$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$8,775	$8,775	$8,775	$8,775	$0	$0	$0	$0	$0
Refining	$10,653		$544	$636	$734	$836	$1,041	$647	$400	$361	$443	$450	$676	$814	$1,106	$907	$690	$320	$32	$8	$5	$3	$0	$0	$0	$0	$0
Total Operating Cost	$3,892,193		$257,113	$250,555	$300,595	$283,295	$260,310	$263,107	$235,119	$249,217	$235,110	$230,303	$229,521	$267,728	$242,516	$198,336	$176,766	$132,404	$20,074	$20,045	$20,041	$20,038	$0	$0	$0	$0	$0
Royalty	$45,772		$1,180	$1,380	$1,592	$1,813	$2,256	$1,405	$896	$786	$972	$1,041	$1,496	$1,963	$6,774	$7,811	$13,610	$692	$70	$18	$12	$6	$0	$0	$0	$0	$0
Salvage Value	$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
Total Production Cost	$3,937,965		$258,293	$251,935	$302,187	$285,108	$262,566	$264,512	$236,015	$250,003	$236,082	$231,344	$231,017	$269,691	$249,290	$206,147	$190,376	$133,096	$20,143	$20,063	$20,053	$20,044	$0	$0	$0	$0	$0
Operating Income	$5,285,712		$212,341	$298,751	$333,418	$438,849	$638,958	$295,860	$110,567	$62,407	$147,403	$157,871	$354,117	$434,703	$708,238	$579,053	$407,217	$143,903	$7,690	-$12,748	-$15,297	-$17,590	$0	$0	$0	$0	$0
Depreciation ($000)																											
Total Capital	$810,493		$82,274	$94,699	$97,956	$104,298	$104,886	$23,380	$12,519	$15,463	$18,278	$19,157	$19,227	$18,082	$15,755	$10,276	$8,890	$8,197	$7,778	$3,904	$226	$146	$12,020	$17,020	$22,020	$27,020	$32,020
Total Depreciation	$810,493		$82,274	$94,699	$97,956	$104,298	$104,886	$23,380	$12,519	$15,463	$18,278	$19,157	$19,227	$18,082	$15,755	$10,276	$8,890	$8,197	$7,778	$3,904	$226	$146	$12,020	$17,020	$22,020	$27,020	$32,020
Net Income after Depreciation	$4,475,218		$130,067	$204,051	$235,462	$334,550	$534,072	$272,480	$98,047	$46,944	$129,125	$138,714	$334,889	$416,621	$692,483	$568,777	$398,328	$135,706	-$88	-$16,652	-$15,523	-$17,736	-$12,020	-$17,020	-$22,020	-$27,020	-$32,020
Taxes ($000)																											
Idaho Property Tax	$61,000		$2,757	$2,947	$3,321	$3,350	$3,385	$3,394	$3,635	$3,891	$3,930	$3,950	$3,964	$3,967	$3,917	$3,865	$3,825	$3,749	$1,070	$1,052	$1,029	$0	$0	$0	$0	$0	$0
Idaho Mine License Tax	$32,276		$624	$941	$1,326	$2,416	$4,353	$1,501	$242	$57	$554	$712	$2,434	$3,081	$5,454	$4,458	$3,084	$937	$29	$73	$0	$0	$0	$0	$0	$0	$0
Idaho Corporate Income Tax	$164,635		$0	$1,776	$7,063	$12,848	$23,123	$7,987	$1,303	$320	$2,959	$3,797	$12,935	$16,370	$28,962	$23,675	$16,378	$4,981	$157	$0	$0	$0	$0	$0	$0	$0	$0
Federal Corporate Tax	$623,365		$0	$6,802	$6,309	$44,574	$95,815	$39,055	$6,285	$1,329	$14,725	$16,981	$50,243	$62,454	$109,662	$89,010	$61,109	$18,284	$583	$0	$0	$0	$0	$0	$0	$145	$0

	LOM	Pre-Prod	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	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Total Taxes	$881,275		$3,381	$12,466	$18,019	$63,189	$126,677	$51,936	$11,465	$5,597	$22,168	$25,440	$69,576	$85,873	$147,996	$121,009	$84,395	$27,951	$1,839	$1,125	$1,029	$0	$0	$0	$0	$145	$0
Net Income after Taxes ($000)	$3,593,943		$126,686	$191,585	$217,443	$271,361	$407,395	$220,544	$86,583	$41,347	$106,957	$113,274	$265,313	$330,748	$544,487	$447,768	$313,933	$107,755	-$1,927	-$17,777	-$16,552	-$17,736	-$12,020	-$17,020	-$22,020	-$27,165	-$32,020
Cash Flow ($000)																											
Net Income before Taxes	$4,475,218	$0	$130,067	$204,051	$235,462	$334,550	$534,072	$272,480	$98,047	$46,944	$129,125	$138,714	$334,889	$416,621	$692,483	$568,777	$398,328	$135,706	-$88	-$16,652	-$15,523	-$17,736	-$12,020	-$17,020	-$22,020	-$27,020	-$32,020
Add back Depreciation	$810,493	$0	$82,274	$94,699	$97,956	$104,298	$104,886	$23,380	$12,519	$15,463	$18,278	$19,157	$19,227	$18,082	$15,755	$10,276	$8,890	$8,197	$7,778	$3,904	$226	$146	$12,020	$17,020	$22,020	$27,020	$32,020
Operating Cash Flow	$5,285,712	$0	$212,341	$298,751	$333,418	$438,849	$638,958	$295,860	$110,567	$62,407	$147,403	$157,871	$354,117	$434,703	$708,238	$579,053	$407,217	$143,903	$7,690	-$12,748	-$15,297	-$17,590	$0	$0	$0	$0	$0
Working Capital ($000)																											
Accounts Receivable	$0	$0	-$12,894	-$2,193	-$2,327	-$2,421	-$4,865	$9,347	$5,857	$936	-$1,947	-$157	-$5,368	-$3,267	-$6,935	$4,721	$5,140	$8,783	$6,826	$562	$70	$63	$67	$0	$0	$0	$0
Accounts Payable	$0	$33,811	-$7,039	-$3,827	$6,400	-$5,697	-$1,820	-$93	$572	$1,354	-$3,857	-$621	-$146	$3,112	-$2,152	-$3,662	-$1,773	-$3,679	-$9,233	-$2	$0	$4,939	-$4,532	$0	$0	$0	-$2,055
Inventory (parts)	$0	$0																									
Total Working Capital	$6,519	$33,811	-$19,933	-$6,020	$4,073	-$8,118	-$6,685	$9,253	$6,430	$2,291	-$5,805	-$778	-$5,514	-$156	-$9,087	$1,059	$3,367	$5,104	-$2,406	$560	$70	$5,003	-$4,465	$0	$0	$0	-$2,055
Initial Capital Expenditures ($000)																											
Pre-stripping	$64,144	$64,144	$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
Mining	$38,854	$38,854	$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
Process	$284,005	$284,005	$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
Owner's Cost	$24,367	$24,367	$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
Expansion Capital Expenditures ($000)																											
Mining	$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
Process	$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
Owner's Cost	$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
Sustaining Capital Expenditures ($000)																											
Mining	$82,056		$47,719	$3,956	$6,982	$1,450	$2,411	$1,339	$1,059	$3,439	$5,846	$2,433	$2,098	$1,750	$775	$400	$400	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process	$168,367		$17,293	$24,657	$49,457	$2,975	$2,860	$0	$35,233	$35,233	$0	$660	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$3,600		$3,600	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation/Closure	$160,100		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$60,100	$25,000	$25,000	$25,000	$25,000	$0
Total Capital	$825,493	$411,370	$68,612	$28,613	$56,439	$4,425	$5,271	$1,339	$36,291	$38,672	$5,846	$3,093	$2,098	$1,750	$775	$400	$400	$0	$0	$0	$0	$60,100	$25,000	$25,000	$25,000	$25,000	$0
Cash Flow before Taxes ($000)	$4,460,218	-$377,559	$123,796	$264,118	$281,053	$426,306	$627,002	$303,774	$80,705	$26,026	$135,753	$154,000	$346,505	$432,798	$698,375	$579,712	$410,184	$149,007	$5,283	-$12,188	-$15,227	-$72,687	-$29,465	-$25,000	-$25,000	-$25,000	-$2,055
Cumulative Cash Flow before Taxes ($000)		-$377,559	-$253,763	$10,355	$291,408	$717,714	$1,344,716	$1,648,491	$1,729,196	$1,755,222	$1,890,975	$2,044,975	$2,391,480	$2,824,278	$3,522,653	$4,102,366	$4,512,550	$4,661,557	$4,666,840	$4,654,652	$4,639,425	$4,566,738	$4,537,273	$4,512,273	$4,487,273	$4,462,273	$4,460,218
Cash Taxes Payable	$881,275	$0	$0	$3,381	$12,466	$18,019	$63,189	$126,677	$51,936	$11,465	$5,597	$22,168	$25,440	$69,576	$85,873	$147,996	$121,009	$84,395	$27,951	$1,839	$1,125	$1,029	$0	$0	$0	$0	$145
Cash Flow after Taxes ($000)	$3,578,943	-$377,559	$123,796	$260,736	$268,587	$408,287	$563,813	$177,098	$28,769	$14,562	$130,156	$131,832	$321,065	$363,221	$612,503	$431,717	$289,176	$64,612	-$22,668	-$14,028	-$16,352	-$73,716	-$29,465	-$25,000	-$25,000	-$25,000	-$2,200
Cumulative Cash Flow after Taxes ($000)		-$377,559	-$253,763	$6,973	$275,561	$683,848	$1,247,661	$1,424,759	$1,453,528	$1,468,090	$1,598,245	$1,730,077	$2,051,142	$2,414,364	$3,026,866	$3,458,583	$3,747,758	$3,812,371	$3,789,703	$3,775,675	$3,759,324	$3,685,607	$3,656,143	$3,631,143	$3,606,143	$3,581,143	$3,578,943
Discounted Cash Flow Parameters																											
Year (Mid-Year Discounting)		0	0.5	1.5	2.5	3.5	4.5	5.5	6.5	7.5	8.5	9.5	10.5	11.5	12.5	13.5	14.5	15.5	16.5	17.5	18.5	19.5	20.5	21.5	22.5	23.5	24.5
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	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Factor @ 5%	5.0%	1.000	0.976	0.929	0.885	0.843	0.803	0.765	0.728	0.694	0.661	0.629	0.599	0.571	0.543	0.518	0.493	0.469	0.447	0.426	0.406	0.386	0.368	0.350	0.334	0.318	0.303
Factor @ 10%	10.0%	1.000	0.953	0.867	0.788	0.716	0.651	0.592	0.538	0.489	0.445	0.404	0.368	0.334	0.304	0.276	0.251	0.228	0.208	0.189	0.171	0.156	0.142	0.129	0.117	0.106	0.097
Discounted Cash Flow before Taxes ($000)																											
DCF @ 0%	$4,460,218	-$377,559	-$253,763	$10,355	$291,408	$717,714	$1,344,716	$1,648,491	$1,729,196	$1,755,222	$1,890,975	$2,044,975	$2,391,480	$2,824,278	$3,522,653	$4,102,366	$4,512,550	$4,661,557	$4,666,840	$4,654,652	$4,639,425	$4,566,738	$4,537,273	$4,512,273	$4,487,273	$4,462,273	$4,460,218
DCF @ 5%	$2,928,582	-$377,559	-$256,747	-$11,268	$237,512	$596,896	$1,100,301	$1,332,580	$1,391,352	$1,409,403	$1,499,071	$1,595,949	$1,803,546	$2,050,495	$2,430,005	$2,730,030	$2,932,208	$3,002,155	$3,004,517	$2,999,328	$2,993,153	$2,965,082	$2,954,244	$2,945,487	$2,937,147	$2,929,204	$2,928,582
DCF @ 10%	$2,016,198	-$377,559	-$259,524	-$30,591	$190,875	$496,259	$904,581	$1,084,423	$1,127,859	$1,140,593	$1,200,975	$1,263,247	$1,390,623	$1,535,256	$1,747,424	$1,907,532	$2,010,519	$2,044,530	$2,045,627	$2,043,328	$2,040,716	$2,029,384	$2,025,209	$2,021,988	$2,019,059	$2,016,397	$2,016,198
Discounted Cash Flow after Taxes ($000)																											
DCF @ 0%	$3,578,943	-$377,559	-$253,763	$6,973	$275,561	$683,848	$1,247,661	$1,424,759	$1,453,528	$1,468,090	$1,598,245	$1,730,077	$2,051,142	$2,414,364	$3,026,866	$3,458,583	$3,747,758	$3,812,371	$3,789,703	$3,775,675	$3,759,324	$3,685,607	$3,656,143	$3,631,143	$3,606,143	$3,581,143	$3,578,943
DCF @ 5%	$2,396,566	-$377,559	-$256,747	-$14,411	$223,335	$567,529	$1,020,200	$1,155,617	$1,176,568	$1,186,667	$1,272,638	$1,355,570	$1,547,926	$1,755,176	$2,088,021	$2,311,452	$2,453,985	$2,484,316	$2,474,182	$2,468,209	$2,461,578	$2,433,109	$2,422,272	$2,413,515	$2,405,175	$2,397,231	$2,396,566
DCF @ 10%	$1,673,650	-$377,559	-$259,524	-$33,522	$178,121	$470,598	$837,768	$942,615	$958,098	$965,223	$1,023,116	$1,076,424	$1,194,448	$1,315,830	$1,501,910	$1,621,143	$1,693,748	$1,708,496	$1,703,793	$1,701,146	$1,698,342	$1,686,850	$1,682,674	$1,679,453	$1,676,525	$1,673,863	$1,673,650
Financial Indicators before Taxes ($000)																											
NPV @ 0%	$4,460,218																										
NPV @ 5%	$2,928,582																										
NPV @ 10%	$2,016,198																										
IRR	63.7%																										

	LOM	Pre-Prod	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	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Payback (years)	2.0		1.0	1.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Financial Indicators after Taxes ($000)																											
NPV @ 0%	$3,578,943																										
NPV @ 5%	$2,396,566																										
NPV @ 10%	$1,673,650																										
IRR	60.5%																										
Payback (years)	2.0		1.0	1.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Au (kozs)	2,838	-	145	169	196	223	277	172	107	96	118	120	180	217	295	242	184	85	9	2	1	1	-	-	-	-	-
Mining	$2,635,819	$0	$186,249	$182,953	$225,716	$208,315	$184,391	$189,056	$165,363	$178,602	$164,169	$156,807	$152,933	$190,956	$165,380	$122,734	$100,876	$61,320	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$898,622	$0	$50,821	$47,466	$54,645	$54,644	$55,377	$53,904	$49,856	$50,754	$50,998	$53,547	$56,412	$56,458	$56,531	$55,195	$55,700	$51,265	$11,267	$11,261	$11,261	$11,260	$0	$0	$0	$0	$0
G&A	$347,100	$0	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$19,500	$8,775	$8,775	$8,775	$8,775	$0	$0	$0	$0	$0
Refining	$10,653	$0	$544	$636	$734	$836	$1,041	$647	$400	$361	$443	$450	$676	$814	$1,106	$907	$690	$320	$32	$8	$5	$3	$0	$0	$0	$0	$0
Royalty	$45,772	$0	$1,180	$1,380	$1,592	$1,813	$2,256	$1,405	$896	$786	$972	$1,041	$1,496	$1,963	$6,774	$7,811	$13,610	$692	$70	$18	$12	$6	$0	$0	$0	$0	$0
Cash Cost before By-Product Credit	$3,937,965	$0	$258,293	$251,935	$302,187	$285,108	$262,566	$264,512	$236,015	$250,003	$236,082	$231,344	$231,017	$269,691	$249,290	$206,147	$190,376	$133,096	$20,143	$20,063	$20,053	$20,044	$0	$0	$0	$0	$0
$/Au oz	$1,388	$0	$1,784	$1,487	$1,545	$1,280	$947	$1,534	$2,213	$2,601	$2,001	$1,932	$1,283	$1,244	$846	$853	$1,035	$1,562	$2,352	$8,914	$13,702	$26,541	$0	$0	$0	$0	$0
Silver Credit	$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
Cash Cost after By-Product Credit	$3,937,965	$0	$258,293	$251,935	$302,187	$285,108	$262,566	$264,512	$236,015	$250,003	$236,082	$231,344	$231,017	$269,691	$249,290	$206,147	$190,376	$133,096	$20,143	$20,063	$20,053	$20,044	$0	$0	$0	$0	$0
$/Au oz	$1,388	$0	$1,784	$1,487	$1,545	$1,280	$947	$1,534	$2,213	$2,601	$2,001	$1,932	$1,283	$1,244	$846	$853	$1,035	$1,562	$2,352	$8,914	$13,702	$26,541	$0	$0	$0	$0	$0
Sustaining Capital Expenditures																											
Mining	$82,056	$0	$47,719	$3,956	$6,982	$1,450	$2,411	$1,339	$1,059	$3,439	$5,846	$2,433	$2,098	$1,750	$775	$400	$400	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process	$168,367	$0	$17,293	$24,657	$49,457	$2,975	$2,860	$0	$35,233	$35,233	$0	$660	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$3,600	$0	$3,600	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value	$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
Reclamation/Closure	$160,100	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$60,100	$25,000	$25,000	$25,000	$25,000	$0
Idaho Property Tax	$61,000	$0	$2,757	$2,947	$3,321	$3,350	$3,385	$3,394	$3,635	$3,891	$3,930	$3,950	$3,964	$3,967	$3,917	$3,865	$3,825	$3,749	$1,070	$1,052	$1,029	$0	$0	$0	$0	$0	$0
Idaho Mine License Tax	$32,276	$0	$624	$941	$1,326	$2,416	$4,353	$1,501	$242	$57	$554	$712	$2,434	$3,081	$5,454	$4,458	$3,084	$937	$29	$73	$0	$0	$0	$0	$0	$0	$0
AISC	$4,445,364	$0	$330,287	$284,436	$363,273	$295,300	$275,576	$270,746	$276,183	$292,622	$246,412	$239,099	$239,513	$278,490	$259,436	$214,870	$197,685	$137,781	$21,243	$21,188	$21,082	$80,144	$25,000	$25,000	$25,000	$25,000	$0
$/Au oz	$1,566	$0	$2,281	$1,679	$1,857	$1,326	$993	$1,570	$2,590	$3,044	$2,088	$1,997	$1,330	$1,285	$881	$889	$1,075	$1,617	$2,480	$9,414	$14,405	$106,121	$0	$0	$0	$0	$0
Total Capital Expenditures ($000)																											
Pre-stripping	$64,144																										
Mining	$120,910																										
Process	$452,372																										
Owner's Cost	$27,967																										
Closure Costs	$160,100																										
Total	**$825,493**																										
Pre-Strip & Mining	$185,054																										
Process & Owner's Cost	$480,340																										
Closure Costs	$160,100																										
LOM Cost per Payable Gold Ounce																											
Mining	$928.74																										
Process Plant	$316.63																										
G&A	$122.30																										
Refining	$3.75																										
Royalty	$16.13																										
Cash Cost before By-Product Credit	$1,387.56																										
Production Cost Minus Royalty	$1,371.43																										

23 ADJACENT PROPERTIES

There are no adjacent properties to report in this section.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 SILVER

The current silver Mineral Resource estimation has only been completed in 2 of the 8 mineralization zones at Black Pine (Discovery and M-zone) because of targeted silver assays completed with a 4-acid ICP-OES analysis (original analysis protocols used 2-acid ICP-OES). A total of 6,524 silver re-assay have been completed. Since 2-acid ICP-OES analysis for silver is not robust enough to build a reliable resource estimation, the other areas of the deposit were omitted from the current estimate. As of the completion of this report, an additional silver re-assaying program has begun for the remaining 6 mineralization zones using the 4-acid ICP-OES method, targeting an additional 5,183 re-assays. Review of the 2-acid silver dataset indicates there are localized zones of high-grade silver mineralization, across the deposit, which have not yet been estimated. With an updated deposit wide 4-acid ICP-OES dataset for silver, a new deposit wide Mineral Resource estimation can be completed and fully integrated into mine planning.

As a result of the limited silver resource estimation Liberty Gold determined it should be excluded from any reserve modeling in this report, however, process considerations have been accounted for with the expectation that some amount of silver would be produced from the leach pad. For context, Pegasus produced 0.47 oz Ag for every 1 oz Au during production from 1993 to 1997 (Annual Report on Form 10-K for the Fiscal year Ended December 31, 1997).

Due to the silver reserve exclusion, it has not been included in any of the financial modeling, which offers an opportunity to recognize additional deposit value when a deposit wide silver estimation can be incorporated into the financial analysis.

24.2 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 can potentially be converted to Indicated with additional infill drilling. Results from previous drill programs completed by Liberty Gold suggest high conversion rate for Inferred to Indicated mineralization.

Table 24-1 below details the quantities of Inferred Mineral Resources which are mined and treated as waste in the production schedule presented in Section 16. A total of 58.4 M tonnes of Inferred Mineral Resources at 0.14 g/t Au and containing 171,000 potentially recoverable ounces of gold are mined over 15 years. Additionally, the 2026 drill program focused on upgrading resource classification will convert much of the Inferred material in the early years of mining to Measured or Indicated.

Table 24-1: Inferred Mineral Resources Mined by Year

Period	Indicated			Inferred		
	Tonnes	Diluted Au Grade (g/t)	Recovered Au Grade (g/t)	Tonnes	Diluted Au Grade (g/t)	Recovered Au Grade (g/t)
pp-1	9,702,967	0.22	0.13	1,364,849	0.13	0.08
y1	24,169,424	0.27	0.17	2,118,345	0.14	0.08
y2	24,139,050	0.40	0.26	2,980,534	0.15	0.09
y3	32,111,128	0.32	0.21	3,156,572	0.15	0.10
y4	31,192,833	0.38	0.25	3,665,817	0.16	0.10
y5	30,762,421	0.42	0.30	6,079,947	0.15	0.10
y6	30,056,620	0.26	0.17	3,784,899	0.14	0.09
y7	16,368,118	0.28	0.19	3,277,256	0.14	0.09
y8	19,259,320	0.24	0.16	2,617,171	0.13	0.08
y9	20,339,894	0.27	0.19	5,361,841	0.14	0.10
y10	28,053,046	0.19	0.13	7,860,444	0.13	0.09
y11	37,216,520	0.21	0.16	7,091,268	0.14	0.10
y12	37,519,386	0.25	0.19	2,735,413	0.14	0.09
y13	37,654,015	0.33	0.25	2,334,661	0.15	0.09
y14	32,695,881	0.30	0.23	1,859,815	0.13	0.09
y15	22,452,116	0.25	0.19	2,121,163	0.14	0.09
Total	433,692,738	0.29	0.20	58,409,996	0.14	0.09

24.3 2026 EARLY YEAR PRODUCTION DRILLING (DTM)

For 2026, a drill program has been designed to upgrade the mineral resource classification from Indicated to Measured for the early years of the mine plan presented in this report. The average nominal drill spacing for the DtM program will provide data at <30 m for an updated resource estimation in order to de-risk the early years of production by upgrading the confidence level of the mineral resources, better define the geology and metallurgical domains, and provide the data for an initial short-range model creation. The completion of the DtM program will also provide baseline data to use for evaluating the cost/benefit of RC grade control drilling during production. The program is expected to be completed in Q3-2026 with final results modeled by year-end 2026.

24.4 CONVERSION OF LEGACY WASTE ROCK FACILITIES AND BACKFILL TO RESERVES

The drill spacing through much of the legacy waste rock facilities and backfill material at Black Pine meet the in-situ definition for Indicated mineral resource (≤ 60 m), but it was determined that a tighter sample density was required to classify the volumes as Indicated due to the nature of the material. To that end, in May 2026 Liberty Gold commenced a drill program to define the legacy waste rock facilities and backfill at a density of ≤ 40 m sample spacing which will allow those volumes to be classified as Indicated and included in future reserve calculations. The program was completed in July 2026 and results are expected in early Q4.

24.5 GEOMETALLURGICAL MODEL UPDATES

The geometallurgical models presented in this study are based on the metallurgical test work completed to date and represent the current understanding of ore variability and processing response across the deposit. As additional metallurgical test work is completed on pending samples and future drilling programs, the geometallurgical models will be refined to incorporate new data and improved understanding of ore characteristics. While the current models are considered accurate for the purposes of this study, future updates may result in minor changes to model parameters,

domain definitions, metallurgical recovery assumptions, and other related inputs used for mine planning and process evaluation.

At the time of this feasibility study, there are five pending metallurgical test programs. Phases 6, 8B, 10, and 11 are in process at KCA and Phase 12 is pending drilling of the historic waste dumps.

- Phase 6 - Bulk sample column tests from across the deposit (7 samples)
- Phase 8B - Continued analysis of the LHLF (7 composites)
- Phase 10 - Infill metallurgical work in Rangefront focused on low grade Pola and Ppos (26 composites)
- Phase 11 - Low grade samples <0.1 g/t (14 composites)
- Phase 12 - Historic waste dump recovery work (no composites yet selected)

24.6 LOW PRICE SENSITIVITY

The Black Pine Project benefits from significant operational flexibility from the multiple mine pits and phases that may enhance its resilience under lower gold price scenarios. Should a sustained decrease in gold price warrant higher cut-off grades, portions of lower-grade material could be deferred through stockpiling or re-classified as waste while prioritizing the mining and processing of higher-grade ore.

The grade distribution within the current mineral reserve provides the opportunity to maintain meaningful gold production levels at reduced heap stacking rates and increased gold grades for a period of time, potentially supporting positive project economics during adverse commodity price conditions. This flexibility may provide management with an additional mechanism to preserve value and optimize mine sequencing in response to changing market conditions. Optimized mine plan and ore routing would likely improve results from low gold price scenarios seen in the sensitivity results in Table 22-6 as those scenarios assume no change in mine plan sequencing or ore routing.

24.7 AUTONOMOUS MINING OPERATIONS

Early evaluations of autonomous haulage have been performed and indicate potential operational efficiencies through reduced operating delays and increased productivity, as well as safety improvements. Autonomous haulage has not been included in the Feasibility Study, however the pits and mining operation have been designed to be “autonomous ready”. In order to support potential future autonomous hauling operations additional road width and other pit design considerations have been included in the Feasibility Study designs. The current data network and communications design has also been built for scalability later to support the increased network coverage and broadband requirements of an autonomous operations. In addition to autonomous haulage, semi or fully autonomous drilling and dozing are also being considered as part of the technology implementation plan for site and further evaluations of these systems are planned to continue prior to potential operations.

24.8 PURCHASE MINING EQUIPMENT

Purchasing the entire mining fleet will reduce the total operating cost through the elimination of financing charges and improve long-term unit costs. The unit operating cost savings per tonne mined by purchasing the mining fleet is estimated to be $0.22-$0.29/mined tonne ($0.65-$0.80/ore tonne). An owner purchased fleet has several advantages including the overall operating cost and tax savings, flexibility on deployment, and maintenance scheduling. Additional capital expenditure both upfront and throughout the mine life would be required with the largest increases during Pre-Production, Year 1, and Year 8/9 when equipment replacements occur. Indications from cashflow modeling of both the lease and capital purchase case indicate the potential to improve life of mine Free Cash Flow (FCF) and NPV through capital purchasing of equipment. At a high level, purchasing of mine equipment could reduce AISC by approximately $40 per ounce of payable gold.

24.9 POTENTIAL FUTURE HLF EXPANSION

The HLF as currently designed has an additional 45 million tonnes of additional capacity, compared to the 433 million tonnes of leachable ore and 30.8 million tonnes of material from the LHLF. As part of the initial FS tradeoff studies, additional future expansion areas were identified to the south (Phase 4) and north (Phase 5) which could create additional capacity of approximately 90 million tonnes. These potential future expansion areas for the HLF are not included in the FS due to permitting and timing limitations. FS infrastructure was sited to generally accommodate the potential future expansions without significant redesign or reconstruction.

24.10 OPTIMIZATION OF FUTURE FRAGMENTATION TARGETS

Current fragmentation targets are based on the analysis described in Section 13.14. These P_{80} values represent an estimate of the fragmentation for each material type, based upon detailed metallurgical core logging and fragmentation analysis completed as part of this feasibility study which show that target P_{80} values are achievable. However, these may not be the optimal P_{80} values given a tradeoff between drill and blast costs and recovery gain or loss.

As discussed in Section 13, the majority of Black Pine metallurgical ore types are insensitive to fragmentation size and do not require any special blasting modifications to achieve their natural target P_{80}'s. This creates an opportunity to save cost through optimizing fragmentation targets. A small portion of the deposit contains overly blocky material (mostly mineralized dolomite rock) that can result in short circuiting issues in the heap which could result in lower recovery than planned. The objective of this work would be to refine the optimum fragmentation strategy by ore type and domain, balancing mining costs against metallurgical performance and overall project value.

24.11 CONTRACT CRUSHING

The Project has an opportunity to replace the currently planned contract crushing of stemming material, road construction material, and potentially overliner with an owner-purchased and operated crushing system. Bringing this work in house could reduce long-term unit costs, improve control over material quality and production schedules, and provide greater flexibility in responding to changing mine requirements. Although this approach would require upfront capital, additional staffing, maintenance capability, and operating oversight, a life-of-project cost and risk assessment may demonstrate attractive value compared with continued reliance on a third-party contractor.

25 INTERPRETATION AND CONCLUSIONS

The authors of this Technical Report consider that Black Pine is a project of merit and warrants advancing the study to detailed engineering and ultimately project permitting and 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 an accurate and reasonable representation of the Black Pine property. Based on the positive results of this FS, 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 presence of 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 ROM 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 pad and processing facilities.

This technical report indicates an average payable gold over the estimated 16-year LOM of approximately 176,700 oz per year, with early mine life peak payable gold in Year 5 of 277,400 oz of gold and ultimate peak payable gold in year 13 of 294,600 oz of gold. Payable gold anticipated in the first 5 years of operation averages 202,000 ounces per year. Cash costs are estimated to be $1,388/oz of payable gold, and the AISC are estimated to be $1,566/oz of payable gold. The resulting after-tax cash flow is $3.58 billion, for an after-tax NPV (5%) of $2.40 billion and an estimated payback period of 2.0 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 - 530 Mt grading 0.30 g/t Au, containing 5.04 Moz Au
- Inferred - 154 Mt grading 0.21 g/t Au, containing 1.04 Moz Au

The 2026 drill program currently underway will provide data to upgrade the Resource classification from Indicated to Measured for the first 3 years of mining in Disco and the first phase of Rangefront. This drilling aims to de-risk the early years of production and increase confidence in the geology and geometallurgical models. Additionally, the 2026 drill data offers the opportunity to outline further Mineral Resources beyond what is stated in this report.

25.2 METALLURGICAL TEST WORK

The Black Pine Project is predominantly a gold oxide deposit with negligible sulfide content and some organic carbon. All carbonaceous (Poc) materials have been geologically identified through 3D modeling and classified as waste in 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 lower process OPEX.

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 Discovery Zone, and blocky dolomites from C/D Pit is planned at the mining face and on top of the heap lifts (before dozing) as required. To ensure optimal gold recovery, Met Types 1, 2, and 3 are planned to be stacked separately on the heap leach pad, or stockpiled if necessary, to avoid intermixing, to prevent potential degradation of recovery performance.

The Black Pine deposits also exhibit low levels of potential cyanide-consuming elements: Sulfides, Cu, Ni, and Zn and very low levels of potentially toxic elements such as As, Sb, Hg and Se further reinforcing the metallurgical and environmental advantages of the project.

25.3 MINERAL RESERVE ESTIMATES

The Mineral Reserve estimate for the Project conforms to industry-accepted practices and is reported using the CIM Standards.

The Project remains subject to active exploration and resource expansion drilling, which is expected to provide additional information that may support future increases to the Mineral Resource estimate. To the extent that additional Mineral Resources are delineated in the appropriate confidence categories and are subsequently demonstrated to be economically mineable following the application of relevant Modifying Factors, such resource growth may also support future additions to the Mineral Reserve estimate. No potential additions resulting from ongoing or future drilling are included in the Mineral Reserve estimate presented in this Technical Report

Factors that may affect the estimate include: changes to long-term metal price assumptions; modifications to recovery assumptions based on further metallurgical test work and determination of material separation on the heap leach due to different metallurgical properties; timely construction of surface haul road construction to allow access to the pits; effective excavation and control of open pit slopes, and maintaining the bench advance rate by efficiently handling 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 feasibility level open pit slope design study has been completed, and the current geotechnical dataset is considered adequate for the feasibility study level designs.

All pits are considered dry, and the requirement for either pit dewatering or de-pressurization is not anticipated. The projected pit bottoms in the later phases of the Rangefront and CD Pits are designed to remain 15 m away from the current phreatic surface. Routine monitoring of groundwater elevations is part of the environmental monitoring and compliance program.

Each pit phase was designed to accommodate the projected 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 the lithology type and kinematic sector. The haul roads will be 37.5 m or 22.8 m in width depending on the fleet selected and have a road grade of 10%.

The mine plan schedule delivers 433 Mt of ore to the leach pad grading 0.35 g/t Au in the first 5 years, 0.30 g/t Au in the first 10 years, and 0.29 g/t Au over the 16 year mine life, including one year of placing only stockpiled ore after in-pit mining has ended. Waste rock tonnage from all pits (including the stockpiling of the LHLF material on the new leach pad) totals 582 Mt and is placed into rock storage facilities including the backfilling of the Discovery Phase 3, CD, and M pits. The overall open pit strip ratio is 1.3:1.

The leach production schedule has a feed rate of 27.1 M tpa (74 ktpd) starting in Year 1, and averages 24.1 Mt from Year 2 through Year 10, 37.0 Mt from Year 10 to Year 13, and 32.5 Mt in Years 14 and 15. Met 1 and Met 2 leach ore stockpiled in pre-production is sent to the leach pad in Year 1.

There are two dedicated rock storage facilities plus three backfilled pits for waste rock storage. Three stockpiles are available based on metallurgical ore types.

The mine equipment fleet is anticipated to be leased to lower capital requirements.

25.5 HEAP LEACH FACILITY

The HLF achieves the design requirements as stipulated by Idaho State Regulations, including an 80-mil double-sided textured HDPE geomembrane liner underlain by 12 in of compacted soil with a hydraulic conductivity equivalent to that of the prescribed 24-in-thick layer of $1x10^{-6}$ cm/s material. Geotechnical investigations and laboratory testing confirmed suitable low permeability materials exist in local borrow sources wholly owned by Liberty Gold for the life-of-mine facility.

Ore will generally be stacked from downhill to uphill and barren solution applied for the leaching process. Rich leachate will drain into the solution collection system installed immediately above the geomembrane liner. The solution collection system consists of a network of perforated pipes surrounded by free-draining gravel, currently expected to be sourced from legacy mining materials or fresh waste rock readily available on site. The perforated pipes will transition to solid-wall HDPE pipes that will convey process solution within the HLF to the process solution tanks located within the double-lined pond system (with leak detection). From the tanks, solution will be recirculated to the heap or processed at the ADR plant.

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 500 ft above the geomembrane liner. Existing infrastructure such as the land application area for the LHLF, existing powerlines, and wells, will need to be removed or relocated prior to constructing the HLF.

25.6 ENVIRONMENTAL AND PERMITTING

The Black Pine Project is advancing through a structured federal and state permitting process, with the 2026 MPO deemed administratively complete by the USFS and BLM and the NEPA process initiated through an NOI to prepare an EIS. Acceptance into the FAST-41 process provides a coordinated permitting timetable, transparency, and interagency accountability. At the time of this report, no material environmental issues have been identified that are expected to preclude permitting; however, the Feasibility-level mine plan will require additional approvals or modifications, including a revised MPO, potential BLM lease modification, updated IDL reclamation plan and financial assurance, cyanidation permit modification, POC evaluation, and review of whether a modified IDEQ air permit is required due to the increased HLF placement rate.

The primary environmental and permitting risks are associated with agency review of the revised mine plan and resolution of liabilities related to the LHLF. Removal and relocation of LHLF material could reduce long-term legacy-site risk and support reuse of suitable material in the new HLF, but the extent of residual liability remains uncertain, including potential issues related to process-affected material, residual cyanide-related constituents, nitrates, metals, contact water, liner and underdrain conditions, subgrade impacts, and financial assurance.

25.7 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.
- 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 58.4M tonnes at 0.14 g/t Au and 171,000 recoverable Au ounces.
- Discovery of additional resources through exploration and expansion drilling. There are numerous exploration targets within the Project boundaries that Liberty continues to evaluate.
- 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 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, and electrification of the mining fleet
- Investigate the relationship between blast fragmentation and gold recovery and redesign drill and blast patterns to optimize HLF feed particle size distribution.
- Modify the pond and tank configuration to allow for process solution tanks and associated pumps to be located outside the pond system. This would include adjusting the pad grading to ensure all phases of pad grading adequately drain by gravity to a higher tank elevation.
- Explore all potential borrow sources for overliner, including the LHLF, legacy RSF, pit development rock, and other potential borrow sources.
- The slope angles for the Rangefront Pit within the Sandstone unit may be steepened if additional drilling confirms that the Sandstone rock mass forming the final pit walls outside the mineralized zone is of better quality than currently interpreted.

25.8 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:

- As with other mining projects, the Project may be subject to broader social, legal, and political risks, including the potential for third-party challenges or litigation, which could affect the schedule for advancement to construction.
- The nature and spatial extent of any residual environmental liability associated with the LHLF has not been fully defined. If future characterization or removal activities identify impacted subgrade soils, residual process solutions, or contact water beneath or adjacent to the LHLF, Liberty Gold could be required to delineate,

excavate, treat, dispose of, or otherwise manage affected materials in accordance with applicable regulatory requirements and agency-approved closure or reclamation plans.

- 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 requirement to establish a road network in difficult terrain the support fleet will be tasked with building and maintaining a considerable amount of surface and access roads in pre-production and early years of the operation. This is considered an important, but manageable operating risk to meet production targets.
- Mining in the steep terrain when opening new mining areas could be inefficient and could result in reduced mining production if not accounted for, 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.
- 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.

25.9 RELATIONSHIP TO ACTIVE PERMITTING AND CONTINUING PROJECT DEVELOPMENT

The Qualified Persons consider the engineering, design criteria, production schedules, capital and operating cost estimates, and supporting technical work developed for the Feasibility Study to be of sufficient scope and level of detail to support the Mineral Reserve estimates, production forecasts, economic analysis and conclusions presented in this Technical Report. The Feasibility Study provides a feasibility-level technical and economic basis for advancing the Project through financing, detailed engineering, procurement, construction planning and continued permitting. Although certain Project components will be further refined through detailed engineering and permit-specific design, such further refinement is not intended to diminish the reliability of the technical and economic evaluations presented herein or the suitability of the designs to support the mining rates, solution flow rates, process capacities and production levels evaluated in the Feasibility Study.

For greater certainty, this Technical Report is not intended to replace, supersede, amend, supplement or otherwise modify the MPO, any pending permit application, or any other document, analysis, consultation, submission or regulatory process currently under review by the applicable federal, state or local regulatory authorities. The MPO and the environmental, engineering and technical information formally submitted in support of the active permitting process remain the basis upon which the applicable regulatory authorities are conducting their respective reviews. Nothing in this Technical Report is intended to request that the applicable regulatory authorities reopen, expand, delay or otherwise alter the scope or schedule of any active permitting activity, or to substitute the Feasibility Study for any information required to be submitted through the established permitting process.

The development configuration evaluated in this Feasibility Study is consistent with the Project described in the current MPO; however, it expands upon the current MPO by incorporating feasibility-level optimizations and refinements not contemplated therein or under existing permitting. The Feasibility Study does not contemplate a change to the fundamental mining or mineral-processing methods described in the MPO. The Project remains a conventional open-pit mining operation in which run-of-mine oxide ore is placed directly on an engineered heap leach facility, and gold and silver are recovered through conventional cyanide leaching and an adsorption, desorption and recovery (ADR) plant and refinery. The Feasibility Study expands on the current MPO and envisions changes to scheduled mining

rates, process flow rates, equipment capacities and the sequencing or configuration of certain facilities. The outcomes of the Feasibility Study will need to be further advanced through additional studies and future approval by Liberty Gold. Where these finalized studies indicate differences with the current MPO, Liberty will engage with applicable regulatory agencies and, when required, address differences through supplemental permitting processes and approvals.

Any differences between the Feasibility Study and the MPO must be evaluated in the future as a separate permitting process with the applicable regulatory authorities and through the established permitting process. To the extent that any such difference requires supplemental information, revised modeling, a permit modification, an amendment, an additional authorization or any other regulatory action, the Company intends to address that requirement through the applicable regulatory procedure before implementing the relevant activity. The inclusion of feasibility-level refinements in this Technical Report should not be interpreted as predetermining whether any additional regulatory review or approval is required, which determination remains within the jurisdiction of the applicable regulatory authorities.

Engineering, mine planning and permitting information will continue to be evaluated as it becomes available. This includes, without limitation, results from the 2026 drilling program and related geological, metallurgical, geotechnical, hydrogeological and environmental investigations that were not available for, and therefore were not incorporated into, the Mineral Resource estimate, Mineral Reserve estimate, conceptual mine plan or economic analysis presented in this Technical Report. New information will be assessed through the Company's established technical, change-management, disclosure and regulatory processes and may result in refinements to the Project design, mine schedule, cost estimates and permitting requirements. Such information will not form part of the current Feasibility Study unless and until it has been appropriately evaluated by the relevant Qualified Persons and, where material, incorporated into the Project's public disclosure in accordance with applicable securities law.

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 drill programs to define early-stage mine life to a Measured Resource confidence level.
- Continue with drill programs aimed at converting unclassified and Inferred classified material to Indicated to expand Reserve conversion capability.
- Continue exploration drilling in the Black Pine district to define new mineralization areas for to expand the resource base.
- Continue to refine the structural geometry model at Black Pine to enhance mineralization domains and the overall Resource estimation
- Enhance the geologic understanding of Black Pine by implementing new geochemical analysis to refine the geology model.

26.2 MINING METHODS

- Additional study is recommended in the following areas above the normal mine planning activities.
- Ore Blending
    - Establish further definition and operational practices regarding blending 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 including larger mining equipment to reduce operating costs.
    - Evaluate alternative shovel buckets and truck boxes to maximize capacity and reduce unit costs.
    - Explore electrification opportunities and regenerative braking opportunities.
- Alternative material handling
    - Evaluate alternative material handling technologies to reduce haulage costs including reduction in the surface road construction/ reclamation costs.
- Scheduling improvements
    - Evaluate mining the smaller pits earlier to bring additional recoverable ounces forward in the plan.
- Haulage
    - Review downhill haul speeds for productivity and safety.
    - Explore tire compounds to reduce tire wear and premature tire failure.

26.3 METALLURGICAL TEST WORK

- Continue development and refinement of geometallurgical models integrating in test work from Phases 6 through 12 and perform additional test programs as needed should the resource expand further
- Complete AMTEL work on Rangefront material, specifically Pola and Ppos units
- Improve modeling of clay by collecting and integrating SWIR information into the geometallurgical data set
- Continue with select metallurgical domain environmental characterization of composite heads and residues to assist project engineering/design and permitting

26.4 LHLF DRAINDOWN SOLUTION

- Confirm that soil from the land application area that may have been exposed to constituents of potential concern (COPCs) from the LHLF 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, subsequently using the data collected to resize pipes and incorporate the flows into the water balance model.
- Further define the water chemistry and potential interaction with the process 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 pond system.

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) identified during the feasibility level field investigation campaign. Laboratory testing would include geochemical sampling on potential construction material borrows. This borrow source investigation should include a detailed laboratory testing program on all potential borrow sources for overliner, including the LHLF, legacy RSF, pit development rock, and other potential sources.
- Confirm the concept of using a low-permeability soil layer with a hydraulic conductivity equivalent to that specified in regulatory documents. This should include continuing discussions with regulatory agencies.
- Complete additional investigations such as geophysics, drilling, sampling, and test pitting within the footprint of the HLF and associated ponds, storm water management infrastructure, truck shop, administration, and process facility areas. The investigations should be developed to further define foundation/subsurface conditions. This should also include geophysical investigations.

26.6 HLF

- Refine the assumed material properties for the various metallurgical material types, including in-place average dry density, hydraulic conductivity, and initial and residual moisture contents.
- Evaluate the possibility of re-routing the LHLF draindown solution directly to the new HLF solution collection piping.
- Evaluate further division of the proposed Phase 1 pad into smaller construction phases to minimize the initial construction efforts. As presented in this study, the Phase 1 pad area is approximately 12 million square feet, which likely could not be fully constructed within one season. Considerations of construction execution and phasing plans will allow for winterization of the on-going works as well as permitting consideration which would expedite placement and potential leaching of ore.
- Refine key associated infrastructure such as road access to the HLF and the solution distribution and application systems to facilitate segregation of the distinct metallurgical material types.

26.7 HYDROLOGY AND HYDRAULICS

- Complete site reconnaissance and/or investigations at the diversions anticipated around the perimeter of the pits and waste rock facilities. If rock is present, the diversions may need to be redesigned or replaced with alternate surface water control measures such as in-pit sumps or other (depending on the extents of the rock).
- Determine if sumps or other storm water controls can be implemented at pits, haul roads, and other mine infrastructure to optimize the complex diversion configurations within steep topography areas.

26.8 PERMITTING

It is understood that Liberty Gold will continue advancing the current environmental review and permitting efforts associated with the 2026 MPO mine plan, including obtaining a Record of Decision and securing the Cyanidation Permit and Air Permits, before developing a formal revised MPO reflecting the conceptual mine plan presented in this Feasibility Study. This approach will allow the current NEPA and state permitting processes to proceed on the established schedule, while preserving the ability to incorporate Feasibility-level modifications through a subsequent revised MPO and related permit amendments.

It is further understood that Liberty Gold will continue to coordinate with IDEQ, IDL, BLM, and USFS, as applicable, to confirm how the proposed excavation, reprocessing, reuse, or placement of LHLF material would affect the modified or amended HLF permit application, existing regulatory obligations, closure requirements, monitoring commitments, and financial assurance associated with the legacy facility.

The relationship between the conceptual feasibility-level Project configuration evaluated in this Technical Report and the Mine Plan of Operations currently under regulatory review and permitting is discussed further in Section 25.5. The Feasibility Study is not intended to replace, supersede, amend or otherwise modify the Mine Plan of Operations or any active permitting submission or regulatory review

26.9 RECOMMENDATION TO PROCEED

Based on the results of the Feasibility Study, the Qualified Persons recommend that the Liberty Gold advances the Project into the engineering, procurement and construction management (EPCM) and project execution phases, with the objective of constructing, commissioning and placing the Project into commercial production.

The Feasibility Study demonstrates that the Project has sufficient technical and economic merit to support advancement, subject to the assumptions, qualifications, risks and opportunities described in this Technical Report. The recommended execution program includes completion of detailed engineering; advancement and maintenance of required permits and authorizations; completion of project financing and commercial arrangements; procurement of long-lead equipment; construction planning and contracting; site preparation; mine development; construction of the process plant and supporting infrastructure; operational-readiness activities; commissioning; and production ramp-up.

The recommended work program is proposed in two phases:

Phase 1 — EPCM and Execution Readiness: Complete detailed engineering, advance current MPO permitting, finalize the execution schedule and control budget, undertake procurement of long-lead equipment, complete financing and material commercial arrangements, and prepare the site and contracting strategy for full construction. The estimated cost of Phase 1 is approximately $35M.

Phase 2 — Construction, Commissioning and Ramp-Up: Proceed with full project construction, mine development, installation of processing and infrastructure facilities, recruitment and training, operational readiness, commissioning, and ramp-up toward commercial production. The estimated cost of Phase 2 is approximately $400M.

Advancement from Phase 1 to Phase 2 would be subject to confirmation that the principal conditions required for construction have been satisfied. These conditions should include, as applicable, receipt of material permits and authorizations; availability of adequate project financing; completion of critical engineering and procurement activities; satisfactory resolution or management of material environmental, social, technical, commercial and implementation risks; and approval by the Company's board of directors.

The Qualified Persons recommend advancement of the Project into EPCM and project execution to be the appropriate next stage of development.

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Appendix A

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 21 years. 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.

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

6. I am independent of the Company as described in Section 1.5 of NI 43-101.

7. I am a contributing author for the preparation of the technical report titled "Black Pine NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.

8. I am responsible for Sections 1.1, 1.2, 1.20, 1.21, 2, 3, 4, 5, 18.1-18.3, 18.8-18.15, 19, 23, 25.7, 25.8, 26.7, 26.9, and 27 of the Technical Report. I have visited the Black Pine project site on October 31, 2023, and May 8, 2024, and reviewed the plant location site.

9. I have been involved in the Black Pine Project and participated in preparing the "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.

10. 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.

11. 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.

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 18th day of September, 2026.

(signed) "Matthew Sletten"

Signature or seal of Qualified Person

Matthew Sletten

Print Name of Qualified Person

Appendix A

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 18 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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.
7. I am responsible for the preparation of Sections 1.14, 1.18, 1.19, 13.14, 17, 21.1, 21.2, 21.5, 21.6 and 22. I have not visited the project site.
8. I have been involved in the Black Pine Project and participated in preparing the "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.
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 18th day of September, 2026.

(signed) "Benjamin Bermudez"

Signature or seal of Qualified Person

Benjamin Bermudez

Print Name of Qualified Person

Appendix A

CERTIFICATE OF QUALIFIED PERSON

TODD CARSTENSEN

I, Todd Carstensen, RM-SME, 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 40 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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.

7. I am responsible for the preparation of Sections 1.12, 1.13, 15, 16 (except 16.5), 21.4, 21.7, 25.3, 25.4, and 26.2. I visited the Black Pine project site on October 31, 2023 and June 17, 2026.

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 and the preparation of "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.

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 18th day of September, 2026.

(signed) "Todd Carstensen"

Signature or seal of Qualified Person

Todd Carstensen

Print Name of Qualified Person

Appendix A

CERTIFICATE OF QUALIFIED PERSON

Kyle Brangers

I, Kyle Brangers, CPG, do hereby certify that:

1. I am currently employed as a Mining Geologist with GSI Environmental Inc., with an office address of 145 W Front St, Missoula, Montana, 59802.
2. I am a graduate of University of Montana and received a Bachelor of Science degree in Geology in 2016.
3. I am a registered member in good standing of the American Institute of Professional Geologists (AIPG) and recognized as a Certified Professional Geologist (CPG #12256).
4. I have worked as an engineer for a total of 11 years. My experience includes underground mining operations, environmental permitting and compliance for exploration and large-scale mining in the U.S. I have managed mineral exploration programs and associated exploration permits, monitoring well drilling and installation, environmental baseline characterization and support, and mine permitting strategy development and regulatory support.
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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.
7. I am responsible for the preparation of Sections 1.17, 20 (except 20.10.2), 25.6, 25.9, and 26.8. I have not visited the project site.
8. I have not been previously involved with the property subject to 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 18th day of September, 2026.

(signed) "Kyle Brangers"

Signature or seal of Qualified Person

Kyle Brangers

Print Name of Qualified Person

Appendix A

CERTIFICATE OF QUALIFIED PERSON

Nicholas Rocco

I, Nicholas Rocco, Ph.D., P.E., do hereby certify that:

1. I am currently employed as a Principal Engineer at NewFields MDTS, 9540 Maroon Circle, Suite 300, Englewood, Colorado 80112.
2. I am a graduate of Missouri University of Science and Technology with a Ph.D. in Civil Engineering.
3. I am a Professional Civil Engineer in good standing in the State of Idaho (No. 15525).
4. I have worked as an engineer for a total of 22 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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.
7. I am responsible for the preparation of Sections 1.15, 1.16, 18.4, 18.5.1-18.5.3, 18.6, 18.7, 20.10.2, 21.3, 25.5, and 26.4-26.6. I visited the project site on April 17, 2024.
8. I have been involved in the Black Pine Project and participated in preparing the "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.
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 18th day of September, 2026.

(signed) "Nicholas Rocco"

Signature or seal of Qualified Person

Nicholas Rocco

Print Name of Qualified Person

Appendix A

CERTICATE OF QUALIFIED PERSON

Gary L. Simmons

I, Gary L Simmons, MMSA, do hereby certify that:

1. I am employed as a Consultant by GL Simmons Consulting, LLC at 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 53 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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.
7. I am a contributing author for Sections 1.10, 13 (except 13.14), 25.2, and 26.3. I have visited the project site on the following dates: June 3, 2019, October 18-19, 2019, June 22-23, 2020, May 3, 2023, August 1-2, 2023, June 2-4, 2025, August 13-15, 2025 and January 14-16, 2026.
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; 2. NI 43-101 Technical Report titled "Technical Report on the Updated Mineral resource Estimate at the Black Pine Gold Project, Cassia and Oneida Counties, Idaho, USA", effective January 21, 2023; and 3. NI 43-101 Technical Report titles "Black Pine Project 43-101F1 Technical Report Pre-Feasibility Study, Cassia County, Idaho, USA", effective June 1, 2024.
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 18th day of September, 2026.

(signed) “Gary L. Simmons”

Signature or seal of Qualified Person

Gary L. Simmons

Print Name of Qualified Person

SLR Consulting (Canada) Ltd.
55 University Ave., Suite 501, Toronto, ON M5J 2H7

Appendix A

Certificate of Qualified Person

Valerie Wilson

I, Valerie Wilson, M.Sc., P.Geo., do hereby certify that:

1 I am a Principal Resource Geologist of SLR Consulting (Canada) Ltd., at 55 University Avenue, Suite 501, Toronto, ON, Canada M5J 2H7

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 20 years since graduation from my bachelor's degree. My relevant experience for the purpose of the Technical Report is: mineral Resource modelling, estimation, and reporting work on numerous Carlin-style gold projects in the Mountain West region of the USA; and 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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.

7 I am responsible for the preparation of Sections 1.3-1.9, 1.11, 6, 7, 8, 9,10, 11, 12, 14 (except 14.16), 24, 25.1, and 26.1. I visited the site on April 11, 2024.

8 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, acting as qualified person for the previous Mineral Resource estimate (effective February 15, 2024) and the preparation of "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.

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 Canadian stock exchanges 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 18th day of September, 2026.

(signed) "Valerie Wilson"

Signature or Seal of Qualified Person

Valerie Wilson

Print Name of Qualified Person

Appendix A

CERTIFICATE OF QUALIFIED PERSON

Brendan Fetter

I, Brendan Fetter, P.E., do hereby certify that:

1. I am currently employed as a Mineral Processing Engineer for Forte Dynamics, Inc., 120 Commerce Drive, Units 3-4, Fort Collins, CO 80524.
2. I graduated with two Bachelor of Science degrees in Civil and Mechanical Engineering in 2013 from the Colorado School of Mines in Golden, CO.
3. I am a Registered Member of the Society of Mining Engineers, and a Professional Engineer licensed in Colorado, Nevada, and Idaho
4. I have worked in the mining industry for a total of 13 years. My experience includes mineral processing, metallurgical evaluations, and financial evaluations.
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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.
7. I am responsible for the preparation of Sections 16.5 and 18.5.4. I have not visited the project site.
8. I have not been previously involved with the property subject to 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 18th day of September, 2026.

(signed) "Brendan Fetter"

Signature or seal of Qualified Person

Brendan Fetter

Print Name of Qualified Person

Appendix A

CERTIFICATE OF QUALIFIED PERSON

Don Hulse

I, Don Hulse, RM-SME, do hereby certify that:

1. I am currently employed as a Director of Mining Resources at Forte Dynamics, Inc., 120 Commerce Drive, Units 3-4, Fort Collins, CO 80524.
2. I am a graduate of the Colorado School of Mines and received a Bachelor of Science in Mining Engineering in 1982.
3. I am a Registered Member of the Society of Mining Engineers, No 1533190RM.
4. I have worked in the mining industry for a total of 38 years. My experience includes technical and general management, permitting coordination, short- and long-term planning, economic pit limit analysis, pit phase and design, cost estimation and analysis, resource estimation, geostatistics, feasibility studies, and reserve audit.
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 NI 43-101 Feasibility Study, Southern Idaho, USA" with an effective date of August 1, 2026 (the "Technical Report"), prepared for Liberty Gold.
7. I am responsible for the preparation of Section 14.16. I visited the project site on June 6, 2025.
8. I have not been previously involved with the property subject to 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 18th day of September, 2026.

(signed) "Don Hulse"

Signature or seal of Qualified Person

Don Hulse

Print Name of Qualified Person