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Idaho Cobalt Operations

Form 43-101F1 Technical Report

Feasibility Study

Idaho, USA

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DATE AND SIGNATURES PAGE

The effective date of this report is 20 January 2020. The issue date of this report is 13 November 2020. 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.

IDAHO COBALT OPERATIONS
FORM 43-101F1 TECHNICAL REPORT
FEASIBILITY STUDY

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1 SUMMARY

1.1 AUTHORIZATION AND PURPOSE

In September 2019, Jervois Mining Limited ("Jervois") through its wholly-owned subsidiary, Jervois Mining USA commissioned DRA Americas Incorporated ("DRA") and M3 Engineering and Technology Corporation ("M3"), and their sub-consultants to prepare a Feasibility Study ("FS") for the production of a bulk cobalt-copper-gold concentrate from its Idaho Cobalt Operations ("ICO"), in east-central Idaho, USA. This report has been prepared in accordance with the reporting requirements of Canadian National Instrument ("NI") 43-101 and The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ('the JORC Code").

The purpose of this report is to support the public disclosure of the ICO mineral resources, reserves and the economic results of the FS.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

The ICO Property is 100% owned by Jervois Mining USA, a wholly owned subsidiary of Jervois, and consists of 163 contiguous unpatented lode mining claims located in east central Idaho, approximately 25.8 miles west of the town of Salmon. The property covers 2,520 acres centered on 45°07'50" north latitude and 114°21'42" west longitude.

At present, the ICO property is not subject to any royalties, other agreements or encumbrances.

1.3 GEOLOGY AND MINERALIZATION

The ICO is hosted in Proterozoic age meta-sediments found on the east side of the central Idaho Batholith comprising granitic-to-granodioritic rocks. The host sedimentary rocks are believed to have been part of a large fault-bounded marine sedimentary basin in which dominantly clastic sediments were deposited. The basin is now part of a supergroup of dominantly quartzite and argillite metasedimentary rock, the base of which is referred to as the Apple Creek Formation. All significant copper-cobalt deposits and occurrences are found in the Apple Creek Formation in a 30- to 35-mile-long linear belt known as the Idaho Cobalt Belt. The deposits are tabular/stratiform, strike north-northwest, with gentle dips of between 50 and 60 degrees to the west. Aside from the Ram deposit, which is the focus of this report, there are two other sub-parallel deposits, namely the Sunshine and East Sunshine which are located about a mile to the south of the Ram.

Mineralization at the ICO is closely associated with the mafic sequences of the middle unit of the Apple Creek Formation. Dominant ore minerals include cobaltite (CoAsS) and chalcopyrite (CuFeS2), with lesser, variable occurrences of gold. Other minerals present in small quantities are pyrite (FeS2), pyrrhotite (FeS), arsenopyrite (FeAsS), linnaeite ((Co Ni)3S4), loellingite (FeAs2), safflorite (CoFeAs2), enargite (Cu3AsS4), and marcasite (FeS2).

1.4 EXPLORATION STATUS

There are no current exploration activities. However, the Ram deposit resource remains open at depth and along strike offering opportunities for expansion. The Sunshine and East Sunshine deposits are within a mile trucking distance of the Ram and represent additional potential to the mineral resources of the ICO.

1.5 MINERAL PROCESSING/METALLURGICAL TESTING

Several historical test work campaigns and studies have been conducted for ICO. The previous study in 2016/2017 focussed on developing a grinding and bulk sulfide flotation process at the mine, followed by subsequent leaching of the flotation concentrate within a Cobalt Hydrometallurgical Facility to ultimately produce cobalt sulphate, copper sulphate and magnesium sulphate crystals (MICON Int Limited, 30 November 2017).

At the start of the current FS in September 2019, Jervois appointed DRA to fulfil the study management and oversight role of the 2019/2020 metallurgical testwork program, in support of this report. A number of metallurgical test work programs comprising batch and continuous tests have been completed using representative samples of the RAM deposit mineralization that support the FS process flowsheet. Testwork programs completed to date include the following:

- Initial milling and flotation test work on bulk samples and drill composites performed by Noranda's (now owned by Glencore) nearby Blackbird Mining Company ("BMC") in the 1980's. BMC reportedly was successful in producing separate copper and cobalt concentrates using a differential flotation flowsheet.
- Early work by The Center for Advanced Mineral and Metallurgical Processing ("CAMMP") in 2001 used approximately 1 ton of large diameter drill core from the RAM deposit. This testwork included a comprehensive milling and flotation test program and nitrogen species catalyzed ("NSC") leaching of the batch flotation concentrate.
- In 2005 SGS Lakefield ("SGS-L") conducted a number of flowsheet development test work programs including detailed comminution and flotation testing as well as preliminary leach testing that confirmed CAMP's NSC test result.
- The initial hydrometallurgical tests completed by SGS-L in 2005 provided the design criteria used for a Mini Pilot Plant Testwork campaign undertaken in 2005 by Mintek, South Africa. This program was directed by Hatch and was successful in developing a basic hydrometallurgical process.
- Pocock Industrial Inc. conducted solids-liquid separation tests in 2005, including settling/thickening and filtration studies on samples of cleaner concentrate and rougher flotation tailings.
- A pilot plant was operated at Mintek in 2007. This work resulted in improved Fe/Cu removal, solution purification steps, consistently high-grade cobalt product (>99.9% Co) and introduced of flash cooling technology.
- In 2015 Hazen Research completed further flotation and hydrometallurgical test work under the direction of Samuel Engineering Inc. (Samuel).
- CYTEC Solvay Group (Cytec), conducted bench scale and continuous pilot plant scale cobalt solvent extraction test work in 2015 using pregnant leach solution ("PLS") generated by Hazen. The objective of this work was to produce a clean cobalt sulphate solution that could be fed to the crystallizers.
- GE Water & Process Technologies ("GE") performed crystallizer bench tests in 2015 with the objective of gathering adequate design data in order to confidently size and estimate the cost of a commercial cobalt sulphate crystallizer. GE also prepared a capital cost estimates for the magnesium sulphate and copper sulphate crystallizer packages for the feasibility study.
- In 2016 and 2017 SGS-L completed a program of bench scale test work to confirm the FS design. This work included differential flotation, copper/iron removal, NSC leaching, leach residue elemental sulphur recovery and gold leaching.
- In 2017 SGS-L completed a series of tests to produce copper and cobalt sulphate crystals.
- In 2018, Dundee Sustainable Technologies processed initially 7 tons and then a further 5 tons of material through a bulk sulphide flotation process (rougher, cleaner scavenger circuit) in order to generate a bulk cobaltite concentrate.
- In 2019 and 2020 six metallurgical test phases were conducted within the 2019/2020 study in support of the design for material from the ICO Ram deposit. Most of the test work was conducted at SGS facilities. All test work conducted for the 2020 FS was in support of a split concentrate flowsheet, where copper was activated with starvation dosages of collector and recovered first, prior to a cobalt flotation using potassium amyl xanthate ("PAX") collector. The two flotation concentrates were then dewatered and bagged separately.

Following financial analysis of this flowsheet, it was subsequently decided to revise the block flow in order to generate a single, bulk (combined copper and cobalt) concentrate based on lock cycle tests of the CAMMP and SGS – 2005 test work campaigns.

1.6 MINERAL RESOURCE ESTIMATE

An updated mineral resource estimate with an effective date of January 20, 2020 was prepared by Orix Geoscience, Inc. ("Orix") for incorporation into the BFS. CSA Global Consultants Canada Ltd ("CSA") audited and validated the Orix estimation procedures.

Compared to previous resource models, the 2020 model is rotated with smaller parent cell sizes of 12 x 12 x 4 ft (3.66 metres x 3.66 meters x 1.22 meters). Prior block models used a minimum block width of 1.8 meters. The rotation is -14° around the Z axis (dominant strike of mineralization is 346°), and -58° around the Y axis. Twenty-four (24) ID2 interpolations were performed to populate the final grades into the block model.

The ore intercepts are best characterized as containing a single very high grade (>1% Co) interval of ~0.6 m length with one to two intervals above cut-off grade on either side resulting in a true width of 2.0 to 2.4 meters. Block rotation to the orientation of the main Ram zone and a reduced cell size has allowed a better reflection of grade distribution within the orebody.

The 2020 updated mineral resources for the Ram deposit as presented in Table 1-1 below. The mineral resources in this table are reported at a cut-off grade of 0.15% Co; the copper and gold resources are those resources carried within the resource blocks which attain the cobalt cut-off grade.

Table 1-1: 2020 Mineral Resource Estimate – Imperial and Metric (2)(3)

Category	Resource (M Tons)	Resource (M tonnes)	Co (%)	Co (M lbs)	Cu (%)	Cu (M lbs)	Au (oz/Ton)	Au (g/tonne)	Au (oz)
Measured (1)	2.92	2.65	0.45	26.2	0.59	34.4	0.013	0.45	38,000
Indicated (1)	2.85	2.59	0.42	23.8	0.80	45.7	0.018	0.62	51,000
M+I	5.77	5.24	0.44	50.1	0.69	80.1	0.015	0.53	89,000
Inferred	1.73	1.57	0.35	12.0	0.44	15.2	0.013	0.45	23,000

1. Mineral Resources are not Mineral Reserves and by definition do not have demonstrated economic viability. The Mineral Resources in this news release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council (2014).
2. The Cobalt cut-off grade for inclusion in the resource is 0.15%, no consideration of copper or gold content was used in determination of cut-off grade.
3. Contained metal values and totals may differ due to rounding of figures.

1.7 MINERAL RESERVE ESTIMATE

For the ICO, the Measured and Indicated mineral resource from the main mineralized horizon was considered in the mine plan for conversion into a mineral reserve.

Conversion of the mineral resource estimates to mineral reserve was inclusive of the modifying factors, diluting material and allowances for losses which are to be expected when the material is mined or extracted.

Stope outlines were generated from two types of 12 ft vertical level interval shells, one being a minimum 15 ft width sill drift and the second being a minimum 6 ft width back stope for the two twelve ft level intervals immediately above the sills. Each stope shape represents two production rounds. A base cut-off grade of 0.30% Co was used to create the sill shapes eligible for conversion to reserve and a cut-off grade of 0.32% Co was used for the back-stope shapes. These shapes were then further filtered to accept only those diluted shapes for which a recovered and payable cobalt equivalent grade of 0.24% was achieved to provide value equal the cash operating cost estimate at a price of US$25.00/lb cobalt. Recoveries used in the calculation were derived from test work conducted as part of this study.

Payable values were based on indicative terms from prospective off-takers. The 2020 updated mineral reserve for ICO is presented in Table 1-2 below.

Table 1-2: Mineral Reserve for at 0.24% Co Recovered and Payable Equivalent Cut-off Grade

Category	Reserve (M short tons)	Co (%)	Co cont. (M lbs)	Cu (%)	Cu cont. (M lbs)	Au (oz / short ton)	Au cont. (oz)
Proven (1,2)	1.59	0.56	17.9	0.67	21.2	0.015	24,633
Probable (1,2)	1.16	0.53	12.3	0.96	22.3	0.023	26,758
Total	**2.75**	**0.55**	**30.1**	**0.80**	**43.6**	**0.019**	**51,391**
Category	**Reserve (M tonnes)**	**Co (%)**	**Co cont. (Tonnes)**	**Cu (%)**	**Cu cont. (Tonnes)**	**Au (g/tonne)**	**Au cont. (oz)**
Proven (1,2)	1.44	0.63	8,100	0.67	9,600	0.53	24,633
Probable (1,2)	1.05	0.53	5,600	0.96	10,100	0.80	26,758
Total	**2.49**	**0.55**	**13,650**	**0.80**	**19,800**	**0.64**	**51,391**

1. Mineral Reserves are based on Measured and Indicated Mineral Resources which have demonstrated economic viability. The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
2. Mineral Reserves are reported as diluted recovered tons with grades considering those Mineral Resource blocks above Resource cut-off grade within the dilutive material as contributing to metal content.
3. The cobalt equivalent cut-off grade for inclusion in the reserve is 0.24% payable equivalent cobalt grade. This includes consideration of copper and gold content as well as recoveries and payability of each commodity.
4. Contained metal figures and totals may differ due to rounding of figures.

1.8 MINING METHODS

The mining methods proposed for the ICO are overhand longitudinal short-hole stoping from 12 ft high sills spaced 36 ft vertically. The sills and backstops will be completely filled with waste rock and cementitious paste fill. Mining sequencing will be overhand with fully paste filled sills forming crowns to terminate the overhand back stoping in a final retreat blind back stope. The selection of these mining methods for the deposit was determined primarily by the geometry of the mineralized horizons, including factors such as its continuity, dip and width, and the geotechnical parameters of the rock mass. The mining method significantly reduces risk of variability in the orebody through detail mapping and sampling of the orebody from the sills to be developed under geologic control.

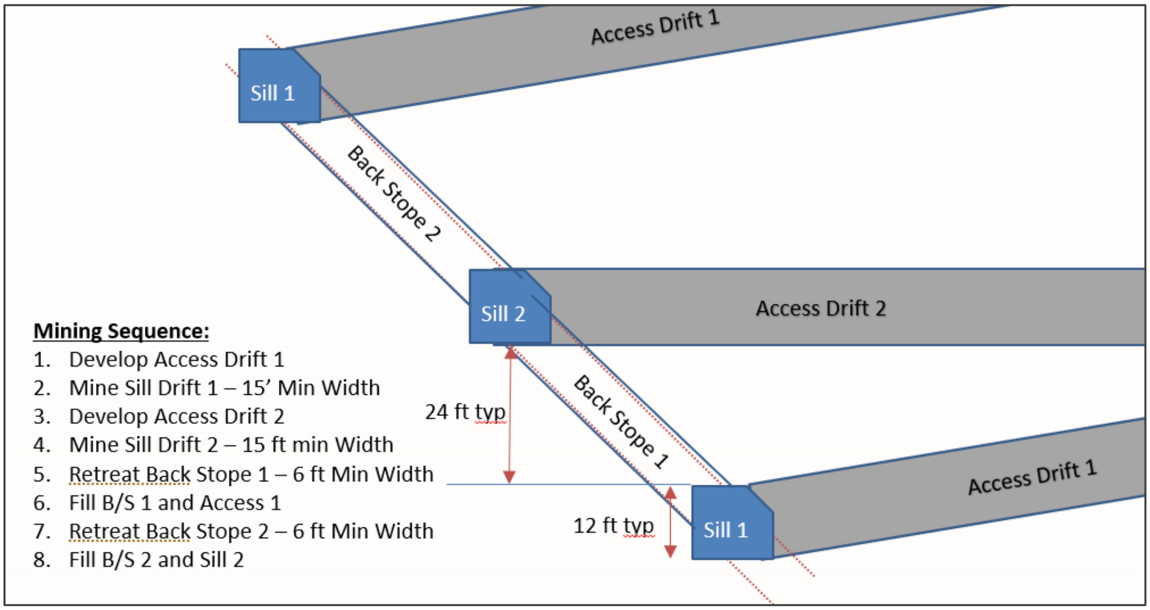


Figure 1-1: Typical Mining Sequence

The Ram deposit is composed of a main mineralized horizon with local variability in width and occasional splays with thickness ranging from one foot to more than 20 ft, at an average dip of 55° (Orix, 2020). All of the measured and Indicated resource occurs within this main horizon.

Mining equipment selection was discussed with contractors submitting tenders for the mine development and will consist of 4 Cubic yard LHD's with remote operation capability and ejector buckets working with 30T capacity haul trucks with ejector boxes to assist with waste rock placement in as fill. Ramp, access drift and sill development will utilize twin boom jumbos with cabs. Ground support will be provided by both jumbos and ro-bolters, selection dependent on drift size.

Conservatively, the mine operating cost estimates and production schedule have been based on supporting the planned mill throughput of 1200 stpd. Development rates were constrained to a maximum of 12 ft per day per available heading to minimize early capital burdens while providing ample stope availability to support the production and fill schedules. The mine will be able to initiate production at 1200 stpd within 10 months of opening the portal and sustain production throughout the planned 7-year mine life. Ore will be cleared from headings after blast and staged in the muck bay of each stope access drift. Ore is then transferred to 30T trucks from transport up the ramp system for staging at the portal area for haulage to the run of mine ("ROM") stockpile utilizing articulated surface haul trucks.

Table 1-3: Mine Development and Production Schedule

	LOM	2021	2022	2023	2024	2025	2026	2027	2028
Ore ST	2,741,520	-	129,252	439,293	438,539	439,317	438,633	438,400	418,085
Co lbs	30,133,351	-	1,597,119	5,836,025	4,745,583	4,545,920	4,039,378	4,860,635	4,508,691
Co grade	0.55%	-	0.62%	0.66%	0.54%	0.52%	0.46%	0.55%	0.54%
Cu lbs	43,600,305	-	1,368,080	9,114,625	7,737,686	7,689,779	10,747,609	3,828,843	3,113,684
Cu grade	0.80%		0.53%	1.04%	0.88%	0.88%	1.23%	0.44%	0.37%
Au oz	51,418	-	2,512	9,516	7,022	8,645	12,079	6,053	5,591
Au grade oz/st	0.0188	-	0.0194	0.0217	0.0160	0.0197	0.0275	0.0138	0.0134
Paste Placed ST	1,145,510	-	16,353	93,218	123,020	223,995	245,456	199,169	244,300
Dev Feet	42,969	3,701	10,543	15,231	9,479	4,015	-	-	-
% Dev Ft		9%	25%	35%	22%	9%	0%	0%	0%
Tails to TWSF	1,917,138	-	112,928	360,169	343,428	281,703	268,827	296,599	253,484
Waste to TWSF	218,789	59,083	215,022	152,153	91,328	(67,882)	(63,674)	(55,898)	(111,343)
Cum total TWSF	10,110,316	59,083	387,034	899,356	1,334,112	1,547,933	1,753,085	1,993,786	2,135,927
Total Ft Dev Drift 15x15	20,873	2,103	5,492	7,013	4,608	1,656	-	-	-
Total Ft Dev Drift 14x14	21,853	775	5,245	7,752	4,687	2,337	413	388	255
Total Ft Dev Raise	2,368	-	644	765	628	331	-	-	-
Total Raises	34	-	9	11	9	5	-	-	-
Total Sill Tons Including non	1,131,294	-	105,882	170,830	196,325	131,977	191,779	212,308	122,193
Total B/S Tons	1,702,041	-	31,827	263,619	259,175	324,212	257,321	245,889	319,997
Waste Haul to/from TWSF	816,384	59,083	215,022	152,153	91,328	67,882	63,674	55,898	111,343
Ore Haul to Mill	2,752,746	-	131,177	440,431	439,957	442,236	439,927	439,865	419,153
Paste Backfill Tons Tails	1,145,510	-	16,353	93,218	123,020	223,995	245,456	199,169	244,300
Cement Use Paste (tons)	45,820	-	654	3,729	4,921	8,960	9,818	7,967	9,772
Total Tons Mined	3,792,311	59,083	376,326	767,084	670,608	546,959	457,654	466,229	448,368

Paste prepared from mill tailings will be utilized as backfill material in combination with waste rock fill arising from mine development. Unused waste rock will be hauled to surface and staged at the portal area for haulage to the Tailings and Waste Rock Facility utilizing articulated surface haul trucks.

1.9 PROCESSING

The process plant metallurgical design is based upon data and design criteria provided by Jervois, DRA, vendor data, test work and regulatory/permitting requirements. These inputs formed the basis for the entire process plant design, including process flowsheet and mass-water balances.

The crushing and grinding circuit design is based upon the design throughput requirements and ore competency and hardness characteristics obtained by test work. The SAG/Ball mill sizing is based on achieving the grind size required for optimal flotation performance and designed utilizing outcomes of the metallurgical test work. Equipment sizing calculations have been completed using energy-based populated balance modelling techniques.

The design and configuration of the bulk sulphide flotation circuit are based upon the locked cycle test results conducted for the 2007 FS under the direction of Samuel Engineering Inc. These results also provided the basis for recovery and grade calculations.

Concentrate and tailings products are thickened and then dewatered using a conventional plate and frame pressure and vacuum disc filtration, respectively. The filtration circuit design is based on common design practices for concentrate and metallurgical test work.

1.10 INFRASTRUCTURE

Infrastructure at the ICO mine/mill site was partly constructed during an earlier stage of project development, including:

- Completion of the access road from highway 93 to the mine site.

- Security/Gate House has been purchased and installed at entrance to the mine site.
- Site preparation including stripping and grading.
- Earthworks for the first cell of the Tailings Waste Storage Facility ("TWSF") was nearly completed during the 2011 construction phase, after testing the liner material on site is unsuitable for use and is budgeted to be replaced.
- Some footings have been installed for the crusher building and the mill and concentrator building.
- The administration building including utilities has been purchased and installed at site.
- The incoming power supply line as well as tie-ins to the supply line and the site distribution system was completed during the last phase of construction.
- The road to the portal location and portal bench has also been completed. A Hilkfiker wall will be constructed during final construction prior to mine development.
- A small warehouse and yard south of Salmon Idaho has been purchased. The Salmon Depot is currently used for storage of the purchased equipment. In future, this site will be used as a mustering point for construction and operations employees who will be bussed to site. It will also serve as temporary storage of concentrate prior to shipment to customers and incoming shipments bound for the mine site.
- Construction of the Water Treatment Plant was largely completed during the previous phase of construction in 2018 and commissioning of the treatment plant will form part of the scope to complete environmental systems to enable mining development.
- The Pumpback equipment has been supplied and is currently in storage in Salmon. The intention is to install and commission the Pumpback system as part of the scope to complete environmental systems to enable mining development.

1.11 MARKET STUDIES AND CONTRACTS

1.11.1 Market Studies

Jervois' plan for ICO is to produce a bulk concentrate, containing annual production of approximately 1,915 metric tonnes of cobalt, 2,900 metric tonnes of copper and 6,700 oz of gold. The concentrate will be low in deleterious elements, with the exception of arsenic as the cobalt is contained in the mineral cobaltite (CoAsS).

To retain marketing flexibility, BFS level engineering (including metallurgical test work and physical product sample generation) has also been completed on separated concentrates and a calcined cobalt product.

ICO's concentrate has been marketed to a variety of customers including nickel and cobalt refineries, precursor and battery manufacturers, and OEM (vehicle manufacturers) directly. Representative samples were sent to customers, indicative off-take offers have been received and negotiations are ongoing.

Commercial terms that underpin the BFS are approximately 75% of the payable metal value in concentrate.

Jervois has also recently agreed to purchase the São Miguel Paulista nickel cobalt refinery ("SMP Refinery"), a transaction which creates marketing flexibility. It introduces an ability for Jervois to further beneficiate ICO concentrate internally and deliver customers refined cobalt. In recent months Jervois has been selectively broaching the concept of delivering refined cobalt to key strategic customers (largely battery makers and OEM's that cannot physically consume concentrate in their industrial facilities). Now that potential concentrate customers are aware that Jervois has an internal processing option, Jervois's negotiating leverage on concentrate sales also improves. Whilst Jervois's current expectation is the bulk concentrate from ICO will be converted at the SMP Refinery, the Company retains an ability to divert to multiple customers should this be commercially advantageous.

1.12 ENVIRONMENT STUDIES, PERMITTING, AND SOCIAL/COMMUNITY IMPACT

The mine and mill are located on National Forest lands managed by the Salmon-Challis National Forest. As such it is subject to the National Environmental Policy Act ("NEPA"). This requires a thorough series of environmental baseline studies and an Environmental Impact Statement that provides the Company and state and federal government agencies a complete property description, identification of all environmental impacts both positive and negative and the development of mitigation methods to reduce or eliminate negative impacts utilizing best practices.

The Final Environmental Impact Statement (FEIS, June 2008) discussed the project, alternatives to the project, environment effects (direct, indirect and cumulative) and consultation with aboriginal groups, communities and other stakeholders. No issues were identified that could not be mitigated using best practices.

An extensive environmental monitoring plan has been developed that covers the following:

- Water Quality Monitoring
- Biological Monitoring
- Wetlands Monitoring
- Storm Water Monitoring
- Weather Monitoring
- Air Quality Monitoring
- Geochemical Monitoring

A list of permits and authorizations required for the project and their current status is given in Section 20 of this report.

1.13 CAPITAL AND OPERATING COSTS

The LOM capital cost estimate is summarized in Table 1-4. The estimate is given in US dollars ($), with a base date of third quarter, 2020.

Table 1-4: Capital Cost Summary by Category (US$ million)

Category	Initial Capital	Sustaining Capital	LOM Total Capital
Process Plant Direct	25.526	-	25.526
Infrastructure	10.807	1.355	12.162
Mining	18.604	55.861	74.465
Indirect	18.192	0.359	18.552
Contingency	5.274	-	5.274
Total	**78.403**	**57.575**	**135.978**

The capital cost estimate for this project presented herein is considered to be at a feasibility study level with an accuracy of +15%/-15% and carrying a contingency totaling approximately 6.6% on initial capital expenditure.

The estimated life-of-mine total project operating costs are summarized in Table 1-5.

Table 1-5: Operating Costs for Life of Mine (US$ million)

Major Project Area	LOM [m US$]	2022 Sept - Dec	2023 Jan - Dec	2024 Jan - Dec	2025 Jan - Dec	2026 Jan - Dec	2027 Jan - Dec	2028 Jan - Dec
Mining Cost	**$203.46**	$8.88	$31.074	$32.291	$32.672	$33.031	$33.194	$32.315
Processing Cost	**$51.98**	$2.42	$8.164	$8.212	$8.321	$8.320	$8.362	$8.175

Major Project Area	LOM [m US$]	2022 Sept - Dec	2023 Jan - Dec	2024 Jan - Dec	2025 Jan - Dec	2026 Jan - Dec	2027 Jan - Dec	2028 Jan - Dec
Concentrate Logistics	**$16.00**	$0.83	$3.114	$2.525	$2.405	$2.130	$2.580	$2.415
G&A Cost	**$33.21**	$1.75	$5.244	$5.244	$5.244	$5.244	$5.244	$5.244
Total Cost	**$304.65**	$13.88	$47.596	$48.272	$48.642	$48.726	$49.380	$48.149
US$ / s Ton	**$111.86**	$124.8	$108.7	$109.9	$111.1	$111.2	$112.7	$114.3
US$ / m Ton	**$123.30**	$137.6	$119.8	$121.2	$122.4	$122.6	$124.3	$126.0

1.14 ECONOMIC ANALYSIS

An economic analysis based on the production and cost parameters of the ICO Project was prepared and selected results are summarized in Table 1-6. The economic assessment of ICO all price projections and cost estimates in US currency are all on a real basis (excluding inflation). Discount rates and Internal Rate of Return ("IRR") are also in real terms. In the analysis, price forecasts of US$25.00/lb for cobalt, US$3.00/lb for copper and US$1,750/oz for gold were assumed.

Table 1-6: Summary of Life of Project Production, Revenues, and Costs

Description	Units	Value
Resource Milled	k tonnes	2,486.1
Bulk Concentrate @ 10 % Co	k tonnes	124.4
Revenue	M USD	667.4
Operating Costs	M USD	315.7
Initial Capital Costs (excludes Working Capital)	M USD	78.4
Sustaining Capital Costs	M USD	56.1
Mine Closure & Rehabilitation Costs	M USD	21.2
Total Pre-Tax Cash Flow	M USD	198.5
Total After-Tax Cash Flow	M USD	170.9

The financial indicators associated with the economic analysis are summarized in Table 1-7.

Table 1-7: Base Case Financial Indicators

Financial Results	Unit	Value
Pre-tax NPV @ 8%	M USD	113.4
Post-tax NPV @ 8%	M USD	95.7
Pre-tax IRR	%	41.8
Post-tax IRR	%	37.6
Pre-tax Payback Period	years	2.6
Post-tax Payback Period	years	2.8

Figure 1-1 and Figure 1-2 show the sensitivity of the post-tax NPV and IRR, respectively, to changes in pre-production (initial) capital expenses (Capex), sustaining capital expenses (Sus Capex), operating costs (Opex), the cobalt price and all prices (varied together).

This FS was compiled according to widely accepted industry standards. However, there is no certainty that the conclusions reached in this study will be realized.

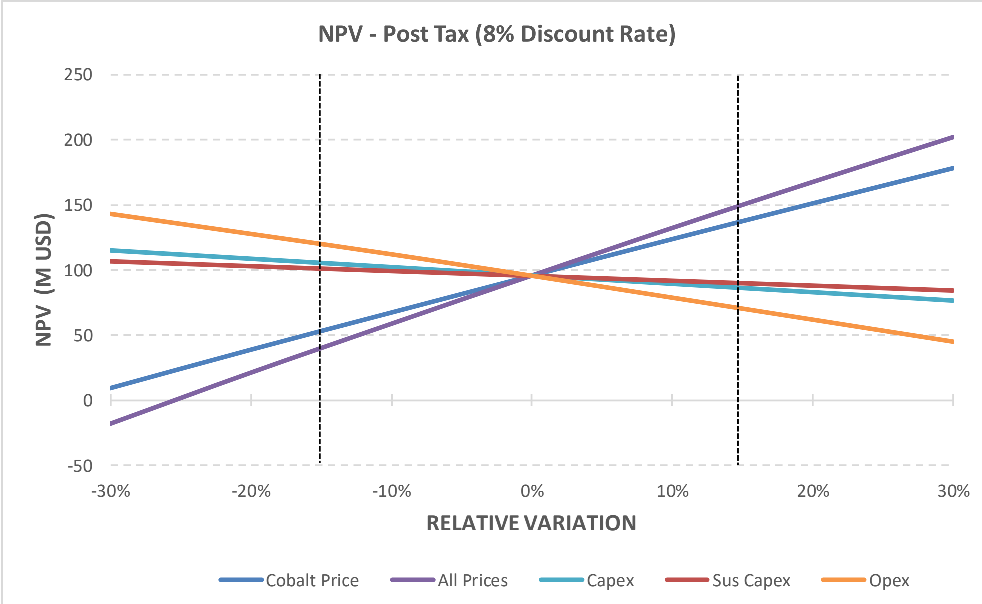


Figure 1-2: Sensitivity of Project NPV @ 8 % (Post Tax)

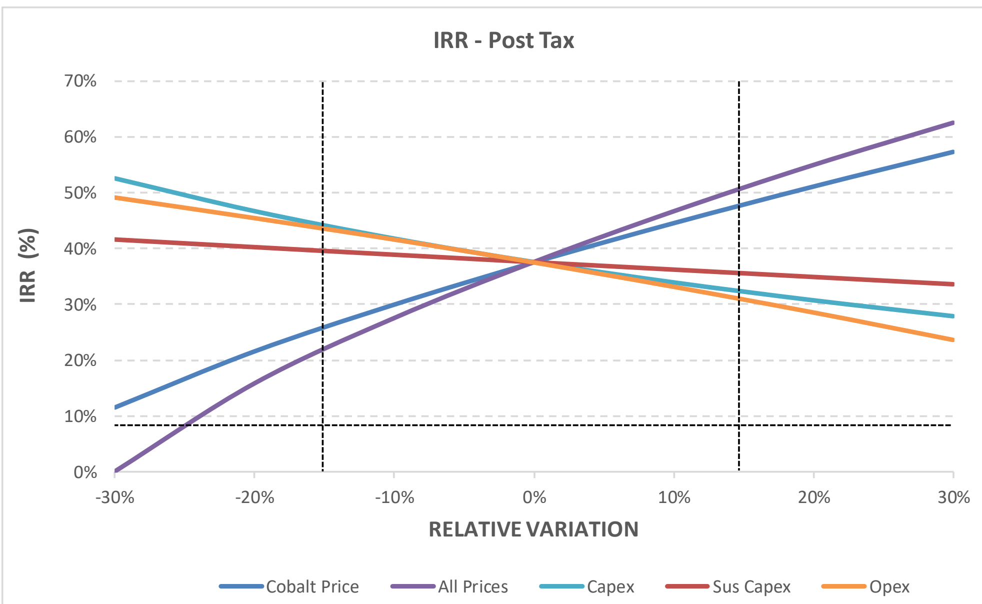


Figure 1-3: Sensitivity of Project IRR (Post Tax)

1.15 CONCLUSION AND RECOMMENDATIONS

1.15.1 Geology and Resources

The following two-phase program is recommended as a result of the geological modeling and resource estimation.

Phase 1 - 1600 metres of drilling is recommended to test up-dip positions on the Blacktail North zone. As development of the underground mine is anticipated, further testing of the hanging wall zones, or Main zone is not recommended at this time.

Phase 2 - Phase 2 includes follow-up drilling based on the results of Phase 1 and definition drilling of the hanging wall zones and Main zone from underground positions (in conjunction with grade control drilling).

- For Blacktail North, an additional 1600 meters of drilling is recommended if Phase 1 produces promising results.
- For the hanging wall and Main zones, 1000 meters of definition drilling from underground (meaning shorter holes).

Orix, Scott Zelligan, and CSA Global recommend the following be implemented for all future programs (including the above two phases).

- The use of four-acid digestion for the over limit arsenic (As) values may be problematic because As, an element of interest, can volatize with this method leading to a potential underreporting. CSA recommends some of the OG62 arsenic over limit pulps to be analyzed with aqua regia digestion OG46 method to determine if there may have been any volatilization of As.
- The use of a regression formula to define densities for the resource calculation proved adequate. However, Standardization of SG measurements potentially using whole core intervals, should be a considered for any oncoming drill program.
- Although the spread use of standard material produced in the late 1990s has yielded decent results, Jervois could re-examine and likely produce new CRMs for ongoing programs.
- QA/QC could be improved by the inclusion of field duplicates more frequently in the sequence, as well as the inclusion of true certified blanks as opposed to the red brick material used during the latest programs.
- Currently all historic and 2019 data is hosted in an excel format database. Ideally a commercial relational database should be used which has built-in error checking, audit documentation and QA/QC.
- Given the gradational nature of the sedimentary package, Orix recommends a detailed analysis of the existing geochemical data, to look for geochemical signatures that could highlight marker horizons. These marker horizons, particularly if found in the hanging wall zones, could prove very useful to determine true lateral continuity of some of those units, which could possibly give a positive impact in the re-definition of the resource categories.
- The current resource model presents the opportunity to test areas where plunges/ore shoots may be present. If said ore shoots do exist and continue at depth, they could represent prospective economic areas at depth.
- The Ram deposit lays less than 2 km north of the famous Blackbird group of deposits, mined by several operators in the 1900s. It is recommended that Jervois use the opportunity to explore the footwall of the Ram, in search of equivalent stratigraphic horizons that exist in the Blackbird mine.

1.15.2 Mining

The following summarizes the recommendations observed during the preparation of the current feasibility:

- Geotechnical – Classification and characterization of the rock mass in relation to it spatial location during initial development and access to ore zones will assist in stope dimension and overall mine design validation in coordination with the geotechnical borehole and study conducted in 2017 at the recommendation of the prior NI 43-101 technical report.
- Ventilation – A mine ventilation study utilizing best practice modeling software was conducted on the proposed mine design. The study identified the potential for a relatively short, bored ventilation raise to surface above the south ramp system would reduce ventilation costs. It is recommended the cost and permitting requirements for a ventilation raise be examined prior to development of the South Ramp system (Y3).
- Electrification – Additional optimization of the mine design, plan and especially the use of battery/electric haulage equipment to enable automation of mine systems should be examined early in the mine life to reduce

costs and the operational carbon footprint. Preliminary trade-off studies indicate additional power supply to the site will be required to support electrification. Studies should be advanced prior to ore production. Unused re-mucks along the ramp system may function as battery bays to reduce excavation requirements in this scenario.

- Contract Mining – The contract for underground mining and portal development should be formalized and executed as the project is approved to move forward.

1.15.3 Processing

It is recommended the following be conducted to for confirmation purposes during the detailed design phase of this project:

- Locked Cycle Test Work – Additional locked cycle test work to be conducted in order to provide additional confirmatory information regarding the metal recoveries of the concentrator.
- Material Handling Tests – Materials handling test work to be conducted in order to confirm design criteria and material flow angles with particular reference to the Fine Storage Bin located prior to the SAG mill.
- HAZOP Studies – During the detailed design phase, it is important to complete a hazard and operability study (HAZOP) in order to identify and evaluate potential risks to personnel or equipment so that the design can mitigate these risks.

2 INTRODUCTION

2.1 TERMS OF REFERENCE

In September 2019, Jervois Mining Limited ("Jervois") through its wholly-owned subsidiary, Jervois Mining USA commissioned DRA Americas Incorporated ("DRA") and M3 Engineering and Technology Corporation ("M3"), and its sub-consultants to prepare a Feasibility Study ("FS") for the production of a bulk cobalt-copper-gold concentrate from its Idaho Cobalt Operations ("ICO"), in east-central Idaho, USA.

A number of specialist consultants were engaged by Jervois Mining USA during the course of the FS; these parties are detailed in Table 2-1. These consultants reported directly to Jervois Mining USA with coordination of their inputs into the FS being undertaken by DRA. DRA also prepared a number of sections of the FS.

The FS scope included definition of project scope and implementation schedule to allow capital (initial and sustaining) and operating costs to be estimated to accuracies of +/-15%, and preparation of a Technical Report prepared in accordance with the reporting requirements of Canadian National Instrument ("N.I.") 43-101 and The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ('the JORC Code').

The requirements of electronic document filing on SEDAR (System for Electronic Document Analysis and Retrieval, www.sedar.com) necessitate the submission of this report as an unlocked, editable pdf (portable document format) file. The authors accept no responsibility for any changes made to the file after it leaves its control.

The authors of this report do not have nor has it previously had any material interest in Jervois Mining USA or related entities. The relationship with Jervois Mining USA is solely a professional association between the client and the independent consultant. This report is prepared in return for fees based upon agreed commercial rates, and the payment of these fees is in no way contingent on the results of this report.

The conclusions and recommendations in this report reflect the authors' best judgment in light of the information available to them at the time of writing. The authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to them subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

This report includes technical information, which requires subsequent calculations or estimates to derive sub-totals, totals and weighted averages. Such calculations or estimations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, they are not considered material.

Table 2-1: ICO Feasibility Study Consultants

Consultant	Primary Responsibility
DRA Americas Inc.	Study Management, Metallurgy and Process Design
M3 Engineering and Technology Corporation	Basic Engineering, Procurement and Capital Cost Estimation
Orix Geoscience Incorporated	Geology and Mineral Resource Estimation
Nick Yugo P. Eng.	Mine Design and Planning
KC Harvey Environmental, LLC.	Environmental Study – TWSF / WMP
SGS Canada Incorporated	Metallurgical Testwork
Grinding Solutions Limited	Comminution Testwork
Paterson & Cooke Canada Incorporated	Paste Testwork
NewFields Mining Design & Technical Services	Geotechnical Testwork

2.2 HISTORY

2.2.1 Discovery History

Copper mineralization in the Blackbird Creek area was discovered in 1892, and the area was soon explored as both a copper and gold prospect. The area was first mined by Union Carbide at the Haynes-Stellite Mine located south of the present Jervois Mining USA claim block, during World War I. Union Carbide mined approximately 4,000 tons of cobalt-bearing ore before ceasing operations, reportedly due to excessive mining costs. From 1938 to 1941, the Uncle Sam Mining and Milling Company operated a mine at the south end of the present Blackbird mine and reportedly mined about 3,600 tons of ore.

Calera Mining Company, a division of Howe Sound Company, developed and mined the Blackbird deposit between 1943 and 1959 under a contract to supply cobalt to the United States ("US") government. Calera mined approximately 1.74 M tons of ore grading 0.63% Co, 1.65% Cu, and 0.03 oz. Au/ton during this period, accounting for the majority of production from the district. Calera stopped mining when the government contract was terminated in 1960. Reportedly, poor payment for cobalt from smelters hindered continued development of the district, with minor exceptions.

Machinery Center Inc. mined 343,000 tons grading 0.36% Co and 0.64% Cu from the district between 1963 and 1966, when Idaho Mining Company (owned by Hanna Mining Company) purchased the property. Noranda optioned the property from Hanna in 1977 and carried out extensive exploration, mine rehabilitation and metallurgical testing. In 1979 Noranda and Hanna formed the Blackbird Mining Company ("BMC") to develop the property. BMC completed an internal feasibility study of their property at the time, including material from the Sunshine deposit in 1982. BMC allowed perimeter claims to lapse in 1994, and Formation Capital Corporation, U.S. ("FCC") re-staked much of that ground.

From 1995 to 2017, FCC completed a number of drilling campaigns and surface geochemical sampling in support of project activities. The Plan of Operations ("PoO") and the United States Environmental Impact Statement ("EIS") were also completed in 2006 and updated in 2008.

In October 2010, the FCC concluded a 5,727.5-ft diamond drill program drilled in six holes in a previously untested area on the Project along the southern extension of the Ram deposit. Data from this drill program was used for subsequent mine plan optimization studies. This drilling extended the previously defined strike length of the Ram deposit an additional 14% from 2,800 to 3,200 ft. The results of this drill program were incorporated into an updated resource estimate for the ICP/ICO and form a part of the 2015 PEA report.

As of the end of 2019, the Ram deposit has been tested with 120 diamond drill holes drilled in 1997 through 2017 by Formation Capital and drilled in 2019 by Jervois totaling 79,682.9 ft. Although drilling has been intermittent over the years, there has been continuity over the campaigns.

Table 2-2: ICO Drilling Campaigns

Year Drilled	Operator	Deposit	Number	Feet
1959	Calera Mining Company	Sunshine	3	982.0
1979 – 1981	Blackbird Mining Company	Sunshine	29	17,826.0
1995 – 1996	Formation Capital	Sunshine	48	29,144.0
1995 – 1996	Formation Capital	East Sunshine	24	14,723.5
	TOTAL Sunshine		**104**	**62,675.5**
1997	Formation Capital	Ram	20	12,045.0
1999	Formation Capital	Ram	11	5,210.5
2000*	Formation Capital	Ram	8	2,613.0
2004	Formation Capital	Ram	28	24,877.0

Year Drilled	Operator	Deposit	Number	Feet
2005	Formation Capital	Ram	9	5,302.5
2006	Formation Capital	Ram	4	4,532.0
2010	Formation Capital	Ram	6	5,727.5
2016	Formation Capital	Ram	9	3,057.5
2017	Formation Capital	Ram	6	6,062.1
2019	Jervois Mining	Ram	19	10,255.8
	TOTAL Ram		**120**	**79,682.9**
Grand Total		**Ram + Sunshine**	**224**	**142,358.4**

**Metallurgical Test holes – Not used in Grade Model*

2.2.1.1 Work Completed to Date

A pre-feasibility-level Technical Report on the ICP/ICO property was prepared by MDA and filed with SEDAR on October 31, 2006. Following this report, FCC decided to push forward with further development work, drilling, new resource model and metallurgical test work.

In September 2007, a technical report on the ICP/ICO (the "Technical Report"), derived from a more comprehensive feasibility study, was filed on SEDAR (www.sedar.com). The Technical Report was subsequently amended and refiled on SEDAR in May 2008.

The United States Department of Agriculture Salmon Challis National Forest (the "Forest Service") issued a revised Record of Decision ("RoD") for the ICO in January 2009. The RoD described the decision to approve a PoO for mining, milling and concentrating mineralized material from the ICP/ICO. The RoD was subsequently affirmed by the Forest Service in April 2009.

Construction on the ICO was planned in three stages; the first two have been completed. Stage I construction commenced in January 2010 and concluded in April 2010. Stage I consisted of timber clearing operations for the tailings waste storage facility ("TWSF"), topsoil stockpile area, roads around the mill site and concentrator pads. Stage II construction comprised primarily of earthworks preparation of all surface structures including mill and concentrator pads, access and haul roads, TWSF and portal bench preparation, and was dependent on securing additional financing discussed below.

In March 2011, FCC announced that it had concluded an equity financing for gross proceeds of CDN$80M. Proceeds of the financing were used to fund the continuation of engineering, procurement and construction at the ICO (Stage II), for reclamation bonding requirements and for general corporate purposes. Stage II construction commenced in July 2011 and concluded in late 2012. Stage II construction also included mine site portal bench development, geotechnical core drilling comprised of three H.Q. sized oriented core holes totaling 575 feet. Drilling was completed in December 2011.

In August 2014, a Technical Disclosure Review by the British Columbia Securities Commission determined that certain information in the Technical Report was deemed to be out of date with respect to, among other things, commodity prices, capital cost estimates and operating cost estimates and as such, was not to be relied upon. In January 2015, FCC commissioned Samuel Engineering to complete a Preliminary Economic Assessment (PEA) for its Idaho Cobalt Project. The PEA was originally completed in March 2015 and the revised Preliminary Economic Assessment was updated by Micon International Limited as part of a Technical Report in January 2017.

FCC continued project development through 2018 to complete construction of the Water Treatment Plant ("WTP"), electrical and site infrastructure as well as development of the portal bench before work was stopped at the end of the 2018 construction session.

Jervois acquired the ICO Project in April 2019 and proceeded with the 2019 drilling program and metallurgic test work.

2.3 PROPERTY DESCRIPTION

2.3.1 Location and Access

The ICO Property consists of 243 contiguous unpatented lode mining claims located in east central Idaho, approximately 25.8 miles (41.5 km) west of the town of Salmon, as shown on the location map provided in Figure 2-1. The property covers 4,475 acres centered on 45°07'50" north latitude and 114°21'42" west longitude. It is within the Gant Mountain 7.5-minute quadrangle of the USGS Topographic Map Series. More specifically, the ICO unpatented mining claims are located in Sections 8, 9, 15, 16, 17, 18, 20, 21, 22, 23, 26, 27, 28, 29, 33, 34, and 35 Township 21 North, Range 18 East (Figure 2-2). The claim block is within the Salmon-Cobalt Ranger District of the Salmon-Challis National Forest (Prenn, 2005), lands under surface use administration by the United States Forest Service ("USFS"). The mine portal is located at an elevation of approximately 7,060 feet above sea level, and the processing plant and most of the site infrastructure is located on Big Flat, which is approximately 930 feet above the mine.

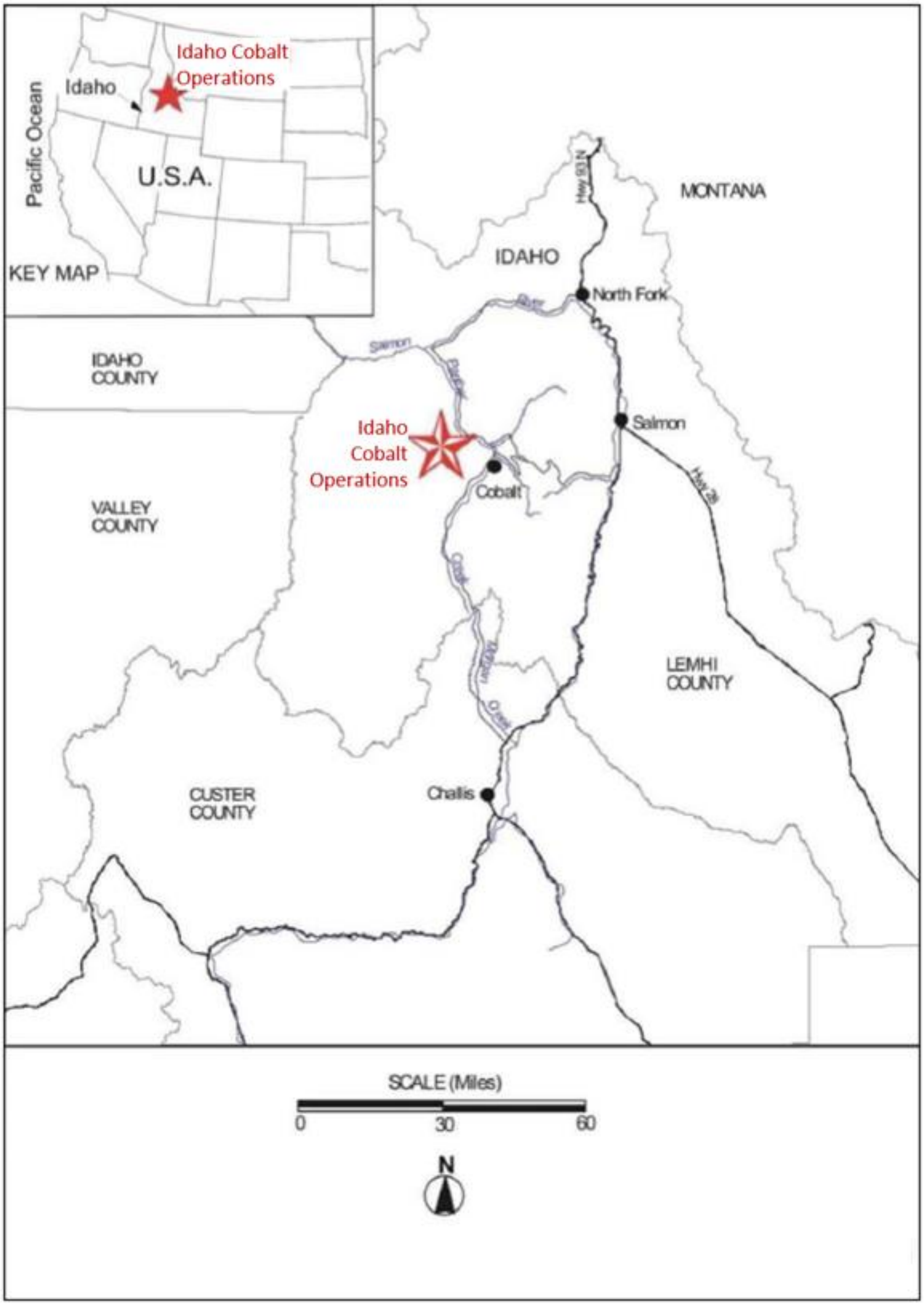
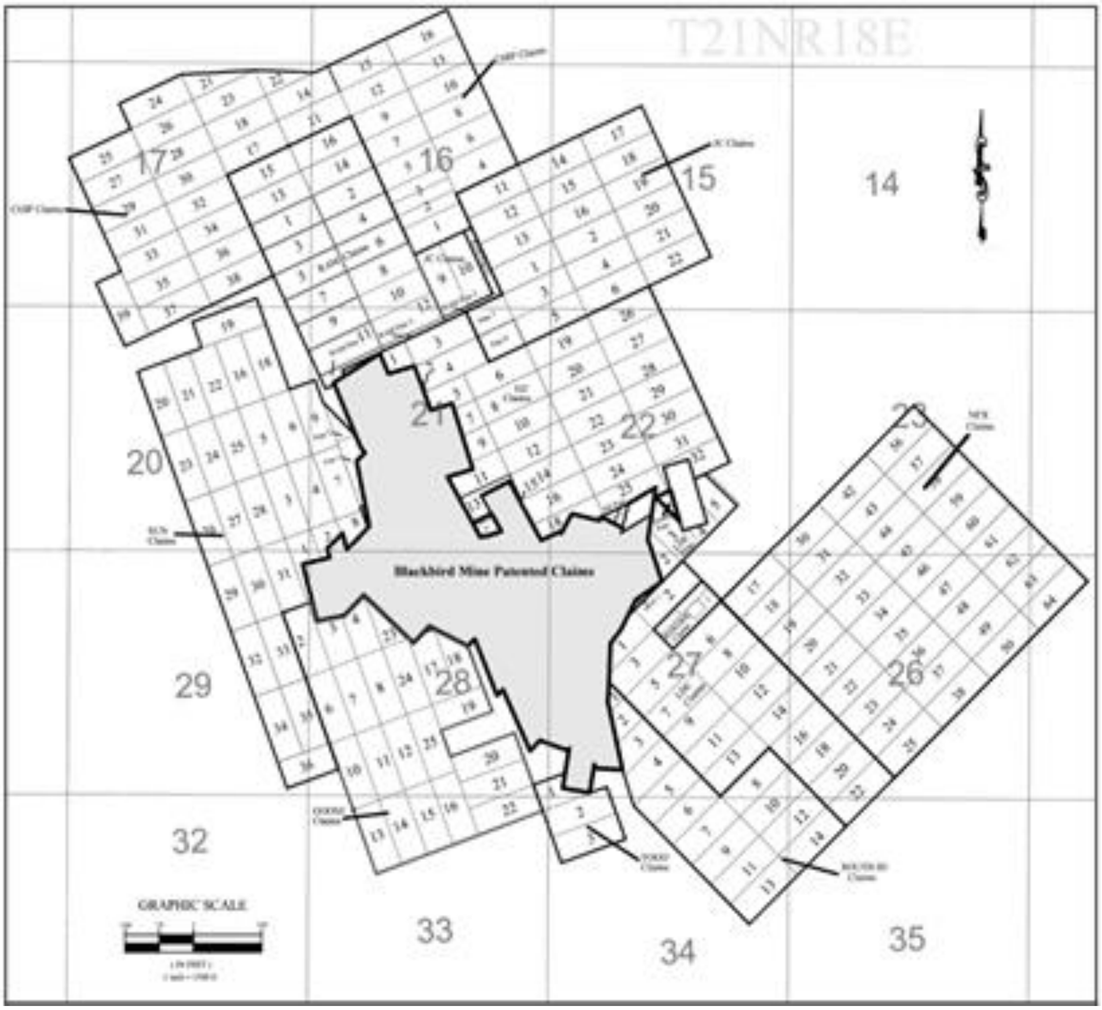


Figure 2-1: Location Map of the Idaho Cobalt Operations
Source: Map supplied by FCC, 2016

2.3.2 Title and Boundaries

The ICO is 100% owned by Jervois Mining USA. The claims that comprise the ICO are listed in Table 4-1. Ownership of unpatented mining claims in the US is in the name of the holder (locator), with ownership of the minerals belonging to the United States of America, under the administration of 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 federal government. The present ICO property position consists of 313 unpatented claims. Copies of individual unpatented mining claim notices and the detailed map showing their locations are on file with the BLM office in Salmon and with the Lemhi County Recorder's office in Salmon.



Idaho Cobalt Operations Claim Holdings - Located in unsurveyed sections 8, 9, 15, 16, 17, 18, 20, 21, 22, 23, 26, 27, 28, 29, 33, 34 and 35, Township 21 North, Range 18 East. Boise Meridian, Lemhi County Idaho

Figure 2-2: Idaho Cobalt Operations Claim Holdings

2.3.3 Climate and Physiography

2.3.3.1 Climate

The Natural Resources Conservation Service ("NRCS") Morgan Creek SnoTel station is located approximately 20 air miles south-southeast of the ICO at an elevation of 7,600-feet (NRCS, 2004). Based on 12 years of data (1991-2003), the average annual temperature at the station is 34.8 degrees Fahrenheit (ºF), with a low of –34.6ºF and a high of 89.4ºF. Based on 23 years of data (1981-2004), annual precipitation is 24.4 inches. About 60 per cent of the precipitation occurs as snow during the winter months (14.7 inches).

2.3.3.2 Physiography

The ICO is located in the Salmon River Mountains of central Idaho, within the Northern Rocky Mountain physiographic province. Major waterways in the area include the Salmon River and Panther Creek. These waterways are located in the upper reaches of the Snake River Basin, which drains to the Columbia River. The ICO is within the Panther Creek sub-basin of the Salmon River.

The Project area contains flat-topped mountains and moderate to steep V-shaped canyons, and covers an area ranging in elevation from 6,100 ft to 8,100 ft. The area that may potentially be affected by mining and mill operations is bounded by the divides of the streams that generally drain the Project area, namely Bucktail Creek and Big Flat Creek. Bucktail Creek drains into the South Fork of Big Deer Creek, which drains to Big Deer Creek, which then drains to Panther Creek. Big Flat Creek drains directly into Panther Creek, which reports to the Salmon River.

The terrain in the mine area is made up of slopes approaching 35% and is cut by narrow valleys. The mineralized material outcrops between elevations of 7,400 ft and 7,800 ft, with most facilities located at 6,850 feet. Soils in the area are generally comprised of sandy loam averaging 5 ft in depth, with frequent rock outcroppings. Bedrock exposure amounts to only about 1% to 3% of the property area. Large boulder fields are found in many areas along the higher mountain ridges.

During the summer of 2000 the Clear Creek Fire burned over 200,000 acres, including the area of the ICO. The severity of the fire was high over most of the area, with all of the canopy cover and most of the litter and duff burned off. A preliminary assessment indicates that the degree of change that occurred was influenced by the various fuel loads, species, ladder fuels, canopy closures, slope and aspect components interacting with fire weather conditions at the site. As a consequence, typical mosaic patterns now prevail that are consistent with large fire behavior in this type of ecosystem. Post-fire vegetation establishment in the Project area in 2004 was variable, with vegetation cover ranging from 30% to 80% depending on slope, aspect, fire intensity and severity, soil type and post-fire seeding.

2.3.4 Existing Infrastructure

Salmon, Idaho, is the nearest town and is located about 26 miles east of the property. The 2000 Census reported a population of about 3,120 people (www.city-data.com, 2005). Salmon is a local supply and transportation centre, with an airport paved with a 5,510 x 75 ft. airstrip at an elevation of 4,044 ft. The nearest railroad is at Dubois, smaller town 100 miles southeast of Salmon. A 4 M.W. power line extends from Salmon to BMC's Blackbird Mine site.

Jervois Mining USA has purchased the Salmon Depot which is located approximately one mile south of Salmon, Idaho along Highway 93. The site is approximately three acres in total area with a single warehouse building at site with a compacted gravel base across the entire property. The depot will store concentrate from the mine site and provide temporary storage for any partial shipments received prior to transport to the mine site to minimize traffic along the mine access road.

The depot is located immediately adjacent to the highway and is surrounded on all side with a 6-ft chain-link fence with lockable gates. All man doors are steel insulated with locking deadbolts and all overhead doors are steel insulated providing protection from any potential break ins. The yard is clearly visible from the highway and any activity in the depot would be readily seen from the highway.

2.4 PRINCIPAL DATA SOURCES

The principal sources of information for this report are:

- Previous NI 43-101 Technical Reports on the ICP filed on SEDAR
- Drill hole databases supplied by Jervois Mining USA
- ICO Ram deposit block model supplied by Mining Development Associates ("MDA")
- Observations made during the site visits
- Discussions/meetings with Jervois management/staff/consultants familiar with the property
- Data/reports supplied by Jervois and its consultants
- Metallurgical Reports by SGS (2015, 2016 and 2017)
- Experience gained while working on similar deposits

The authors are pleased to acknowledge the helpful cooperation of Jervois's management, staff and consultants who made all data requested available and responded openly and helpfully to all questions, queries and requests for material. The qualified persons and their associated responsibilities are listed in Table 2-3.

Table 2-3: Qualified Persons

QP Name	Company	Qualification	Site Visit Date	Area of Responsibility
Matthew Sletten	M3 Engineering & Technology Corporation – Tucson, AZ	P.E.	August 13, 2019	Sections 2, 3, 4, 5, 6, 18, 21, 23, and corresponding sections of 1, 24, 25, 26 and 27
Scott Zelligan	Independent Resource Geologist and Associate to Orix Geoscience	P. Geo.	October 4 to 6, 2019	Sections 7, 8, 9, 10, 11, 12, 14, and corresponding sections of 1, 24, 25, 26 and 27
Dave Frost	DRA Americas	FAusIMM	August 13, 2019	Sections 13, 17, 21 and corresponding sections of 1, 24, 25, 26, and 27
Nick Yugo	9140697 CANADA Inc.	P. Eng.	December 11 to 13, 2018	Sections 15, 16 and corresponding sections of 1, 24, 25, 26, and 27
Celine Charbonneau	DRA Americas	P. Eng.	N/A	Sections 19, 22 and corresponding sections of 1, 24, 25, 26, and 27.
David P. Cameron	KC Harvey Environmental, LLC	P.E.	September 23 to 24, 2020	Section 20 and corresponding sections of 1, 24, 25, 26, and 27.

The following notable site visits were performed, as previously indicated in the table above:

- Matt Sletten of M3 visited the site on August 13, 2019 to perform a general review and walkthrough for engineering purposes.
- Scott Zelligan, an independent geologist, visited the site on October 4 to 6, 2019 to review the core shack, the active drilling operation, the historical core storage, and the general site and infrastructure. This included a review of current sampling practices and historical assay results.
- Dave Frost of DRA Americas visited the site on August 13, 2019 to review the plant location site including plant, equipment and infrastructure already installed at the site.

- Nick Yugo, of 9140697 CANADA Inc., visited the site on December 11 to 13, 2018 to perform a general review and walkthrough.
- David P. Cameron of KC Harvey Environmental, LLC visited the site on September 23 to 24, 2020 to review the site for environmental purposes.

2.5 UNITS, CURRENCY AND ABBREVIATIONS

Currency values are expressed in United States dollars ($ or USD), unless otherwise indicated.

Abbreviations used in this report are listed in Table 2-4.

Table 2-4: List of Abbreviations

Abbreviation	Term
°	Degree(s)
°C	Degree(s) Centigrade
°C-days	Degree Centigrade days
°F	Degree(s) Fahrenheit
<	Less than
>	Greater than
µg/L	Micrograms per litre
µm	Micrometre(s) (micron = 0.001 mm)
%	Percent, percentage
'	Minutes of latitude and longitude
3D	Three dimensional
A	Ampere(s)
AAS	Atomic absorption spectroscopy
ABA	Acid based accounting
Ag	Silver
Al	Aluminum
amsl	Above mean see level
ANFO	Ammonium nitrate-fuel oil
ARD	Acid Rock Drainage
As	Arsenic
Au	Gold
B	Billion
BMPs	Best Management Practices
BO	Biological Opinion
C	Carbon
Ca	Calcium
CCR	Central control room
CERCLA	Comprehensive Environmental Response, Compensation, and Liability Act
cfm	Cubic feet per minute
cfm/bhp	Cubic feet per minute per brake horsepower
cfs	Cubic feed per second
CGP	General Permit for Discharges from Construction Activities
cm	Centimetre(s)
CPF	Cobalt Processing Facility
Co	Cobalt
Cu	Copper
CV	Coefficient of variation
CWA	Clean Water Act

Abbreviation	Term
d	Day(s)
dB(A)	Decibel(s) (adjusted)
dmt	Dry metric tonne(s)
dtpd	Dry metric tonnes per day
dwt	Dead weight tonnes
EPCM	Engineering, procurement and construction management
F	Fluorine
FA	Financial Assurance
Fe	Iron
FEIS	Final Environmental Impact Statement
FOB	Free on board
ft	Foot, feet
FW	Footwall
g	Gram(s)
g	Acceleration due to gravity
g/L	Grams per litre
g/t	Grams per tonne
Ga	Billion years (old, ago)
GA	General arrangement
gal	Gallon(s) (US)
GHG	Green House Gas (emissions)
gpm	Gallons per minute
GPS	Global positioning system
GWh	Gigawatt-hour
H	Hydrogen
h	Hour(s)
h/d	Hours per day
h/w	Hours per week
ha	Hectare(s)
HAZOP	Hazard and operability study
HCT	Humidity Cell Tests
HDPE	High density polyethylene
HP	Horsepower
HQ	Diamond drill core size 63.5 mm (inside diameter of core tube)
HU	Habitat unit
Hz	Hertz
IATF	Inter-Agency Task Force
IBA	Impact benefit agreement
ICP	Inductively Coupled Plasma
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICO	Idaho Cobalt Operations
IDEQ	Idaho Department of Environmental Quality
IDWR	Idaho Department of Water Resources
in	Inch(es)
IPDES	Idaho Pollutant Discharge Elimination System
IRR	Internal Rate of Return
IX	Ion Exchange
J	Joule(s)
JRP	Joint Review Process
K	Potassium
k	Kilo (thousand)

Abbreviation	Term
kcfm	Thousand cubic feet per minute
kg	Kilogram(s)
kg/h	Kilograms per hour
kg/m³	Kilograms per cubic metre
km	Kilometre(s)
km/h	Kilometres per hour
kPa	Kilopascal(s)
kV	Kilovolt(s)
kVA	Kilovolt-ampere(s)
kW	Kilowatt(s)
kWh	Kilowatt hour
kWh/t	Kilowatt hours per tonne
L	Litre(s)
L/h	Litres per hour
L/s	Litres per second
LAN	Local area network
lb	Pound(s)
LCT	Locked cycle test
LHD	Load-haul-dump
LME	London Metal Exchange
LOI	Loss on ignition
LOM	Life of mine
LRMP	Salmon National Forest Land and Resource Management
M	mega (million)
m	Metre(s)
m³/h	Cubic metres per hour
m/min	Metres per minute
m/s	Metres per second
mA	Milliampere(s)
Ma	Million years (old, ago)
masl	Metres above sea level
MBBR	Moving Bed Biofilm Reactor
MCC	Motor control centre
Mgal	Million gallons
min	Minute(s)
ML	Million litres
mL	Millilitres
ML/d	Million litres per day
mm	Millimetre(s)
mg/L	Milligrams per litre
Mg	Magnesium
MPa	Megapascal(s)
MSGP	Multi-Sector General Permit for Stormwater Discharges associated with Industrial Activity
MSHA	Mining Safety and Health Administration
MW	Megawatt(s)
MW	Megawatt(s)
MWh	Megawatt hour(s)
Na	Sodium
NAG	Net acid generating
NEPA	National Environmental Policy Act

Abbreviation	Term
NI 43-101	Canadian National Instrument 43-101
NMFS	National Oceanic and Atmospheric Association National Marine Fisheries Service
NNP	Net-Neutralization Potential
NO2	Nitrous oxide
non-PAG	non- Potentially Acid Generating
NPDES	National Pollutant Discharge Elimination System
NPV	Net present value
NSR	Net smelter return
NWT	Northwest Territories
oz	Ounce(s), troy ounces
oz/ton	Ounces per ton (short ton, 2,000 pounds)
P&ID	Process and instrumentation diagram
Pa	Pascal(s)
Pa·s	Pascal-second
PAG	Potentially Acid Generating
Pb	Lead
ppb	Parts per billion
ppm	Parts per million
POC	Point of Compliance
PoO	Plan of Operations
P3	Public private partnerships
PVC	Polyvinyl chloride
QA	Quality assurance
QA/QC	Quality assurance/quality control
QC	Quality control
RBC	Rotating biological contactor
RMB	Chinese Renminbi
ROD	Record of Decision
ROM	Run-of-mine
rpm	Revolutions per minute
RQD	Rock quality designation
s	Second(s)
S	Sulphur
SAG	Semi-autogenous grinding
Sb	Antimony
SCNF	Salmon-Challis National Forest
SEM	Scanning electron microscope
SG	Specific gravity
SI	International system of units
Si	Silicon
SO_2	Sulphur dioxide
SWPPP	Stormwater Pollution Prevention Plan
T	Ton(s) – short (2,000 lb)
t/h	Tons per hour
t/y	Ton per year
t/d	Tons per day
TOR	Terms of reference
TSP	Total suspended particulates
TSS	Total suspended solids
TWSF	Tailings and Waste Rock Storage Facility

Abbreviation	Term
UCS	Unconfined compressive strength
UF	Ultrafiltration
USACE	US Army Corps of Engineers
USDA	United States Department of Agriculture
USFS	US Forest Service
USEPA	US Environmental Protection Agency
USFWS	US Fish and Wildlife Service
US$	United States dollar(s)
V	Volt(s)
WMP	Water Management Pond
WTP	Water Treatment Plant
XRF	Energy-dispersive x-ray fluorescence
y	Year(s)
yd^3	Cubic yard(s)
Zn	Zinc

3 RELIANCE ON OTHER EXPERTS

The authors are not experts in legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements in the United States. The authors did not conduct any investigations of the environmental, permitting, or social-economic issues associated with the ICO property, and the authors are not experts with respect to these issues. The authors have fully relied on ICO to provide complete information concerning the legal status of Jervois and related companies, as well as current legal title, material terms of all agreements, and material environmental and permitting information that pertain to the ICO property.

Section 4 in its entirety is based on information provided by ICO, and the authors offer no professional opinions regarding the provided information. The authors have relied on this information and have supplemented it by their own observations on site which have generally corroborated the presented information.

The authors present this information to fulfill reporting requirements of NI 43-101 and express no opinion regarding the legal or environmental status of the ICO property. The description of the property is presented here for general information purposes only, as required by NI 43-101. The authors are not qualified to provide professional opinion on issues related to mining and exploration lands title or tenure, royalties, permitting and legal and environmental matters. Accordingly, the authors have relied upon the representations of the issuer, Jervois, for Section 4.0 of this report, and have not verified the information presented therein.

Those portions of the report that relate to the location, property description, infrastructure and history (Sections 4.0 to 6.0) are taken, at least in part, from current and previous texts prepared for ICO. Some of the figures and tables for this report were reproduced or derived from reports written for ICO during previous report development.

Mr. Klaus Wollhaf, a Director of KSSM Consulting Pty Limited, Mindarie, Australia supplied the data and elaborated the basis of Section 19 – Market Studies and Contracts. Mr. Wollhaf has over 30 years of diverse experience in the mining industry and has held senior operational, technical, marketing, and commercial roles, including over 15 years in marketing of nickel, copper and cobalt products for Falconbridge and Xstrata, and as an independent consultant. He holds a Bachelor of Engineering – Mineral Process Engineering from the University of the Witwatersrand and a Bachelor of Commerce in Management and Finance from the University of South Africa. DRA has access to a confidential memo written by Mr. Wollhaf to Jervois Mining Limited dated September 18, 2020 titled "Idaho Cobalt Operations Product Pricing".

Mr. Andrew van Dinter from Ernst & Young, Melbourne, Australia reviewed the tax assumptions used in Jervois Financial Model. After reviewing Jervois' financial model, EY wrote "nothing came to their attention to suggest that the tax assumptions were not constructed appropriately, as they are logical and reasonable, so as to materially achieve the objective described above under the base case assumptions".

4 PROPERTY DESCRIPTION AND LOCATION

4.1 SUMMARY

4.1.1 Ownership and Access

The Idaho Cobalt Operations ("ICO" or "Project") is a primary high-grade cobalt deposit located in Lemhi County, Idaho held by the Company's 100% owned subsidiary, Jervois Mining USA. Limited and was extensively explored before the initial phases of construction. The Project covers an area of approximately 5,990 acres and includes 313 contiguous unpatented lode mining claims. This property is not subject to any royalty payments.

Vehicle access to the ICO is via a series of well-maintained, public-access gravel roads that lead west from a point on paved Highway 93, approximately 6 miles south of Salmon, Idaho, as shown in the figure below. This gravel road leads to the Blackbird Mine, which is currently not operating; however, the road is kept open year-round, and a potential mining operation can operate year-round. The total driving distance from Salmon to the ICO proposed mill site is approximately 48 miles.

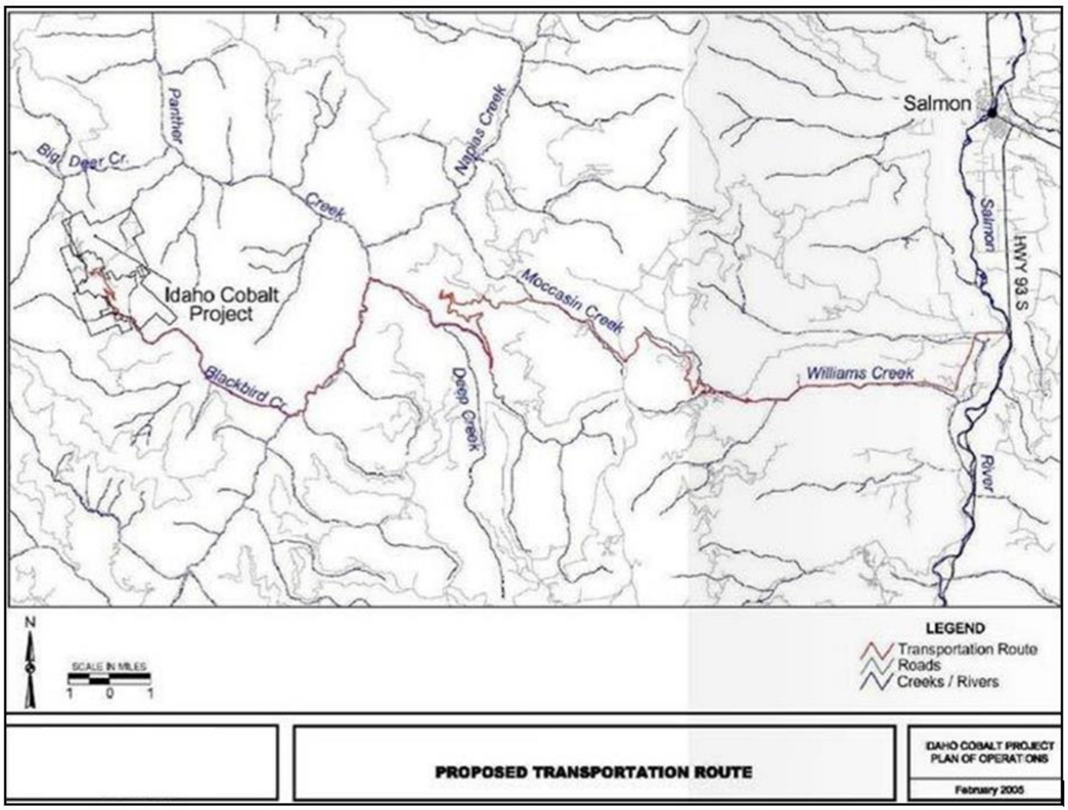


Figure 4-1: Idaho Cobalt Operations Location

4.1.2 Corporate

The current corporate structure is as follows:

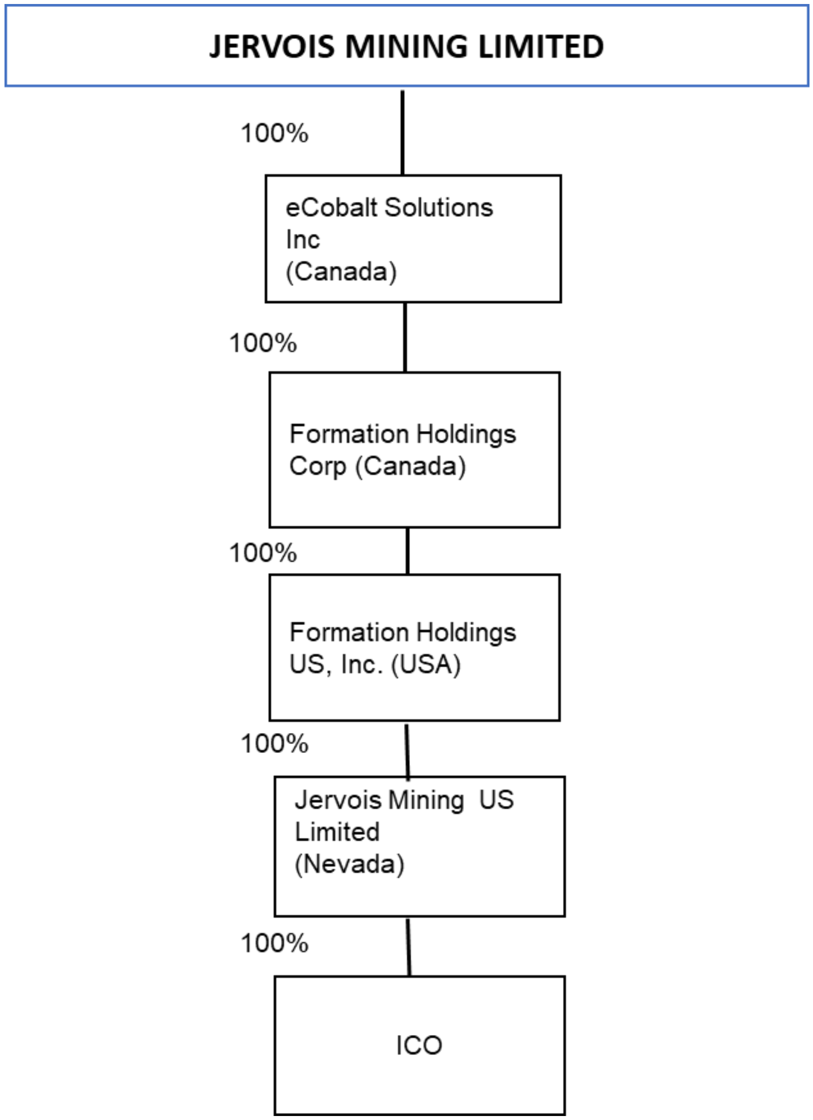


Figure 4-2: ICO Corporate Structure

4.1.3 Land and Resources

4.1.3.1 Land

The ICO property consists of 313 contiguous unpatented lode mining claims located in east-central Idaho, approximately 25.8 miles (41.5 km) west of the town of Salmon, as shown on the location map provided in the figure below.

The property covers approximately 5,990 acres centered on 45°07’50” north latitude and 114°21’42” west longitude. It is within the Gant Mountain 7.5-minute quadrangle of the USGS Topographic Map Series. More specifically, the ICO unpatented mining claims are located in Sections 8, 9, 15, 16, 17, 18, 20, 21, 22, 23, 26, 27, 28, 29, 33, 34, and 35, Township 21 North, Range 18 East.

The claim block is within the Salmon-Cobalt Ranger District of the Salmon-Challis National Forest (Prenn, 2005), lands under surface use administration by the United States Forest Service (“USFS”). The mine portal is located at an elevation of approximately 7,060 ft above sea level, and the processing plant and most of the site infrastructure is located on a plateau unique to the area known as the Big Flat, which is approximately 930 ft above the mine.

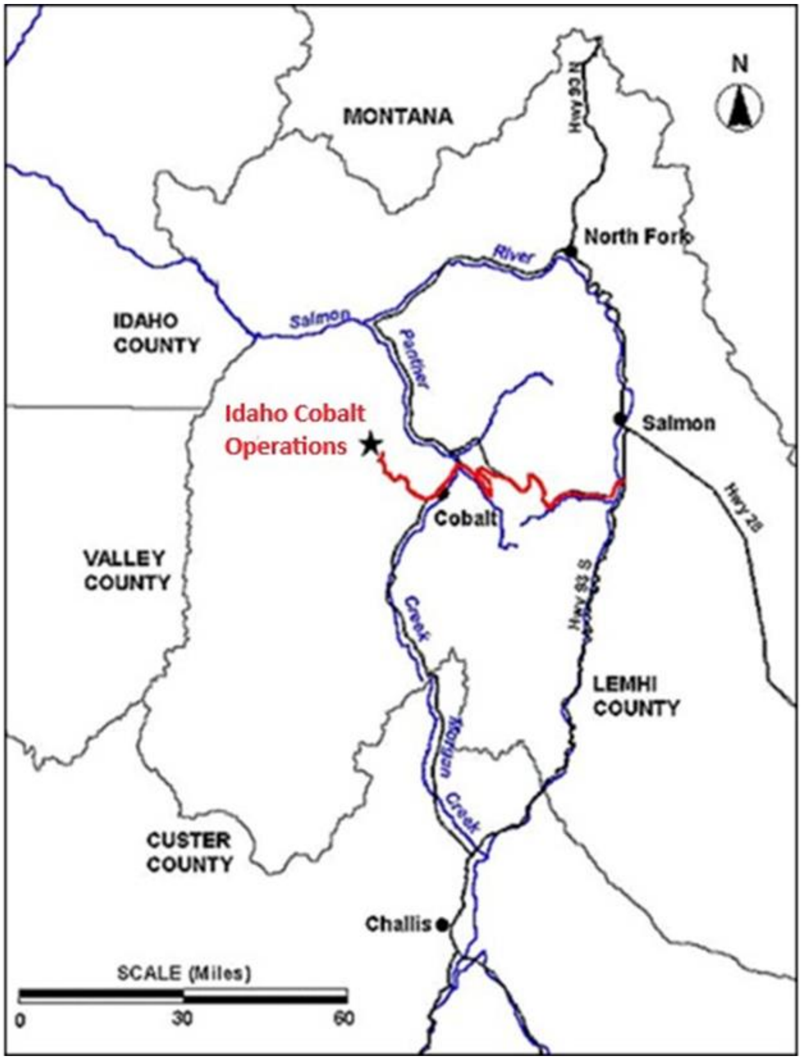


Figure 4-3: Idaho Cobalt Operations

4.1.3.2 Documentation and Status of the Mineral Claims

Ownership of unpatented mining claims in the United States ("U.S.") is in the name of the holder (locator), with ownership of the minerals belonging to the United States of America, under the administration of 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 federal government. It should also be noted that in recent years there have been U.S. Congressional efforts to change the 1872 mining law to include the provision of federal production royalties. Currently, however, annual claim maintenance and filing fees are the only federal encumbrances to unpatented mining claims.

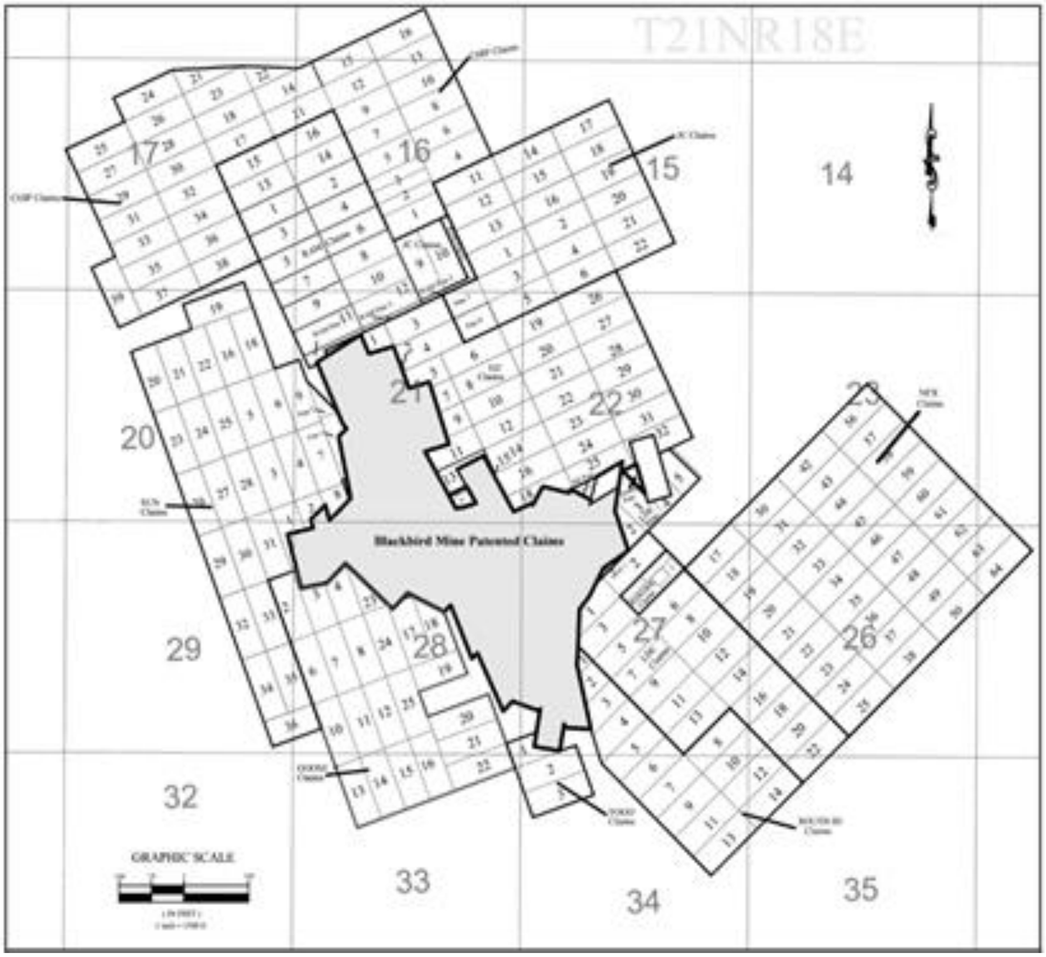
Taylor Mountain Survey surveyed the mining claims covering the northwest end of the property which includes the Ram deposit, mill site and the tailings and waste rock storage facility; fractional claims were located to cover all fractions.

Table 4-1 lists the claims that comprise the ICO.

Table 4-1: Claims that Comprise the ICO

ICO Mining Claims								
Claim Name	**County #**	**IMC #**	**Claim Name**	**County #**	**IMC #**	**Claim Name**	**County #**	**IMC #**
Chelan No. 1 (A)	248345	175861	HZ 22	224194	174660	NFX 49	307262	218717
Chip 1	248956	184883	HZ 23	224195	174661	NFX 50	307263	218718
Chip 2	248957	184884	HZ 24	224196	174662	NFX 56	307269	218724
Chip 3 (A)	277465	196402	HZ 25	224197	174663	NFX 57	307270	218725
Chip 4 (A)	277466	196403	HZ 26	224198	174664	NFX 58	307271	218726
Chip 5 (A)	277467	196404	HZ 27	224199	174665	NFX 59	307272	218727
Chip 6 (A)	277468	196405	HZ 28	224200	174666	NFX 60	307273	218728
Chip 7 (A)	277469	196406	HZ 29	224201	174667	NFX 61	307274	218729
Chip 8 (A)	277470	196407	HZ 30	224202	174668	NFX 62	307275	218730
Chip 9 (A)	277471	196408	HZ 31	224203	174669	NFX 63	307276	218731
Chip 10 (A)	277472	196409	HZ 32	224204	174670	NFX 64	307277	218732
Chip 11 (A)	277473	196410	HZ Frac.	228967	177254	Powder 1	269506	190491
Chip 12 (A)	277474	196411	JC 1	224165	174631	Powder 2	269505	190492
Chip 13 (A)	277475	196412	JC 2	224166	174632	Ram 1	228501	176757
Chip 14 (A)	277476	196413	JC 3	224167	174633	Ram 2	228502	176758
Chip 15 (A)	277477	196414	JC 4	224168	174634	Ram 3	228503	176759
Chip 16 (A)	277478	196415	JC 5 (A)	245689	174635	Ram 4	228504	176760
Chip 17 (A)	277479	196416	JC 6	224170	174636	Ram 5	228505	176761
Chip 18 (A)	277480	196417	JC 7 Frac.	224171	174637	Ram 6	228506	176762
Chip 21 Frac.	306059	218113	JC 8 Frac.	224172	174638	Ram 7	228507	176763
Chip 22 Frac.	306060	218114	JC 9	228054	176750	Ram 8	228508	176764
Chip 23	306025	218115	JC 10	228055	176751	Ram 9	228509	176765
Chip 24	306026	218116	JC 11	228056	176752	Ram 10	228510	176766
Chip 25	306027	218117	JC-12	228057	176753	Ram 11	228511	176767
Chip 26	306028	218118	JC-13	228058	176754	Ram 12	228512	176768
Chip 27	306029	218119	JC 14	228971	177250	Ram 13 (A)	245700	181276
Chip 28	306030	218120	JC 15	228970	177251	Ram 14 (A)	245699	181277
Chip 29	306031	218121	JC 16	228969	177252	Ram 15 (A)	245698	181278
Chip 30	306032	218122	JC 17	259006	187091	Ram 16 (A)	245697	181279
Chip 31	306033	218123	JC 18	259007	187092	Ram Frac.1 (A)	245696	178081
Chip 32	306034	218124	JC 19	259008	187093	Ram Frac.2 (A)	245695	178082
Chip 33	306035	218125	JC 20	259009	187094	Ram Frac.3 (A)	245694	178083
Chip 34	306036	218126	JC 21	259010	187095	Ram Frac.4 (A)	245693	178084
Chip 35	306037	218127	JC 22	259011	187096	South ID 1 (A)	248725	175874
Chip 36	306038	218128	LDC Frac.1 (A)	248720	175880	South ID 2 (A)	248726	175875
Chip 37	306039	218129	LDC Frac.2 (A)	248721	175881	South ID 3 (A)	248727	175876
Chip 38	306040	218130	LDC Frac.3 (A)	248722	175882	South ID 4 (A)	248717	175877
Chip 39	306041	218131	LDC Frac.4 (A)	248723	175883	South ID 5 (A)	248715	176743
DEWEY Frac. (A)	248739	177253	LDC Frac.5 (A)	248724	175884	South ID 6 (A)	248716	176744
Goose 2 (A)	259554	175863	LDC-1	224140	174579	South ID 7	306433	218216

ICO Mining Claims								
Claim Name	County #	IMC #	Claim Name	County #	IMC #	Claim Name	County #	IMC #
Goose 3	227285	175864	LDC-2	224141	174580	South ID 8	306434	218217
Goose 4 (A)	259553	175865	LDC-3	224142	174581	South ID 9	306435	218218
Goose 6	227282	175867	LDC-5	224144	174583	South ID 10	306436	218219
Goose 7 (A)	259552	175868	LDC-6	224145	174584	South ID 11	306437	218220
Goose 8 (A)	259551	175869	LDC-7	224146	174585	South ID 12	306438	218221
Goose 10 (A)	259550	175871	LDC-8	224147	174586	South ID 13	306439	218222
Goose 11 (A)	259549	175872	LDC-9	224148	174587	South ID 14	306440	218223
Goose 12 (A)	259548	175873	LDC-10	224149	174588	Sun 1	222991	174156
Goose 13	228028	176729	LDC-11	224150	174589	Sun 2	222992	174157
Goose 14 (A)	259547	176730	LDC-12	224151	174590	Sun 3 (A)	245690	174158
Goose 15	228030	176731	LDC-13 (A)	248718	174591	Sun 4	222994	174159
Goose 16	228031	176732	LDC-14 (A)	248719	174592	Sun 5	222995	174160
Goose 17	228032	176733	LDC-16	224155	174594	Sun 6	222996	174161
Goose 18 (A)	259546	176734	LDC-18	224157	174596	Sun 7	224162	174628
Goose 19 (A)	259545	176735	LDC-20	224159	174598	Sun 8	224163	174629
Goose 20	228035	176736	LDC-22	224161	174600	Sun 9	224164	174630
Goose 21	228036	176737	NFX 17	307230	218685	Sun 16 (A)	245691	177247
Goose 22	228037	176738	NFX 18	307231	218686	Sun 18 (A)	245692	177249
Goose 23	228038	176739	NFX 19	307232	218687	Sun 19	277457	196394
Goose 24	228039	176740	NFX 20	307233	218688	Sun 20	306042	218133
Goose 25	228040	176741	NFX 21	307234	218689	Sun 21	306043	218134
HZ 1	224173	174639	NFX 22	307235	218690	Sun 22	306044	218135
HZ 2	224174	174640	NFX 23	307236	218691	Sun 23	306045	218136
HZ 3	224175	174641	NFX 24	307237	218692	Sun 24	306046	218137
HZ 4	224176	174642	NFX 25	307238	218693	Sun 25	306047	218138
HZ 5	224413	174643	NFX 30	307243	218698	Sun 26	306048	218139
HZ 6	224414	174644	NFX 31	307244	218699	Sun 27	306049	218140
HZ 7	224415	174645	NFX 32	307245	218700	Sun 28	306050	218141
HZ 8	224416	174646	NFX 33	307246	218701	Sun 29	306051	218142
HZ 9	224417	174647	NFX 34	307247	218702	Sun 30	306052	218143
HZ 10	224418	174648	NFX 35	307248	218703	Sun 31	306053	218144
HZ 11	224419	174649	NFX 36	307249	218704	Sun 32	306054	218145
HZ 12	224420	174650	NFX 37	307250	218705	Sun 33	306055	218146
HZ 13	224421	174651	NFX 38	307251	218706	Sun 34	306056	218147
HZ 14	224422	174652	NFX 42	307255	218710	Sun 35	306057	218148
HZ 15	231338	178085	NFX 43	307256	218711	Sun 36	306058	218149
HZ 16	231339	178086	NFX 44	307257	218712	Sun Frac.1	228059	176755
HZ 18	231340	178087	NFX 45	307258	218713	Sun Frac.2	228060	176756
HZ 19	224427	174657	NFX 46	307259	218714	Togo 1	228049	176769
HZ 20	224428	174658	NFX 47	307260	218715	Togo 2	228050	176770
HZ 21	224193	174659	NFX 48	307261	218716	Togo 3	228051	176771



Idaho Cobalt Operations Claim Holdings
Located in unsurveyed sections 8, 9, 15, 16, 17, 18, 20, 21, 22, 23, 26, 27, 28, 29, 33, 34 and 35, Township 21 North, Range 18 East. Boise Meridian, Lemhi County Idaho

Figure 4-4: Idaho Cobalt Operations Claim Holdings

The present ICO property position consists of 313 unpatented claims. Copies of individual unpatented mining claim notices and the detailed map showing their locations are on file with the BLM office in Salmon and with the Lemhi County Recorder's office in Salmon. The claim notices and maps on file with the BLM and Lemhi County constitute the legal descriptions of the unpatented mining claims. The claim locations in the field take precedence should there be a discrepancy between descriptions and maps. The BLM serial numbers (IMC numbers) for each claim or claim group were previously listed in the table above. These numbers provide sufficient information to identify specific claims and their detailed description and map which are on file.

To maintain the claims in good standing, the Company pays annual claim maintenance and filing fees to the BLM before September 1 of each calendar year. Other than maintenance and filing fees, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the ICO property.

4.1.3.3 Local Resources and Infrastructure

Salmon, Idaho, is the nearest town and is located about 26 miles east of the property. The 2000 Census reported a population of about 3,120 people (www.city-data.com, 2005). Salmon is a local supply and transportation centre, with an airport paved with a 5,510 x 75-ft. airstrip at an elevation of 4,044 ft. The nearest railroad is at Dubois, a smaller town 100 miles southeast of Salmon. A 4 MW power line extends from Salmon to Glencore's Blackbird Mine site.

Although Salmon currently does not provide services for mining activities, it has functioned in this manner for past mining activities at Noranda's (now owned by Glencore) former Blackbird mine, and at Meridian Gold's former Beartrack gold mine. Salmon has, and can again, serve as a location for personnel housing and a staging point for mine support services.

The area covered by the Idaho claims is sufficiently large to accommodate underground operations, including ancillary installations.

4.1.4 Intellectual Property and Technology

No intellectual property currently exists in relation to any process technologies or equipment relating to the Project, and no new technology is foreseen as a result of the implementation of the Project.

4.2 LEGAL

4.2.1 Operating Legal Framework

The Project may be subject to United States Federal and State legislation and government agency regulatory systems, including the following:

- USFS regulations for mineral activities on National Forest lands (36 CFR 228A)
- National Environmental Policy Act ("NEPA")

4.2.2 Sovereign Risk

The project is located in Idaho, U.S. and will be subject to the various political, economic, and other risks and uncertainties associated with operating in that country. The U.S. is a developed country with a multi-party democracy and an established mining industry.

Any future material adverse changes in government policies or legislation in the U.S. that affect foreign ownership, mineral exploration, development or mining activities, may affect the viability and profitability of the project. Operations may be affected in varying degrees by government regulations with respect to, but not limited to, restrictions on exploration, development, mining production, export controls, income taxes, foreign investment, maintenance of claims, environmental legislation, land use, land claims of local people, water use, employment, contractor selection, and mine safety. Failure to comply strictly with applicable laws, regulations, and local practices relating to mineral right applications and tenure, could result in loss, reduction or expropriation of entitlements. The occurrence of these various factors adds uncertainties that cannot be accurately predicted and could have an adverse effect on the project's operations or profitability.

4.2.3 Landowners

There are no landowners impacted by the Project with all mineral claims residing on federal land managed by the USFS.

4.2.4 Taxation, Tenure, and Royalties

The following taxation issues are relevant for the Project:

- The ICO project is not subject to any royalties.
- An aggregated income tax rate of 26.4% is payable to the US Federal and Idaho State governments.
- A 1% Idaho State mining licence tax.
- A 1% Lemhi County real property tax.

4.2.5 Closure

The closure of the mine and mill will meet the requirements of the USFS and reduce and eliminate future reclamation and liabilities on the site. A phased approach to the reclamation is recommended to even out the Company's operating expenses. It is hoped that the post-closure operations, maintenance, and water treatment at the mine and mill will only be required for 5 to 10 years before the USFS releases the Company from any future obligations; however, there is provision for treatment to continue for 100 years as mandated by the government. In particular this relates to the requirement for post-closure water treatment.

The FS financial model includes an estimate of US$21.2M to cover mine closure and rehabilitation requirements.

4.3 CONTRACTUAL AGREEMENTS

4.3.1 Facility Inputs

Facility input agreements have been negotiated for the following infrastructure and services (telecommunications, power, water) in relation to the Project:

- Power
- Water
- Telecoms

4.3.2 Operational Agreements

Proposals have been received for the following services however no final agreements have been negotiated:

- Mining Contractor – Small Mine Development L.L.C. ("SMD")
- The Company has a Road User Permit issued by the Department of Agriculture Forest Service pursuant to Section 4 and Section 6 of the National Forest Roads and Trails Act, 16 USC. 535, and 537. This permit is valid from December 1 to November 30 and requires an application for a new permit each year to renew the use authorized by the current permit. Integral to the application is an Annual Operating Plan with respect to the level of road use in relation to the project. The current Road Use Permit is valid until November 30, 2020 and includes sufficient use to allow the early works contemplated for the ICO through to November 30, 2020.

4.3.3 Facility Outputs

No facility output agreements have been negotiated as yet regarding infrastructure and services in relation to the Project, and formal approvals still need to be sought from the relevant Government departments/agencies based on the Project scope.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following description is excerpted from the March 2015 PEA Technical Report by Samuel Engineering Inc. with minor edits and additions.

5.1 ACCESSIBILITY

Vehicle access to the ICO is via a series of well-maintained, public-access gravel roads that lead west from a point on paved Highway 93, approximately 6 miles south of Salmon, Idaho, as shown in Figure 5-1. This gravel road leads to the Blackbird Mine, which is currently not operating; however, the road is kept open year-round and a potential mining operation can operate year-round. The total driving distance from Salmon to the ICO proposed mill site is approximately 48 miles.

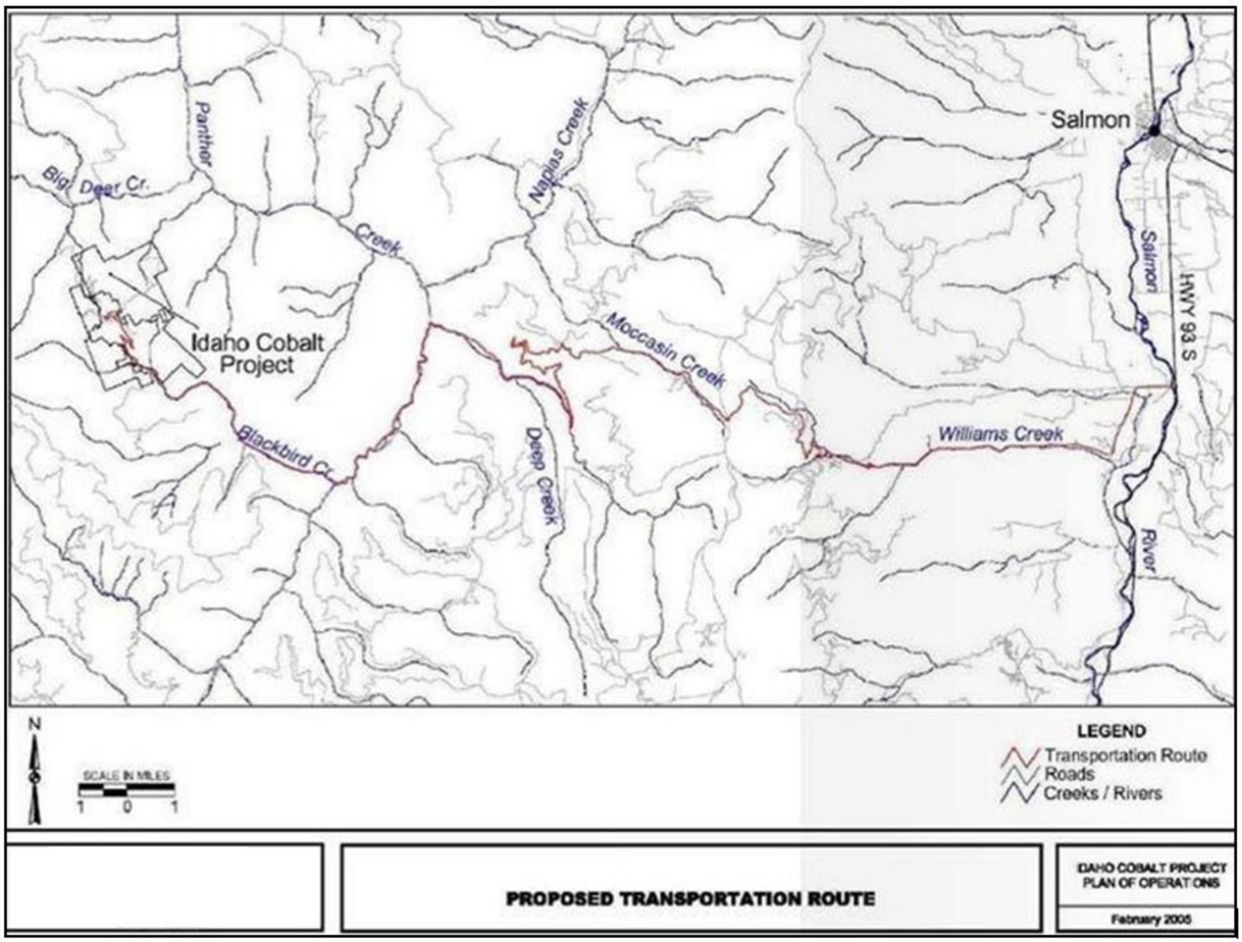


Figure 5-1: Idaho Project Site Access Roads
Source: Map supplied by FCC, 2016

5.2 CLIMATE

The Natural Resources Conservation Service ("NRCS") Morgan Creek SnoTel station is located approximately 20 air miles south-southeast of the ICO at an elevation of 7,600-feet (NRCS, 2004). Based on 12 years of data (1991-2003), the average annual temperature at the station is 34.8 degrees Fahrenheit (ºF), with a low of –34.6ºF and a high of

89.4ºF. Based on 23 years of data (1981-2004), annual precipitation is 24.4 inches. About 60 percent of the precipitation occurs as snow during the winter months (14.7 inches).

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

Salmon, Idaho, is the nearest town and is located about 26 miles east of the property. The 2000 Census reported a population of about 3,120 people (www.city-data.com, 2005). Salmon is a local supply and transportation center, with an airport paved with a 5,510 x 75-ft. airstrip at an elevation of 4,044 ft. The nearest railroad is at Dubois, a smaller town 100 miles southeast of Salmon. A 4 MW power line extends from Salmon to BMC's Blackbird Mine site.

Although Salmon currently does not provide services for mining activities, it has functioned in this manner for past mining activities at Noranda's former Blackbird mine, and at Meridian Gold's former Beartrack gold mine. Salmon has, and can again, serve as a location for personnel housing and a staging point for mine support services.

The area covered by the Idaho claims is sufficiently large to accommodate open pit and underground operations, including ancillary installations.

5.4 PHYSIOGRAPHY

The ICO is located in the Salmon River Mountains of central Idaho, within the Northern Rocky Mountain physiographic province. Major waterways in the area include the Salmon River and Panther Creek. These waterways are located in the upper reaches of the Snake River Basin, which drains to the Columbia River. The ICO is within the Panther Creek sub-basin of the Salmon River. The Project area contains flat-topped mountains and moderate to steep V-shaped canyons, and covers an area ranging in elevation from 6,100 ft. to 8,100 ft. The area that may potentially be affected by mining and mill operations is bounded by the divides of the streams that generally drain the Project area which are Bucktail Creek and Big Flat Creek. Bucktail Creek drains into the South Fork of Big Deer Creek, which drains to Big Deer Creek, which then drains to Panther Creek. Big Flat Creek drains directly into Panther Creek, which reports to the Salmon River.

The terrain in the mine area is made up of slopes approaching 35% and cut by narrow valleys. The mineralized material outcrops between elevations of 7,400 ft. and 7,800 ft., with most facilities located at 6850 feet. Soils in the area are generally comprised of sandy loam averaging 5 ft. in depth, with frequent rock outcroppings. Bedrock exposure amounts to only about 1% to 3% of the property area. Large boulder fields are found in many areas along the higher mountain ridges.

During the summer of 2000 the Clear Creek Fire burned over 200,000 acres, including the area of the ICO. The severity of the fire was high over most of the area, with all of the canopy cover and most of the litter and duff burned off. A preliminary assessment indicates that the degree of change that occurred was influenced by the various fuel loads, species, ladder fuels, canopy closures, slope and aspect components interacting with fire weather conditions at the site. As a consequence, typical mosaic patterns now prevail that are consistent with large fire behavior in this type of ecosystem. Post-fire vegetation establishment in the Project area in 2004 was variable, with vegetation cover ranging from 30% to 80% depending on slope, aspect, fire intensity and severity, soil type, and post- fire seeding.

6 HISTORY

Sections 6.1 and 6.2 of this chapter have been derived from earlier feasibility studies with some minor edits and additions.

6.1 DISCOVERY HISTORY

Section 2.2.1, Discovery History, details the regional history of the discovery of mineralization within the Idaho Cobalt Belt. This drew attraction to the northern area beyond the limits of the Blackbird mine workings and FCC pegged the claims for the prospectivity of further cobalt-copper mineralization outside of what had been historically discovered. As detailed in Section 9, Exploration, a soil sampling program in 1995 defined a surface geochemistry anomaly that would ultimately lead to the discovery of the Ram deposit in 1996 when a surface trenching exercise exposed the discovery mineralized boulder. This then led to a series of drilling campaigns detailed in Section 10 which defined the Ram deposit as it is currently modelled.

6.2 HISTORICAL STUDY AND EVALUATION WORK

As detailed in Section 2.2.1.1, Work Completed to Date, Feasibility Studies commenced in 2006 with a pre-feasibility level study. This was followed up with a more detailed study in 2007 which was refiled on SEDAR in 2008. A RoD was issued in 2009 and first construction commenced in 2010. In 2011 further studies and evaluation including engineering, procurement and construction progressed to further development including commencement of the mine portal bench. This completed the first 2 stages of construction of a proposed 3 stage plan.

A Google Earth Image from September 2013 outlines the earthworks completed to date in Figure 6-1 below.

Figure 6-1: Image of ICO Showing Mill Site and Completed Earthworks after Completion of Stages I & II Construction

The decision was made to defer Stage III construction of the ICO in May 2013. This final stage of construction was to include the commencement of the underground development of mine workings and the construction of all surface buildings including the mill, concentrator and water treatment plant. The decision to defer construction was made in response to weakened commodity prices and the enhanced adversity to risk by potential financiers in the prevailing turbulent financial and commodity markets.

Falling commodity prices also affected Formation Metal Inc.'s ability to operate its Sunshine Precious Metals refinery at a profit and in October 2013, the refinery was sold. The sale of the refinery included land adjacent to the refinery building that was originally intended to house the Cobalt Production Facility ("CPF"). As a consequence, a trade-off study was undertaken to determine the optimal location of the new CPF which is to be located along a railhead in southern Idaho. Blackfoot, Idaho Falls, and Pocatello were all considered to be potential future locations of the CPF.

Positive developments in the cobalt sector were realized in early 2014, fueled largely by expansion projects for the development of electric vehicles, grid storage and the associated projected explosive growth in the demand for rechargeable batteries requiring cobalt. In August 2014, the price of cobalt metal attained a twenty-nine-month high of US$16.00/lb. In response to these developments, in early 2014, FCC undertook a review of the cobalt chemicals utilized in this sector and determined that pursuing the viability of producing cobalt chemicals for the rechargeable battery sector was warranted. This developed into an in-house economic analysis returning positive results and by August 2014, Requests for Proposals by independent engineering firms to review the in-house engineering work was initiated and awarded to Samuel Engineering, Inc. The results of these efforts culminated in the completion of the 2015 PEA and the initiation of feasibility level metallurgical test work by Hazen Research Inc. of Golden, CO, on ICO core and rejects. This metallurgical test work was completed in Q2 2015.

A number of changes from the Technical Report were proposed, primarily at the CPF, with the goal of maximizing the economic viability of producing cobalt chemicals for the rechargeable battery sector. These changes included:

- Increase in resources by including data from the 2010 drilling.
- Inclusion of inferred material in resources in the 2015 PEA.
- Re-estimated the resource and created a block model.
- Redesign of a block mine model and mine schedule using the block model.
- Reduction of development workings on the north end of the Ram deposit where narrower, lower grade and isolated mineralization occurs.
- Attention to dilution factors by utilizing slusher mining as opposed to LHD's in smaller width stopes.
- Relocation of the CPF on a railhead.
- Scalping of copper at the CPF.
- The use of Cyanex 272 solvent extraction reagent at the CPF.
- The production of battery grade cobalt sulphate heptahydrate chemicals at the CPF resulting in the removal of numerous circuits required for high purity cobalt metal production.
- The production of copper sulphate as opposed to copper metal at the CPF.
- The use of MgO instead of lime for neutralization resulting in less residue disposal at the CPF.
- The production of saleable $MgSO_4$.
- Inclusion of gold revenues (at 85% recovery).

6.3 HISTORICAL MINERAL RESOURCE ESTIMATES

Several mineral resource estimates have been prepared for the ICO prior to the 2020 estimate of mineral resources presented herein. The reader is cautioned that these prior mineral resource estimates are being treated as historical in nature and therefore are not confirmed to be NI 43-101 compliant. They were prepared prior to the involvement of Jervois and a Qualified Person ("QP") has not verified them as current. The relevance and reliability of the estimates are not known. The 2005 and 2006 ICO estimates are classified using the categories set out in the then current versions of the Canadian Institute of Mining, Metallurgy and Petroleum's CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines as required by NI 43-101. It is not known what reporting codes were used for the earlier estimates. It should also be noted that all existing mineral resource estimates prepared prior to this report have since been superseded by the 2020 mineral resource estimate for the ICO, as described in Section 14 of this report.

6.3.1 1981 and 1997 ICO Mineral Resource Estimates

The resource estimates conducted in 1981 and 1997 pertain to the Sunshine and Sunshine East deposits and not the Ram deposit which is the subject of this Technical Report.

6.3.2 1998 ICO Mineral Resource Estimate

The 1998 mineral resource estimate was conducted by FCC for the Ram, Sunshine and East Sunshine deposits utilizing data from 92 drill holes, including historic and FCC's drill campaigns in 1995, 1996, and 1997. FCC performed the estimation by means of long-sectional polygonal methods for the various stratiform mineralized horizons in each target area. The resources are summarized in Table 6-1 at a cut-off grade of 0.20% Co.

Table 6-1: FCC's 1998 ICO Mineral Resources at 0.20% Co Cut-off

Deposit	Short Tons	%Co	%Cu	Oz Au/t
Measured & Indicated (M & I)				
Ram (I)	770,921	0.496	0.68	0.015
Sunshine (M & I)	245,554	0.965	0.47	0.022
East Sunshine (I)	100,466	0.422	0.94	0.014
Project Total (M & I)	**1,116,941**	**0.592**	**0.657**	**0.016**
Inferred				
Ram	1,722,822	0.463	0.47	0.012
Sunshine	96,830	0.624	1.29	0.027
East Sunshine	430,748	0.404	1.06	0.017
Project Total (Inferred)	**2,250,400**	**0.459**	**0.618**	**0.014**

Caution: This resource is historical. It does not conform to NI 43-101 and the CIM Definition Standards (2014); it is superseded by Jervois' resource estimate presented in Section 14 of this report.

FCC's resource estimates were independently audited by MDA in 1998, 1999, and again as part of the 2001 MDA pre-feasibility study.

6.3.3 MDA 2001 Resource Estimate

FCC conducted additional drilling on the Ram deposit in 1999 and 2000 and this drilling was included in the 2001 MDA resource estimate. Resource estimates for the Sunshine and East Sunshine deposits remained unchanged from the 1998 estimate (see Table 6-1). The updated Ram deposit resources were reported at a cut-off of 0.30% Co and are summarized in Table 6-2.

Table 6-2: MDA 2001 Ram Deposit Mineral Resource Estimate @ 0.030% Co Cut-off

Deposit	Short Tons	%Co	%Cu	Oz Au/t	Lbs Co	Lbs Cu	Oz Au
Measured & Indicated (M & I)					(000's)	(000's)	
Ram (M & I)	945,00	0.690	0.57	0.018	13,043	10,824	16,700
Inferred							
Ram	1,807,000	0.644	0.47	0.021	23,298	17,128	38,560

Caution: This resource does not follow the CIM Definition Standards (2014) and is superseded by Jervois's resource estimate presented in Section 14 of this report.

6.3.4 MDA 2005 Resource Estimate

MDA updated the ICO mineral resources during 2005. For the Ram deposit, correlation of horizons between drill holes and between cross sections were made based on a combination of lithology, structure, style of mineralization, and grade.

The 2005 resource estimates for the Ram and Sunshine deposit were based on a long-section polygonal method.

The 2005 combined Measured and Indicated Resources for the Sunshine and Ram deposits are summarized in Table 6-3.

Table 6-3: MDA 2005 Resource Estimate (Ram & Sunshine Deposits) at 0.20% Co & 0.30% Co Cut-off

Ram & Sunshine Deposits	Cut-off %	Tons	%Co	%Cu	Oz Au/t	Avg TH Ft
Measured & Indicated (M & I)	0.03	1,895,400	0.667	0.598	0.016	7.7
Measured & Indicated (M & I)	0.02	2,282,300	0.596	0.561	0.014	6.1

Caution: This resource does not follow the CIM Definition Standards (2014) and is superseded by resource estimate presented in Section 14 of this report.

6.3.5 MDA 2006 Resource Estimate

The MDA 2006 resource estimate was disclosed in the May 2008 Technical Report and Feasibility Study prepared by Samuel Engineering Inc. (Kunter and Prenn, 2008). The estimated resources are summarized in Table 6-4.

Table 6-4: MDA 2006 Resource Estimate at 0.30% Co Cut-off

Deposit	Category	Tons	%Co	%Cu	Oz Au/t	Avg TH Ft
Ram	Measured & Indicated	2,393,700	0.631	0.651	0.016	8.2
Sunshine	Measured & Indicated	260,700	0.604	0.327	0.013	3.8
Total	**Measured & Indicated**	**2,654,400**	**0.628**	**0.619**	**0.016**	**7.8**

Caution: This resource does not follow the CIM Definition Standards (2014) and is superseded by resource estimate presented in Section 14 of this report.

6.3.6 2017 and 2018 Resource Estimates

The 2017 resource estimate was produced by Micon and further updated in February 2018 to include information from 3 drill holes drilled in the latter part of 2017. These resource estimates are tabled in Section 14.13.

6.3.7 Comparison to previous Resource Estimates

The 2020 mineral resource estimate is compared to the 2017 and 2018 resource estimates in Section 14.13 showing an increased in Measured and Indicated resources and a decrease in Inferred resources. The comparison as done at the 2017 and 2018 resource cobalt cut-off grade of 0.2% Co.

6.4 HISTORICAL MINERAL RESERVES ESTIMATES

The only prior mineral reserves estimate was reported in the November 2017 feasibility study by Micon and is tabled below.

Table 6-5: 2017 Reserves (Micon)

Category	Reserve (M short tons)	Co (%)	Co cont. (M lbs.)	Cu (%)	Cu cont. (M lbs.)	Au (oz / short ton)	Au cont. (oz)
Proven (1,2)	1.99	0.43	17.1	0.69	27.4	0.013	25,276
Probable (1,2)	1.68	0.52	17.4	0.67	22.4	0.017	28,009
Total	**3.66**	**0.47**	**34.5**	**0.68**	**49.8**	**0.015**	**53,286**

In 2020 the Reserves were updated and are presented in Section 15. Comparison to the 2017 reserves is depicted below.

Table 6-6: Comparison of Current Reserves to 2017 Reserves

Category	Reserve (ΔM short tons)	Co (Δ%)	Co cont. (ΔM lbs.)	Cu (Δ%)	Cu cont. (ΔM lbs.)	Au (Δoz / short ton)	Au cont. (Δoz)
Proven (1,2)	-0.4	0.13	0.8	-0.02	-6.2	0.002	-643
Probable (1,2)	-0.52	0.01	-5.1	0.29	-0.1	0.006	-1,251
Total	**-0.91**	**0.08**	**-4.4**	**0.12**	**-6.2**	**0.004**	**-1,895**

It can be noted that while total reserves have decreased, grades have been enhanced across all metal categories. A more selective mining method with greater mining shape discretization was employed for the new update, enhancing economics by removing lower grade material from the ore feed. A finer resolution updated resource block model aided in achieving this goal.

6.5 HISTORICAL PRODUCTION

There has been no prior production from the ICO project.

7 GEOLOGICAL SETTING AND MINERALIZATION

Section 7 of this report relies heavily upon material contained in the November 2017 NI 43-101 Technical Report by Micon International Limited ("Micon") with minor edits/additions.

7.1 INTRODUCTION

The ICO is located on the east side of the central Idaho Batholith Cretaceous age granitic to granodioritic rocks, hosted in Proterozoic age sedimentary rock. The host sedimentary rocks are on the southern flank of, and perhaps were part of, a large Proterozoic age marine sedimentary basin in which dominantly clastic sediments were deposited; now these metamorphosed rocks are known as the Belt Supergroup and consist of dominantly quartzite, metagreywacke and argillite.

Unique to the Proterozoic rocks in this region are cobalt-copper ("Co-Cu") occurrences in the Proterozoic age Apple Creek Formation of east-central Idaho. The Co-Cu mineralization at the Blackbird Mine has been described as a type locality for this occurrence of stratiform Co-Cu mineralization. The ICO is located to the North of and directly adjacent to the former Co-Cu producing Blackbird Mine.

Previously identified as the middle Yellow Jacket Formation, the Apple Creek Formation was renamed based on a correlation of rocks of the Lemhi Range with the rocks of the Salmon River Range. The Apple Creek Formation includes the cobalt-bearing strata (Tysdal, 2000).

7.2 REGIONAL GEOLOGICAL SETTING

The regional geology is summarized in Figure 7-1. The ICO is situated in the Idaho Cobalt Belt ("ICB"), a 40-50 km long metallogenic district characterized by stratiform/tabular copper-cobalt deposits. The deposits are hosted by a Middle Proterozoic age, thick, dominantly clastic sequence, sandwiched between late Proterozoic quartz monzonitic intrusions. The clastic sediments were deposited in a large fault-bounded basin, probably as large submarine fan complexes and or deltas that were frequently submerged by continuing subsidence within the basin. All significant copper-cobalt deposits and occurrences are found in the Proterozoic Apple Creek Formation, which constitutes the base of this sequence. This formation was originally correlated with Pritchard Formation metasediments of the Belt supergroup to the North, its age being constrained by dates of 1.37 Ga for adamellites intruding the sequence and 1.7 Ga from mafic dykes and sills emplaced along the basin margin faults (Hughes, 1983).

The regional rift structure dominates the structure of the Apple Creek Formation. Cobalt-copper-gold mineralization occurs along a northwest-southeast trending structure parallel to and west of the central axis of the rift.

There is a series of northerly trending faults that are considered to represent initial growth faults, reactivated by Laramide and younger events. The district has also been affected by north-easterly structures of the Trans-Challis Fault Zone (Gow, 1995).

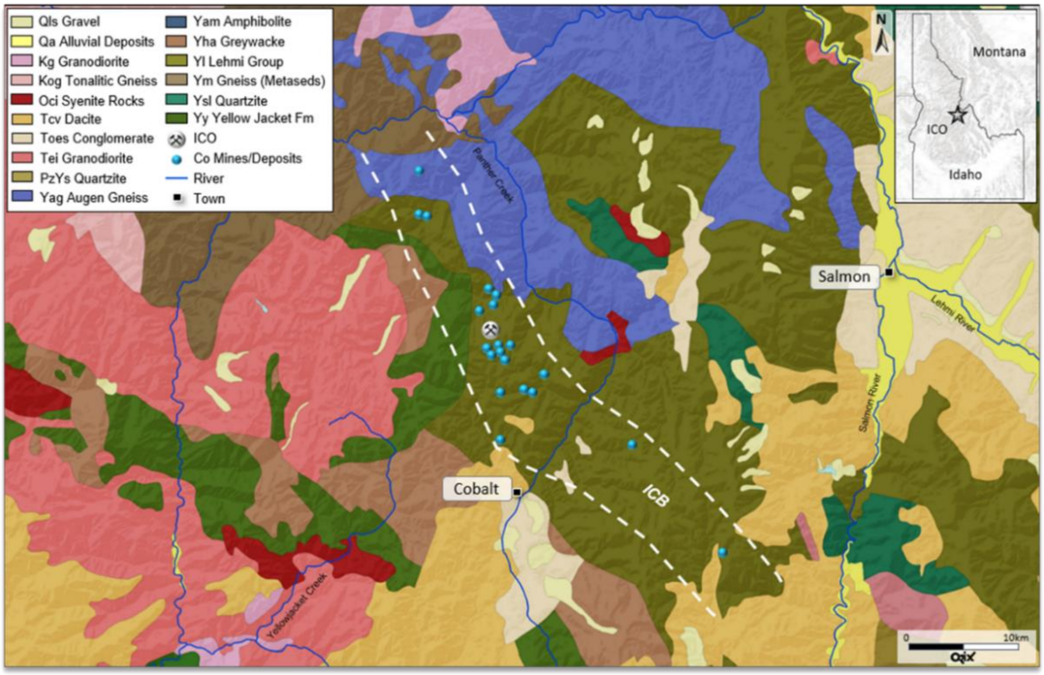


Figure 7-1: Regional Geology of the ICB
Source: IGS, 2019.

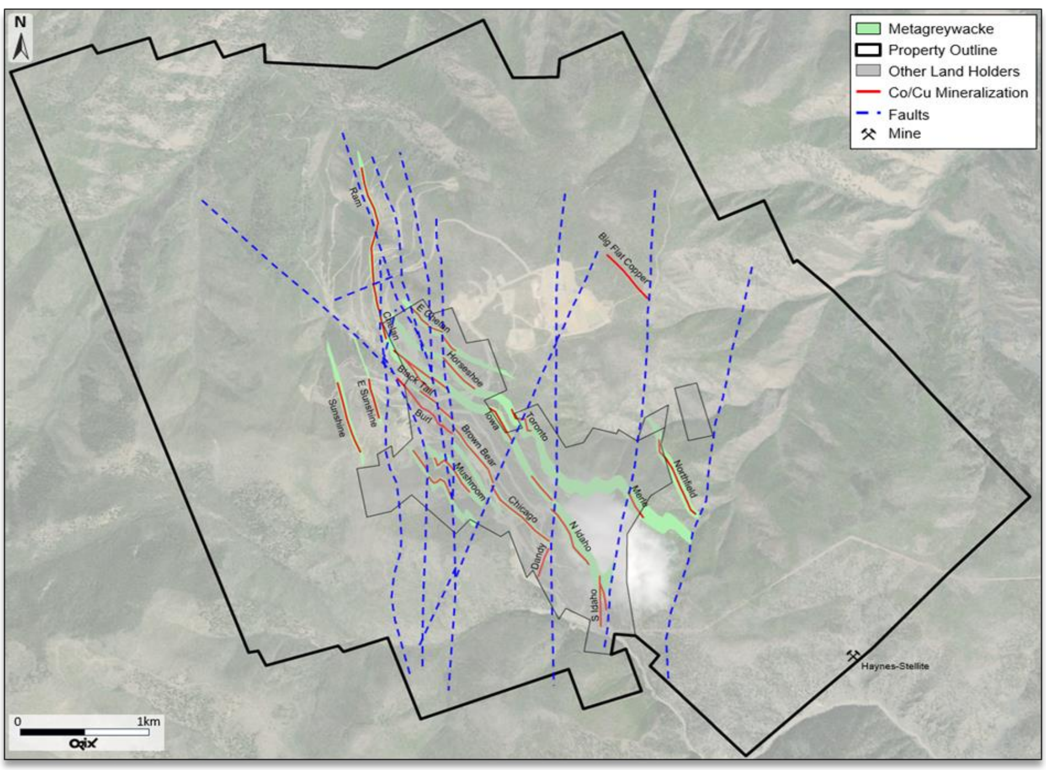


Figure 7-2: Local Geology of the ICO
Source: Map supplied by Jervois, 2016.

7.3 DEPOSIT GEOLOGY

The ICO is hosted in Proterozoic age meta-sediments found on the east side of the central Idaho Batholith comprising granitic-to-granodioritic rocks. The local geology is summarized in Figure 7-2. Most of the following geologic discussion (except where otherwise indicated) is summarized from an internal report dated April 1998 and entitled "Report on the Reserve/Resource Estimates for Sunshine Lode, East Sunshine, and Ram Prospects, Sunshine Property, Idaho, USA" by Jervois field staff.

7.3.1 Lithology and Stratigraphy

The Idaho Cobalt Belt represents a distinct region dominated by stratabound cobalt + copper + gold mineralization, with a remobilized component. The region is underlain by strata of the middle Proterozoic age Apple Creek Formation, which is an upward-thickening, upward- coarsening clastic sequence at least 15,000 m (49,000 ft) thick (Nash, 1989) that represents a major basin-filling episode (Connor, 1990) and is formerly considered part of the Yellow Jacket Formation.

Detailed work by Noranda geologists and the USGS showed that the Apple Creek could be divided into three units. The lower unit of the Apple Creek Formation is over 4,500 m (15,000 ft) thick and consists mainly of argillite and siltite, with lesser occurrences of fine-grained quartzite and carbonates. Graded bedding and planar to wavy laminae are common in the lower unit, which is locally metamorphosed to phyllite. The middle unit of the Apple Creek Formation is up to 1,100 m (3,600 ft) thick and contains several upward-coarsening sequences of argillite, siltite, metagreywacke and quartzite, with distinctive biotite-rich interbeds (Nash, 1989) that generally have a direct correlation to mineralization. The middle unit hosts the majority of the known cobalt, copper and gold occurrences in the Idaho Cobalt Belt. The upper unit exceeds 3,000 m (9,800 ft) in thickness and is predominantly composed of thin- to thick bedded, very fine- to fine-grained quartzite (Connor, 1990).

Mafic tuffs within the Apple Creek Formation are the oldest igneous rocks exposed in the Sunshine-Blackpine district. They are accompanied by felsic tuffs and carbonatitic tuffs. Some mafic dikes and sills intrude the Apple Creek Formation and may be comagmatic with the mafic tuff beds. Several small lamproitic diatremes may also be coeval with mafic volcanism (Gow, 1995).

The Apple Creek Formation has undergone varying degrees of regional metamorphism, ranging from greenschist facies in the southern part of the district to amphibolite grade facies in the northern part of the district. Several types of mafic dikes and sills, ranging from 1 to 30 m (3 ft to 100 ft) thick, intrude the Apple Creek Formation and are interpreted as feeders to the exhalative mafic tuffs, which are most abundant in areas of intrusive activity.

7.3.2 Structural Geology of the Deposits

The dominant structures in the area are steep, north- to northwest-trending normal faults and shear zones. The prominent White Ledge Shear, which displays substantial apparent strike-slip movement, marks the western extent of the mafic strata and associated stratiform mineralization in the project area (Nash, 1989).

Noranda Exploration Inc. interpreted the Sunshine stratigraphy as having been folded into a tight syncline about a northerly-plunging axis (Daggett and Baer, 1981). Small-scale fold hinges and transposed bedding visible in the Sunshine Trench indicate parasitic folding and locally severe deformation. Large-scale transposition faults roughly parallel the axial plane of the Sunshine syncline.

7.3.3 Ram Deposit Stratigraphy

The Ram deposit is one of the two deposits part of the ICO, the other being Sunshine. Stratigraphy in the Ram deposit area is predominantly medium- to fine-grained metagreywacke (previously described as quartzite) metamorphosed to upper greenschist to amphibolite facies. Stratigraphically, the Ram deposit is subdivided into three zones: Hanging wall, Main and Footwall zones, with each zone containing distinct mineralized horizons. Typical cross-sections are provided in Section 7.6 that deals with drilling results.

7.3.3.1 Hanging wall Zone

The Ram Hanging wall zone had been previously subdivided into three similar and gradational lithological packages. The upper hanging wall contains medium- to coarse-grained, locally poorly bedded to well-bedded metagreywacke. Occurrences of biotite-rich interbeds, previously identified as biotitic tuffaceous exhalite ("BTE") are generally restricted to discontinuous, irregular coarse-grained garnetiferous pods. The middle hanging wall is dominated by medium-grained, generally well-bedded metagreywacke that is locally conformably interbedded with chloritic/biotitic cobaltiferous horizons. The lower hanging wall includes medium- to coarse-grained metagreywacke with poorly defined, chaotic bedding. BTE material is restricted to sporadic, irregular interbedded lenses.

The chaotic bedding described in previous reports is interpreted here as soft sediment deformation structures with load textures, small scale slump structures, and deformed lamination, being identified during the core logging of the 2019 drill program.

The current resource model described in Section 14 contains six BTE-rich horizons that occur in between the three hanging wall zones. Each of the six has limited interpreted spatial extent, but this may be due to the soft sediment deformation obscuring the lateral continuity of such horizons.

7.3.3.2 Main Zone

The Main zone is dominated by fine- to medium-grained, thin- to medium-bedded metagreywackes that are interbedded with biotite and chlorite-rich horizons (BTE), and locally silicified lenses, previously interpreted as siliceous tuffaceous exhalites ("STE"). Mineralization in the Ram Main zone is generally found within a confined stratigraphic package containing up to three, closely spaced, stratiform BTE horizons, of variable thickness and continuity, which strike between 340° and 355° and dip between 50° and 55° to the northeast.

The main mineralized zone as described in logs (litho/alteration combination) varies in thickness between ~4.5 m (15 ft), up to 24 m (80 ft) in some areas. However, it is clear by looking at the assays, that Co mineralization is restricted to the heavily biotitic/chloritic lenses (BTE), where a single lense will rarely exceed 3 m (10 ft) in thickness, with more frequent shoulders of lower grade material.

The Main zone BTE horizons contain fine- to coarse-grained disseminations, bands, blebs, and stringers of cobaltite, chalcopyrite, and minor pyrite. This mineralization is dominantly concordant with bedding but locally has been remobilized into thin quartz veins (i.e., 'sweat veins') or crosscutting structures. The main zone represents the bulk of the potentially economic mineralization identified in the Ram deposit to date.

7.3.3.3 Footwall Zone

The Footwall zone was subdivided into two rock packages. The upper footwall is characterized by poorly to well-bedded silty quartzite to metagreywacke, often intercalated with chloritic and biotitic lenses (BTE). Frequently distorted bedding (soft sediment deformation) and a lack of BTE differentiate the lower footwall from the upper.

7.3.3.4 Faulting/Structures

During the geological modelling of the deposit in 2019, several structural features were identified from downhole drill hole intersects. A total of seven major faults and five minor splays were identified in the area of the Ram deposit. Some of these faults continue to the south and match previous structural interpretations obtained in the blackbird deposit.

In the Ram area, a north-trending vertical to steeply west-dipping normal fault is evident in drill core. The fault cuts the main zone in the south near the centre of the deposit and diverges to the North. Two other parallel and semi-vertical faults cut the main zone, showing predominantly normal to normal oblique displacements, with the strongest displacements near the south of the deposit. Two NE-SW trending semi-vertical faults occur near the southern and northern ends of the main zone with a stronger strike-slip component than the more predominant N-S faults. Minor faults or splays are structures that may not show significant displacement but do display strongly fractured and brecciated zones.

Soft sediment deformation is predominant near the centre of the deposit and it appears to affect some of the hanging wall horizons strongly. Irregular laminations and flame structures are seen in the core, likely represent a deposit scale presence of slump structures that affect the lateral continuity of some of those stratigraphic horizons.

7.3.4 Sunshine Deposit Stratigraphy

Sunshine is the second deposit part of the ICO. Stratigraphy, including the BTE horizons, strikes north northwest and dips moderately to steeply to the east-northeast. Individual sulphide-bearing beds may not be continuous over a distance of a few hundred feet. Still, generally, the overall mineralized zones within the BTE horizons can be traced along strike for over 500 m (~1,500 ft).

The description that follows is copied from the 1998 resource report that was completed by Formation Capital Corporation ("FCC"), now a subsidiary of Jervois Mining Limited:

The Sunshine Lode's Main Zone is comprised of fine- to medium-grained metaquartzite interbedded with siltite and mafic sequences. The mafic sequences, consist of green biotite and lesser chlorite, have been interpreted to be metamorphosed tuffs or exhalites (BTE) (Clark, L.A., 1995). Portions of the mafic sequence contain significant amounts of chert of exhalative origin (STE) (Clark, L.A., 1995).

The hanging wall stratigraphy is dominated by upward coarsening and thickening quartzite. In the lower hanging wall, quartzite is intercalated with local siltite and minor mafic sequences (BTE), while in the upper hanging wall quartzite contains little siltite and no mafic sequences.

The footwall stratigraphy is dominated by a thick sequence of monotonous siltite or pelite with minor interbedded sandy units. Mafic sequences are rare and cannot be correlated except locally. Shearing is prevalent within this package.

The boundary between the footwall and the main zone is defined by a sedimentary interface based on grain size, indicating a change between shallow and deeper water.

Concordant to sub-concordant discontinuous quartz veins are found throughout the Sunshine Lode's stratigraphy. These are diagenetic in origin, and while they occasionally carry grade, they are not traceable for any considerable distance along strike or down dip.

Folding, at least locally and on the bedding scale, has been noted within the (drill) core from changes in bedding attitudes and fold noses. This may lend support to the idea that the horizons are folded repetitions. However, no definitive evidence of overturning could be documented in the core.

The Sunshine Lode mineralization appears to be cut by several discontinuous, shallow to moderate, west-dipping, dip-slip faults/shears. In addition, drilling has revealed several discontinuous, crosscutting tectonic breccias, which may affect the continuity of the Sunshine Lode, at least locally.

The north-trending, steeply west-dipping, Green Dyke fault, which parallels the Sunshine Lode for much of its strike length, may truncate the mineralization down-dip and to the south. Drilling below the fault has been limited, and some of the holes may not have reached the mineralized horizons. Two Noranda drill holes, 80-03A and 80-13A, which do penetrate below the fault, intersected a core length of 2.30 ft. of 0.320% cobalt, 0.08% copper and 0.003 oz gold/ton and 4.00 ft. of 0.217% cobalt, 0.21% copper 0.003 oz gold/ton respectively. These holes suggest that higher-grade pods of mineralization may remain undiscovered below the fault. Neither a sense of movement nor a displacement has been determined for this fault.

The Sunshine Lode's mineralized zone is found within a confined stratigraphic section that contains a main mineralized horizon (1003), a lower footwall horizon (1001), and an upper hanging wall horizon (1007). Although the mineralized zone is continuous along strike, the individual horizons do not always display good continuity along strike or down dip. The footwall and hanging wall horizons attenuate rapidly both along strike and down dip. However, within the main horizon and hanging wall horizon, tabular deposits of mineralization with sufficient grade and size exist, which should be mineable. These deposits appear to have their long axis down plunge towards North.

The stratabound mineralization revealed by the drilling consists of fine- to medium-grained disseminations, blebs and stringers of cobaltite and minor chalcopyrite and pyrite. Two types of (mineralization) occur within the Sunshine Lode, fine- to coarse-grained cobaltite within siliceous gangue and fine-grained cobaltite within micaceous gangue. The micas are black biotite, green biotite and chlorite. The horizons are typically composed of both mineralized material types and are hosted by medium-grained biotite rich quartzites.

7.3.5 Alteration

The Apple Creek Formation has been subjected to varying degrees of regional metamorphism resulting in the southern part of the district displaying greenschist facies. In contrast, the northern part is dominated by amphibolite grade facies. On a broad scale, alteration related to mineralizing events is manifested by the presence of tourmaline and ankerite in the central unit that hosts the mineralized zones. However, these alteration minerals can be found up to several thousands of feet from the nearest known sulphide occurrences, and thus, do not provide any reliable indications of proximity to ore targets. In places, ankerite changes to disseminated siderite. Silicification and chloritization have been noted within the mineralized zones, but chlorite-rich rocks may be found as much as several hundreds of feet from known mineralization. Alteration at the ICO has been likened by Jervois personnel to that found at the nearby Blackbird deposits which have been described as being strata-bound and coincident with biotite and intercalated rocks with the alteration zoning consisting of pyrite-siderite-quartz-muscovite in the core zone and grading outward into quartz-muscovite- lesser pyrite.

As of late, the mineralized horizons labelled previously as BTE are here interpreted as hydrothermally altered meta-argillites, at least near the Ram deposit where no evidence of co-existing volcanism has been seen.

7.4 MINERALIZATION

Several significant stratiform/tabular cobalt-copper-gold deposits and prospects define the Idaho Cobalt Belt. As far as can be determined at this point, they are associated with two or more distinctive, regional stratigraphic horizons within the Apple Creek Formation that are distinguished by diagnostic Fe minerals. In the Blackbird area, the mineralized sequence is characterized by the presence of biotite-rich beds often referred to as "biotitic" within a sequence of up to 900 m (~3,000 ft) of interbedded metagreywacke, siltite argillite and minor quartzite. Approximately 16 km (~10 miles) to the southeast, probably within the same stratigraphic sequence, FCC in the past explored stratiform copper-cobalt mineralization at their Blackpine project.

Three types of cobalt-copper-gold occurrences have been reported in the Idaho Cobalt Belt (Nash, 1989, reported in Pegg, 1997):

- Type 1: Cobalt-copper-arsenic rich deposits of the Blackbird Mine type. Generally, these contain approximately equal amounts of cobalt and copper, with variable amounts of gold and pyrite. The dominant minerals include cobaltite (CoAsS) and chalcopyrite (CuFeS2). The cobaltite accounts for nearly all the arsenic content in these occurrences. This syngenetic and stratabound mineralization is closely associated with “mafic sequences” of the Apple Creek Formation, although such rock types have not been identified in the latest drilling campaign. The deposits are found in tabular form. Examples of these types of deposits include the Blackbird Mine and the mineralized zones found within Jervois’s Sunshine and Ram deposits.

- Type 2: Cobaltiferous-pyrite-magnetite deposits with variable chalcopyrite and low arsenic content. These occurrences are hosted by fine-grained metasediments from the lower unit of the Apple Creek Formation. Mineralization is stratabound, locally stratiform and is found within syn-sedimentary soft sediment structures. The deposits are located in the area of Iron Creek, approximately 27 km (~17 miles) southeast of the Blackbird Mine.

- Type 3: Cobaltiferous, tourmaline-cemented breccias. These are relatively common in the lower unit of the Apple Creek Formation, especially south and east of the Blackbird Mine. Only a few of these, apparently, contain more than 0.1% cobalt.

7.4.1 ICO Mineralization

Mineralization at the ICO is of Type 1 characterized as syngenetic, stratiform/tabular exhalative deposits; however, the presumably associated mafic sequences of the Apple Creek Formation have not been identified at this time. This mineralization is dominantly bedding concordant, and the deposits range from nearly massive to disseminated. Some crosscutting mineralization is present that may be in feeder zones to the stratiform mineralization or may be due to remobilization locally into fracture quartz veins and/or crosscutting structures.

Dominant minerals include cobaltite (CoAsS) and chalcopyrite ($CuFeS_2$). Other minerals present in small quantities are pyrite (FeS_2), pyrrhotite (FeS), arsenopyrite (FeAsS), linnaeite ($(Co\ Ni)_3S_4$), loellingite ($FeAs_2$), safflorite ($CoFeAs_2$), enargite (Cu_3AsS_4), and marcasite (FeS_2).

Recently, rare-earth minerals have been identified in samples from the deposit as monazite, xenotime and allanite. At this time, these minerals have not been considered for potential recovery as by-products.

The Ram deposit consists of a Hanging wall Zone with six minor somewhat discontinuous horizons, a Main Zone comprising of up to three BTE-rich horizons, and a Footwall Zone somewhat discontinuous on strike (Figure 7-7). These sub-parallel horizons generally strike N15°W and dip 50° – 60° to the northeast. Most of the significant Co mineralization is associated with biotitic-chloritic heavily altered interlayered horizons, previously described as biotite tuffaceous exhalates (“BTE”), silicified somewhat locally brecciated sections previously identified as siliceous tuffaceous exhalates (“STE”), and metagreywacke with interlayered biotitic horizons (“QTZ/BTE”) or siliceous horizons (“QTZ/STE”).

The Sunshine/East Sunshine deposit is Jervois’s second deposit within the ICO area and is located about 1 km (~0.6 miles) south of the Ram deposit. Mineralized zones are stacked sulphide-bearing beds. Individual mineralized beds or horizons are intimately associated with biotite-rich (BTE) horizons. An increase in silica content generally indicates an increase in cobalt, copper and gold grades.

7.5 DEPOSIT TYPES

7.5.1 Current Interpretation

Identification and classification of the ICO deposit as a specific type has fluctuated throughout time. Geoscientific work/observations prior to 2005 suggested a sedimentary exhalative deposit class for the ICO deposits. According to Evans et al. (1986), "These deposits are stratabound iron-, cobalt-, copper-, and arsenic-rich sulphide mineral accumulations in nearly carbonate-free argillite/siltite couplets and quartzites".

The deposits comprising the ICO belong to a class of deposits variably described as "Blackbird Co-Cu" (Evans et al., 1986) or "Blackbird Sediment-hosted Cu-Co" (Hõy, 1995). Hoy suggested the following "associated deposit types: Possibly Besshi volcanogenic massive sulphide deposits, Fe formations, base metal veins, tourmaline breccias."

However, as of 2019, the identification of volcanic or intrusive rocks in the Ram deposit has been elusive, with the only exception being, some late lamprophyre and mafic dykes cutting across stratigraphy. At this point, at least for the Ram immediate area, there doesn't seem to be evidence of coeval volcanism associated with the Mesoproterozoic synsedimentary mineralization. It is likely, however, that such source type may have played a bigger role near the south in the Blackbird deposit.

Later in 2006, Geoscientific work and observations suggested an iron oxide-copper-gold ("IOCG") deposit class with a magmatic-hydrothermal origin for the ICO deposits. The following is an excerpt from the abstract of a paper by Slack J. F. (2006).

"Analysis of 11 samples of strata-bound Co-Cu-Au ore from the Blackbird district in Idaho shows previously unknown high concentrations of rare earth elements ("REE") and Y, averaging 0.53 wt. per cent ∑REE + Y oxides. Scanning electron microscopy indicates REE and Y residence in monazite, xenotime, and allanite that form complex intergrowths with cobaltite, suggesting coeval Co and REE + Y mineralization during the Mesoproterozoic. The occurrence of high REE and Y concentrations in the Blackbird ores, together with previously documented saline-rich fluid inclusions and Cl-rich biotite, suggest that these are not volcanogenic massive sulphide or sedimentary exhalative deposits but instead are iron oxide-copper-gold (IOCG) deposits."

On the other hand, mineralogy seen in the 2019 program, as well as recorded in all previous drilling campaigns, fails to mention any tangible content of IOCG related assemblages. Therefore, making it difficult to assign such deposit type to this mineralization.

Instead, the current understanding indicates that the Ram area is a Metasedimentary rock hosted Co-Cu-Au package with strata bound zones of semi-massive sulphides. The origins of these deposits are thought to be varied; a range of mineralizing processes, from diagenetic to epigenetic are thought to be involved; however, the sources of the hydrothermal fluids and metals are still enigmatic. (Bookstrom et al. 2016).

7.6 EXPLORATION

7.6.1 Programs

7.6.1.1 1995-1996 Campaign

In 1995, soil sampling of selected areas was conducted on lines spaced ~60 m (200 ft) and ~120 m (400 ft) apart, with samples collected at intervals of ~30 m (100 ft) along the lines. This program discovered the southern end of the previously unknown Ram target.

In 1996, the soil grid was extended North, and soil samples were collected on lines spaced ~60 m (200 ft) apart with samples collected at ~8 m (25-ft) intervals along the lines. Some infill samples were collected from the 1995 soil grid. Other parts of the grid were also extended and sampled on ~8 m (25 ft) intervals where it was deemed warranted.

A total of 8,427 soil samples were collected during the 1995/1996 campaign. Geochemical contours were created for Co, Cu, As, and Au and helped to narrow and confirm the location of the RAM anomaly (Figure 7-3).

Other exploration activities conducted during 1995/1996 included surface geological mapping at a scale of 1 in to 100 ft, mapping of old trenches and prospect pits, and collection of 979 surface rock samples including those from trenches.

7.6.1.2 1997 Campaign

The Ram soil grid was extended northward, with the collection of an additional 95 soil samples; concurrently, the north and south extensions of the Ram prospect were geologically mapped.

In the same year, Jervois built ~950 m (3,100 ft) of benched drill road into the Ram zone; the road was laid out to cross the Ram soil geochemical anomaly, in order to facilitate trenching. Three trenches, ~190 m (623 ft) long in aggregate, were excavated within the "prism" of the road; the trenches were mapped, and 83 rock samples were collected. The newly opened 6,930 drift was mapped, and 163 rock samples were collected.

For a topographic base, Jervois had a five-feet contour map of the project area, produced photogrammetrically, using aerial photography.

7.6.1.3 1998-2001 Campaign

Permitting baseline studies were initiated.

7.6.1.4 2002-2006 Campaign

Various baseline studies were completed in support of project activities. The Plan of Operations ("PoO") and the United States ("USFS") Environmental Impact Statement ("EIS") were also completed. An updated PoO was submitted in April 2006.

7.6.1.5 2007-2019 Campaign

No exploration works other than drilling was carried out.

7.7 EXPLORATION RESULTS

The surface geological and geochemical work were important contributors to the discovery and expansion of the Ram deposit both in the northerly and southerly directions. While both soil and rock chip samples are not representative; they serve primarily to detect mineralization for further investigation by trenching and ultimately drilling.

7.8 DRILLING

7.8.1 Drilling Campaigns

The ICO drilling campaigns are summarized in Table 7-1. Total drilling in the Property is 224 holes for 142,358.4 ft.

As of the end of 2019, the Ram deposit has been tested with 120 diamond drill holes drilled in 1997 through 2017 by Formation Capital and drilled in 2019 by Jervois Mining totalling 79,682.9 ft. Although drilling has been intermittent over the years, there has been continuity over the campaigns.

The Ram deposit comprises several sub-parallel horizons which generally strike N15°W and dip 50°-60° to the northeast and was drill tested to depths of 1,200 ft vertically. The Main zone, which is the most extensive and laterally continuous, has been tested drill tested over 3,300 ft (~1,000 m) in strike length, and have true thicknesses that average about 20 ft. However, the main zone consists of minor layers of differentially altered and mineralized sub-horizons, most of which range between 3 to 6 ft.

Figure 7-4 shows locations of the drill collars, and the surface projection of drill-hole traces (azimuths) for the Ram deposit.

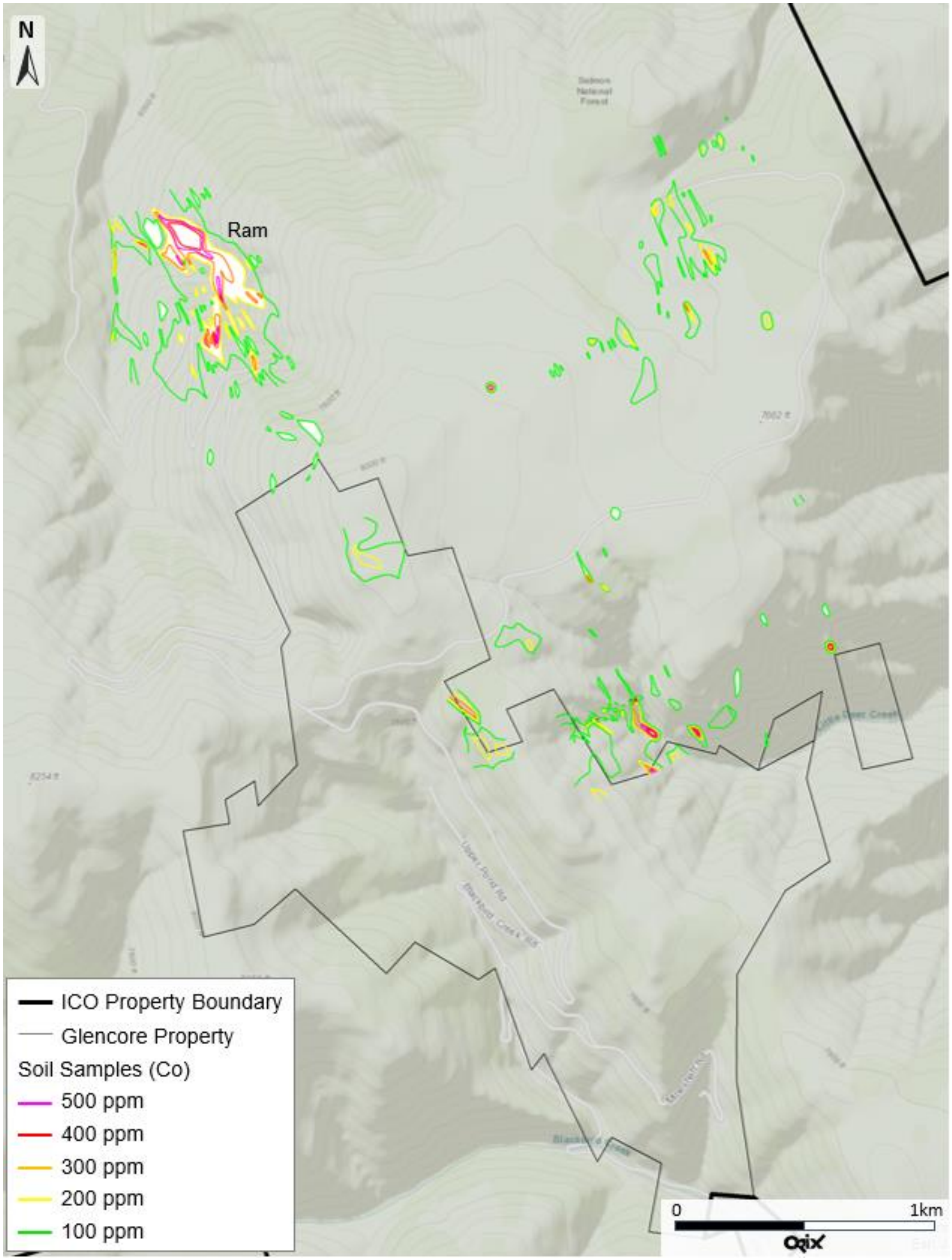


Figure 7-3: Soil Co Contours in the ICO

Table 7-1: ICO Drilling Campaigns

Year Drilled	Operator	Deposit	Number	Feet
1959	Calera Mining Company	Sunshine	3	982
1979 – 1981	Blackbird Mining Company	Sunshine	29	17,826.0
1995 – 1996	Formation Capital	Sunshine	48	29,144.0
1995 – 1996	Formation Capital	East Sunshine	24	14,723.5
	TOTAL Sunshine		**104**	**62,675.5**
1997	Formation Capital	Ram	20	12,045.0
1999	Formation Capital	Ram	11	5,210.5
2000*	Formation Capital	Ram	8	2,613.0
2004	Formation Capital	Ram	28	24,877.0
2005	Formation Capital	Ram	9	5,302.5
2006	Formation Capital	Ram	4	4,532.0
2010	Formation Capital	Ram	6	5,727.5
2016	Formation Capital	Ram	9	3,057.5
2017	Formation Capital	Ram	6	6,062.1
2019	Jervois Mining	Ram	19	10,255.8
	TOTAL Ram		**120**	**79,682.9**
Grand Total		**Ram + Sunshine**	**224**	**142,358.4**

**Metallurgical Test holes – Not used in Grade Model*

The Sunshine deposit is located about a mile (~1.6 km) due south of the Ram deposit (Figure 7-2). It consists of multiple, stacked sulphide-bearing beds of limited strike length. Individual mineralized beds or horizons range in thickness from inches to several feet and are associated with biotite-rich tuffaceous exhalative (BTE) horizons. The deposit horizons strike north- northwest and dip moderately to steeply to the east-northeast.

The resources considered in the current Technical Report are those of the Ram deposit only. The Sunshine and other deposits within the Property represent additional potential for the ICO resources. All holes drilled on the Ram deposit are diamond core holes.

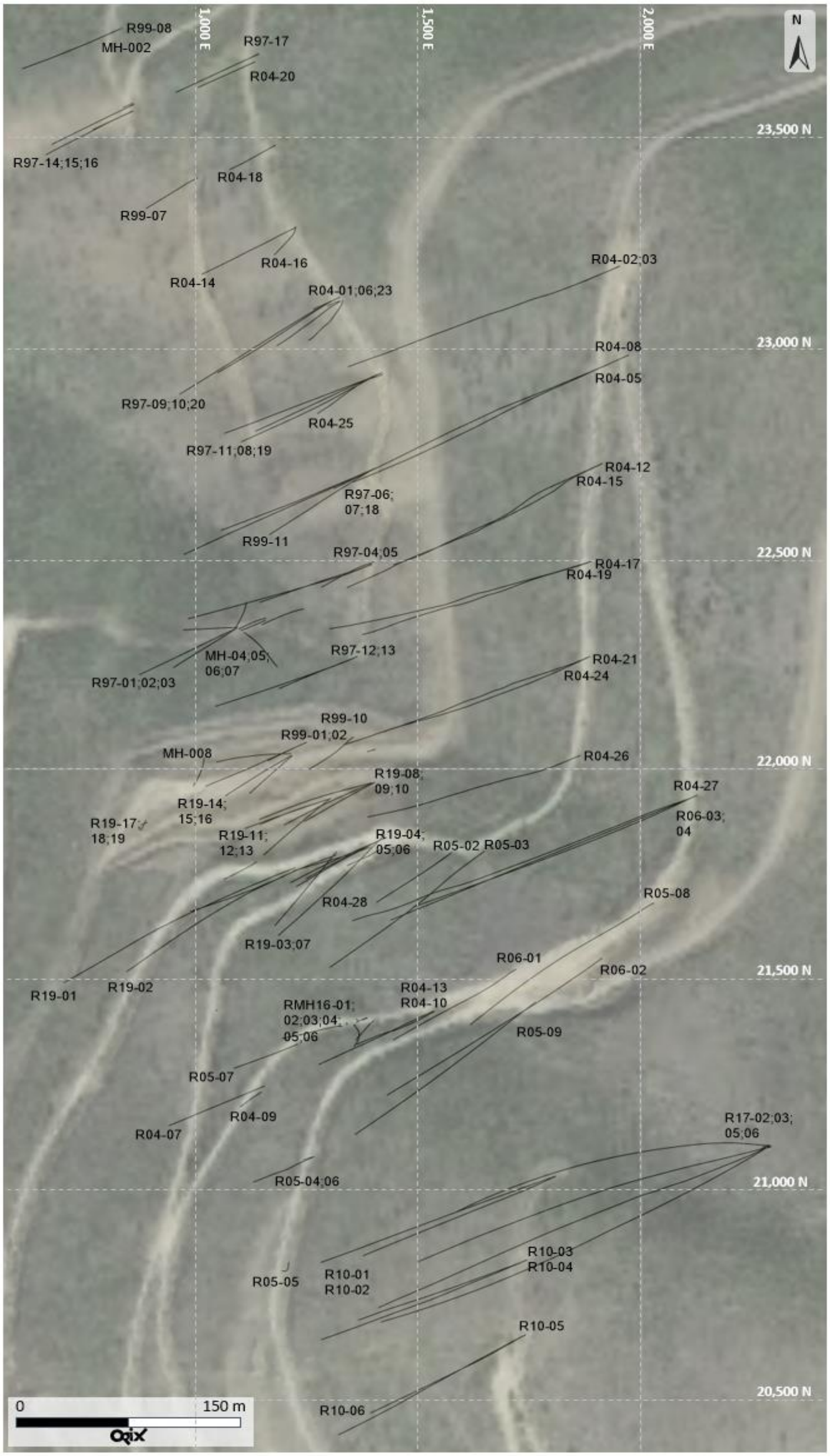


Figure 7-4: Ram Deposit Drill Hole Locations

Source: Map modified by Orix, 2020.

7.8.2 Drilling Procedures

7.8.2.1 Historic Drilling

The following description has been excerpted from the March 2015 PEA Technical Report by Samuel Engineering Inc. and is based on observations from Mining Development Associates (MDA) between 1998 to 2010. In addition, MDA also provided their expertise in the development of the first ICO Ram Block model.

All drill data was obtained by core drilling, except for reverse circulation collars for the holes completed by FCC in 2000 to obtain metallurgical samples. Exploration holes were drilled with either NQ- or HQ-size core; the metallurgical holes were drilled with PQ- size core. NQ, HQ, and PQ core have diameters of 1.875 inches (47.6 mm), 2.500 inches (63.5 mm), and 3.345 inches (85.0 mm), respectively.

FCC routinely logged the drill core in considerable detail, with particular emphasis placed on mineralized intervals.

The collars of all drill holes were located using tight chain and compass from the nearest known point. Most of the pre-1998 drill-hole collar locations were resurveyed by Harper-Leavitt Engineering Inc., using a transit (1998 report by FCC Staff). Collar locations for the 2010 drill holes were professionally surveyed by Taylor Mountain Surveying, of Salmon, Idaho, using a combination of Global Positioning Systems and conventional survey methods.

A single-shot, Sperry Sun instrument was used for down-hole surveys to check the drill-hole orientations. Down-hole surveys were done every 150 feet in the hole.

Drilling was conducted as angle holes oriented approximately normal to the strike of the mineralized horizons and crosscutting mineralized horizons at appropriate angles that allowed true thicknesses of mineralization to be determined.

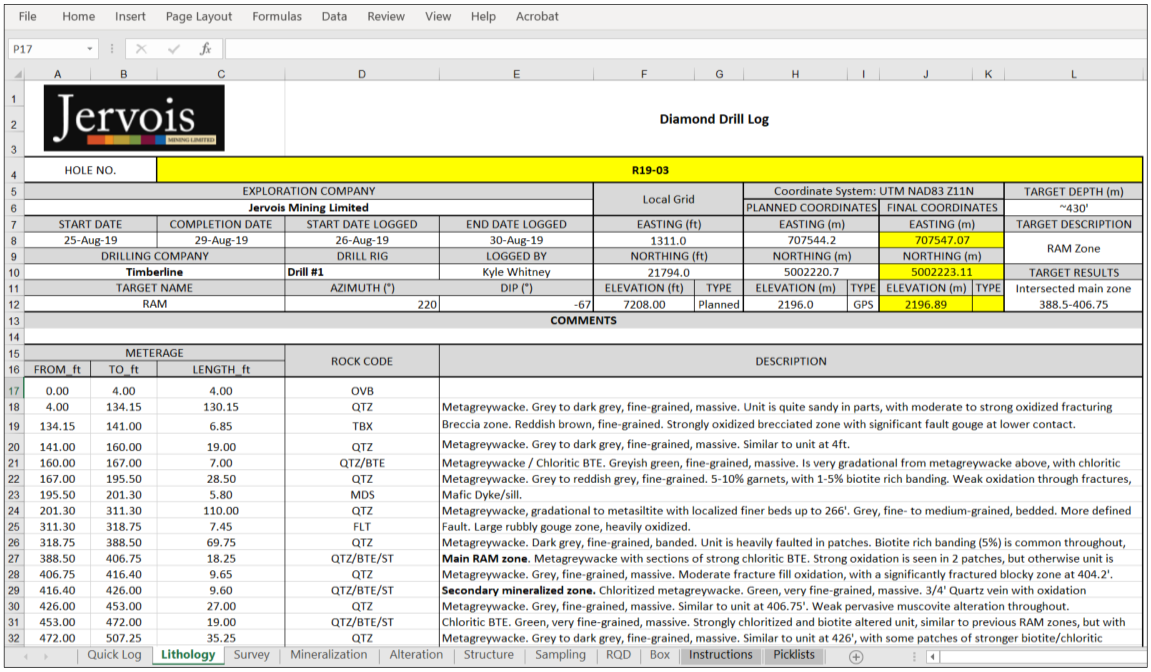
It was MDA's opinion that FCC's drilling methods used at the Ram Deposit followed industry standard procedures and were appropriate methods to adequately interpret the geology and mineralized zones used in the resource model.

7.8.2.2 Jervois Mining 2019 Drilling

In the Spring of 2019, Jervois Mining approached Orix Geoscience Inc., to support and manage the Summer 2019 diamond drill hole program. Orix created new logging templates and sent geologists for rotation work on site during July to October. All logging templates and procedures (Appendix 3-1) were previously discussed and edited with input from the senior geologist of the program, George King, as well as the Geology Group Manager David Selfe from Jervois Mining.

All drill data was obtained by diamond core drilling. Exploration holes were drilled with HQ -size core; the metallurgical holes were drilled with PQ- size core. HQ and PQ core have diameters of 2.500 inches (63.5 mm) and 3.345 inches (85.0 mm), respectively.

Orix systematically logged the drill core in considerable detail using an excel sheet with multiple tabs for different sources of information as follows: Quicklog; Detailed Lithology; Survey; Mineralization; Alteration; Structure; Assay; RQD; Box Ends (Figure 7-5).



Diamond Drill Log

HOLE NO.	R19-03						
EXPLORATION COMPANY				Local Grid	Coordinate System: UTM NAD83 Z11N		TARGET DEPTH (m)
Jervois Mining Limited					PLANNED COORDINATES	FINAL COORDINATES	~430'
START DATE	COMPLETION DATE	START DATE LOGGED	END DATE LOGGED	EASTING (ft)	EASTING (m)	EASTING (m)	TARGET DESCRIPTION
25-Aug-19	29-Aug-19	26-Aug-19	30-Aug-19	1311.0	707544.2	707547.07	RAM Zone
DRILLING COMPANY		DRILL RIG	LOGGED BY	NORTHING (ft)	NORTHING (m)	NORTHING (m)	
Timberline		Drill #1	Kyle Whitney	21794.0	5002220.7	5002223.11	TARGET RESULTS
TARGET NAME		AZIMUTH (°)	DIP (°)	ELEVATION (ft) / TYPE	ELEVATION (m) / TYPE	ELEVATION (m) / TYPE	Intersected main zone
RAM		220	-67	7208.00 / Planned	2196.0 / GPS	2196.89	388.5-406.75

COMMENTS

FROM_ft	TO_ft	LENGTH_ft	ROCK CODE	DESCRIPTION
0.00	4.00	4.00	OVB	
4.00	134.15	130.15	QTZ	Metagreywacke. Grey to dark grey, fine-grained, massive. Unit is quite sandy in parts, with moderate to strong oxidized fracturing
134.15	141.00	6.85	TBX	Breccia zone. Reddish brown, fine-grained. Strongly oxidized brecciated zone with significant fault gouge at lower contact.
141.00	160.00	19.00	QTZ	Metagreywacke. Grey to dark grey, fine-grained, massive. Similar to unit at 4ft.
160.00	167.00	7.00	QTZ/BTE	Metagreywacke / Chloritic BTE. Greyish green, fine-grained, massive. Is very gradational from metagreywacke above, with chloritic
167.00	195.50	28.50	QTZ	Metagreywacke. Grey to reddish grey, fine-grained. 5-10% garnets, with 1-5% biotite rich banding. Weak oxidation through fractures,
195.50	201.30	5.80	MDS	Mafic Dyke/sill.
201.30	311.30	110.00	QTZ	Metagreywacke, gradational to metasiltite with localized finer beds up to 266'. Grey, fine- to medium-grained, bedded. More defined
311.30	318.75	7.45	FLT	Fault. Large rubbly gouge zone, heavily oxidized.
318.75	388.50	69.75	QTZ	Metagreywacke. Dark grey, fine-grained, banded. Unit is heavily faulted in patches. Biotite rich banding (5%) is common throughout,
388.50	406.75	18.25	QTZ/BTE/ST	**Main RAM zone.** Metagreywacke with sections of strong chloritic BTE. Strong oxidation is seen in 2 patches, but otherwise unit is
406.75	416.40	9.65	QTZ	Metagreywacke. Grey, fine-grained, massive. Moderate fracture fill oxidation, with a significantly fractured blocky zone at 404.2'.
416.40	426.00	9.60	QTZ/BTE/ST	**Secondary mineralized zone.** Chloritized metagreywacke. Green, very fine-grained, massive. 3/4' Quartz vein with oxidation
426.00	453.00	27.00	QTZ	Metagreywacke. Grey, fine-grained, massive. Similar to unit at 406.75'. Weak pervasive muscovite alteration throughout.
453.00	472.00	19.00	QTZ/BTE/ST	Chloritic BTE. Green, very fine-grained, massive. Strongly chloritized and biotite altered unit, similar to previous RAM zones, but with
472.00	507.25	35.25	QTZ	Metagreywacke. Grey to dark grey, fine-grained, massive. Similar to unit at 426', with some patches of stronger biotite/chloritic

Quick Log | Lithology | Survey | Mineralization | Alteration | Structure | Sampling | RQD | Box | Instructions | Picklists

Figure 7-5: Excel Log Sample

The scale/detail of logging was predetermined and set at 1 m (~3 ft), meaning any lithological or structural feature larger than 1 m will be broken into its own separate unit, and anything lesser than 1m would have to be briefly included in the description box.

The planned collar of all drill holes was located using a handheld Garmin GPS; the rigs were then aligned by the geologist on site using a Suunto compass as well as back or front picket sights when possible. At the end of the program, all of the 2019 drill-hole collar locations were resurveyed by Wade Surveying, from Salmon, Idaho, using two types of GPS units. The final collar coordinates for the first six holes were surveyed using a Leica SR500 GPS unit, and the remaining drill holes were surveyed with a Trimble R8 Model 3 GPS Unit. Final surveyed coordinates have an expected horizontal accuracy of +/- 1 cm, and vertical accuracy of +/- 5 cm.

For every drill hole, verbal and written instructions were given to the drillers on site indicating the location, azimuth, dip and frequency of downhole surveys. Downhole surveys were taken every 50 ft using a Reflex EZ-Trac unit. For the infill drill holes R19-08 to R19-16, a Reflex TN14 Gyrocompass was placed on the rods at the start of the hole, to confirm azimuth and dip (Figure 7-6).

CSA has audited the 2019 drill program procedures, and it is their opinion that the drilling methods used at the Ram Deposit followed industry standard procedures and were appropriate to interpret the geology and mineralized zones.

Jervois MINING LIMITED		**Drillhole Summary Sheet**			
		Hole ID:	**R19-02**		
Coordinates:	5002218.6 N 707540 E UTM NAD83 Z11N	**Azimuth:**	245	**Estimated Depth (ft):**	~1350
Start Date:	11/08/2019	**Dip:**	-75	**Rig:**	**Pad:**
Reflex Instructions:		• Reflex to be taken at depths of 50, 100, 150, 200, etc.			
Geology Contact:		David Selfe: +61 439 030-921 Edwin Escarraga: +1 416-8854371			

Figure 7-6: Drill Hole Instructions Sheet and Drill Hole Survey Tools

7.8.3 Drilling Results

Drill hole logging, sampling and assay results have confirmed the following:

- The Ram deposit consists of somewhat discontinuous hanging wall zones composed of 6 main horizons, the Main zone identified in terms of a combination of lithology and alteration, and a Footwall Zone. These sub-parallel horizons generally strike N15°W and dip 50° – 60° to the northeast.
- The mineralized zones are lenticular/stratiform with most of the significant Co mineralization associated with biotite/chlorite hydrothermally altered horizons, previously identified as exhalative, i.e. biotitic tuffaceous exhalate ("BTE"), siliceous tuffaceous exhalate ("STE"), and quartzite with impregnations of biotitic tuffaceous exhalate ("QTZ/BTE") or siliceous tuffaceous exhalate ("QTZ/STE").
- True thickness of the lithological units modelled for the hanging wall units have a wide range as they occur as lenses, on the other hand, the main unit is continuous on strike length and dip and has an average thickness of about 30 ft. However, the strongly mineralized horizons occurring within this main unit, average only about 3-5 ft and range from less than 2 ft up to 13 ft.

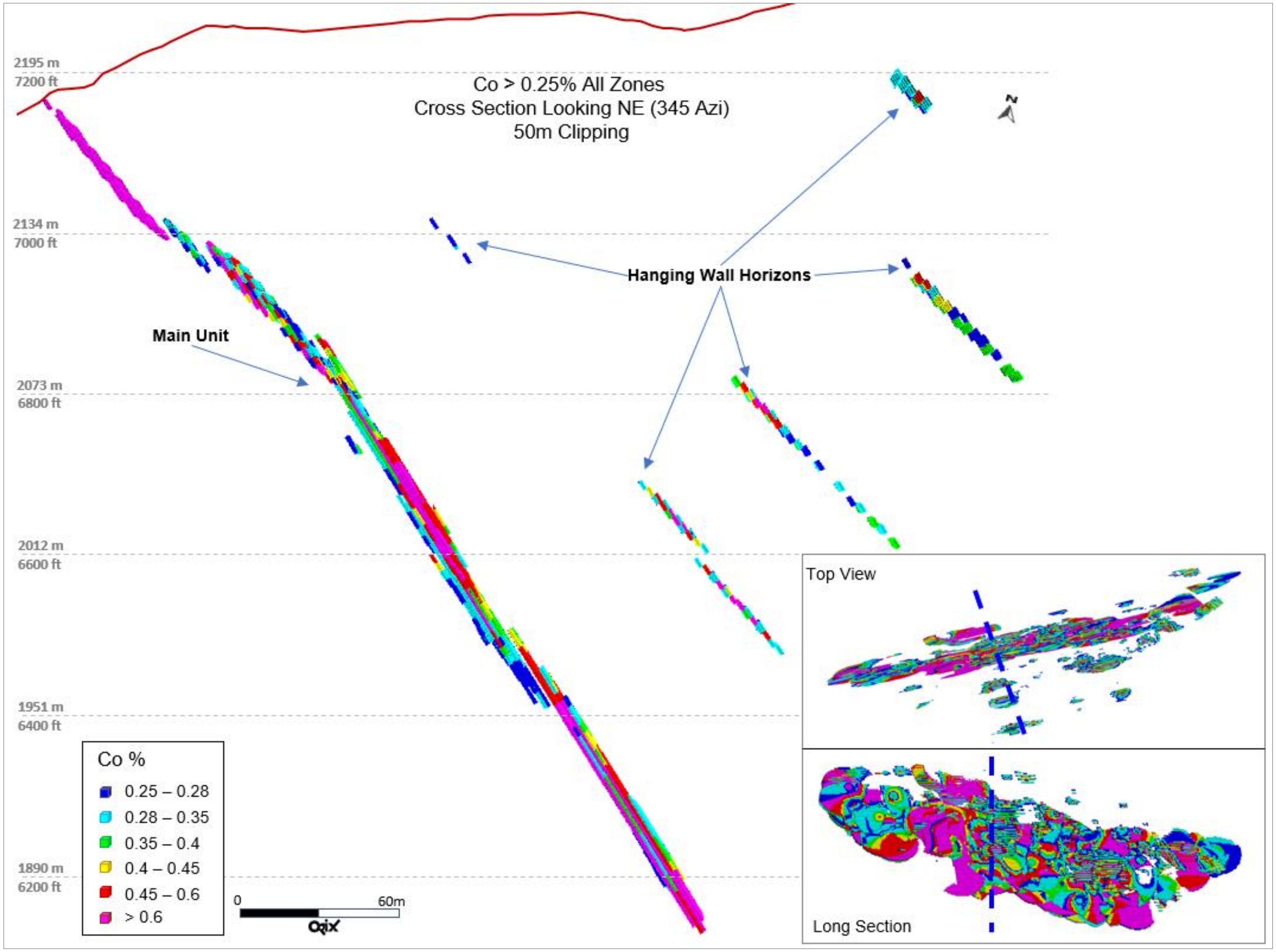


Figure 7-7: Cross Section through the Ram Deposit

7.9 SAMPLE PREPARATION, ANALYSES AND SECURITY

7.9.1 2019 Drill Program, Sample Preparation

7.9.1.1 Sample Preparation at Site

The drilling crew delivered the core at the end of each shift; the boxes were cross piled on pallets for temporary storage at the core logging building. The core was then moved to core benches to be quick logged by Senior Geologist George King with assistance from the Orix Geologist on site. Once the core was laid out on the logging tables, RQD, and footage-marks on the boxes and the core were completed using China Markers.

At this stage Orix personnel would proceed to do the detailed examination and description of the core, adding markings to relevant sections of the core, leaving for last the marking of sample intervals (Figure 7-8).

Figure 7-8: Core Logging Image
Source: Orix, 2019 Ram Drilling, R19-16

Sample lengths/intervals were defined based on lithological, alteration and mineralogical changes; an effort was made to not sample over lithological boundaries or drastic changes in mineralogy/alteration segments. Sampling lengths in 2019 ranged from 1.0 ft to 6 ft, with most samples between 2 -3 ft (average 2.8 ft). Mineralized/anomalous zones were bracketed by taking two or more samples on the margins as shoulders.

Once the logging was completed, and wet photos were taken, a hired local technician would cut the drill core selected for sampling with a diamond blade core saw, into symmetrical halves resulting in two equally representative samples. One-half of the drill core was placed in a plastic sample bag with a sample identification tag before being sealed. The other half of the drill core was returned to its original position in the core box, and the corresponding tag for each sample interval was placed at the end of the sample position in the core box. The only exception to this procedure was selected samples from the main mineralized unit in holes R19-04 and R19-06 that were submitted as whole core to SGS for Metallurgical testing.

Quality control was achieved by inserting one barren control sample (blank), two different certified reference materials ("CRMs"), and field duplicates at regular intervals into the sample stream for each batch of core samples. Blanks were inserted approximately every 40 samples or immediately after a sample suspected to run high (strong visible cobaltite mineralization). Standards were inserted approximately every 20 samples. Field duplicates occurred approximately every 60 samples outside of the main unit but in mineralization in order to test the variability of metal values. In general, the goal was to place a QA/QC sample approximately every 20th sample (Standard, Blank, Duplicate).

Other than the insertion of control samples, there is no other action taken at the site.

7.9.1.2 Laboratory Sample Preparation

Once at the laboratory, the samples are entered into the internal system. Samples are prepared by drying, if necessary, then the entire sample is crushed in its entirety to ≥70% at <2 mm, riffle split to obtain a 250 g sub-sample, which was pulverized to ≥ 85% at < 75 microns.

7.9.2 Analyses

Over the course of all drilling programs in the past, the Ram deposit has been selectively sampled and analysed by a few different laboratories. In 1996 check-sample Analysis was completed in EcoTech Laboratories Ltd. of Kamloops, British Columbia. However, sample analyses through 2006 were performed by Chemex Labs, Inc., of Sparks, Nevada, and Vancouver, British Columbia, and by Bondar Clegg Laboratories, Inc. (USA), of Reno, Nevada, and Bondar Clegg Laboratories, Inc. (Canada), of Vancouver, British Columbia.

Cobalt and copper analyses for drill samples up through the 2000 drilling were done by 4-acid (HNO3- HClO4-HF-HCl) digestion and an atomic absorption ("AA") finish; gold was analyzed by 30-gram fire assay followed by an AA finish. Cobalt and copper analyses for the 2004 through 2006 drill samples were done by aqua regia digestion and an AA finish; gold was again analyzed by 30-gram fire assay followed by an AA finish. Multi-element geochemical analyses for all drill campaigns were performed using aqua regia digestion followed by induction-coupled plasma atomic-emission spectrometry (ICO-AES). These are all industry standard analytical techniques appropriate for the types of rocks and mineralization at the ICO.

Chemex Labs, Inc., which became ALS Chemex and subsequently ALS Global, holds ISO 9002:1994 certification at its North American and Peruvian laboratories and ISO 9001:2000 certification in North America. ALS Global is the successor to Chemex and Bondar Clegg, the laboratories that did most of FCC`s analyses. Neither Micon nor MDA has determined the date that ALS Global or its predecessors first obtained ISO 9002 certification, but it is probable that much of the work for FCC was done before that date.

Sample analyses in 2010 were performed by ALS Minerals, a division of ALS Global. Analytical techniques similar to those used prior to 2010 were employed, including aqua regia digestion and AA or ICO-AES finish for cobalt and copper, and 30-gram fire assay with AA finish for gold. Multi-element geochemical analyses were performed using lithium metaborate fusion, acid digest and ICO- AES-Mass Spectrometry. Duplicate samples for verification purposes were analyzed at ACT Labs of Ontario, Canada and were analyzed for cobalt and copper by sodium peroxide fusion and ICO-AES finish, and for gold by 30-gram fire assay with AA finish.

For the 2019 Drill Program, Jervois submitted samples to two different Labs. Regular assay samples were submitted to ALS in Reno Nevada, and SGS in Lakefield, Canada. Assays included cobalt, copper and gold as part of their routine analytical procedure. In addition, multi-element geochemical analyses were completed on all the samples submitted using aqua regia digestion and AA or ICO-AES finish. The set of samples submitted to SGS were then kept for further metallurgical analysis.

All the laboratories involved in the analyses of samples are independent of the issuer.

7.9.3 Security

All activities pertaining to data collection, i.e. sampling, insertion of control samples, packaging and transportation, were/are conducted under the direct supervision of the project manager.

Jervois's core and sample security measures were typical for exploration projects in North America at the time the work was done. All historical core was received at the drill by the geologist on site and taken to the company's facility in Salmon for storage after logging and sampling were completed. For the 2019 drill program, the core was kept on site, a portion of the core is cross piled on wooden pallets inside the logging facility, and the remaining portion is stored in locked sea can containers.

That facility is a warehouse-like building with lockable doors. The sawed core was placed in labelled sample bags that were closed with plastic zip ties (Figure 7-9). During the 2019 program, Shana Hilton, the Administration Manager, prepared the samples for shipment and took them to "Bri-Easy Shipping" in Salmon. The samples were weighed, labelled and picked up by FedEx who notifies Jervois personnel when sample batches have been delivered to the laboratory. Thus, the core has been under Jervois's control from receipt at the drill, and the parts of core not used for the analytical samples remained under Jervois's control.

Figure 7-9: Jervois Core Storage and Core Sampling
Source: Photos taken by Orix, 2019.

7.9.4 Quality Control/Assurance (QA/QC)

7.9.4.1 Historic Core MDA Verification

MDA examined Jervois's data related to QA/QC in 1998 and established that the assays of the check samples, blanks and standards were in good agreement with the expected values. MDA also examined the 1999 Ram drilling QA/QC and a further check on assay QA/QC data was completed in 2004. MDA's conclusion was "Overall, Jervois has demonstrated diligence in monitoring check assays and standards and blanks results, which is critical to the maintenance of an accurate database". In addition to these checks, MDA independently selected ten samples from the 2005-2006 drilling program and sent them to ACME laboratories for check assaying from which they obtained a good agreement between the original assays and the check assays.

7.9.4.2 Micon Verification 2017

Micon noted that FCC used both blanks and standards in its QA/QC protocols but did not compile control charts. A blank sample was inserted in the sample batch sequence immediately after a highly mineralized sample expected to return high values of cobalt and/or copper. A standard or certified reference material ("CRM") was inserted at the rate of 1 in every 20 samples. Warning limits were set at +/-2 standard deviations, and control limits were set at +/-3 standard deviations. When a quality control sample fell outside the control limits, the cause was thoroughly investigated, and if need be, the entire sample batch was automatically re-assayed, and all the initial test results are rejected.

7.9.4.3 Blanks

FCC used a barren Apple Creek meta-siltite as a blank to monitor and control contamination between samples. The assay was considered a failure if the value was higher than three times the detection limit (DL). A control chart incorporating cobalt, copper and gold can be seen in Figure 7-10. Except for only two samples, the control chart demonstrates that there was no contamination between samples; if any, then it was insignificant. It has been suggested that the failures indicated in Figure 7-10 are most likely due to typo errors.

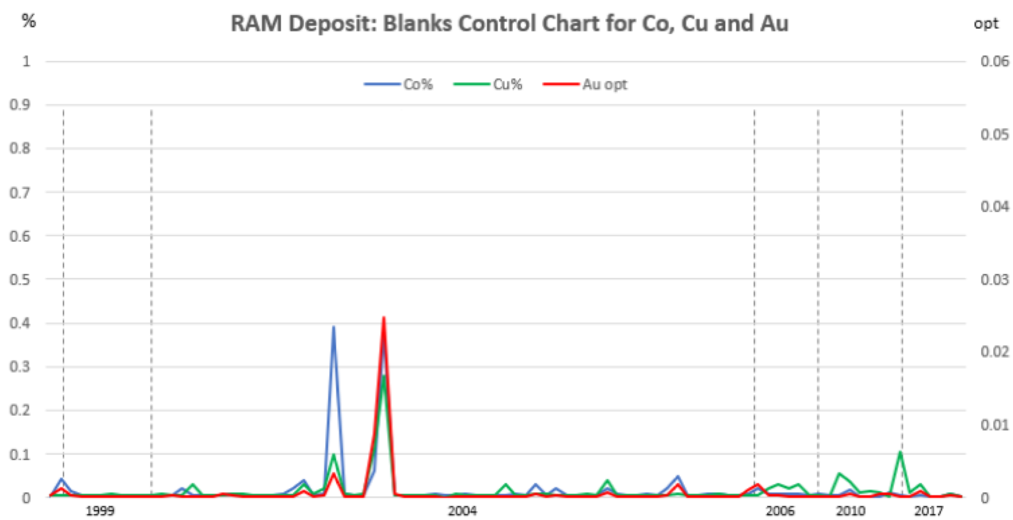


Figure 7-10: Summary of Blank Samples Results: 1997 to 2017 Drilling

7.9.4.4 Standards/CRMs

In the past, FCC used three varieties of CRMs, i.e. low grade, medium grade and high grade. Although the use of the higher standard was never consistent, all CRMS were prepared at Chemex Laboratories Inc. from mineralized material obtained from the ICO area. Given that throughout the years of 1996 – 2017, different methods of analysis were used, it was decided that for control charts of the historical (pre-2019 data), a new calculation of standard deviations and means needed to be created, as some of the original data was unfortunately lost and true certificates of some elements were not found. The values summarized in Table 7-2 below are based on the averages of QA/QC assays obtained from all drill hole programs.

Table 7-2: Summary of Certified Values from Standards used at the ICO

Item	Cobalt %	Copper %	Gold oz/t
Standard 1	0.834	0.071	0.061
Standard 2	0.254	0.034	0.030

Any results falling outside the failure limit of +/-3 SD (standard deviation) were rejected pending an investigation into the source of error. Jervois's general practice was to use standards 1 and 2. Orix has summarized the QA/QC results by compiling control charts for the drilling periods from 1997 to 2017 for standards 1 and 2 (see Figure 7-11 to Figure 7-16).

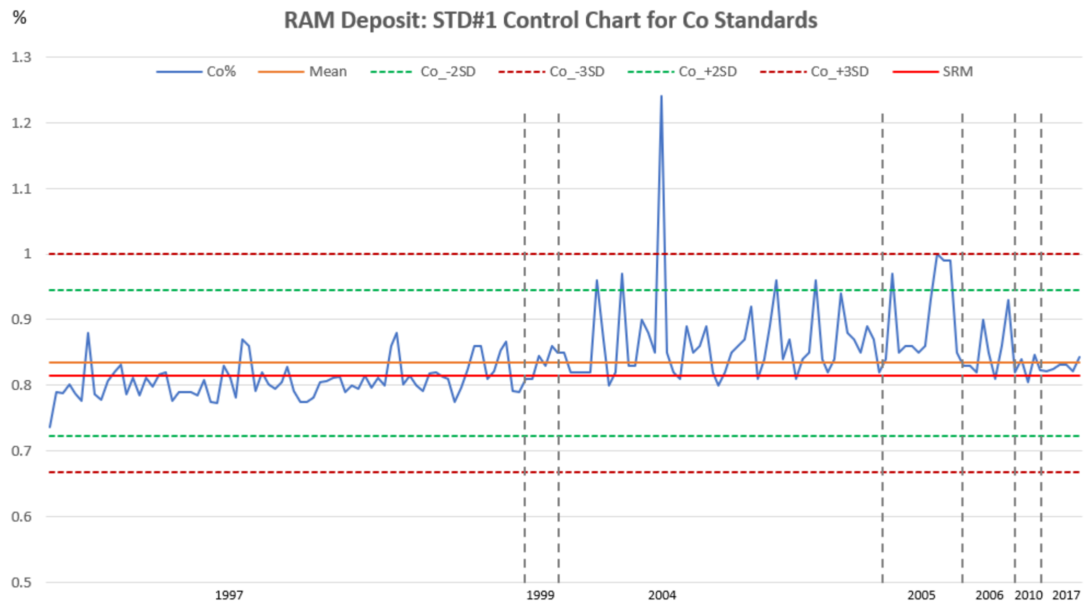


Figure 7-11: Control Chart for Co: Standard 1

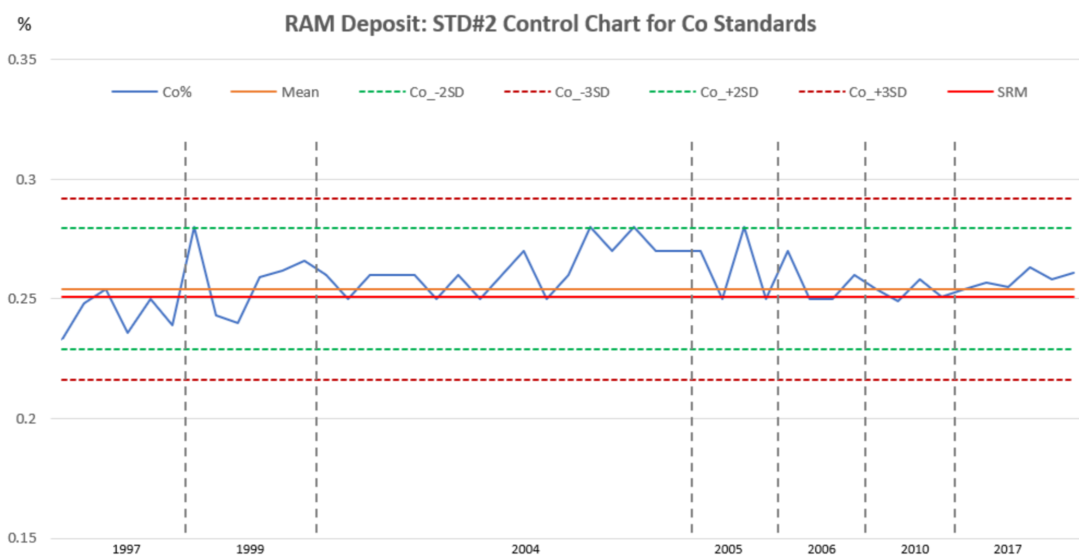


Figure 7-12: Control Chart for Co: Standard 2

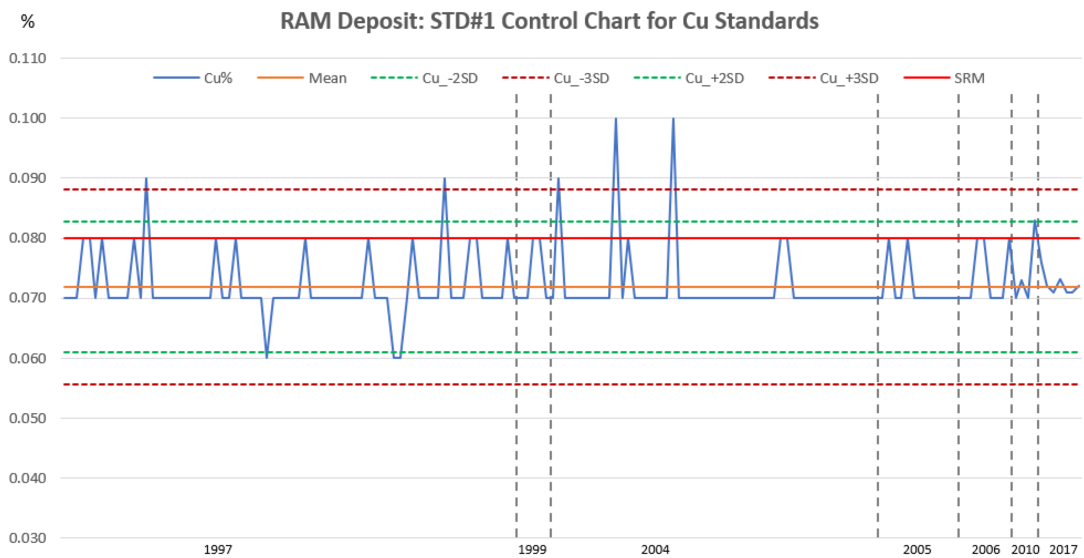


Figure 7-13: Control Chart for Cu: Standard 1

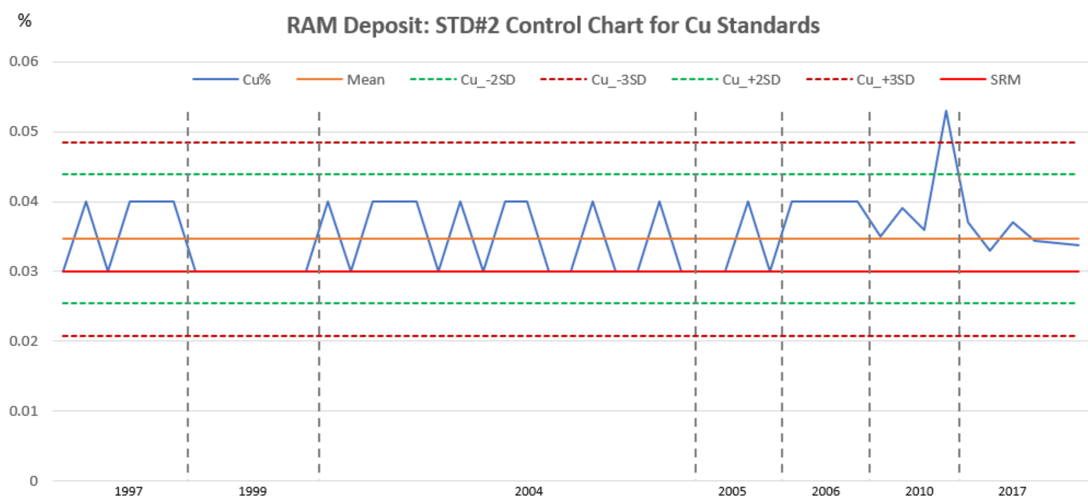


Figure 7-14: Control Chart for Cu: Standard 2

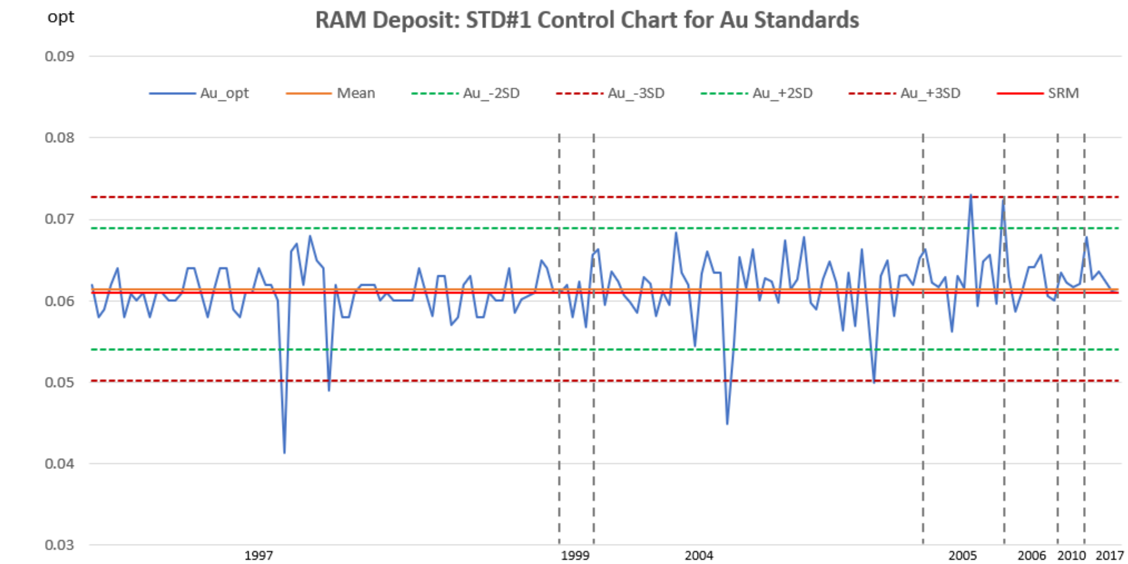


Figure 7-15: Control Chart for Au: Standard 1

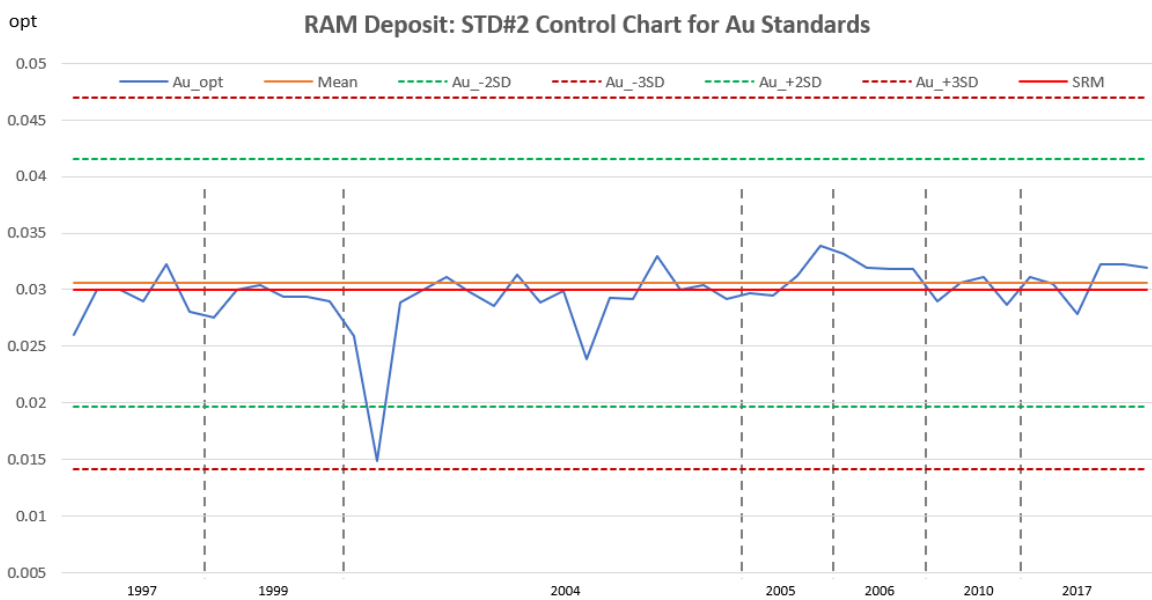


Figure 7-16: Control Chart for Au: Standard 2

For Standard 1, Figure 7-11 shows only one significant failure plus three borderline failures for Cobalt; Figure 7-13 shows five failures for copper, three of them during the 2004 program and Figure 7-15 shows three failures for gold. For Standard 2, there is only one failure (Figure 7-14) for copper. Difference in analytical methods likely contribute to the wide variation as well as some of the failure. Overall, however, the number of failures/borderline failures is insignificant to have a material impact on the assay database.

7.9.4.5 Orix Verification 2019

During the 2019 drill program, Orix's personnel used blanks, standards and field duplicates in its QA/QC protocols. A blank sample was inserted in the sample batch sequence immediately after a highly mineralized sample expected to return high values of cobalt and/or copper. Two different standards or certified reference material ("CRM") were used, such alternating standard was inserted approximately every 20 samples. Warning limits were set at +/-2 standard deviations, and control limits were set at +/-3 standard deviations. When a quality control sample fell outside the control limits, the cause was thoroughly investigated, and if need be, the entire sample batch was automatically re-assayed, and all the initial test results are rejected.

7.9.4.6 Blanks

Jervois used barren construction brick as a blank to monitor and control contamination between samples. The assay was considered a failure if the value was higher than half the average Co background of the assays in the area. Figure 7-17 incorporates cobalt, copper and gold. Except for only one sample, the control chart demonstrates that there was no contamination between samples; if any, then it was insignificant. It has been suggested and is strongly recommended by Orix that Jervois stop the use of brick material as a blank and incorporate the use of certified blank silica material for any upcoming programs, in order to obtain better consistency. In addition to this, the laboratory performs its internal blanks which are reported and seen by the user via their webtrieve sites.

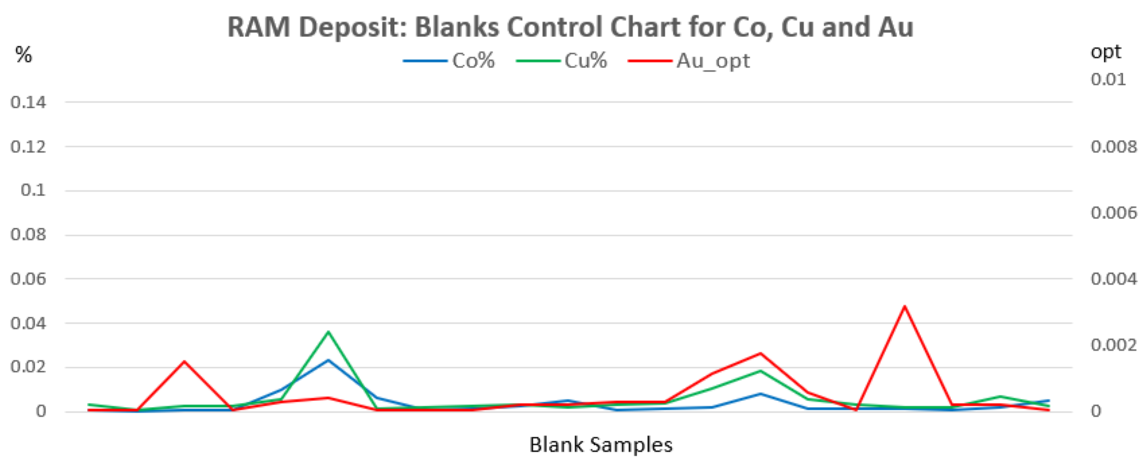


Figure 7-17: 2019 Control Charts: Blanks

7.9.4.7 Standards/CRMs

The 2019 drill program used two varieties of CRMs. The certified values summarized in Table 7-3 below are based on the averages of assays obtained from several different reputable laboratories using mineralized material obtained from the ICO area.

Table 7-3: Summary of Certified Values for Standards used during 2019 at the Ram Deposit

	Standard	Mean	SD	H2SD	L2SD	H3SD	L3SD
Co (%)	1	0.814	0.013	0.84	0.788	0.853	0.775
Cu (%)	1	0.08	0.01	0.10	0.06	0.11	0.05
Au (oz/t)	1	0.061	0.003	0.067	0.055	0.070	0.052
	Standard	Mean	SD	H2SD	L2SD	H3SD	L3SD
Co (%)	2	0.251	0.025	0.301	0.201	0.326	0.176
Cu (%)	2	0.03	0.01	0.05	0.01	0.06	0.00
Au (oz/t)	2	0.030	0.002	0.034	0.026	0.036	0.024

Any results falling outside the failure limit of +/-3 SD (standard deviation) were rejected pending an investigation into the source of error. The 2019 QA/QC results are summarized in Figure 7-18 to Figure 7-23.

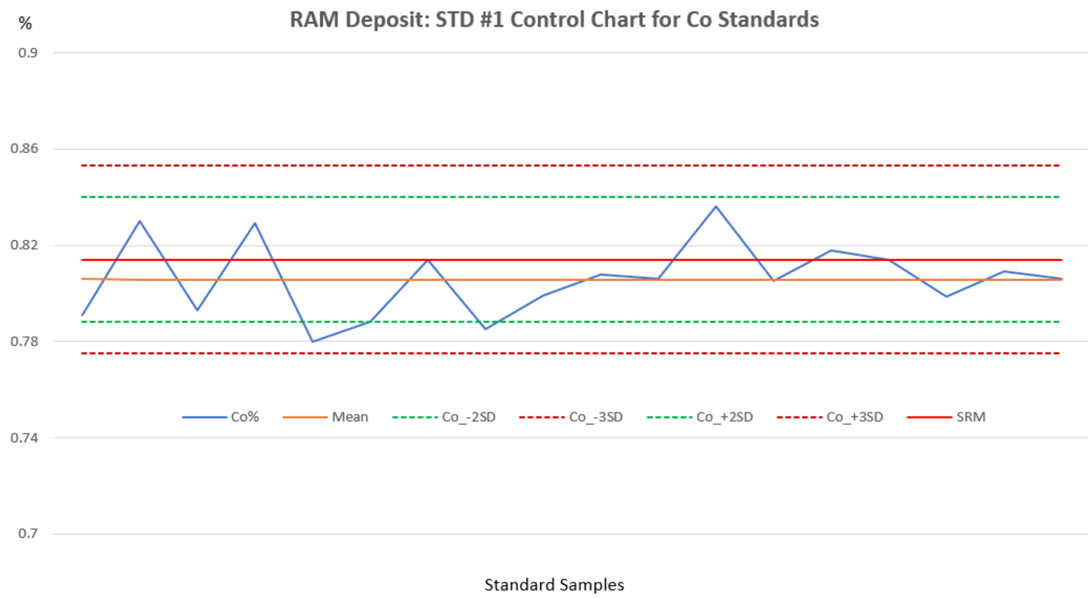


Figure 7-18: 2019 Control Chart for Co: Standard 1

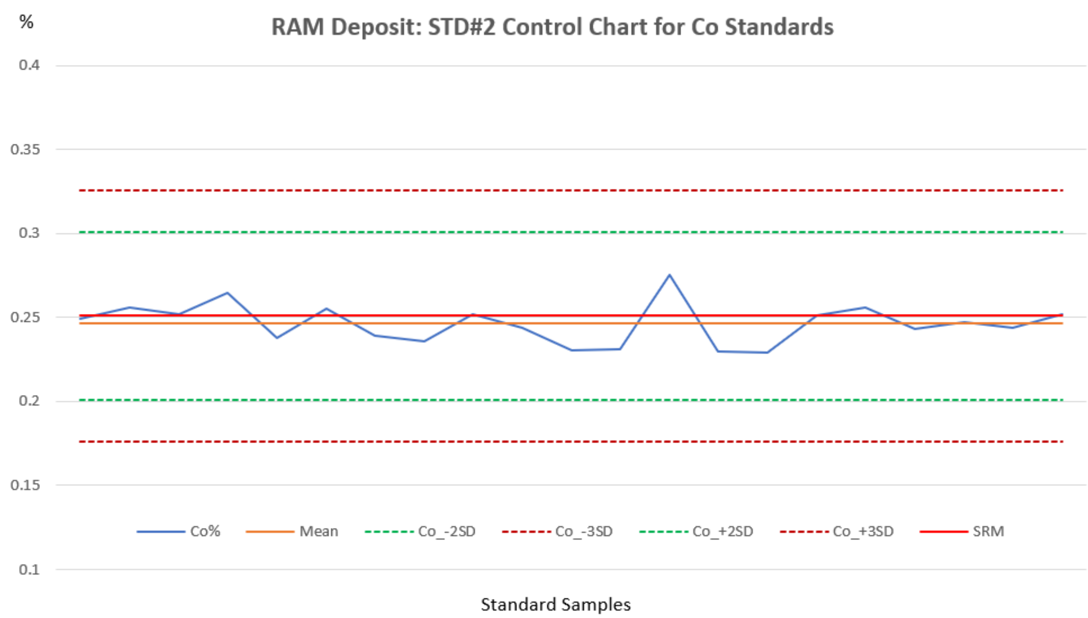


Figure 7-19: 2019 Control Chart for Co: Standard 2

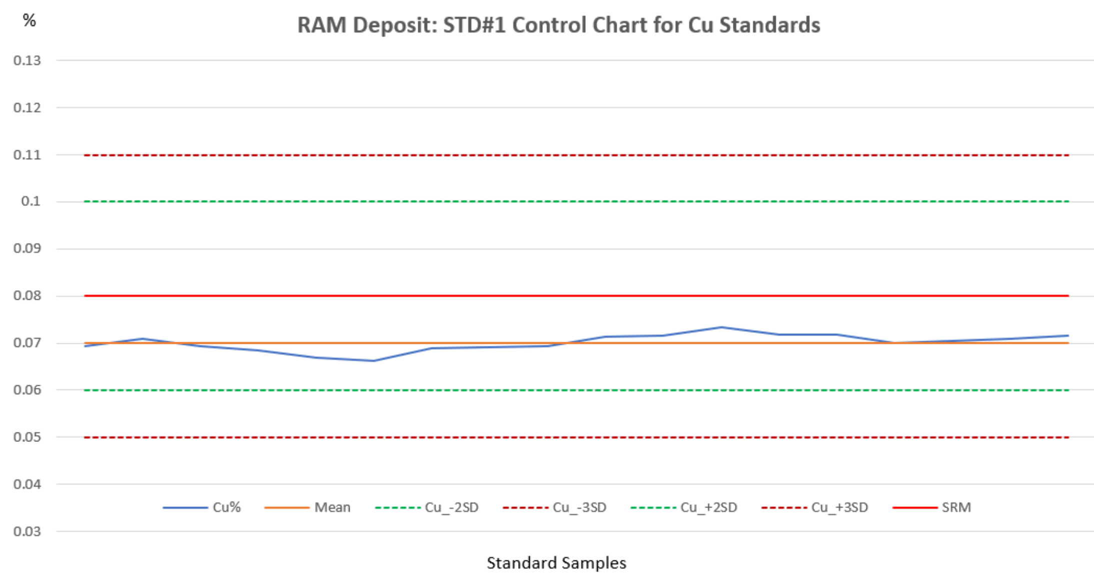


Figure 7-20: 2019 Control Chart for Cu: Standard 1

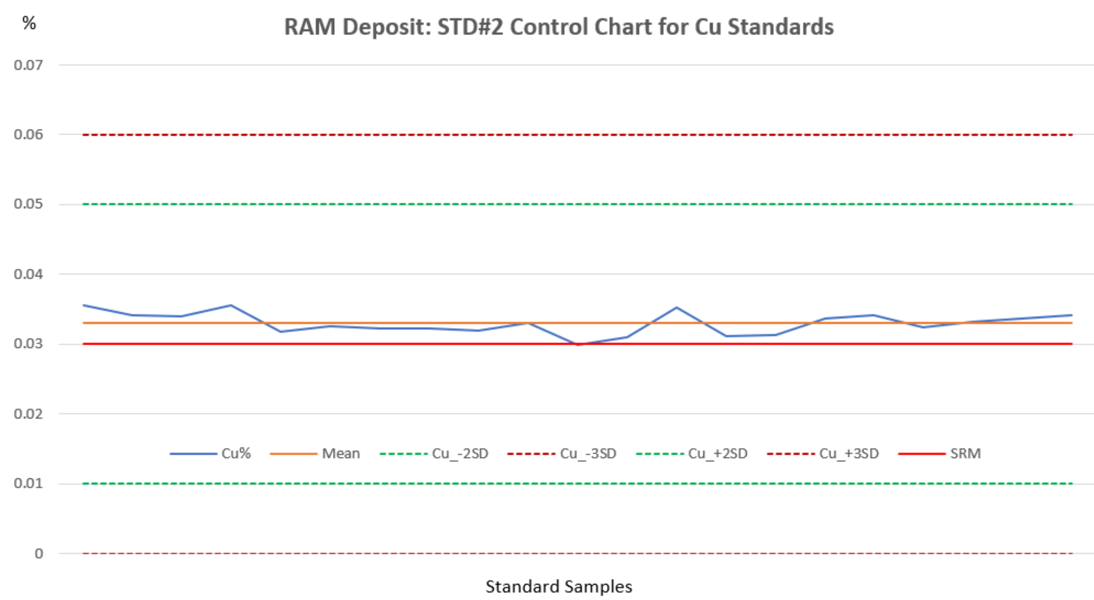


Figure 7-21: 2019 Control Chart for Cu: Standard 2

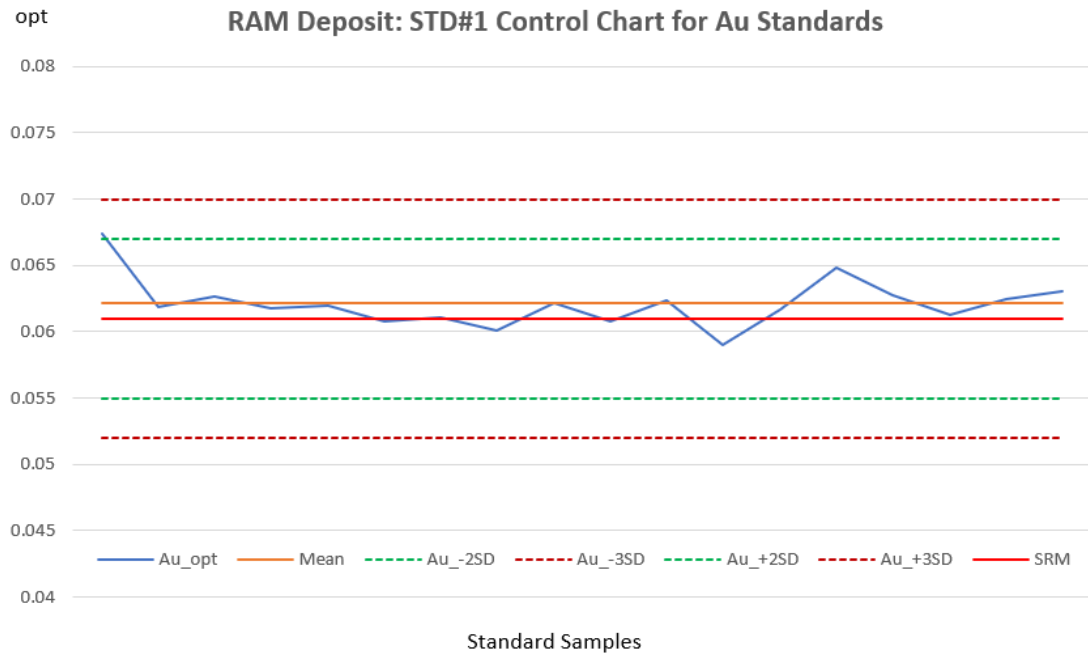


Figure 7-22: 2019 Control Chart for Au: Standard 1

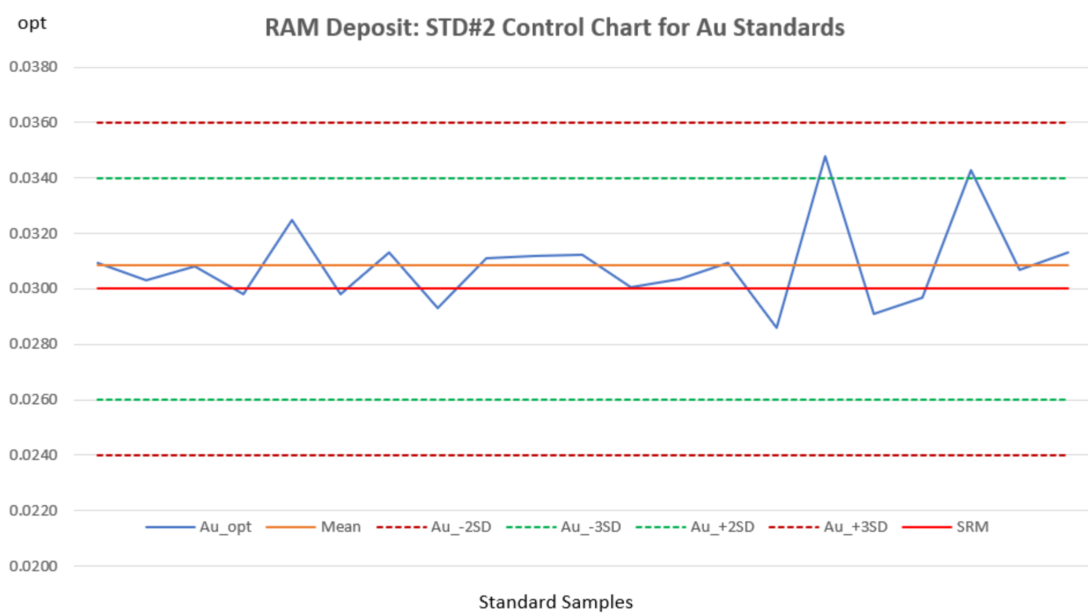


Figure 7-23: 2019 Control Chart for Au: Standard 2

7.9.4.8 Field Duplicates

Field duplicates, or quarter core samples, were taken on mineralized material outside of the main zone (Hanging wall). A relatively high variation in Co values Table 7-4 was seen between original and duplicate values. Such difference can be explained by looking at the texture in which cobaltite occurs in the rock. Cobaltite mineralization was seen as stringers, disseminated and spotty patches. Given the different textures and the uneven distribution of the mineral on any given surface, some variation is expected and explained in terms of a "nugget" effect. Therefore, for duplicate samples, it was considered a failure if the difference between samples was higher than 100%. None of the samples collected exceeded this limit.

Cu values had lesser variations, thus confirming the idea of Cu belonging to a second and much later mineralization event not associated with the Co mineralization, and as the samples were primarily picked in terms of looking for cobaltite mineralization, a small range in Cu values was expected.

Table 7-4: Summary of Duplicate Samples

Parent Sample	Dup Sample	Co_ppm	Co_ppm_DUP	Cu_ppm	Cu_ppm_DUP	Co_%diff	Cu_% diff	COA
14202	14203	317	322	1210	1450	1.56	18.05	EL19235258
14330	14331	7740	3520	37	31	74.96	17.65	EL19260563
14473	14474	507	261	418	515	64.06	20.79	EL19269269
14603	14604	177	183	742	790	3.33	6.27	EL19269269
14620	14621	725	577	1055	1025	22.73	2.88	EL19274215
14458	14459	114	116	642	629	1.74	2.05	EL19275112
14660	14661	152	130	1440	1340	15.6	7.19	EL19281890

7.9.5 Summary Statement/Comments

Orix and CSA auditors consider the sample preparation, security and analytical procedures to have been adequate to ensure the integrity and credibility of the analytical results used in the mineral resource estimation. Orix believes that the QA/QC aspects of the project have been adequately addressed.

7.10 DATA VERIFICATION

Prior to 2019, all drilling on the Ram deposit had been conducted by FCC, and thus, protocols pertaining to the exploration history of the deposit had progressive continuity. During 2019, Jervois retained the assistance of the resident geologist George King, who had been involved in all the historical drilling campaigns, in order to preserve continuity and consistency.

During the summer of 2019, Orix analyzed the state of the historic drill hole database in order to support and plan the drill program. During this analysis, several validation checks were performed outlining missing data from late drill programs. Missing data was entered, compiled, and a new current database created.

7.10.1 Discussions on Geological Attributes

Discussions held with Jervois's resident geologist centred on the geological attributes of the Ram-Sunshine-Blackbird deposits including the genetic model, mineralization trends, and the role of structures lithology and alteration, lead to the current interpretation and understanding of the deposit with the following attributes for Ram specifically:

- Continuity of the mineralization in the main zone is distinctly stratiform, less lenticular.
- Continuity of mineralization in the hanging walls is not as clear likely due to a combination of lenticular ore zones that pinch in and out, as well as soft-sediment deformation highlighted by likely large slump features.
- Copper related mineralization is visibly a later event in the form of sulphide-rich veinlets/stockworks that cut-cross earlier Cobaltite mineralization.
- The strong association of heavily altered chlorite/biotite horizons ("BTE") of the Apple Creek Formation with mineralization. BTE was previously interpreted as associated with mafic sequences, however, during the drill program no such rocks were identified, and the current interpretation suggests these horizons correspond to heavily hydrothermally altered meta-argillites.
- One of the main challenges in the interpretation of the RAM geological model is the combination of "Lithology/ Alteration" in the logs. The combination of lithology/alteration provided a general sedimentary package that

defined areas well mineralized. But the combination of lithology/alteration does not define individual mineralized lenses.

7.10.2 Data Collection Techniques/Sampling

The logging geologist photographed drill core after logging to include relevant distance markers, lithological contacts, alteration phases, strong mineralization, and sample intervals selected for assay analysis (Figure 7-8).

Drill hole log templates in excel were produced before the start of the 2019 drill program by Orix, and further edited/modified versions were created with input from the resident geologist George King and Jervois' Geology group manager David Selfe. The log sheets are very detailed and include separate tables such as Quicklog, Survey, Lithology, Mineralization, Alteration, Structure, Sampling, RQD and Box ends (Figure 7-5).

Sample intervals varied from 1 ft. to 6 ft., with the vast majority being 2 and 3 ft. long. Samples were determined based on geologic, mineralogic and alteration features. Particular attention was placed on not crossing over lithological breaks and/or major changes in mineralization/alteration.

7.10.2.1 Collar Coordinates

During data validation conducted by Orix, it was discovered that historic collar coordinates from selected drill holes in 2004-2006 had slight inconsistencies within the database and the values recorded on the logs. After careful interrogation, a compiled survey sheet generated by local surveyors was provided by George King. The survey sheet highlighted the inconsistencies, and the resident geologist identified them as planned coordinates recorded in the logs as opposed to final coordinates as seen in the compiled surveying sheet.

In the 2019 drill program, planned collar coordinates were generated in ArcGIS and holes were spotted using handheld Garmin GPS units. After the drilling was completed, Jervois hired the services of Wade Surveying to use high precision equipment to obtained final coordinates. The final coordinates were then given to Orix to add to the drill hole database.

7.10.2.2 Down-hole Surveys

Historic down-hole surveys were done at 150 ft. intervals. According to Micon, the reliability of down-hole surveys was difficult to confirm, particularly for the deep drill holes. But ultimately no surveys were removed from the database.

In the 2019 drill program, downhole surveys were taken every 50 ft. using a Reflex EX Trac unit, and survey results were closely monitored by the geologist on site, following up with the drill foreman immediately after any odd result. Overall, no issues were found with the 2019 down-hole survey data.

7.10.2.3 Lithology

During data validation and preparing the data for geological modelling, it became apparent that some important information available in the handwritten logs and excel quick logs was not available in the database. Two cases were found:

- Omissions of BTE/STE horizons described in the logs but entered as comments in the database, instead of entered as discrete mappable units.
- Omissions of BTE/STE horizons described in the "quick logs" but not captured in the database.

Orix incorporated this information in order to obtain a new lithology file that could guide the 3D modelling efforts. It is, however, important to note that the combination of lithology and alteration provided a general sedimentary package that defined areas more or less well mineralized. But the combination of lithology/alteration does not define individual mineralized lenses.

7.10.2.4 Analysis of QA/QC Monitoring Charts

Monitoring charts on quality control samples have already been discussed in Section 7.9 of this report. The use of quality control samples appears to have been in line with prevailing industry standards over the drill campaign periods. Overall, Orix and CSA auditors considered the sample preparation, security and analytical procedures were adequate over the different drill campaigns to ensure the integrity and credibility of the analytical results used for mineral resource estimation.

7.10.2.5 Specific Gravity

Prior to 2019, specific gravity measurements were performed by Geotechs in conjunction with sampling. The geologist would select an interval around 1.5-2 inches in length that was generally deemed representative of the given sample. This portion would be separated and weighted.

Measurements were taken using a triple beam balance scale. The sample was attached to the base of the weighing pan with a length of fishing line; then it was weighed while suspended in the air and subsequently weighed while suspended in a container of water. The water was tap water from the city of Salmon or a well near the Panther Creek Inn or water pumped from a pond on the property, which was the same water used for sawing core and drilling. No measures were in place for controlling the temperature of the water, and no measures were taken to prevent porous samples from taking on water during the measuring process.

The specific gravity value was computed with a handheld calculator using the following formula:

SG = (Weight in air) / (Weight in air-Weight in water).

Due to the inconsistencies seen in terms of the source of water used for such measurements, some concerns were raised about the validity of that historical dataset.

At the end of the 2019 drill program, in an effort to reconcile and validate historic densities measured on site with measurements under a controlled environment in the laboratory, a total of 100 samples were selected and submitted for density analysis to the SGS laboratory in Lakefield. Fifty samples from the historic holes stored in the core warehouse in Salmon, and another 50 samples from the 2019 drill program. Samples were carefully selected to be representative of the higher grade and shoulder zones in the main mineralized horizon, found on each of the 2019 holes, and a selected subset of historic holes. Samples were between 3-5 inches in length of PQ core; vacuum-sealed using a commercial plastic bag sealer, the sealed bag would then be placed inside thick sample bags and closed with zip ties. For samples that appeared fragile and/or oxidized, instructions were given to the lab to wax the samples prior to analysis. The footage and location of the samples were recorded on an excel sheet as well as individually on the core boxes, to guarantee the return of the sample at the correct position on the core box.

A comparison between historical samples measured on site with the 50 samples submitted in 2019 and analyzed by SGS, shows that the discrepancy of values is not significant (<10%) and is therefore assumed that the specific gravity data can be used for resource calculation. A historical compilation of specific gravity measurements from 813 samples is discussed in detail in Section 14.6.

7.10.3 Review of Previous Verifications/Audits

MDA has been an independent consultant on the Idaho project for several years dating back to 1995 and has been involved in all Jervois's previous independent mineral resource estimates and Independent Technical Reports.

7.10.3.1 Database Audit for the 2006 Resource

In 2005 MDA made numerous site visits to the Idaho project area during which time they reviewed and checked original assays, check assays and QA/QC procedures and results; reviewed and audited the digital database; examined geologic data and interpretations; and reviewed and re-sampled representative core intervals. Spot re-sampling produced comparable results to the original assays.

For drill data prior to the 1999 Ram drilling program, MDA checked about five per cent of the sample intervals in the project database for data entry errors. No errors were found for entries of cobalt, copper, or gold values; however, the footage for one interval was entered incorrectly. Approximately 10 per cent of the 1999 Ram drill data was audited, and no errors were found.

7.10.3.2 Database Audit for the 2015 Resource

Edwin Peralta of MDA visited the ICO site on December 10 to 12, 2014. Data verification of the 2010 drill data was completed to bring the 2012 resource estimate and block model to status as current and compliant with NI 43-101. Collar and downhole surveys were checked against original data supplied by a third-party surveyor while the assay data was digitally checked against the original assay lab data. No errors or missing data were encountered, and no changes were made to the database.

7.10.3.3 QA/QC for the 2006/2015 Resource

MDA reports that "Jervois's QA/QC analytical procedures including assays of check samples, standard reference material samples, and blanks, all show that the ICO assay data is reliable and verifiable and is adequate for estimating the ICO mineral resource".

7.10.3.4 Database Audit by CSA in 2019

CSA Global Consultants Canada Ltd ("CSA") was commissioned by Jervois to conduct an audit of the recent work completed at the ICO in Idaho. The aim was to review the processes and procedures used to collect the data, review the proposed methodology for the estimation of Mineral Resources, and audit the results. The quality of the informing data was discussed. The key points are:

- The confidence in the historical data is acceptable.
- There were initial concerns with density data. However, after compiling historical and 2019 data, a conclusion was made on using a regression formula that would represent density values adequately (see more in Section 4.6).
- There are long intervals logged as Main Zone that contain only isolated and very narrow mineralized layers. These layers were reported as a percentage of the horizon in the main logging files, and the detailed location is provided in separated logging files (in non-easy to use files in MS Word format). Some of these intervals were manually corrected. These intervals seem to be localized in certain peripheral areas and may represent a localized risk/uncertainty on resources.
- CSA reviewed the geological and mineralization interpretation and considered it appropriate for mineral resource estimation. However, CSA observed that some non-assayed intervals and low-grade intervals were incorporated within the wireframe. This was reported to Orix who removed the non-assayed metallurgical drill holes.
- The drill program appears to have been conducted at industry standards.
- Sampling was conducted methodically with 2019 similar to previous drill campaigns at ICO. Samples ranged from approximately 2 to 5 feet in length. Analytical methods appear fit for purpose using industry standard

techniques such as aqua regia and four-acid digests, lithium borate and fire assay fusions and ICP and AA instrumentation.

- The database presented appears to be fit for purpose. However, the present format (Excel spreadsheets) lowers confidence in the data, and there are few issues which need to be addressed to improve the data quality.
- Overall, no fatal flaws or major concerns remained.

7.10.4 Database Validation

Orix verified the database doing spot checks in excel before importing files on to Leapfrog. From Leapfrog, summaries of errors and warnings were exported and fixed back again in the database. This process was repeated until no errors were highlighted on Leapfrog. Section by section checks was also made to ensure that down-the-hole surveys were making sense and that all drill hole collars conformed to the DTM.

When compared to the access database received from Jervois and authored by MDA, Orix proceeded to include information from the last drill hole programs in order to have a complete database. Orix proceeded to compile QA/QC samples not included in the access database.

7.10.5 Data Verification Conclusions

Based on the verification procedures described above, Orix considers that the current database of the Ram deposit compiled during 2019 has been generated in a credible manner and has compiled all available data, is, therefore, suitable for use in mineral resource estimation. As highlighted previously by Micon, the lack of sampling beyond mineralized zones is a notable weakness in the database. It does not allow for the proper determination of dilution grades. Notwithstanding this shortfall, Jervois's exploration databases were professionally constructed and are sufficiently error-free to support mineral resource estimates.

7.10.6 Site Visit

Scott Zelligan, P.Geo., visited the site on October 4-6, 2019. The site visit included:

- A review of selected drill intervals from the active drill program,
- A review of selected drill intervals from historical drill programs,
- A review of historical core storage facility,
- A review of procedures and facilities of the active drill program,
- A review of Property access,
- A survey of collars from the active drill program, and
- A general review of the site and infrastructure.

Independent sampling was not completed. The Property has an extensive history, including NI 43-101 Technical Report reviews. The author reviewed these reports. Additionally, base metal mineralization is visible in the core, and all historical (and current) intervals were available for review. The author reviewed these and determined independent sampling was not necessary.

All current facilities and procedures were deemed appropriate and in keeping with CIM best practices. The Property was accessed from Salmon, Idaho, via well-maintained public-access gravel roads (Figure 7-24).

Table 7-5 displays the surveyed collars by the author compared to the database value. Measurements were taken with a Garmin GPS map 60CSx GPS unit and were taken in UTM NAD83 11T coordinates. The accuracy of the unit ranges from 5-10m horizontally. Vertical measurements with handheld units are generally not accurate enough to be useful for verification purposes. The terrain can play a factor in accuracy as well, and the Property does display abrupt elevation changes. Considering these factors, the review of collars yielded quite good results, confirming the location of five of the seven drill pad locations used for the 2019 program.

Table 7-5: Collar Review Measurements (UTM NAD83 11T)

Drill Collar	Review Survey		Database Value	
	Easting	Northing	Easting	Northing
R19-01	707,518	5,002,212	707,519	5,002,211
R19-02	707,546	5,002,222	707,547	5,002,221
R19-08	707,570	5,002,273	707,569	5,002,274
R19-11	707,541	5,002,260	707,541	5,002,261
R19-14	707,512	5,002,298	707,515	5,002,292

Figure 7-24: Map of site visit tracks

Core from the current program was easily accessible and stored within the core-logging facility or directly outside in shipping containers. The author reviewed core from throughout the stratigraphic sequence with the supervising geologist George King. Mineralized intervals were reviewed for six holes from the current program. The lithology and mineralization observed confirm the interpretation and, in the opinion of the author, the current approach accurately portrays the observed geology.

Core from the historical programs is stored at a building in Salmon. The building is in slight disrepair, but the core storage itself is in quite a good condition and very well organized. The author chose eight mineralized intervals from the main zone (mmh) at random from four of the historical drill programs (1997, 2004, 2005, and 2006) and was easily able to find the intervals and verify the mineralization within 5 minutes for each interval.

Figure 7-25: Jervois office in Salmon Idaho

Figure 7-26: Gravel Access Road to Site

Figure 7-27: ICO Mine office

Figure 7-28: Core-Logging Facility

Figure 7-29: Main Zone (mmh) Core from R19-13

Figure 7-30: Core Storage at Site

Figure 7-31: Active Drilling and Access Roads

Figure 7-32: Historical Core Storage in Salmon Idaho

Figure 7-33: Mineralized Intersection from R04-03

8 DEPOSIT TYPES

Identification and classification of the ICO deposit as a specific type has fluctuated throughout time. Geoscientific work/observations prior to 2005 suggested a sedimentary exhalative deposit class for the ICO deposits.

According to Evans et. al. (1986), "These deposits are strata bound iron-, cobalt-, copper-, and arsenic-rich sulphide mineral accumulations in nearly carbonate-free argillite/siltite couplets and quartzites."

The deposits comprising the ICO belong to a class of deposits variably described as "Blackbird Co-Cu" (Evans et. al., 1986) or "Blackbird Sediment-hosted Cu-Co" (Hõy, 1995).

Hoy (1995) suggested the following "associated deposit types: Possibly Besshi volcanogenic massive sulphide deposits, Fe formations, base metal veins, tourmaline breccias."

However, as of 2019, the identification of volcanic or intrusive rocks in the Ram deposit has been elusive, with the only exception being, some late lamprophyre and mafic dykes cutting across stratigraphy. At this point, at least for the Ram immediate area there doesn't seem to be evidence of coeval volcanism associated to the mesoproterozoic synsedimentary mineralization. It is likely however that such source type may have played a bigger role near the south in the blackbird deposit.

Later in 2006, Geoscientific work and observations suggested an iron oxide-copper-gold ("IOCG") deposit class with a magmatic-hydrothermal origin for the ICO deposits. The following is an excerpt from the abstract of a paper by Slack J. F. (2006).

"Analysis of 11 samples of strata-bound Co-Cu-Au ore from the Blackbird district in Idaho shows previously unknown high concentrations of rare earth elements ("REE") and Y, averaging 0.53 wt. percent ∑REE + Y oxides. Scanning electron microscopy indicates REE and Y residence in monazite, xenotime, and allanite that form complex intergrowths with cobaltite, suggesting coeval Co and REE + Y mineralization during the Mesoproterozoic. Occurrence of high REE and Y concentrations in the Blackbird ores, together with previously documented saline-rich fluid inclusions and Cl-rich biotite, suggest that these are not volcanogenic massive sulphide or sedimentary exhalative deposits but instead are IOCG deposits."

On the other hand, mineralogy seen in the 2019 program, as well as recorded in all previous drilling campaigns fails to mention any tangible content of IOCG related assemblages. Therefore, making it difficult to assign such deposit type to this mineralization.

Instead, the current understanding indicates that the Ram area is a Metasedimentary rock hosted Co-Cu-Au package with strata bound zones of semi massive sulphides. The origins of these deposits are thought to be varied; a range of mineralizing processes, from diagenetic to epigenetic are thought to be involved, however the sources of the hydrothermal fluids and metals are still enigmatic (Bookstrom et al. 2016).

9 EXPLORATION

9.1 PROGRAMS

9.1.1 1995-1996 Campaign

In 1995, soil sampling of selected areas was conducted on lines spaced ~60 m (200 ft) and ~120 m (400 ft) apart, with samples collected at intervals of ~30 m (100 ft) along the lines. This program discovered the southern end of the previously unknown Ram target.

In 1996, the soil grid was extended north, and soil samples were collected on lines spaced ~60 m (200 ft) apart with samples collected at ~8 m (25-ft) intervals along the lines. Some infill samples were collected from the 1995 soil grid. Other parts of the grid were also extended and sampled on ~8 m (25 ft) intervals where it was deemed warranted.

A total of 8,427 soil samples were collected during the 1995/1996 campaign. Geochemical contours were created for Co, Cu, As, and Au and helped to narrow and confirm the location of the Ram anomaly (Figure 9-1).

Other exploration activities conducted during 1995/1996 included surface geological mapping at a scale of 1 in to 100 ft, mapping of old trenches and prospect pits, and collection of 979 surface rock samples including those from trenches.

9.1.2 1997 Campaign

The Ram soil grid was extended northward, with the collection of an additional 95 soil samples; concurrently, the north and south extensions of the Ram prospect were geologically mapped.

In the same year, Jervois built ~950 m (3,100 ft) of benched drill road into the Ram zone; the road was laid out to cross the Ram soil geochemical anomaly, in order to facilitate trenching. Three trenches, ~190 m (623 ft) long in aggregate, were excavated within the "prism" of the road; the trenches were mapped, and 83 rock samples were collected. The newly opened 6,930 drift was mapped, and 163 rock samples were collected.

For a topographic base, Jervois had a five-feet contour map of the project area, produced photogrammetrically, using aerial photography.

9.1.3 1998-2001 Campaign

Permitting baseline studies were initiated.

9.1.4 2002-2006 Campaign

Various baseline studies were completed in support of project activities. The Plan of Operations ("PoO") and the United States ("USFS") Environmental Impact Statement ("EIS") were also completed. An updated PoO was submitted in April 2006.

9.1.5 2007-2019 Campaign

No exploration work other than drilling was carried out.

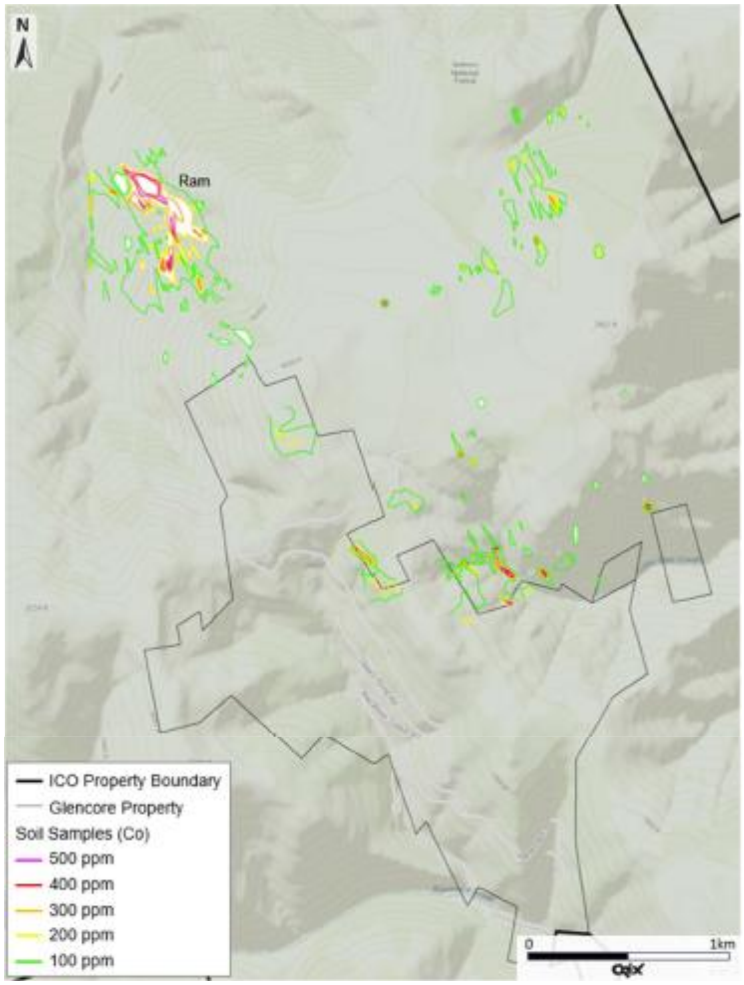


Figure 9-1: Soil Co Contours in the ICO

9.2 EXPLORATION RESULTS

The surface geological and geochemical work were important contributors to the discovery and expansion of the Ram deposit both in the northerly and southerly directions. Whilst both soil and rock chip samples are not representative, they serve primarily to detect mineralization for further investigation by trenching and ultimately drilling.

10 DRILLING

10.1 DRILLING CAMPAIGNS

The ICO drilling campaigns are summarized in Table 10-1. Total drilling in the property is 224 holes for 142,358.4 ft.

As of the end of 2019, the Ram deposit has been tested with 120 diamond drill holes drilled in 1997 through 2017 by Formation Capital and drilled in 2019 by Jervois Mining totaling 79,682.9 ft. Although drilling has been intermittent over the years, there has been continuity over the campaigns.

The Ram deposit comprises several sub-parallel horizons which generally strike N15°W and dip 50°-60° to the northeast and were drill tested to depths of 1,200 ft vertically. The Main zone, which is the most extensive and laterally continuous, has been tested drill tested over 3,300 ft (~1,000 m) in strike length, and have true thicknesses that average about 20 ft. However, the main zone consists of minor layers of differentially altered and mineralized sub horizons, most of which range between 3 to 6 ft.

Table 10-1 shows locations of the drill collars, and the surface projection of drill-hole traces (azimuths) for the Ram deposit.

Table 10-1: ICO Drilling Campaigns

Year Drilled	Operator	Deposit	Number	Feet
1959	Calera Mining Company	Sunshine	3	982.0
1979 – 1981	Blackbird Mining Company	Sunshine	29	17,826.0
1995 – 1996	Formation Capital	Sunshine	48	29,144.0
1995 – 1996	Formation Capital	East Sunshine	24	14,723.5
	TOTAL Sunshine		**104**	**62,675.5**
1997	Formation Capital	Ram	20	12,045.0
1999	Formation Capital	Ram	11	5,210.5
2000*	Formation Capital	Ram	8	2,613.0
2004	Formation Capital	Ram	28	24,877.0
2005	Formation Capital	Ram	9	5,302.5
2006	Formation Capital	Ram	4	4,532.0
2010	Formation Capital	Ram	6	5,727.5
2016	Formation Capital	Ram	9	3,057.5
2017	Formation Capital	Ram	6	6,062.1
2019	Jervois Mining	Ram	19	10,255.8
	TOTAL Ram		**120**	**79,682.9**
Grand Total		**Ram+Sunshine**	**224**	**142,358.4**

*Metallurgical Test holes – Not used in Grade Model

The Sunshine deposit is located about a mile (approximately 1.6 km) due south of the Ram deposit (Figure 10-1). It consists of multiple, stacked sulphide-bearing beds of limited strike length. Individual mineralized beds or horizons range in thickness from inches to several feet and are associated with biotite-rich tuffaceous exhalative (BTE) horizons. The deposit horizons strike north- northwest and dip moderately to steeply to the east-northeast.

The resources and reserves considered in the current Technical Report are those of the Ram deposit only. The Sunshine and other deposits within the property represent future potential for the ICO resource and reserve base. All holes drilled on the Ram deposit are diamond core holes.

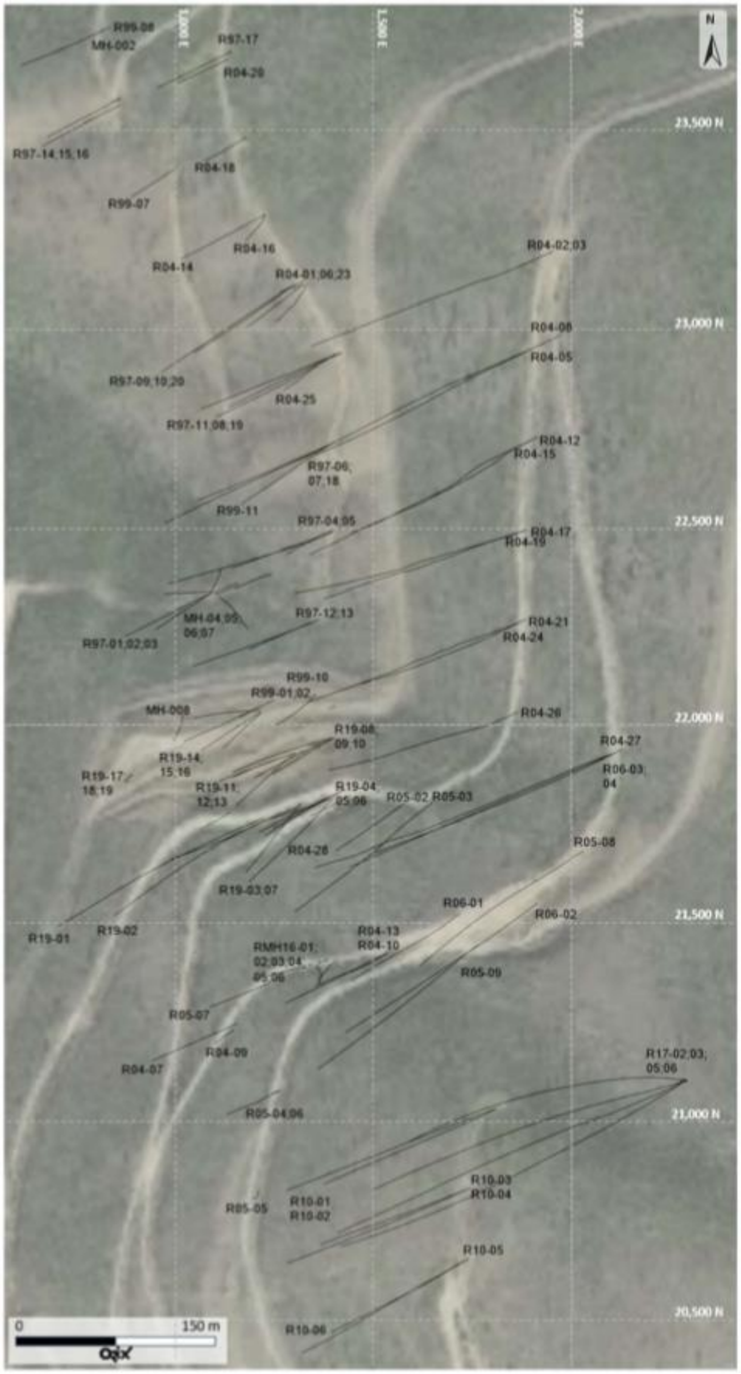


Figure 10-1: Ram Deposit Drill Hole Locations

10.2 DRILLING PROCEDURES

10.2.1 Historic Drilling

The following description has been excerpted from the March 2015 PEA Technical Report by Samuel Engineering Inc. and is based on observations from Mining Development Associates ("MDA") between 1998 to 2010. In addition, MDA also provided their expertise in the development of the first ICO Ram Block model.

All drill data was obtained by core drilling, except for reverse circulation collars for the holes completed by FCC in 2000 to obtain metallurgical samples. Exploration holes were drilled with either NQ- or HQ-size core; the metallurgical holes were drilled with PQ- size core. NQ, HQ and PQ core have diameters of 1.875 inches (47.6 mm), 2.500 inches (63.5 mm), and 3.345 inches (85.0 mm), respectively.

FCC routinely logged the drill core in considerable detail, with particular emphasis placed on mineralized intervals.

The collars of all drill holes were located using tight chain and compass from the nearest known point. Most of the pre-1998 drill-hole collar locations were resurveyed by Harper-Leavitt Engineering Inc., using a transit (1998 report by FCC Staff). Collar locations for the 2010 drill holes were professionally surveyed by Taylor Mountain Surveying, of Salmon, Idaho, using a combination of Global Positioning Systems and conventional survey methods.

A single-shot, Sperry Sun instrument was used for down-hole surveys to check the drill-hole orientations. Down-hole surveys were done every 150 feet in the hole.

Drilling was conducted as angle holes oriented approximately normal to the strike of the mineralized horizons and crosscutting mineralized horizons at appropriate angles that allowed true thicknesses of mineralization to be determined.

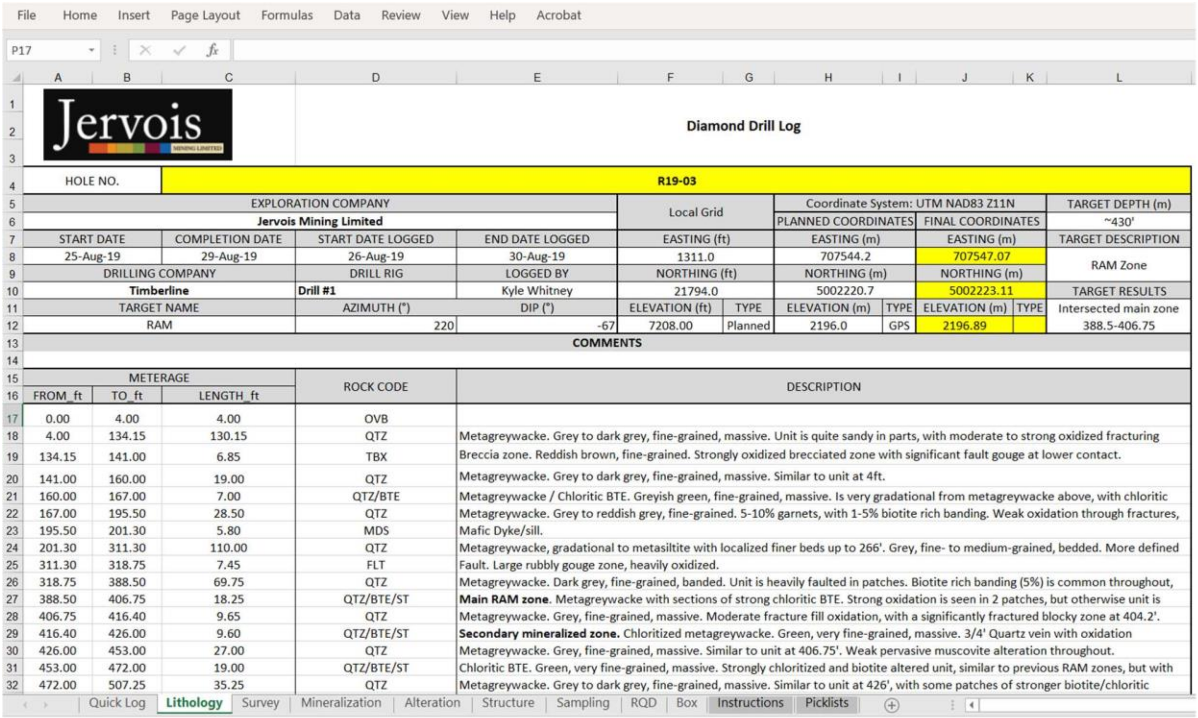
It was MDA's opinion that FCC's drilling methods used at the Ram Deposit follow industry standard procedures and were appropriate methods to adequately interpret the geology and mineralized zones used in the resource model.

10.2.2 Jervois Mining 2019 Drilling

In the Spring of 2019, Jervois Mining approached Orix Geoscience Inc., to support and manage the summer 2019 diamond drill hole program. Orix, created new logging templates and sent geologists for rotation work on site during the months of July to October. All logging templates were previously discussed and edited with input from the senior geologist of the program, George King, as well as the geology group manager David Selfe from Jervois Mining.

All drill data was obtained by diamond core drilling. Exploration holes were drilled with HQ -size core; the metallurgical holes were drilled with PQ- size core. HQ and PQ core have diameters of 2.500 inches (63.5 mm) and 3.345 inches (85.0 mm), respectively.

Orix systematically logged the drill core in considerable detail using an excel sheet with multiple tabs for different sources of information as follows: Quick log; Detailed Lithology; Survey; Mineralization; Alteration; Structure; Assay; RQD; Box Ends (Figure 10-2).



Diamond Drill Log

HOLE NO.	R19-03										
EXPLORATION COMPANY				Local Grid		Coordinate System: UTM NAD83 Z11N				TARGET DEPTH (m)
Jervois Mining Limited						PLANNED COORDINATES		FINAL COORDINATES		~430'
START DATE	COMPLETION DATE	START DATE LOGGED	END DATE LOGGED	EASTING (ft)		EASTING (m)		EASTING (m)		TARGET DESCRIPTION
25-Aug-19	29-Aug-19	26-Aug-19	30-Aug-19	1311.0		707544.2		707547.07		RAM Zone
DRILLING COMPANY		DRILL RIG	LOGGED BY	NORTHING (ft)		NORTHING (m)		NORTHING (m)		
Timberline		Drill #1	Kyle Whitney	21794.0		5002220.7		5002223.11		TARGET RESULTS
TARGET NAME		AZIMUTH (°)	DIP (°)	ELEVATION (ft)	TYPE	ELEVATION (m)	TYPE	ELEVATION (m)	TYPE	Intersected main zone
RAM		220	-67	7208.00	Planned	2196.0	GPS	2196.89		388.5-406.75

COMMENTS

FROM_ft	TO_ft	LENGTH_ft	ROCK CODE	DESCRIPTION
0.00	4.00	4.00	OVB	
4.00	134.15	130.15	QTZ	Metagreywacke. Grey to dark grey, fine-grained, massive. Unit is quite sandy in parts, with moderate to strong oxidized fracturing
134.15	141.00	6.85	TBX	Breccia zone. Reddish brown, fine-grained. Strongly oxidized brecciated zone with significant fault gouge at lower contact.
141.00	160.00	19.00	QTZ	Metagreywacke. Grey to dark grey, fine-grained, massive. Similar to unit at 4ft.
160.00	167.00	7.00	QTZ/BTE	Metagreywacke / Chloritic BTE. Greyish green, fine-grained, massive. Is very gradational from metagreywacke above, with chloritic
167.00	195.50	28.50	QTZ	Metagreywacke. Grey to reddish grey, fine-grained. 5-10% garnets, with 1-5% biotite rich banding. Weak oxidation through fractures,
195.50	201.30	5.80	MDS	Mafic Dyke/sill.
201.30	311.30	110.00	QTZ	Metagreywacke, gradational to metasiltite with localized finer beds up to 266'. Grey, fine- to medium-grained, bedded. More defined
311.30	318.75	7.45	FLT	Fault. Large rubbly gouge zone, heavily oxidized.
318.75	388.50	69.75	QTZ	Metagreywacke. Dark grey, fine-grained, banded. Unit is heavily faulted in patches. Biotite rich banding (5%) is common throughout,
388.50	406.75	18.25	QTZ/BTE/ST	**Main RAM zone**. Metagreywacke with sections of strong chloritic BTE. Strong oxidation is seen in 2 patches, but otherwise unit is
406.75	416.40	9.65	QTZ	Metagreywacke. Grey, fine-grained, massive. Moderate fracture fill oxidation, with a significantly fractured blocky zone at 404.2'.
416.40	426.00	9.60	QTZ/BTE/ST	**Secondary mineralized zone.** Chloritized metagreywacke. Green, very fine-grained, massive. 3/4' Quartz vein with oxidation
426.00	453.00	27.00	QTZ	Metagreywacke. Grey, fine-grained, massive. Similar to unit at 406.75'. Weak pervasive muscovite alteration throughout.
453.00	472.00	19.00	QTZ/BTE/ST	Chloritic BTE. Green, very fine-grained, massive. Strongly chloritized and biotite altered unit, similar to previous RAM zones, but with
472.00	507.25	35.25	QTZ	Metagreywacke. Grey to dark grey, fine-grained, massive. Similar to unit at 426', with some patches of stronger biotite/chloritic

Figure 10-2: Excel Log Example

The scale/detail of logging was predetermined and set at 1 m (~3 ft), meaning any lithological or structural feature larger than 1 m will be broken into its own separate unit, and anything lesser than 1m would have to be briefly included in the description box.

The planned collar of all drill holes was located using a handheld Garmin GPS, the rigs were then aligned by the geologist on site using a Suunto compass as well as back or front picket sights when possible. At the end of the program, all of the 2019 drill-hole collar locations were resurveyed by Wade Surveying, from Salmon, Idaho, using two types of GPS units. The final collar coordinates for the first 6 holes were surveyed using a Leica SR500 GPS unit, and the remaining drill holes were surveyed with a Trimble R8 model 3 GPS Unit. Final surveyed coordinates have an expected horizontal accuracy of +/- 1 cm, and a vertical accuracy of +/- 5 cm.

For every drill hole, verbal and written instructions were given to the drillers on site indicating the location, azimuth, dip and frequency of downhole surveys. Downhole surveys were taken every 50 ft using a Reflex EZ-Trac unit. For the infill drill holes R19-08 to R19-16 a Reflex TN14 Gyrocompass was placed on the rods at the start of the hole, to confirm azimuth and dip (Figure 10-3).

The 2019 drill program procedures have been audited by CSA and it is their opinion that the drilling methods used at the Ram Deposit follow industry standard procedures and were appropriate to interpret the geology and mineralized zones.

Jervois MINING LIMITED		**Drillhole Summary Sheet**			
		Hole ID:	**R19-02**		
Coordinates:	5002218.6 N 707540 E UTM NAD83 Z11N	**Azimuth:**	245	**Estimated Depth (ft):**	~1350
Start Date:	11/08/2019	**Dip:**	-75	**Rig:**	**Pad:**
Reflex Instructions:		• Reflex to be taken at depths of 50, 100, 150, 200, etc.			
Geology Contact:		David Selfe: +61 439 030-921 Edwin Escarraga: +1 416-8854371			

Figure 10-3: Drill hole Instructions sheet and Drill hole Survey Tools

10.3 DRILLING RESULTS

Drill hole logging, sampling and assay results have confirmed the following:

- The Ram deposit consists of somewhat discontinuous hanging wall zones composed of 6 main horizons, a Main zone identified in terms of a combination of lithology and alteration., and a Footwall Zone (Figure 10-4). These sub-parallel horizons generally strike N15°W and dip 50° – 60° to the northeast.
- The mineralized zones are lenticular/stratiform with most of the significant Co mineralization associated with biotite/chlorite hydrothermally altered horizons, previously identified as exhalative, i.e. biotitic tuffaceous exhalate ("BTE"), siliceous tuffaceous exhalate ("STE"), and quartzite with impregnations of biotitic tuffaceous exhalate (QTZ/BTE) or siliceous tuffaceous exhalate (QTZ/STE).
- True thickness of the lithological units modelled for the hanging wall units have a wide range as they occur as lenses, on the other hand the main unit is continuous on strike length and dip and has an average thickness of about 30 ft. However, the strongly mineralized horizons occurring within this main unit, average only about 3-5 ft and range from less than 2 ft up to 13 ft.

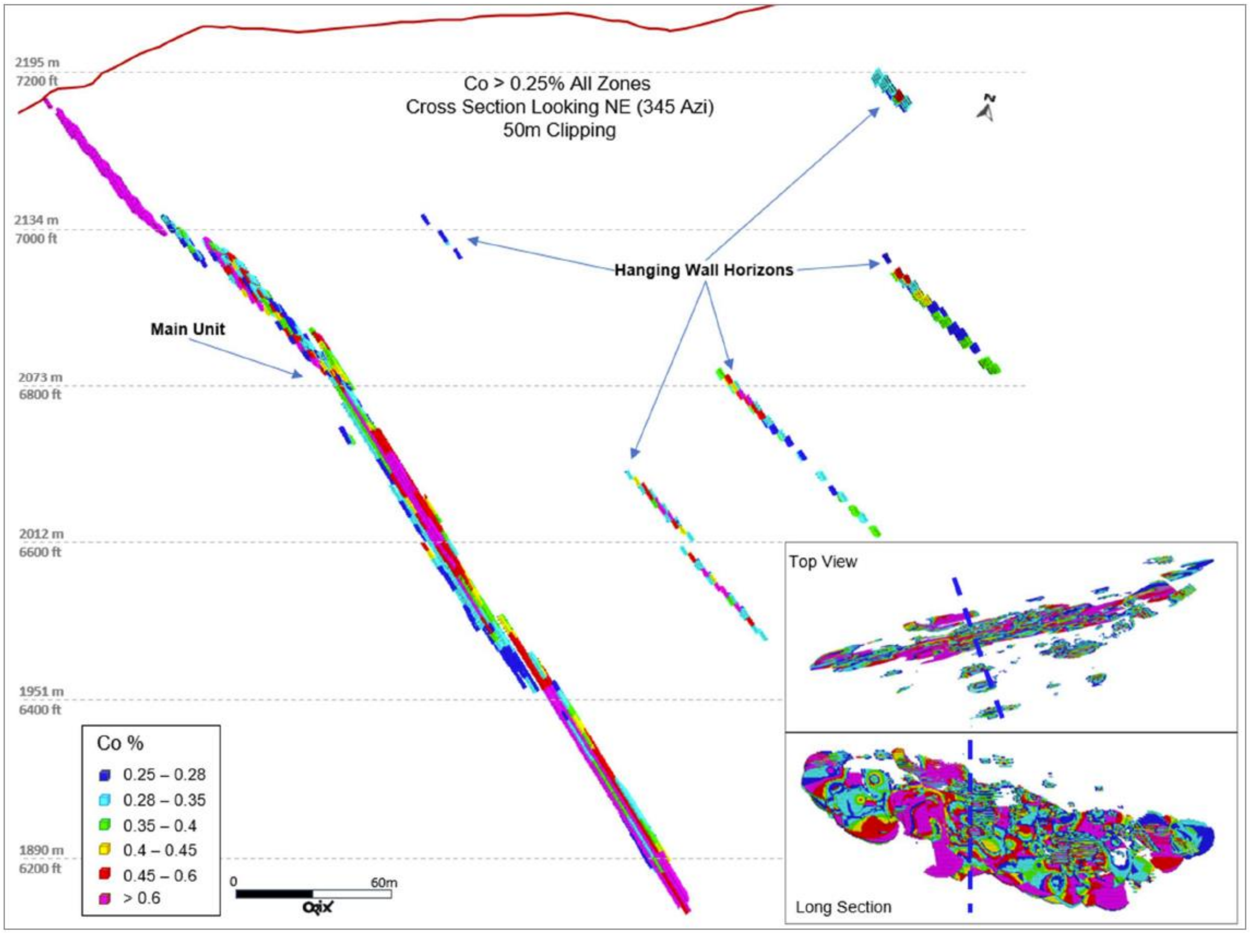


Figure 10-4: Cross Section through the Ram Deposit

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 2019 DRILL PROGRAM, SAMPLE PREPARATION

11.1.1 Sample Preparation at Site

The core was delivered by the drilling crew at the end of each shift, the boxes were cross piled on pallets for temporary storage at the core logging building. The core was then moved to core benches to be quick logged by Senior Geologist George King with assistance from the Orix Geologist on site. Once the core was laid out on the logging tables, RQD, and footage-marks on the boxes and the core were completed using china markers.

At this stage Orix personnel would proceed to do the detailed examination and description of the core, adding markings to relevant sections of the core, leaving for last the marking of sample intervals (Figure 11-1).

Figure 11-1: Core Logging Image
Source: Orix, 2019 Ram Drilling, R19-16.

Sample lengths/intervals were defined based on lithological, alteration and mineralogical changes, an effort was made to not sample over lithological boundaries or drastic changes in mineralogy/alteration segments. Sampling lengths in 2019 ranged from 1.0 ft to 6 ft, with most samples between 2-3 ft (average 2.8 ft). Mineralized/anomalous zones were bracketed by taking two or more samples on the margins as shoulders.

Once the logging was completed and wet photos were taken, a hired local technician would cut the drill core selected for sampling with a diamond blade core saw, into symmetrical halves resulting in two equally representative samples. One-half of the drill core was placed in a plastic sample bag with a sample identification tag before being sealed. The other half of the drill core was returned to its original position in the core box and the corresponding tag for each sample interval was placed at the end of the sample position in the core box. The only exception to this procedure were selected samples from the main mineralized unit in holes R19-04 and R19-06, that were submitted as whole core to SGS for metallurgical testing.

Quality control was achieved by inserting one barren control sample (blank), two different certified reference materials ("CRMs") and field duplicates at regular intervals into the sample stream for each batch of core samples. Blanks were inserted approximately every 40 samples or immediately after a sample suspected to run high (strong visible cobaltite mineralization). Standards were inserted approximately every 20 samples. Field duplicates occurred approximately every 60 samples outside of the main unit but in mineralization in order to test variability of metal values. In general, the goal was to place a QA/QC sample approximately every 20th sample (Standard, Blank, Duplicate).

Other than the insertion of control samples, there is no other action taken at site.

11.1.2 Laboratory Sample Preparation

Once at the laboratory, the samples are entered into the internal system. Samples are prepared by drying, if necessary, then the entire sample is crushed in its entirety to ≥70% at <2 mm, riffle split to obtain a 250 g sub-sample, which was pulverized to ≥ 85% at < 75 microns.

11.2 ANALYSES

Over the course of all drilling programs in the past, the Ram deposit has been selectively sampled and analysed by a few different laboratories. In 1996 check-sample analysis was completed in EcoTech Laboratories Ltd. of Kamloops, British Columbia. However, sample analyses through 2006 were performed by Chemex Labs, Inc., of Sparks, Nevada, and Vancouver, British Columbia, and by Bondar Clegg Laboratories, Inc. (USA), of Reno, Nevada, and Bondar Clegg Laboratories, Inc. (Canada), of Vancouver, British Columbia.

Cobalt and copper analyses for drill samples up through the 2000 drilling were done by 4-acid (HNO3- HClO4-HF-HCl) digestion and an atomic absorption (“AA”) finish; gold was analyzed by 30-gram fire assay followed by an AA finish. Cobalt and copper analyses for the 2004 through 2006 drill samples were done by aqua regia digestion and an AA finish; gold was again analyzed by 30-gram fire assay followed by an AA finish. Multi-element geochemical analyses for all drill campaigns were performed using aqua regia digestion followed by induction-coupled plasma atomic-emission spectrometry (ICO-AES). These are all industry standard analytical techniques appropriate for the types of rocks and mineralization at the ICO.

Chemex Labs, Inc., which became ALS Chemex and subsequently ALS Global, holds ISO 9002:1994 certification at its North American and Peruvian laboratories and ISO 9001:2000 certification in North America. ALS Global is the successor to Chemex and Bondar Clegg, the laboratories that did most of Jervois` analyses. Neither Micon nor MDA has determined the date that ALS Global or its predecessors first obtained ISO 9002 certification, but it is probable that much of the work for Jervois was done before that date.

Sample analyses in 2010 were performed by ALS Minerals, a division of ALS Global. Analytical techniques similar to those used prior to 2010 were employed, including aqua regia digestion and AA or ICO-AES finish for cobalt and copper, and 30-gram fire assay with AA finish for gold. Multi-element geochemical analyses were performed using lithium metaborate fusion, acid digest and ICO- AES-Mass Spectrometry. Duplicate samples for verification purposes were analyzed at ACT Labs of Ontario, Canada and were analyzed for cobalt and copper by sodium peroxide fusion and ICO-AES finish, and for gold by 30-gram fire assay with AA finish.

For the 2019 Drill Program, Jervois submitted samples to two different Labs. Regular assay samples were submitted to ALS in Reno Nevada, and SGS in Lakefield, Canada. Assays included cobalt, copper and gold as part of their routine analytical procedure. In addition, multi-element geochemical analyses were completed on all the samples submitted using aqua regia digestion and AA or ICO-AES finish. The set of samples submitted to SGS were then kept for further metallurgical analysis.

All the laboratories involved in the analyses of samples are independent of the issuer.

11.3 SECURITY

All activities pertaining to data collection, i.e. sampling, insertion of control samples, packaging and transportation, were/are conducted under the direct supervision of the project manager.

Jervois's core and sample security measures were typical for exploration projects in North America at the time the work was done. All historic core was received at the drill by the geologist on site and taken to the company's facility in Salmon for storage after logging and sampling was completed. For the 2019 drill program the core was kept on site, a portion of the core is cross piled on wooden pallets inside the logging facility and the remaining portion is stored in locked sea can containers.

That facility is a warehouse-like building with lockable doors. Sawed core was placed in labeled sample bags that were closed with plastic zip ties (Figure 11-2). During the 2019 program, Shana Hilton, the Administration Manager, prepared the samples for shipment and took them to "Bri-Easy Shipping" in Salmon. The samples were weighted, labeled and picked up by FedEx who notifies Jervois personnel when sample batches have been delivered to the laboratory. Thus, the core has been under Jervois's control from receipt at the drill, and the parts of core not used for the analytical samples remained under Jervois's control.

Figure 11-2: Jervois Core Storage and Core Sampling
Source: Photos taken by Orix, 2019.

11.4 QUALITY CONTROL/ASSURANCE (QA/QC)

11.4.1 Historic Core MDA Verification

MDA examined Jervois's data related to QA/QC in 1998 and established that the assays of the check samples, blanks and standards were in good agreement with the expected values. MDA also examined the 1999 Ram drilling QA/QC and a further check on assay QA/QC data was completed in 2004. MDA's conclusion was "Overall, Jervois has demonstrated diligence in monitoring check assays and standards and blanks results, which is critical to the maintenance of an accurate database". In addition to these checks, MDA independently selected 10 samples from the 2005-2006 drilling program and sent them to ACME laboratories for check assaying from which they obtained a good agreement between the original assays and the check assays.

11.4.2 Micon Verification 2017

Micon noted that Jervois used both blanks and standards in its QA/QC protocols but did not compile control charts. A blank sample was inserted in the sample batch sequence immediately after a highly mineralized sample expected to return high values of cobalt and/or copper. A standard or certified reference material ("CRM") was inserted at the rate of 1 in every 20 samples. Warning limits were set at +/-2 standard deviations, and control limits were set at +/-3 standard deviations. When a quality control sample fell outside the control limits, the cause was thoroughly investigated, and if need be, the entire sample batch was automatically re-assayed, and all the initial test results are rejected.

11.4.2.1 Blanks

Jervois used a barren Apple Creek meta-siltite as a blank to monitor and control contamination between samples. The assay was considered a failure if the value was higher than three times the detection limit ("DL"). A control chart incorporating cobalt, copper and gold can be seen in Figure 11-3. Except for only two samples, the control chart demonstrates that there was no contamination between samples; if any, then it was insignificant. It has been suggested that the failures indicated in Figure 11-3 are most likely due to typo errors.

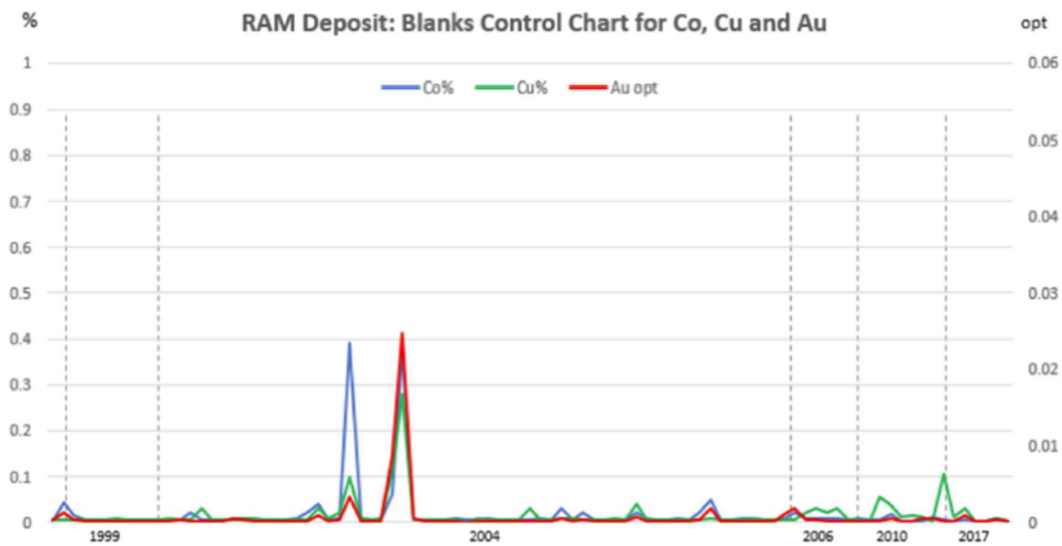


Figure 11-3: Summary of blank Samples Results: 1997 to 2017 Drilling

11.4.2.2 Standards/CRMs

In the past Jervois used three varieties of CRMs i.e., low grade, medium grade and high grade. Although the use of the higher standard was never consistent. All CRMs were prepared at Chemex Laboratories Inc. from mineralized material obtained from the ICO area. Given that throughout the years of 1996 – 2017, different methods of analysis were used, it was decided that for control charts of the historical (pre-2019 data), a new calculation of standard deviations and means needed to be created, as some of the original data was unfortunately lost and true certificates of some elements were not found. The values summarized in Table 11-1 below are based on the averages of QA/QC assays obtained from all drill hole programs.

Table 11-1: Summary of Certified Values for Standards used at the ICO

Item	Cobalt %	Copper %	Gold oz/st
Standard 1	0.834	0.071	0.061
Standard 2	0.254	0.034	0.030

Any results falling outside the failure limit of +/-3 SD (standard deviation) were rejected pending investigation into the source of error. Jervois's general practice was to use standards 1 and 2. Orix has summarized the QA/QC results by compiling control charts for the drilling periods from 1997 to 2017 for standards 1 and 2 (see Figure 11-4 to Figure 11-9).

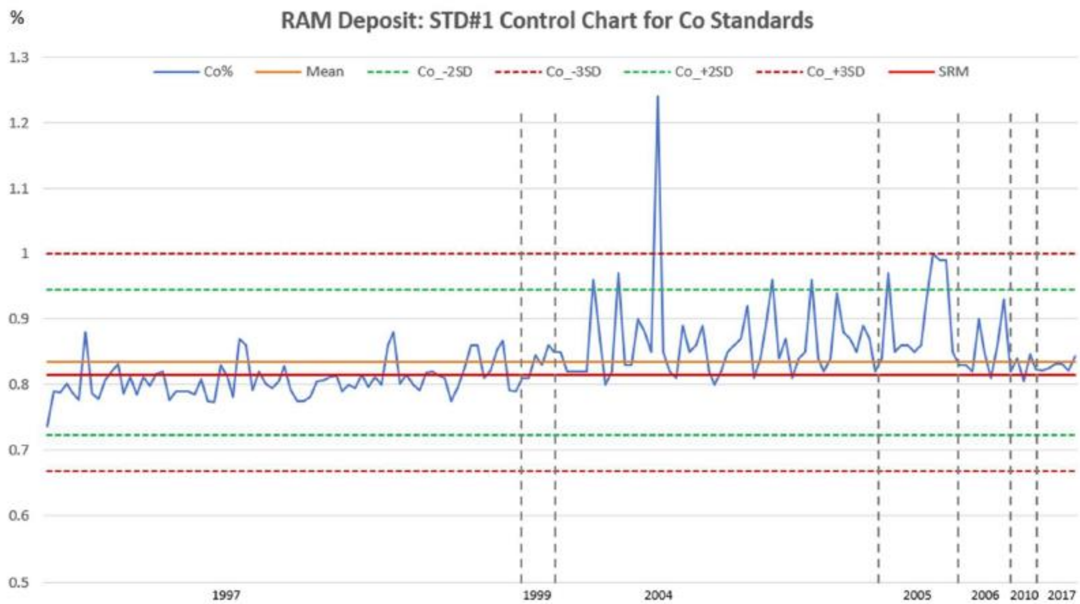


Figure 11-4: Control Chart for Co: Standard 1

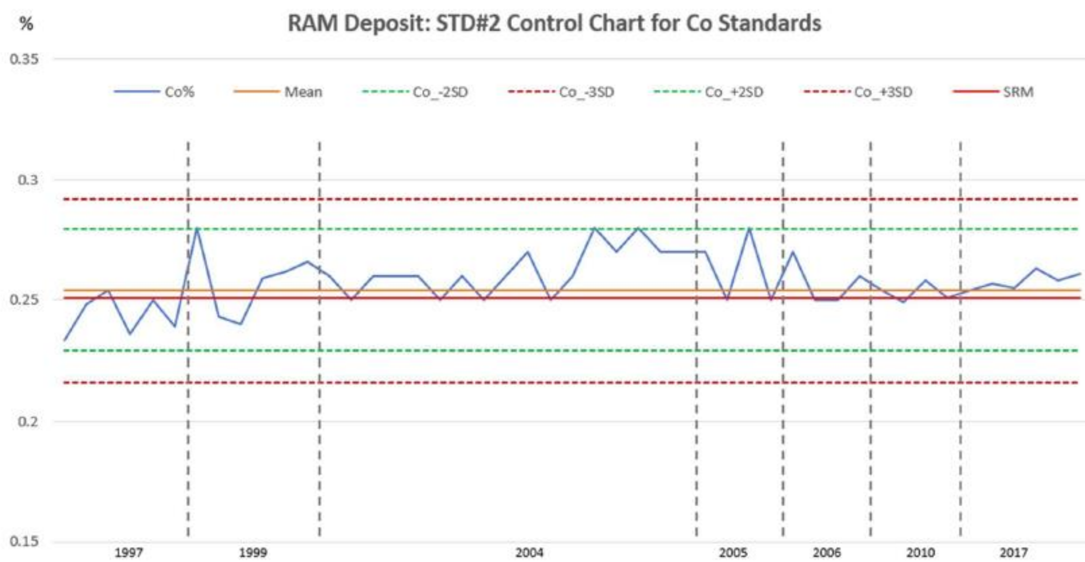


Figure 11-5: Control Chart for Co: Standard 2

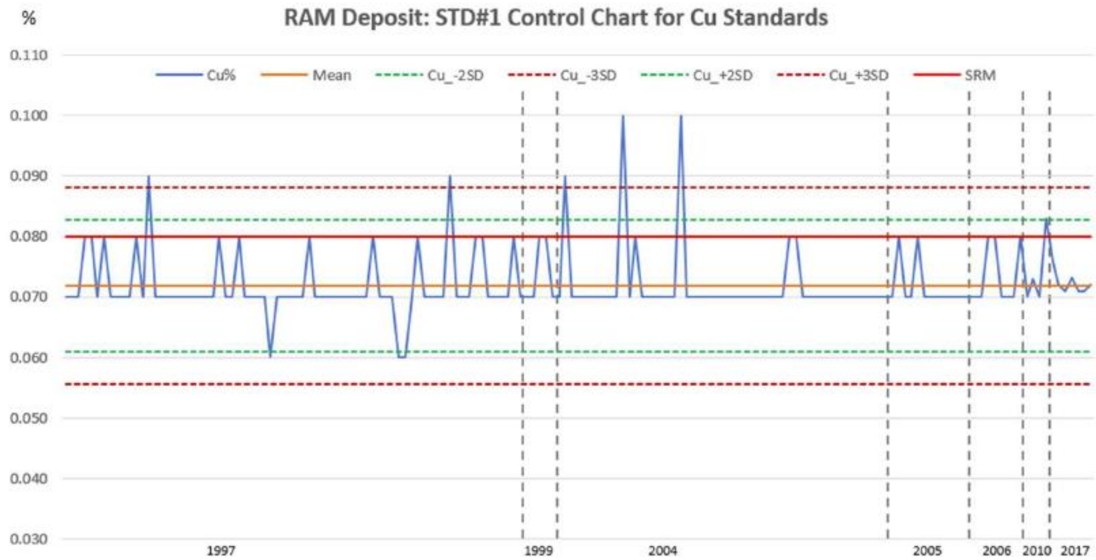


Figure 11-6: Control Chart for Cu: Standard 1

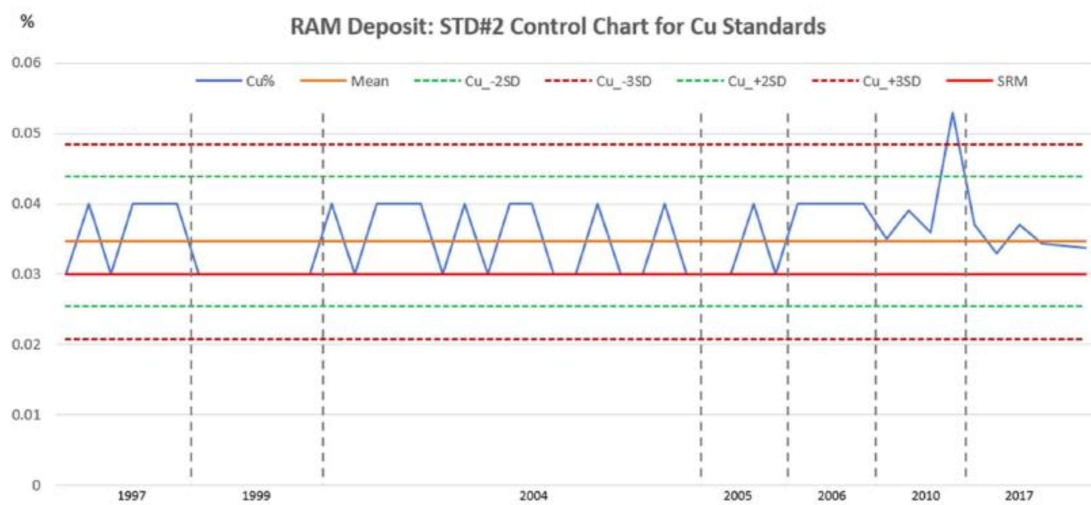


Figure 11-7: Control Chart for Cu: Standard 2

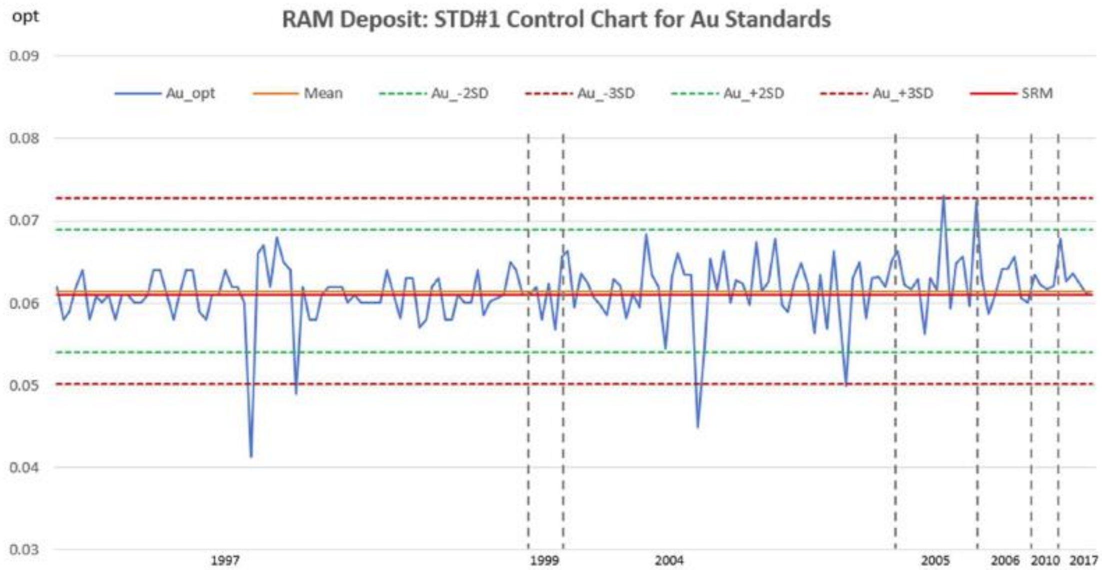


Figure 11-8: Control Chart for Au: Standard 1

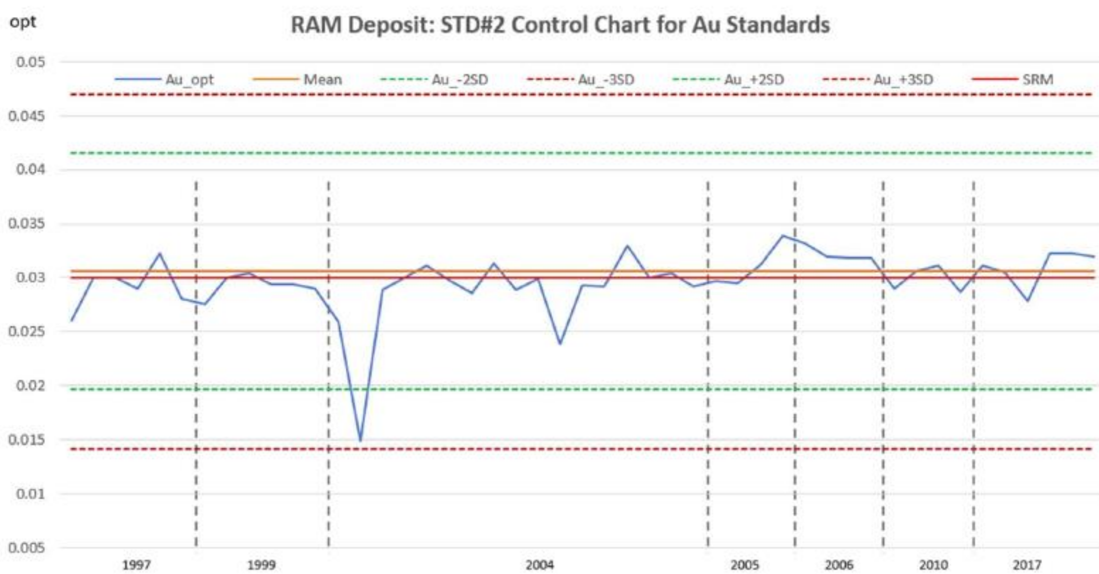


Figure 11-9: Control Chart for Au: Standard 2

For Standard 1, Figure 11-4 shows only one significant failure plus three borderline failures for Cobalt; Figure 11-5 shows five failures for copper, three of them during the 2004 program and Figure 11-8 shows three failures for gold. For Standard 2, there is only one failure (Figure 11-7) for copper. Difference in analytical methods likely contribute to the wide variation as well as some of the failure. Overall, however the number of failures/borderline failures is insignificant to have material impact on the assay database.

11.4.3 Orix Verification 2019

During the 2019 drill program, Orix's personnel used blanks, standards and field duplicates in its QA/QC protocols. A blank sample was inserted in the sample batch sequence immediately after a highly mineralized sample expected to return high values of cobalt and/or copper. Two different standards or CRM were used, such alternating standard was inserted approximately every 20 samples. Warning limits were set at +/-2 standard deviations, and control limits were set at +/-3 standard deviations. When a quality control sample fell outside the control limits, the cause was thoroughly investigated, and if need be, the entire sample batch was automatically re-assayed, and all the initial test results are rejected.

11.4.3.1 Blanks

Jervois used barren construction brick as a blank to monitor and control contamination between samples. The assay was considered a failure if the value was higher than half the average Co background of the assays in the area. Figure 11-10 incorporates cobalt, copper and gold. Except for only one sample, the control chart demonstrates that there was no contamination between samples; if any, then it was insignificant. It has been suggested and is strongly recommended by Orix, that Jervois stop the use of brick material as a blank and incorporate the use of certified blank silica material for any upcoming programs, in order to obtain better consistency. In addition to this, the laboratory performs their internal blanks which are reported and seen by the user via their webtrieve sites.

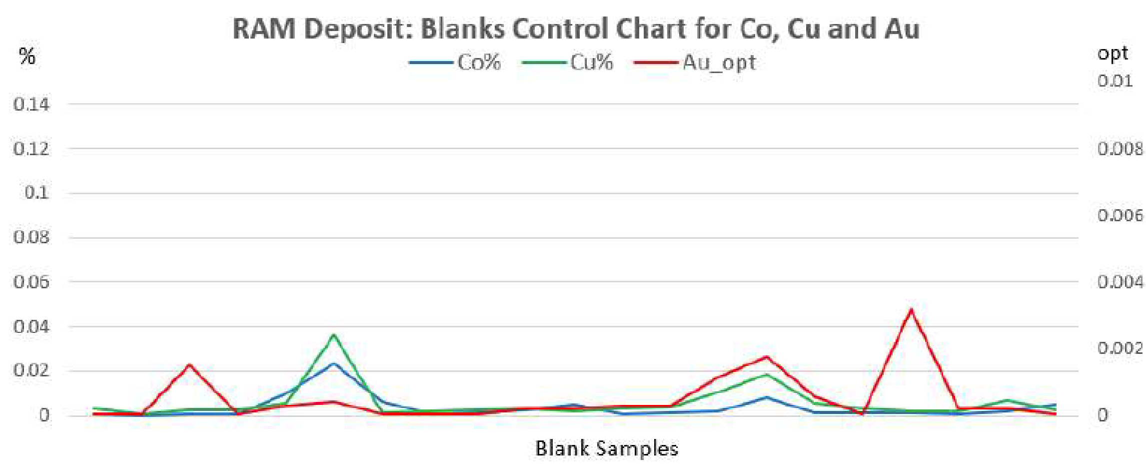


Figure 11-10: 2019 Control Charts: Blanks

11.4.3.2 Standards/CRMs

The 2019 drill program used two varieties of CRMs. The certified values summarized in Table 11-2 below are based on the averages of assays obtained from several different reputable laboratories using mineralized material obtained from the ICO area.

Table 11-2: Summary of Certified Values for Standards used during 2019 at the Ram Deposit

	Standard	Mean	SD	H2SD	L2SD	H3SD	L3SD
Co (%)	1	0.814	0.013	0.84	0.788	0.853	0.775
Cu (%)	1	0.08	0.01	0.1	0.06	0.11	0.05
Au (oz/t)	1	0.061	0.003	0.067	0.055	0.07	0.052
	Standard	**Mean**	**SD**	**H2SD**	**L2SD**	**H3SD**	**L3SD**
Co (%)	2	0.251	0.025	0.301	0.201	0.326	0.176
Cu (%)	2	0.03	0.01	0.05	0.01	0.06	0
Au (oz/t)	2	0.03	0.002	0.034	0.026	0.036	0.024

Any results falling outside the failure limit of +/-3 SD (standard deviation) were rejected pending investigation into the source of error. 2019 QA/QC results are summarized in Figure 11-11 to Figure 11-16.

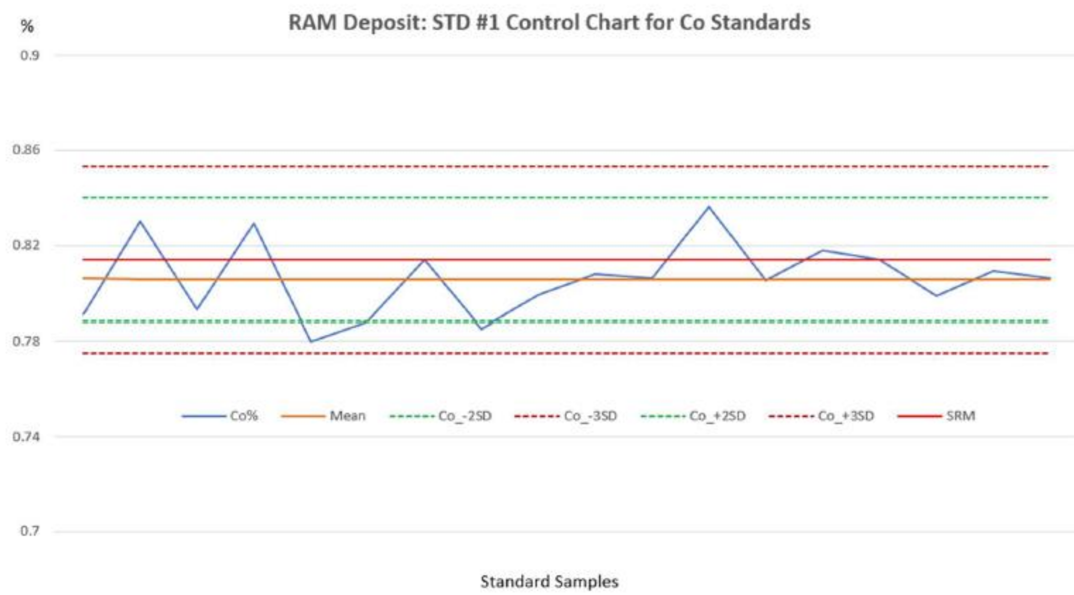


Figure 11-11: 2019 Control Chart for Co: Standard 1

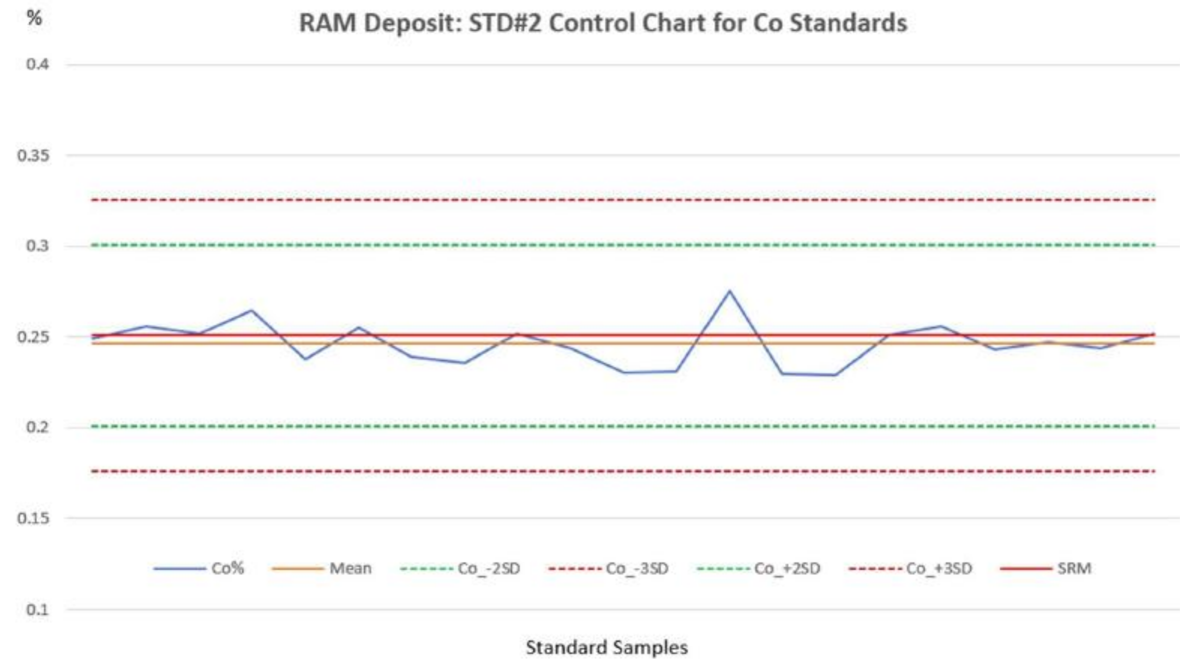


Figure 11-12: 2019 Control Chart for Co: Standard 2

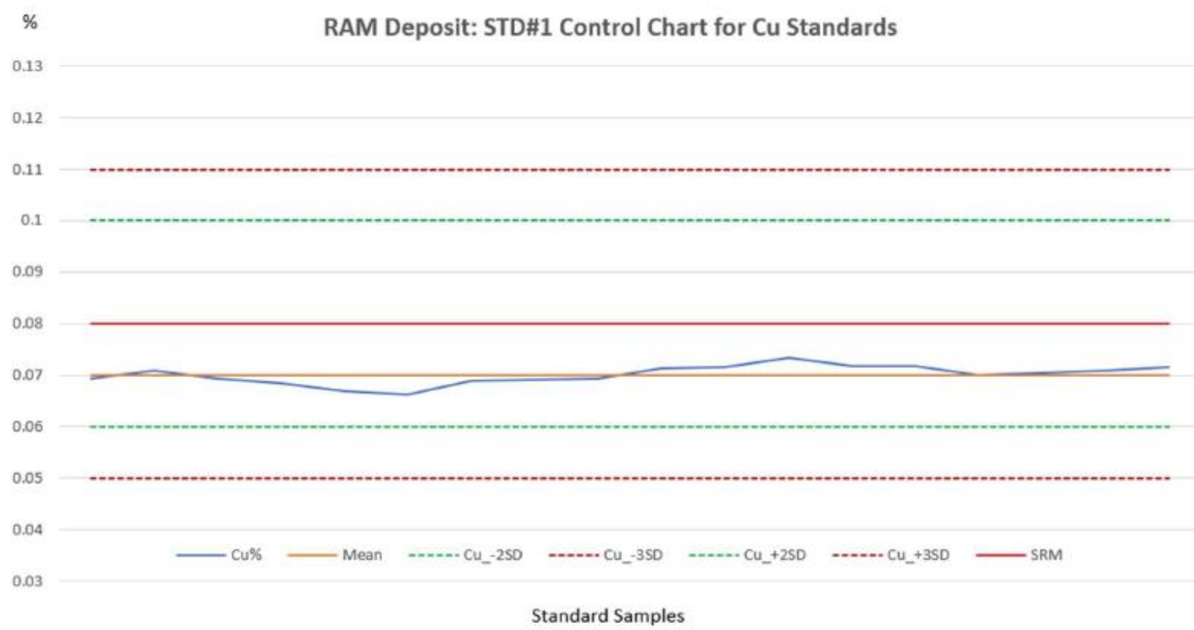


Figure 11-13: 2019 Control Chart for Cu: Standard 1

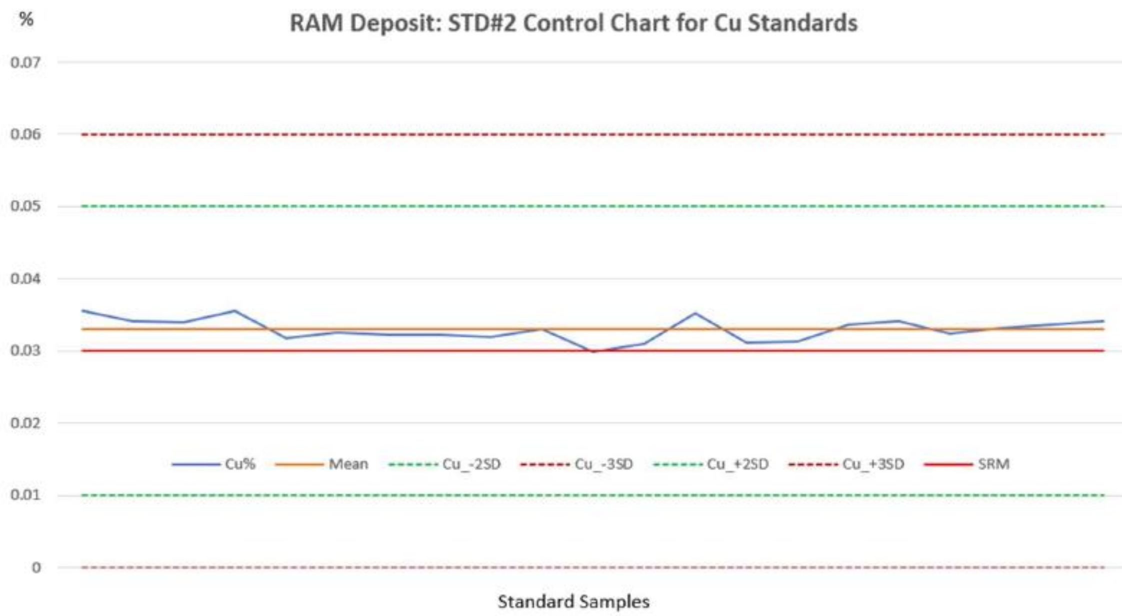


Figure 11-14: 2019 Control Chart for Cu: Standard 2

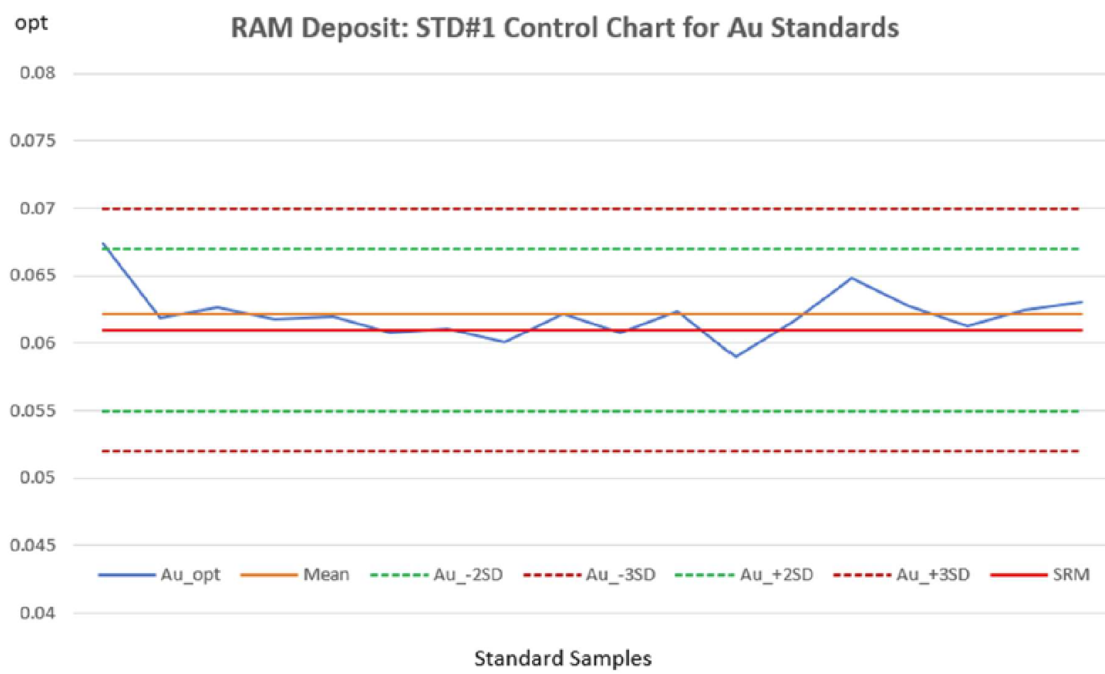


Figure 11-15: 2019 Control Chart for Au: Standard 1

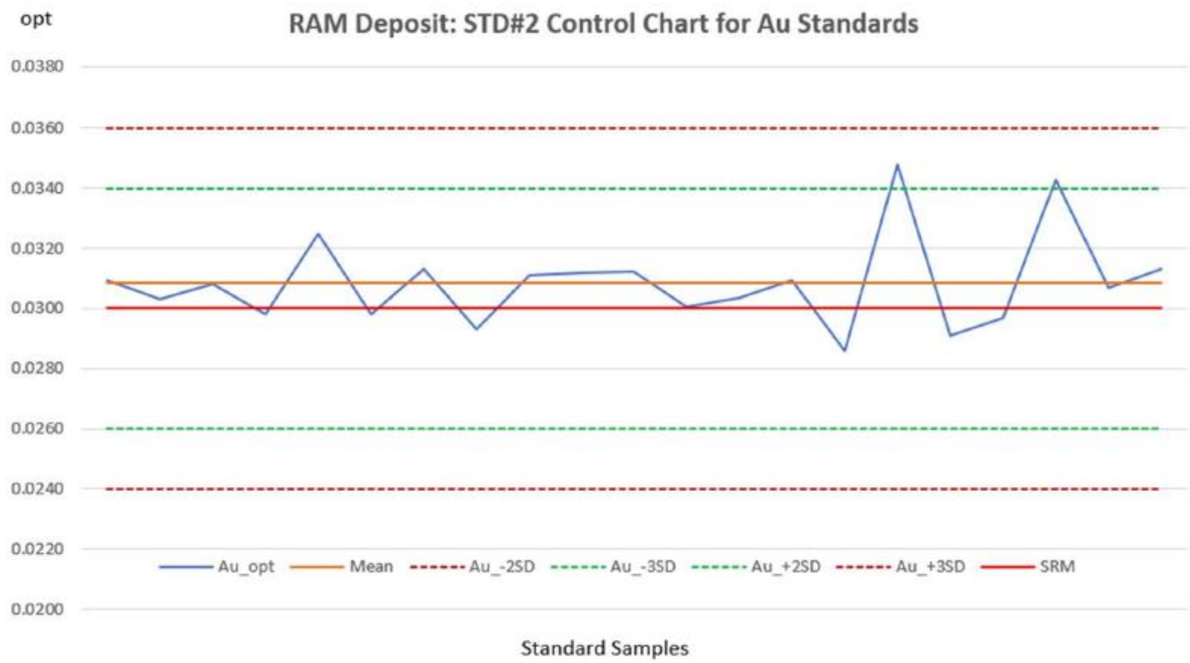


Figure 11-16: 2019 Control Chart for Au: Standard 2

11.4.3.3 Field Duplicates

Field duplicates, or quarter core samples, where taken on mineralized material outside of the main zone (hanging wall). A relatively high variation in Co values in Table 11-3 was seen between original and duplicate values. Such difference can be explained by looking at the texture in which cobaltite occurs in the rock. Cobaltite mineralization was seen as stringers, disseminated and spotty patches. Given the different textures and the uneven distribution of the mineral on any give surface, some variation is expected and explained in terms of a "nugget" effect. Therefore, for duplicate samples it was considered a failure if the difference between samples was higher than 100%. None of the samples collected, exceeded this limit.

Cu values had lesser variations, this confirming the idea of Cu belonging to a second and much later mineralization event not associated with the Co mineralization, and as the samples were primarily picked in terms of looking for cobaltite mineralization, a small range in Cu values was expected.

Table 11-3: Summary of Duplicate Samples

Parent Sample	Dup Sample	Co_ppm	Co_ppm_DUP	Cu_ppm	Cu_ppm_DUP	Co_%diff	Cu_% diff	COA
14202	14203	317	322	1210	1450	1.56	18.05	EL19235258
14330	14331	7740	3520	37	31	74.96	17.65	EL19260563
14473	14474	507	261	418	515	64.06	20.79	EL19269269
14603	14604	177	183	742	790	3.33	6.27	EL19269269
14620	14621	725	577	1055	1025	22.73	2.88	EL19274215
14458	14459	114	116	642	629	1.74	2.05	EL19275112
14660	14661	152	130	1440	1340	15.6	7.19	EL19281890

11.5 SUMMARY STATEMENT/COMMENTS

Orix and CSA auditors consider the sample preparation, security and analytical procedures to have been adequate to ensure the integrity and credibility of the analytical results used in the mineral resource estimation. Orix believes that the QA/QC aspects of the project have been adequately addressed.

12 DATA VERIFICATION

Prior to 2019, all drilling on the Ram deposit had been conducted by FCC and thus, protocols pertaining to the exploration history of the deposit, had progressive continuity. During 2019, Jervois retained the assistance of the resident geologist George King, who had been involved in all the historic drilling campaigns, in order to preserve continuity and consistency.

During the summer of 2019, Orix analyzed the state of the historic drill hole database in order to support and plan the drill program. During this analysis, several validation checks were performed outlining missing data from late drill programs. Missing data was entered, compiled, and a new current database created.

12.1 DISCUSSIONS ON GEOLOGICAL ATTRIBUTES

Discussions held with Jervois's resident geologist centred on the geological attributes of the Ram-Sunshine-Blackbird deposits including the genetic model, mineralization trends, and the role of structures lithology and alteration, lead to the current interpretation and understanding of the deposit with the following attributes for Ram specifically:

- Continuity of the mineralization in in the main zone is distinctly stratiform, less lenticular.
- Continuity of mineralization in the hanging walls is not as clear likely due to a combination of lenticular ore zones that pinch in and out, as well as soft sediment deformation highlighted by likely large slump features.
- Copper related mineralization is visibly a later event in the form of sulphide-rich veinlets/stockworks that cut-cross earlier cobaltite mineralization.
- The strong association of heavily altered chlorite/biotite horizons (“BTE”) of the Apple Creek Formation with mineralization. BTE was previously interpreted as associated with mafic sequences, however, during the drill program no such rocks were identified, and the current interpretation suggest these horizons correspond to heavily hydrothermally altered meta-argillites.
- One of the main challenges in the interpretation of the Ram geological model, is the combination of "Lithology/ Alteration" in the logs. The combination of lithology/alteration provided a general sedimentary package that defined areas well mineralized. But the combination of lithology/alteration does not define individual mineralized lenses.

12.2 DATA COLLECTION TECHNIQUES/SAMPLING

Drill core was photographed by the logging geologist after logging to include relevant distance markers, lithological contacts, alteration phases, strong mineralization, and sample intervals selected for assay analysis (Figure 11-1).

Drill hole log templates in excel, were produced before the start of the 2019 drill program by Orix, and further edited/modified versions were created with input from the resident geologist George King and Jervois' Geology Group Manager David Selfe. The log sheets are very detailed and include separate tables such as: Quicklog, Survey, Lithology, Mineralization, Alteration, Structure, Sampling, RQD and Box ends (Figure 10-2).

Sample intervals varied from 1 ft to 6 ft, with the vast majority being 2 and 3 ft long. Samples were determined based on geologic, mineralogic and alteration features. Particular attention was placed on not crossing over lithological breaks and/or major changes in mineralization/alteration.

12.2.1 Collar Coordinates

During data validation conducted by Orix, it was discovered that historic collar coordinates from selected drill holes in 2004-2006 had slight inconsistencies within the database and the values recorded on the logs. After careful interrogation a compiled survey sheet generated by local surveyors was provided by George King. The survey sheet highlighted the inconsistencies and the resident geologist identified them as planned coordinates recorded in the logs as opposed to final coordinates as seen in the compiled surveying sheet.

In the 2019 drill program, planned collar coordinates were generated in ArcGIS and holes were spotted using handheld Garmin GPS units. After the drilling was completed, Jervois hired the services of Wade Surveying to use high precision equipment to obtained final coordinates. The final coordinates were then given to Orix to add to the drill hole database.

12.2.2 Down-hole Surveys

Historic down-hole surveys were done at 150 ft intervals. According to Micon, the reliability of down-hole surveys was difficult to confirm, particularly for the deep drill holes. But ultimately no surveys were removed from the database.

In the 2019 drill program, downhole surveys were taken every 50 ft using a Reflex EX trac unit, and survey results were closely monitored by the geologist on site, following up with the drill foreman immediately after any odd result. Overall, no issues were found with the 2019 down-hole survey data.

12.2.3 Lithology

During data validation and preparing the data for geological modelling, it became apparent that some important information available in the handwritten logs and excel quicklogs was not available in the database. Two cases were found:

- Omissions of BTE/STE horizons described in the logs but entered as comments in the database, instead of entered as discrete mappable units.
- Omissions of BTE/STE horizons described in the "quicklogs" but not captured in the database.

Orix incorporated this information in order to obtain a new lithology file that could guide the 3D modelling efforts. It is, however, important to note that the combination of lithology and alteration provided a general sedimentary package that defined areas more or less well mineralized. But the combination of lithology /alteration does not define individual mineralized lenses.

12.2.4 Analysis of QA/QC Monitoring Charts

Monitoring charts on quality control samples have already been discussed in Section 11 of this report. The use of quality control samples appears to have been in line with prevailing industry standards over the drill campaign periods. Overall, Orix and CSA auditors considered the sample preparation, security and analytical procedures were adequate over the different drill campaigns to ensure the integrity and credibility of the analytical results used for mineral resource estimation.

12.2.5 Specific Gravity

Prior to 2019, specific gravity measurements were performed by geotechs in conjunction with sampling. The geologist would select an interval around 1.5-2 inches in length that was deemed generally representative of the given sample. This portion would be separated and weighted.

Measurements were taken using a triple beam balance scale. The sample was attached to the base of the weighing pan with a length of fishing line, then it was weighed while suspended in the air and subsequently weighed while suspended in a container of water. The water was tap water from the city of Salmon or a well near the Panther Creek Inn, or water pumped from a pond on the property, which was the same water used for sawing core and drilling. No measures were in place for controlling temperature of the water, and no measures were taken to prevent porous samples from taking on water during the measuring process.

The specific gravity value was computed with a handheld calculator using the following formula:

SG = (Weight in air)/ (Weight in air-Weight in Water).

Due to the inconsistencies seen in terms of the source of water used for such measurements, some concerns were raised about the validity of that historic dataset.

At the end of the 2019 drill program, in an effort to reconciliate and validate historic densities measured on site with measurements under a controlled environment in the laboratory, a total of 100 samples were selected and submitted for density analysis to the SGS laboratory in Lakefield. 50 samples from the historic holes stored in the core warehouse in Salmon, and another 50 samples from the 2019 drill program. Samples were carefully selected to be representative of the higher grade and shoulder zones in the main mineralized horizon, found on each of the 2019 holes, and a selected subset of historic holes. Samples were between 3-5 inches in length of PQ core, vacuum sealed using a commercial plastic bag sealer, the sealed bag would then be placed inside thick sample bags and closed with a zip tie. For samples that appeared fragile and/or oxidized, instructions were given to the lab to wax the samples prior to analysis. The footage and location of the samples was recorded on an excel sheet as well as individually on the core boxes, to guarantee the return of the sample at the correct position on the core box.

A comparison between historical samples measured on site with the 50 samples submitted in 2019 and analyzed by SGS, shows that the discrepancy of values is not significant (<10%) and is therefore assumed that the specific gravity data can be used for resource calculation. A historic compilation of specific gravity measurements from 813 samples is discussed in detailed in Section 14.6.

12.3 REVIEW OF PREVIOUS VERIFICATIONS/AUDITS

MDA has been an independent consultant on the Idaho project for several years dating back to 1995 and have been involved in all Jervois's previous independent mineral resource estimates and Independent Technical Reports.

12.3.1 Database Audit for the 2006 Resource

In 2005, MDA made numerous site visits to the Idaho project area during which time they reviewed and checked original assays, check assays and QA/QC procedures and results; reviewed and audited the digital database; examined geologic data and interpretations; and reviewed and re-sampled representative core intervals. Spot re-sampling produced comparable results to the original assays.

For drill data prior to the 1999 Ram drilling program, MDA checked about five percent of the sample intervals in the project database for data entry errors. No errors were found for entries of cobalt, copper, or gold values; however, the footage for one interval was entered incorrectly. Approximately 10 percent of the 1999 Ram drill data was audited, and no errors were found.

12.3.2 Database Audit for the 2015

Edwin Peralta of MDA visited the ICO site on December 10 to 12, 2014. Data verification of the 2010 drill data was completed to bring the 2012 resource estimate and block model to status as current and compliant with NI 43-101. Collar and downhole surveys were checked against original data supplied by a third-party surveyor while the assay data was digitally checked against the original assay lab data. No errors or missing data were encountered, and no changes were made to the database.

12.3.3 QA/QC for the 2006/2015 Resources

MDA reports that "Jervois's QA/QC analytical procedures including assays of check samples, standard reference material samples, and blanks, all show that the ICO assay data is reliable and verifiable and is adequate for estimating the ICO mineral resource."

12.3.4 Database Audit by CSA in 2019

CSA Global Consultants Canada Ltd ("CSA") was commissioned by Jervois to conduct an audit of the recent work completed at the ICO in Idaho. The aim was to review the processes and procedures used to collect the data, review the proposed methodology for the estimation of Mineral Resources, and audit the results. The quality of the informing data was discussed. The key points are:

- The confidence in the historical data is acceptable.
- There were initial concerns with density data. However, after compiling historic and 2019 data a conclusion was made on using a regression formula that would represent adequately density values (see more in Section 14.6).
- There are long intervals logged as Main Zone that contain only isolated and very narrow mineralized layers. These layers were reported as percentage of the horizon in the main logging files, and detailed location is provided in separated logging files (in non-easy to use files in MS Word format). Some of these intervals were manually corrected. These intervals seem to be localized in certain peripheral areas and may represent a localized risk/uncertainty on resources.
- CSA reviewed the geological and mineralization interpretation and considers it appropriate for mineral resource estimation. However, CSA observed that some non-assayed intervals and low-grade intervals were incorporated within the wireframe. This was reported to Orix who removed the non-assayed metallurgical drill holes.
- The drill program appears to have been conducted at industry standards.
- Sampling was conducted in methodical manner with 2019 similar to previous drill campaigns at ICO. Samples ranged from approximately 2 to 5 feet in length. Analytical methods appear fit for purpose using industry standard techniques such as aqua regia and four-acid digests, lithium borate and fire assay fusions and ICP and AA instrumentation.
- The database presented appears to be fit for purpose. However, the present format (Excel spreadsheets) lowers confidence in the data and there are few issues which need to be addressed to improve the data quality.
- Overall, no fatal flaws or major concerns remained.

12.4 DATABASE VALIDATION

Orix verified the database doing spot checks in excel before importing files on to Leapfrog. From leapfrog, summaries of errors and warnings were exported and fixed back again in the database. This process was repeated until no errors were highlighted on Leapfrog. Section by section checks were also made to ensure that down-the-hole surveys were making sense and that all drill hole collars conformed to the DTM.

When compared to the access database received from Jervois and authored by MDA, Orix proceeded to include information from the last drill hole programs in order to have a complete database. Orix proceeded to compile QA/QC samples not included in the access database.

12.5 DATA VERIFICATION CONCLUSIONS

Based on the verification procedures described above, Orix considers that the current database of the Ram deposit compiled during 2019 has been generated in a credible manner and has compiled all available data, is therefore suitable for use in mineral resource estimation. As highlighted previously by Micon, the lack of sampling beyond mineralized zones is a notable weakness in the database. It does not allow for the proper determination dilution grades. Notwithstanding this shortfall, Jervois' s exploration databases were professionally constructed and are sufficiently error free to support mineral resource estimates.

12.6 SITE VISIT

Scott Zelligan, P.Geo., visited the site on October 4-6, 2019. The site visit included:

- A review of selected drill intervals from the active drill program,
- A review of selected drill intervals from historical drill programs,
- A review of historical core storage facility,
- A review of procedures and facilities of the active drill program,
- A review of Property access,
- A survey of collars from the active drill program,
- A general review of the site and infrastructure.

Independent sampling was not completed. The Property has an extensive history including NI 43-101 Technical Report reviews. These reports were reviewed by the author. Additionally, base metal mineralization is visible in the core, and all historical (and current) intervals were available for review. The author reviewed these and determined independent sampling was not necessary.

All current facilities and procedures were deemed appropriate and in keeping with CIM best practices. The Property was accessed from Salmon, Idaho, via well-maintained public-access gravel roads (Figure 12-1).

Table 12-1 displays the surveyed collars by the author compared to the database value. Measurements were taken with a Garmin GPS map 60CSx gps unit and were taken in UTM NAD83 11 T coordinates. The accuracy of the unit ranges from 5-1 Om horizontally. Vertical measurements with handheld units are generally not accurate enough to be useful for verification purposes. Terrain can play a factor in accuracy as well, and the Property does display steep elevation changes. Considering these factors, the review of collars yielded quite good results, confirming the location of five of the seven drill pad locations used for the 2019 program.

Table 12-1: Collar Review Measurements (UTM NAD83 IIT)

Drill Collar	Review Survey		Database Value	
	Easting	Northing	Easting	Northing
R19-01	707,518	5,002,212	707,519	5,002,211
R19-02	707,546	5,002,222	707,547	5,002,221
R19-08	707,570	5,002,273	707,569	5,002,274
R19-11	707,541	5,002,260	707,541	5,002,261
R19-14	707,512	5,002,298	707,515	5,002,292

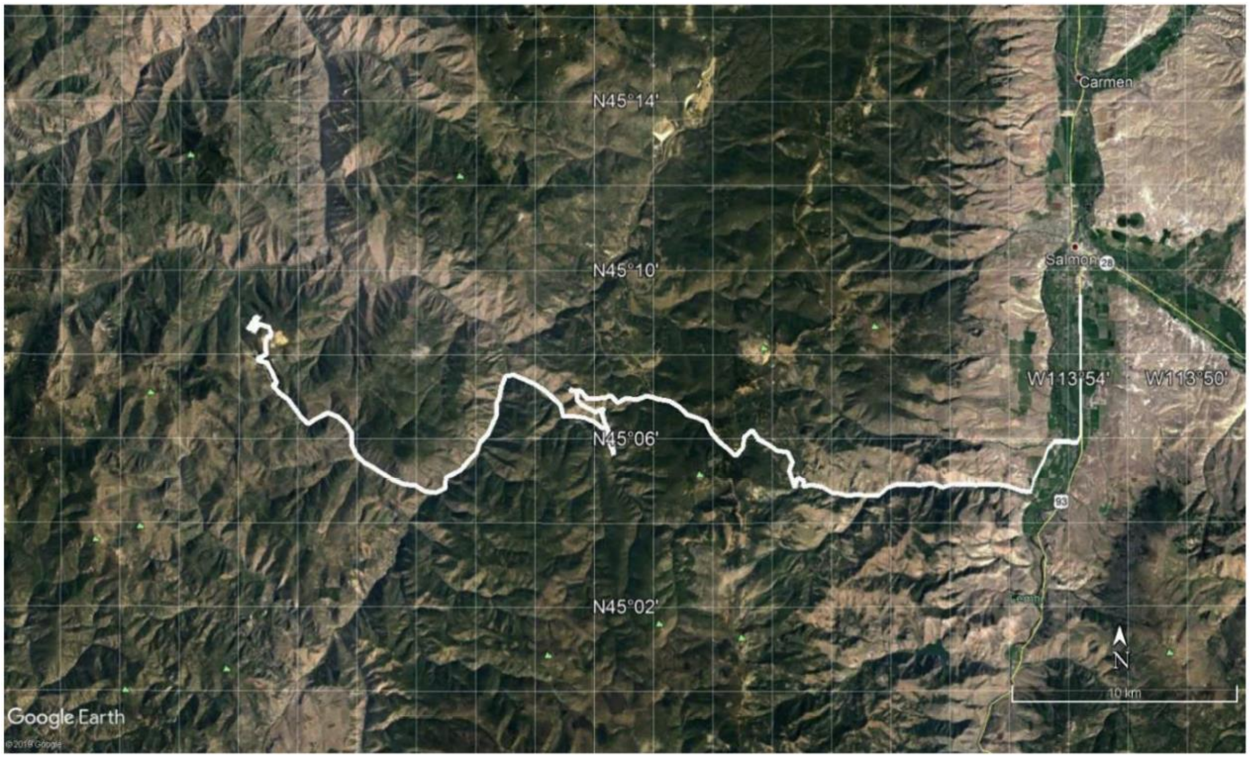


Figure 12-1: Map of Site Visit Tracks

Core from the current program was easily accessible and stored within the core-logging facility or directly outside in shipping containers. The author reviewed core from throughout the stratigraphic sequence with the supervising geologist George King. Mineralized intervals were reviewed for six holes from the current program. The lithology and mineralization observed confirms the interpretation and, in the opinion of the author, the current approach accurately portrays the observed geology.

Core from the historical programs is stored at a building in Salmon. The building is in slight disrepair, but the core storage itself is in quite good condition and very well organized. The author chose eight mineralized intervals from the main zone ("mmh") at random from four of the historical drill programs (1997, 2004, 2005, and 2006) and was easily able to find the intervals and verify the mineralization within 5 minutes for each interval.

Figure 12-2: Jervois office in Salmon Idaho

Figure 12-3: Gravel Access Road to Site

Figure 12-4: ICO Mine Office

Figure 12-5: Core-Logging Facility

Figure 12-6: Main Zone (mmh} Core from R19-13

Figure 12-7: Core Storage at Site

Figure 12-8: Active Drilling and Access Roads

Figure 12-9: Historical Core Storage in Salmon Idaho

Figure 12-10: Mineralized intersection from R04-03

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Several historical testwork campaigns and studies have been conducted for the Idaho Cobalt Operation ("ICO") deposit. The previous study in 2016/2017 focused on developing a grinding and bulk sulfide flotation process at the mine, followed by subsequent leaching of the flotation concentrate within a Cobalt Hydrometallurgical Facility to ultimately produce cobalt sulphate, copper sulphate and magnesium sulphate crystals (MICON Int Limited, 30 November 2017).

At the start of the current feasibility study in September 2019, Jervois appointed DRA to fulfil the study management and oversight role, as well as being responsible for the process engineering deliverables. The feasibility study design was originally for a 1200 stpd concentrator treating Ram deposit material, consisting of primary crushing and SAG/ball milling circuit, followed by copper-cobalt two-stage sequential flotation, concentrate and tails dewatering, paste backfill tails pumping and additional ancillary facilities. The two flotation products (copper and cobalt concentrate) were to be bagged separately for sale to offtake customers. All testwork conducted in 2019/2020 was in support of this split concentrate flowsheet.

Jervois subsequently determined it was preferable to adjust the flowsheet back to a bulk flotation process where a single combined copper/cobalt product would be bagged and sold to offtake customers, including a refinery in Brazil it purchased during the study period. All other processes were left unchanged.

13.2 HISTORICAL TESTWORK

Several historical metallurgical testwork programs comprising batch and continuous tests have been completed.

13.2.1 Noranda Blackbird Mining Company (1980's)

Preliminary bulk sample and drill composite milling and flotation testwork by Noranda's nearby Blackbird Mining Company ("BMC") in the 1980s. It is reported that BMC was successful in producing separate copper and cobalt concentrates using a sequential flotation flowsheet. On a 60% Siliceous/40% chloritic ore blend with 1.05% Co feed grade, recoveries of over 80% could be achieved at a 13% Co concentrate grade by batch flotation tests. Near-surface oxidised ore tested resulted in a drop-off of recoveries to approximately 50% (Noranda, 1982) (Noranda, 1986).

13.2.2 CAMMP (2001)

In 2001, The Center for Advanced Mineral and Metallurgical Processing ("CAMMP") used approximately 1 ton of large diameter drill core from the Ram deposit on a comprehensive milling and bulk flotation (copper and cobalt combined product) test program which also included nitrogen species-catalyzed ("NSC") leaching of the batch flotation concentrate. The optimal grind was found to be 80% passing 200 mesh (74 µm) (CAMMP, 2001). One bulk concentrate locked cycle test was conducted during this testwork which is used within this study to support the recovery model estimations.

13.2.3 SGS – Lakefield (2005)

In 2005, SGS - Lakefield (SGS-L) conducted several flowsheet development testwork programs, utilising material from both the Ram and Sunshine deposits. The testwork included detailed comminution and flotation testing as well as preliminary leach testing that confirmed CAMMP's NSC test result. After adjusting the initial program, approximately 1.5 tonnes of Ram and Sunshine deposit material was used in bulk flotation batch tests. Three locked cycle tests and three batch variability tests were conducted on Ram deposit material to produce a bulk copper/cobalt concentrate. These locked cycle tests have been used within this study to support the recovery model estimations (SGS Lakefield

Research Limited, April 2005) (SGS Lakefield Research Limited, May 2005). Samuel Engineering supervised the entire testwork programme in support of the 2007 feasibility study (Samuel Engineering Inc, 2007).

Pocock Industrial Inc. conducted solid-liquid separation tests in 2005 from material generated at SGS. This included settling/thickening and filtration studies on samples of cleaner concentrate (P80 of 88 μm) and rougher flotation tailings (P80 of 100 μm) (Pocock Industrial Inc, April 2005).

13.2.4 Mintek (2005 & 2007)

A mini Pilot Plant testwork campaign undertaken in 2005 by Mintek, South Africa, was successful in developing a basic hydrometallurgical process (Mintek, 2007).

Following batch scale tests, a full-scale pilot plant was operated at Mintek in 2007 in support of a feasibility study design by Samuel Engineering to produce high-purity cobalt metal, copper cathodes and by-product streams nickel hydroxide and magnesium sulphate (Mintek, 2007).

13.2.5 Hazen Research (2015)

In 2015 Hazen Research completed additional flotation and hydrometallurgical testwork under the direction of Samuel Engineering (Samuel Engineering, 2015). The objective of this flotation work was to investigate sequential flotation, subsequent to a bulk sulfide flotation, in order to produce separate copper and cobalt concentrates (Hazen Research Inc., 2015).

Sequential flotation, after the bulk sulfide flotation, resulted in poor selectivity possibly because of residual potassium amyl xanthate ("PAX") collectors from the bulk sulfide flotation.

Following on from this, selective copper flotation alone was investigated on one of the composites, based on the fast copper flotation kinetics observed during bulk sulfide flotation. Using low (starvation) doses of PAX, copper rougher concentrates were produced, which assayed 24 to 27% Cu and 1.1 to 1.5% Co and contained 56–71% of the copper and 5–6% of the cobalt in about 2% mass pull. Cleaner concentrates produced by reprocessing the selective copper concentrates assayed 31–32% Cu and 0.5–0.6% Co and contained, on a whole-ore basis, 22–33% of the copper and 0.8–0.9% of the cobalt in 0.6–0.9% mass pull.

In 2015, bench-scale and continuous pilot plant scale cobalt solvent extraction testwork using pregnant leach solution ("PLS") generated by Hazen were conducted. The objective of this work was to produce a clean, cobalt sulphate solution that could be fed to the crystallizers.

13.2.6 SGS – Lakefield (2016 & 2017)

In 2016 and 2017 SGS completed a program of bench-scale testwork in support of a feasibility study design, managed by Micon International Limited. This work included grinding, sequential flotation, as well as hydrometallurgical testwork (SGS Canada Inc, 2017). Two composite samples used in batch testwork resulted in the following in support of a sequential split flotation process:

- Batch rougher flotation kinetics tests showed that similar metallurgy was achievable in the copper rougher circuit regardless of primary grind size in the range tested (P80 of 58 - 86 μm). Starvation dosages of PAX collector with MIBC as a frother yielded copper rougher recoveries of 91-92% in 6-7% mass pull. Approximately 12-16% of the contained cobalt was carried into the copper rougher.
- Cobalt rougher flotation recoveries ranged from 80.4 – 85.4% for P80 of 57 - 86 μm grind sizes. Approximately 7-8% of the ore's copper reported to the cobalt rougher concentrates.

- Copper cleaner flotation concentrate grades of ~32-33% Cu were generated with copper losses of 5-11% between the rougher and the first cleaner concentrate. All tests employed a regrind stage prior to cleaning (P80 =19 µm). A second cleaner stage yielded only a marginal increase in copper concentrate grade while a further ~13% copper loss was incurred.
- Cobalt cleaner grades of 12.4% and 14.1% were realised at 79.1% and 73% cobalt recovery respectively for a feed grade of 1% Co.
- Locked cycle tests for a split concentrate flotation flowsheet were conducted on the two test composites.

13.2.7 Dundee Sustainable Technologies (2018)

In 2018, Dundee Sustainable Technologies processed initially 7 tons and then a further 5 tons of material through a bulk sulphide flotation process (rougher, cleaner scavenger circuit) in order to generate a bulk cobaltite concentrate. This was in order to demonstrate a process for the removal and stabilization of arsenic from a combined cobaltite/copper concentrate. Both samples of surface material had been oxidised to an extent resulting in lower than anticipated flotation recoveries. Pilot plant grinding targeted a P80 of 75 µm for both samples. No bulk sulphide flotation locked cycle tests were conducted during this testwork campaign (Dundee Sustainable Technologies, 2018).

Details of the samples used in historical testwork programs are tabulated below in Table 13-1.

Table 13-1: Historical Metallurgical Samples

Test Laboratory	Quantity	Sample	Description
CAMMP (2001)	Approx. 1,000 kg	Bulk Composite of 13 samples	PQ core from 1999/2000 Ram drilling. Main zones, 3021, 3022 and 3023. Met testing comp 0.57%Co, 0.29%Cu and 0.02g/t Au
SGS-L (April 2005) Head Sample	778 kg	Not specified	Ram composite, Not specified
SGS-L (April 2005) Core Composite	275 kg	Not specified	Ram Drill Core Not specified
SGS-L (April 2005) Reject Composite	155 kg	Not specified	Ram Reject Not specified
SGS-L (April 2005) Reject Composite 2	90 kg	Not specified	Ram Reject Not specified
SGS (May 2005) - Composite 1	54 kg	Assay Rejects	Ram Ore
SGS (May 2005) - Composite 2	65 kg	Assay Rejects	Ram Ore
SGS (May 2005) - Composite 3	47 kg	Assay Rejects	Ram Ore
SGS (May 2005) – Variability Sample S	26 kg	$^1/_4$ core Ram samples	Siliceous material
SGS (May 2005) – Variability Sample M	47 kg	$^1/_4$ core samples	Micaceous material
SGS (May 2005) – Variability Sample Q	50 kg	$^1/_4$ core samples	Quarzitic material
Hazen (2015)	228 kg	$^1/_4$ core samples	Four composites with grades 0.46 - 1.89%Cu and 0.44 - 0.58%Co Selective flotation comps with 0.9 & 1.20%Cu and 0.51 & 0.58%Co
SGS-L (2016 / 2017) - Composite 1 and 2	258 kg	Met Hole RMH-16-06 $^1/_2$ core samples	Two drill core composites 1.61 & 2.10%Cu and 1.03 & 0.49%Co
Dundee Sustainable Technologies (2018)	7 ton and 5 ton	Bulk Sample	Surface material

Details of the head assays for the samples used in testwork programs are tabulated below in Table 13-2. Included in the table are the composite details for the feasibility study of this report.

Table 13-2: Metallurgical Sample Head Assays

Composite / Sample	Cu [%]	Co [%]	Cu / Co Ratio	Au [g/ metric t]	As [%]	As / Co Ratio	S [%]
CAMMP (2001) Composite	0.29	0.57	0.51	0.68	-		-
SGS-L (April 2005) Head Sample	0.35	0.36	0.97	-	0.59	1.64	0.44
SGS-L (April 2005) Core Composite	0.45	0.49	0.92	-	0.63	1.29	0.94
SGS-L (April 2005) Rej Composite	0.68	0.69	0.99	-	0.91	1.32	1.22
SGS-L (April 2005) Rej Composite 2	0.43	0.42	1.02	-	0.61	1.45	0.93
SGS (May 2005) – Locked Cycle Test Composite 1	0.44	0.59	0.75	0.35	0.75	1.27	0.99
SGS (May 2005) - Locked Cycle Test Composite 2	1.1	0.72	1.53	0.69	0.98	1.36	2.07
SGS (May 2005) - Locked Cycle Test Composite 3	0.45	1.2	0.38	0.67	1.48	1.23	1.24
SGS (May 2005) – Variability Sample S	1.2	1.14	1.05	1.57	2.46	2.16	2.09
SGS (May 2005) – Variability Sample M	0.33	0.92	0.36	0.64	1.25	1.36	0.73
SGS (May 2005) – Variability Sample Q	0.7	0.41	1.71	0.60	0.57	1.39	1.40
Hazen (2015) - Composite C	1.89	0.51	3.71	0.80	0.68	1.33	2.91
Hazen (2015) - Composite A	0.46	0.44	1.05	0.60	0.74	1.68	0.90
Hazen (2015) - Composite B	0.9	0.51	1.76	0.80	0.68	1.33	1.66
Hazen (2015) - Composite D	1.2	0.58	2.07	1.00	0.87	1.50	2.03
SGS-L (2016/17Ph1) - Composite 1	1.62	0.99	1.64	0.74	1.27	1.28	3.50
SGS-L (2016/17Ph1) - Composite 2	1.99	0.47	4.23	0.71	0.58	1.23	3.14
Dundee 7t Sample (2018	0.8	0.58	1.37		1.92	3.31	
Dundee 5t Sample (2018	0.05	0.73	0.07		1.58	2.16	
SGS-L (2020 Ph1) – New Comp	1.50	0.44	3.41	-	-	-	3.04
SGS - L (2020 Ph2) – Composite HL	0.43	0.56	0.77	0.40	0.72	1.29	0.99
SGS - L (2020 Ph2) - Composite HH	1.06	0.72	1.47	0.57	0.87	1.21	2.06
SGS - L (2020 Ph2) - Composite LL	0.33	0.4	0.83	0.29	0.49	1.23	0.70
SGS - L (2020 Ph2) - Composite LH	0.66	0.43	1.53	0.36	0.52	1.21	1.26
SGS - L (2020 Ph2) - Composite AG	0.51	0.46	1.11	0.34	0.54	1.17	1.07
SGS - L (2020 Ph2) - Composite OX	0.59	0.5	1.18	1.22	1.31	2.62	1.45
SGS - L (2020 Ph3) – Bulk Comp 1	0.89	0.62	1.44	0.59	0.85	1.37	1.74
SGS- L (2020 Ph 4) - Surface Outcrop	0.04	1.45	0.23	1.28	3.04	2.10	1.74

13.3 TESTWORK FOR THE 2020 FEASIBILITY STUDY

13.3.1 Introduction

Six metallurgical test phases were conducted within the 2019/2020 study in support of the design for material from the ICO Ram deposit. Most of the testwork was conducted at SGS facilities. All testwork conducted for the 2020 FS was in support of a split concentrate flowsheet, where copper was activated with starvation dosages of collector and recovered first, prior to a cobalt flotation using PAX collector. The two flotation concentrates were then dewatered and bagged separately.

Following financial analysis of this flowsheet, it was subsequently decided to revise the blockflow in order to generate a single, bulk (combined copper and cobalt) concentrate. The lock cycle tests of the CAMMP and SGS – 2005 testwork campaigns are utilised in order to estimate the copper and cobalt recoveries in a single bulk concentrate.

The testwork for the 2020 feasibility flowsheet, where applicable to the bulk concentrate flowsheet is described below.

13.3.1.1 Phase 1 – Preliminary Flowsheet Development

Two, previously crushed, composites of drill core material left over from the previous testwork conducted at SGS in 2016/2017 were used in order to develop a Preliminary Economic Assessment ("PEA") level metallurgical flowsheet (SGS Canada Inc, March 2020). The phase 1 program consisted of:

- Head characterisation analysis and mineralogy
- Grind calibration tests
- Wet High-Intensity Magnetic Separation ("WHIMS") testwork on cobalt rougher concentrate for pyrite/cobalt separation evaluation
- Heavy Liquid Separation ("HLS") testwork to evaluate the potential for dense media pre-concentration
- Gravity separation (Knelson and Mozley) was tested in both scavenging of Cobalt Rougher tails as well as upgrading Cobalt Cleaner Scavenger concentrate.
- Twenty-one batch flotation tests on two composites, one locked cycle test and a Qemscan analysis of the lock cycle test products in support of the split concentrate flowsheet were conducted. In addition, further batch flotation testwork was also completed in order to generate greater quantities of concentrate via nine 10 kg batch flotation tests. Although this flotation testwork has contributed greatly to further understanding of the general characteristics of the ore body, they are no longer applicable in the development of the single bulk concentrate flowsheet.

13.3.1.2 Phase 2 – Flowsheet Detailed Development

Fresh drill core material comprising of 94 intervals (approx. 1 tonne), from the 2019 drilling campaign was used at SGS to verify the flowsheet developed in Phase 1 on composites with various Cu:Co ratios, ore grades, and different degrees of sample oxidation (SGS Canada Inc, 2020). The phase 2 program consisted of:

- Head grade characterisation analysis of the drill cores material which consisted of chlorite, silicified, biotite-chlorite, biotite-silicified and separate oxidised alterations.
- Five composites were made using both interstitial and shoulder material for varied copper and cobalt feed grades, as well as one separate, additional oxidised ore composite for use at SGS for concentrator testwork.
- Semi-quantitative X-Ray Diffraction Analysis ("XRD"), as well as Qemscan analysis, was conducted on the composites.
- Three separate grinding composites (biotite chlorite, biotite silicified, and oxide) were delivered to Grinding Solutions Mineral Processing Services for physical characterisation testwork. The two non-oxide composites were also used at SGS for additional comminution grind indices testwork.
- Grind time calibration testing on both laboratory scale rod mill and ball mills was conducted to support the batch and locked cycle flotation testwork.
- Flocculant screening tests on combined plant tailings was conducted, followed by static and dynamic thickener tests and finally, vacuum filtration testwork.
- Compaction and environmental tests were conducted in support of the Tailings Waste Facility ("TWSF") design.
- Forty-one (41) batch flotation tests were conducted to optimise the split concentrate flowsheet. This included optimising grind sizes, reagents, flotation times and energy, while simultaneously considering the impact of varied pulp densities, recycle water and oxidised material. In addition, three locked cycle tests were conducted

on the optimised, split concentrate plant flowsheet utilising fresh mine sourced water. The locked cycle test products were assayed and submitted for Qemscan analysis. Although this flotation testwork has contributed greatly to further understanding of the general characteristics of the ore body, they are no longer applicable in the development of the single bulk concentrate flowsheet.

13.3.1.3 Phase 3 - Batch Test Concentrate Generation

Material not utilised in the Phase 2 composites plus additional drill core material were combined and used for concentrate production via twenty-seven 12 kg batch flotation, spit concentrate tests. Although the focus of these tests was to generate more concentrate material for offtake agreement analysis, each test allowed for additional optimisation and testwork to be conducted in support of the split concentrate flowsheet. In total, 335 kg of the composite material was treated through batch flotation tests in order to generate 6.6 kg of copper concentrate (32.4% Cu) and 11.8 kg of cobalt concentrate (11.9% Co) (SGS Canada Inc, 2020).

13.3.1.4 Phase 4 - Pilot Plant Concentrate Generation

Surface outcrop material with a very low copper grade from the ICO deposit was shipped to SGS for continuous pilot-plant scale flotation in order to generate more concentrate material for marketing analysis. Although the focus of this phase was to generate a representative cobalt concentrate and not metallurgical testing, the operation of a continuous bulk flotation pilot plant at SGS provided useful information for metallurgical plant design. The low copper feed grade allowed the pilot plant to operate with a bulk sulphide flotation flowsheet and still generate an acceptable cobalt concentrate quality. In total, 3.1 tonnes of material were treated to generate 156 kg of cobalt concentrate.

13.3.1.5 Phase 5 – Xanthate Destruction

High level clarified water treatment options were investigated with a view to reducing the quantity of residual bulk sulphide collector reagent in the recirculated water to the milling process. This included measuring the destruction of the xanthate via water ageing, Ultra-Violet light irradiation, carbon capture and ozone oxidation. The ozone destruction method was then utilised in batch flotation tests to demonstrate its effectiveness successfully (SGS Canada Inc, 2020). This testwork is no longer of relevance in the development of the single bulk concentrate flowsheet.

13.3.1.6 Phase 6 - Gold Deportment

The results of the locked cycle tests in Phase 2 indicated that gold reported predominantly to the cobalt concentrate within the sequential flotation flowsheet. Subsequent testwork was conducted to consider the gold mineralogy, the efficacy of gravity separation at varied points within the flowsheet, alternative copper flotation collector reagents and gold diagnostic leach tests on the cobalt concentrate.

The results of this work concluded that gravity separation was not economically viable in any of the locations considered within the split or bulk concentrate flowsheets.

13.3.2 Sample Selection for the 2020 Feasibility Study Testwork

Ores present in the Ram deposit comprise of siliceous and chloritic ore types with typically low total sulfide contents. The main cobalt ore mineral is cobaltite (Co, Fe) AsS while the main copper mineral is chalcopyrite ($CuFeS_2$). Oxidation is present in exposed ores where the arsenic: cobalt ratio is elevated. According to the project's resource model, oxide ores make up only a very small portion of the Ram orebody, and ICO advised that the oxide ore type will not exceed 15% of the total proportion of the plant feed. Low-grade gold is present throughout the ore body.

13.3.2.1 Phase 1

Phase 1 material was leftover from the previous testwork conducted at SGS, utilising drill core RMH16-06.

13.3.2.2 Phase 2

For phase 2, seven drill cores were initially drilled for metallurgical testing (R19 – 01 through 07). Upon further evaluation, the R19-01 drill core section was too highly oxidised for realistic flotation testwork as its cobalt grades were below or close to the project's cut-off grades. This drill core was then utilised as the worst-case oxide material for comminution testwork. An additional three drill cores (R19 – 17, 18 & 19) were subsequently provided for additional material that could be used in developing a realistic oxide composite that would be representative of plant feed processed within the split-concentrate flowsheet.

Drill core sections provided for testwork comprised of interstitial and shoulder material from the resource of the Ram ore body. The locations of the drill cores within the ore body are depicted below in Figure 13-1.

13.3.2.3 Phase 3

The Bulk Comp 1 composite material utilised for all testwork in this phase was derived from the following material:

- Six high-grade drill core intervals not utilised in Phase 2. These came from drill holes R19-05, 06, and 07
- Thirty drill core intervals delivered separately to SGS. The drill holes from which this material was derived were R19 – 08, 09, 10, 11, 12, 13, and 15
- Material composite not utilised in phase 2 comprising of HH, LH, and LL composites

The additional drill core locations are depicted below in Figure 13-5 and Figure 13-6.

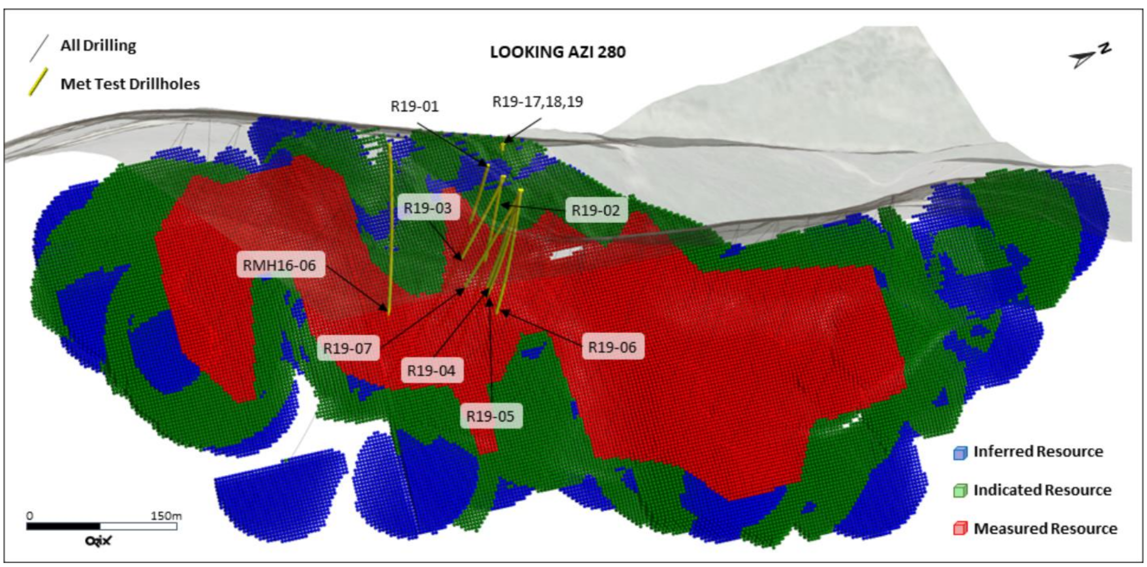


Figure 13-1: Phase 1 and 2 Drill Core Location in Ram Ore Resource
Source: Orix Geoscience Inc.

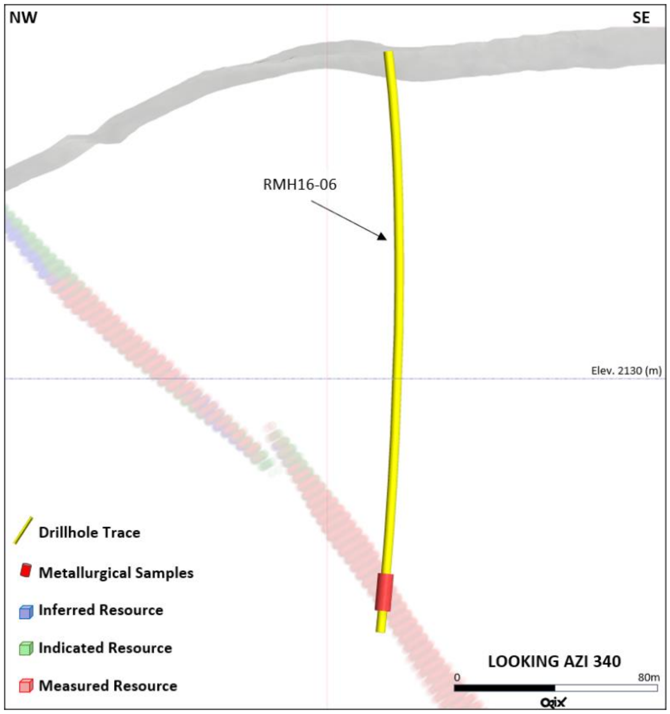


Figure 13-2: Phase 1 Drill Core Metallurgical Sample Cross Sections

Source: Orix Geoscience Inc.

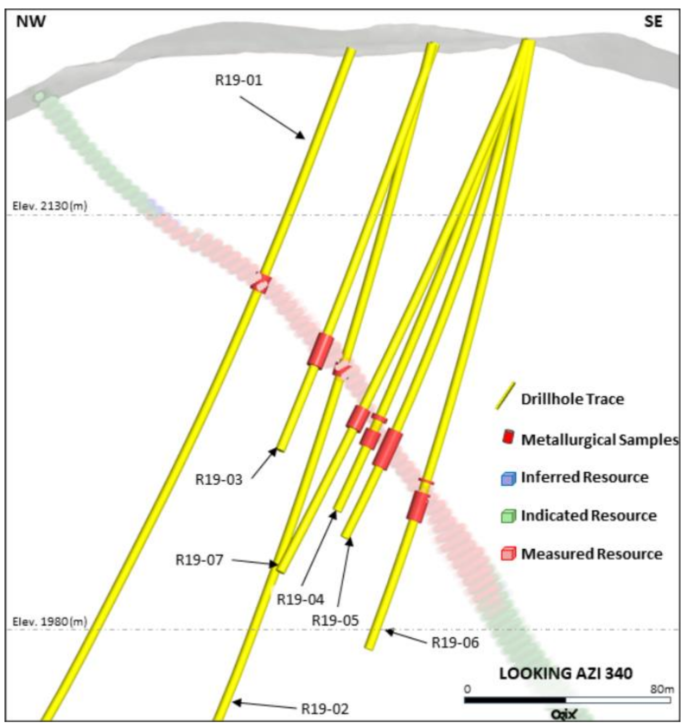


Figure 13-3: Phase 2 Drill Core Metallurgical Sample Cross Sections

Source: Orix Geoscience Inc.

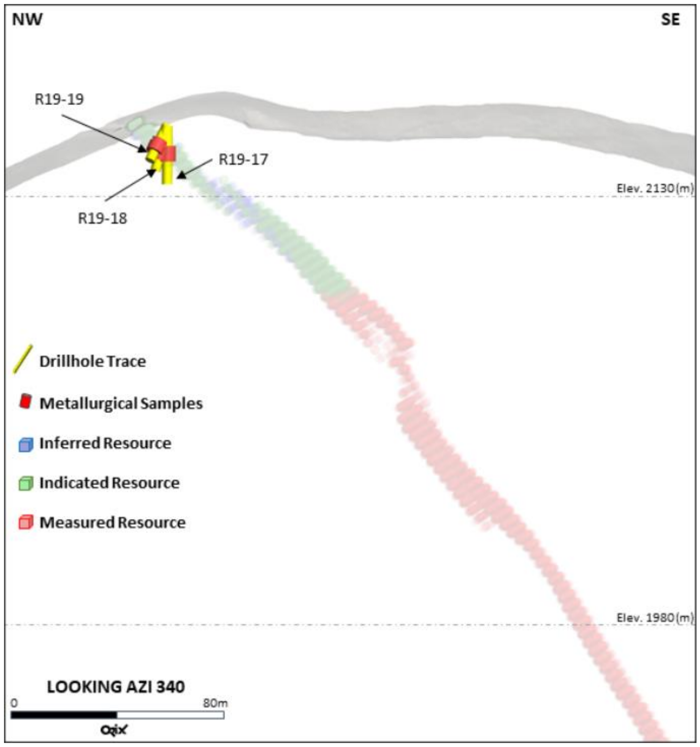


Figure 13-4: Phase 1 and 2 Drill Core Metallurgical Sample Cross Sections

Source: Orix Geoscience Inc.

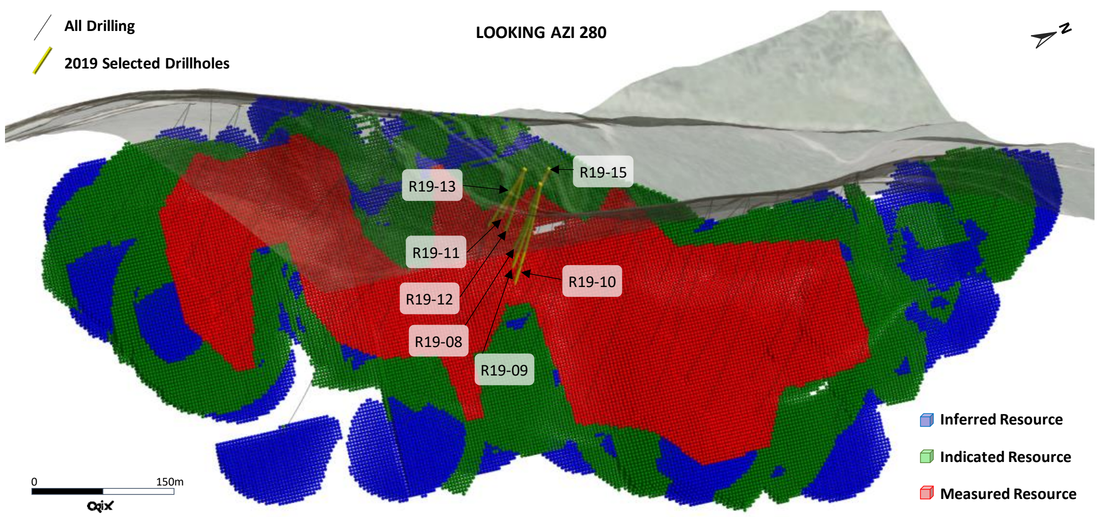


Figure 13-5: Additional Phase 3 Drill Core Location in the Ram Resource

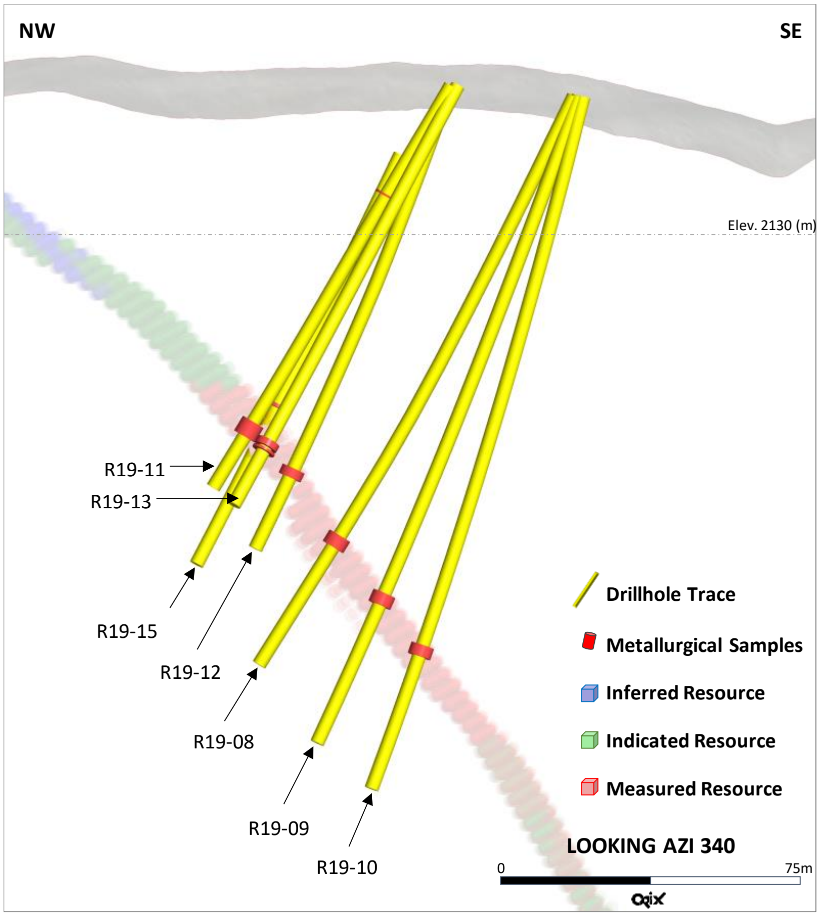


Figure 13-6: Additional Phase 3 Drill Core Cross Sections

Source: Orix Geoscience Inc.

13.3.3 Sample Selection for the 2007 Feasibility Study Testwork

All testwork conducted by SGS in 2005 was supervised and detailed by Samuel Engineering within the 2007 feasibility study (Samuel Engineering Inc, 2007). A brief summary overview of this is described below:

- CAMMP pre-feasibility testwork completed flotation and pressure leaching testwork on core samples that were collected during the 1999 to 2000 drilling of the Ram deposit derived from drill cores MH00-02 through MH00-07
- SGS Lakefield testwork in 2005 conducted three locked cycle tests from material derived from drill cores R04-01, 03, 05, 07, 08, 10, 12 through 21

The location of these drill cores in respect to the ore body is depicted below in Figure 13-7 and Figure 13-8.

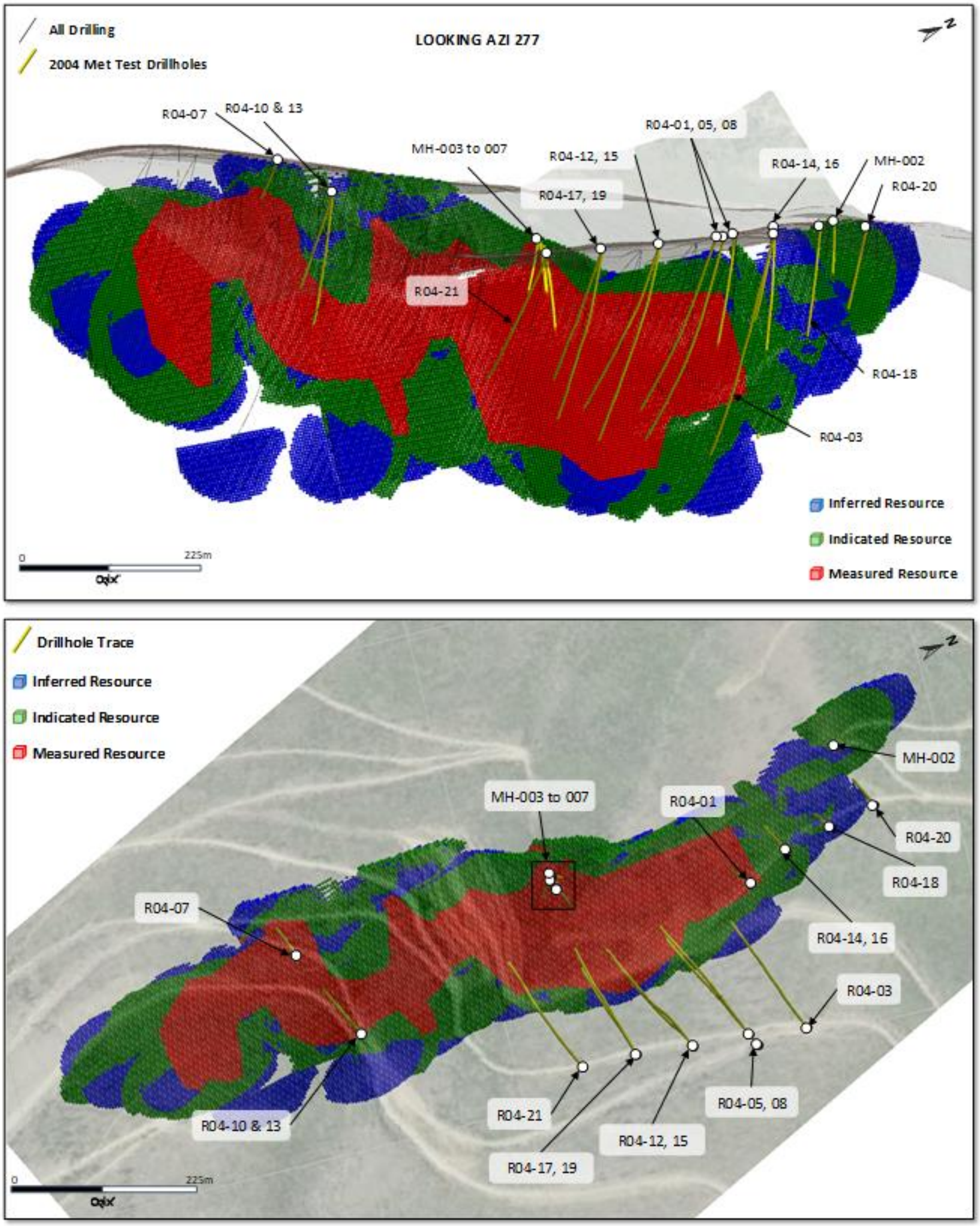


Figure 13-7: CAMMP and 2007 Feasibility Study Drill Core Location in the Ram Resource

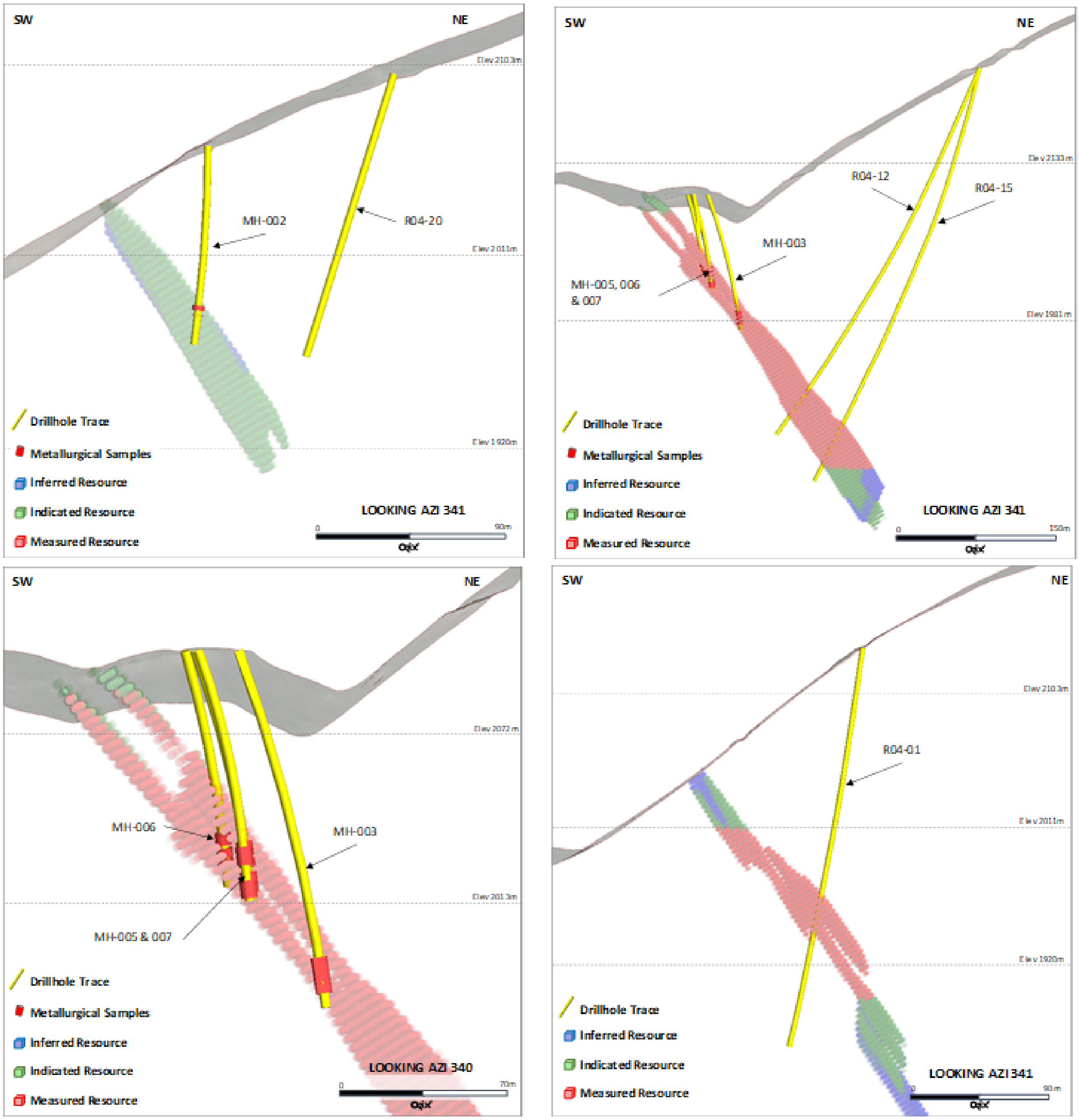


Figure 13-8: CAMMP and 2007 Feasibility Study Drill Core Cross Sections
Source: Orix Geoscience Inc.

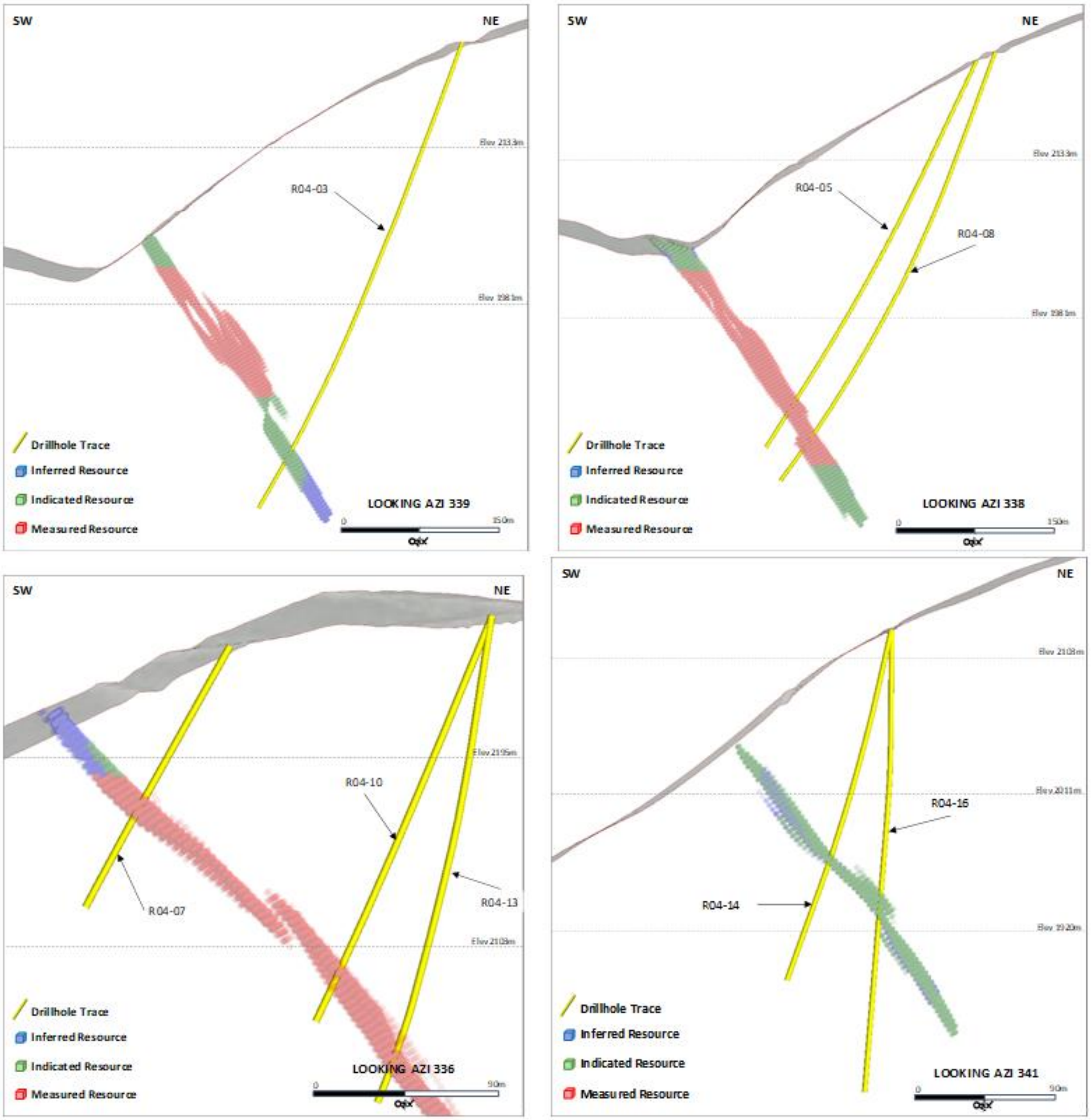


Figure 13-8: CAMMP and 2007 Feasibility Study Drill Core Cross Sections
Source: Orix Geoscience Inc.

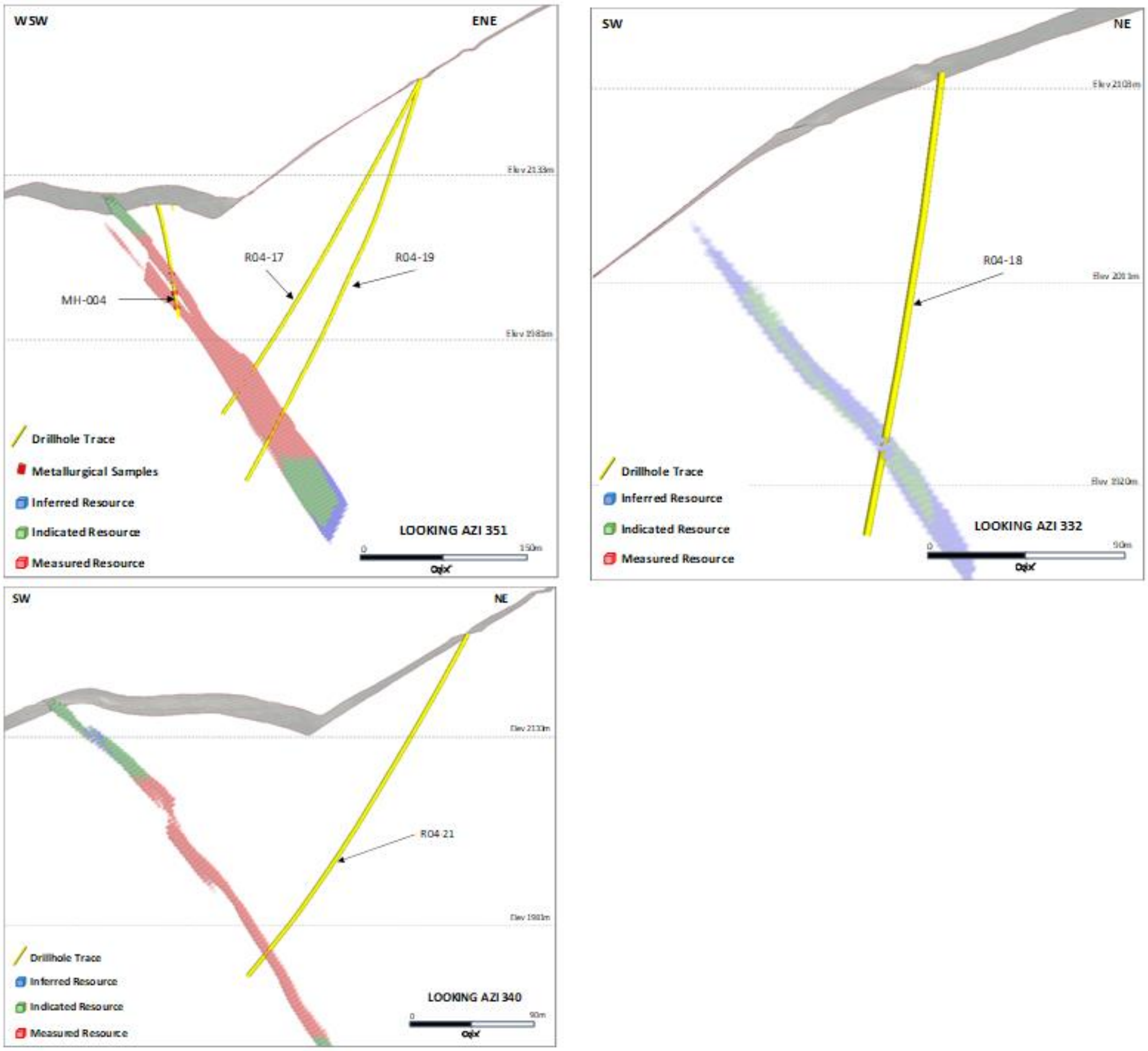


Figure 13-8: CAMMP and 2007 Feasibility Study Drill Core Cross Sections
Source: Orix Geoscience Inc.

13.3.4 Composite

13.3.4.1 Phase 1 Composites

Phase 1 material was left over from the previous testwork conducted at SGS, utilising drill core RMH16-06 from 657' to 699.4' in-depth. Mineralization and alteration information is not available. Approximately 60 kg of Composite 1 and 80 kg of Composite 2, had been stage crushed to 1.7 mm and stored in a freezer in order to minimise degradation. This material was used to develop the feasibility study flowsheet while awaiting the arrival of the freshly drilled core material. Once most of the Composite 1 material had been utilised, all remaining material, together with an additional 60 kg SMC comminution test rejects were combined into a composite (New Comp) for additional batch flotation, to produce representative concentrate for offtake analysis. Heavy Liquid Separation (HLS) testwork was also conducted on this new composite material.

The composite assays are tabulated in Table 13-2. Copper grades of 1.5 to 1.99% Cu within the composites were relatively high, resulting in very high Copper: Cobalt ratios (1.64 to 4.23), particularly of Composite 2 and New Comp, in comparison with the resource Life of Mine ("LOM") estimates. However, the purpose of this phase of the project was to develop an initial flowsheet and flotation conditions which would be optimised in phase 2.

13.3.4.2 Phase 2 Composites

Orix Geoscience Inc. Drill cores supervised the generation and tagging of the drill cores for the phase 2 testwork from the main zone intervals were placed in vacuum bags and sealed on site. Each vacuum-sealed bag was placed into a heavy-duty plastic bag with the sample identity tag. The heavy-duty bag was then labelled with sample identity, hole identity and interval. The heavy-duty plastic bag was then closed with zip ties and transported to the Salmon warehouse and placed in a freezer. No nitrogen was used during storage.

On arrival at SGS, fifty-five interstitial and shoulder intervals were cut to quarter and half cores. One quarter was submitted for geochemical analysis, while the half was used in the development of four composites for batch flotation purposes. The remaining quarter was returned to Salmon for storage. A further twenty-eight intervals were stage crushed to 1.7 mm, assayed and then used in the development of the four composites for batch flotation purposes. The four composites represented the realistically expected variation within the ore body in terms of copper and cobalt grades and ratios which the concentrator would be expected to process. Some material not used in the batch flotation composites was stored and utilised in Phase 3 of the project for additional product generation. The batch flotation composite drill core information is tabulated in Table 13-3 below.

The names of the composites were selected based on:

- The composite cobalt grade (High grade or Low grade)
- The Cu:Co ratio (High ratio or Low ratio)

Table 13-3: Phase 2 Flotation Composite Drill Core Information

Composite	Drill Core ID	Combined Length [ft]			Actual Mass	Mineralization	Alteration
		From [ft]	To [ft]	Length [ft]	kg		
High-Grade Low Ratio (HL)	R19-03	420	423	3	14.5	Cob	Biotite
	R19-05	524.3	525.9	1.6	8	Cpy, Po, Cob	Biotite, Chlorite
	R19-05	525.9	528.5	2.6	12.5	Cpy, Po, Cob	Biotite, Chlorite
	R19-06	571	573.5	2.5	13	Po, Cpy, Cob	Biotite, Chlorite
	R19-07	508	510.1	2.1	10.5	Cpy, Py, Po	Biotite Chlorite
	R19-07	515.3	518	2.7	14	Cpy, Py, Cob	Biotite Silicified
	R19-07	518	520.9	2.9	14.5	Cpy, Py, Cob	Qtz - Biotite
	R19-07	520.9	523	2.1	10	Cpy, Py	Biotite Silicified
	R19-03	404	406.5	2.5	6.5	Garnet	Chlorite
	R19-04	510	512	2	4.5	Garnet	Biotite-Chlorite
	R19-05	511	513.5	2.5	5.75	Garnet	Biotite, Chlorite
	R19-05	530.9	533	2.1	4.75	Garnet, Chloritoid	Biotite, Silicified
	R19-05	533	536	3	7	Garnet, Chloritoid	Biotite
	R19-05	536	540.5	4.5	10	Garnet, Chloritoid	Silicified

Composite	Drill Core ID	Combined Length [ft]			Actual Mass	Mineralization	Alteration
		From [ft]	To [ft]	Length [ft]	kg		
	R19-05	540.5	544.5	4	10.25	Garnet, Chloritoid	Chlorite, Qtz
	R19-06	581	584.3	3.3	6.75	Garnet	Silicified
	R19-06	584.3	587.1	2.8	7.25	Garnet	Chlorite
	R19-06	587.1	589	1.9	4.75	Garnet	Chlorite
	R19-06	589	591.5	2.5	6.5	Garnet	Chlorite
	R19-06	591.5	595	3.5	7.75	Garnet	Clay Silicified
	R19-06	595	597.5	2.5	5.75	Garnet	Clay Silicified
High-Grade High Ratio (HH)	R19-05	522	524.3	2.3	12	Cpy, Po, Cob	Biotite, Chlorite
	R19-05	528.5	530.9	2.4	11.5	Cpy, Po	Biotite, Chlorite
	R19-06	566.1	568	1.9	8.5	Po, Cpy, Cob	Biotite, Chlorite
	R19-06	568	571	3	15.5	Cpy, Po, Cob	Biotite, Chlorite
	R19-07	512.2	515.3	3.1	15.5	Cpy, Po, Py, Cob	Biotite
	R19-03	412	416	4	8.5	Garnet	Biotite
	R19-03	416	418	2	5.25	Garnet	Chlorite
Low-Grade Low Ratio (LL)	R19-04	512	515	3	7	Garnet	Silicified
	R19-04	515	517.8	2.8	6.5	Garnet	Silicified
	R19-04	517.8	520	2.2	5.25	Py	Chlorite
	R19-05	544.5	548	3.5	7.75	Garnet, Chloritoid	Biotite
	R19-06	546	549	3	7	Garnet	Biotite Silicified
	R19-06	573.5	575.5	2	5	Garnet	Silicified
Low-Grade High Ratio (LH)	R19-04	520	522	2	4.25	Py	Chlorite
	R19-04	522	524.5	2.5	6	Py	Chlorite
	R19-04	524.5	528	3.5	7.75	Garnet	Biotite
	R19-06	575.5	578	2.5	5.25	Garnet	Silicified
	R19-06	578	581	3	7	Garnet	Silicified

Halfway through the batch flotation testwork, the two low-grade composites were combined to form an Average Grade ("AG") composite following advice that the mining method had become more selective and the cobalt feed grades would increase over the life of mine. The composite assays are tabulated in Table 13-2.

13.3.4.3 Phase 2 Oxide Composite

The oxide material composite was selected from the drill cores that contained a cobalt grade higher than the cut-off grade, while simultaneously having a high As:Co ratio. The batch flotation oxide composite drill core information is tabulated in Table 13-4 below.

Table 13-4: Phase 2 Flotation Oxide Composite Drill Core Information

Drill Core ID	Combined Length [ft]			Act Mass	As %	Cu %	Co %	S %	As: Co	Mineralogy	Alteration From
	From	To	Length	kg							
R19-02	403	405.6	2.6	11.1	0.8	0.1	0.3	0.2	2.4	Cob, Cpy	Silicified
R19-03	393	395.5	2.5	12.7	1.6	1.5	1.0	3.4	1.6	Cob, Cpy	Chlorite
R19-03	398	401	3.0	12.1	2.1	0.5	0.2	0.9	10.5	Missing	Oxidised
R19-16	268	270.4	2.4	5.5	0.6	0.1	0.2	0.3	2.8	Missing	Missing
R19-17	29	31	2.0	3.7	1.0	0.0	0.2	0.1	4.7	Missing	Missing

The oxide composite assay is tabulated in Table 13-2.

13.3.4.4 Phase 2 Grind Composites

For comminution testwork, eleven intervals were combined to form two 30 kg composites providing both a hard silicified and a softer chloritic alteration composite. These two composites were split approximately in half and were used at both SGS - Lakefield and Grinding Solutions Mineral Processing Services. In addition, heavily oxidised material from the R19-01 drill core was also shipped for comminution testwork at Grinding Solutions. The grind composite information is tabulated in Table 13-5.

Table 13-5: Phase 2 Grind Composites Drill Core Information

Grind Composite	Drill Core	Combined Length			Estimated Mass [kg]	Mineralization	Alteration
		From [ft]	To [ft]	Length [ft]			
Grind Comp 1	R19-04	493	506	13.5	58.4	Cpy, Py, Garnet, Cob	Biotite - Chlorite
Grind Comp 2	R19-06	549	562.1	13.1	56.7	Cpy, Cpy Tr, Py, Po, Garnet, Cob	Biotite-Silicified
Grind - Oxidised	R19-01	292.3	313.8	21.5	46.49		Oxidised

In overseeing the selection of the composites, DRA was limited on the availability of sample for metallurgical testing due to the narrow vein nature of the deposit. The fact that variable copper/cobalt grades and copper/cobalt ratios have been tested across the four fresh samples plus other composites is important with respect to predicting the concentrator recoveries in relation with the LOM plan.

13.3.4.5 Phase 3 Bulk Composite

Tabulated below in Table 13-6 is the additional drill core selections delivered to SGS for bulk concentrate production, which were above the 0.1% Co grade.

High-grade drill core material not utilised in developing phase 2 composites was also used in developing the bulk composite for phase 3.

Table 13-6: Additional Drill Core Material Provided for Phase 3*

Drill Hole ID	Combined Length [ft]			As [%]	Co [%]	Cu [%]	Mineralogy
	From [ft]	To [ft]	Length [ft]				
R19-08	415.2	426.2	11.0	0.7	0.5	0.5	Cpy, Co
R19-09	445.3	455.7	10.4	1.1	0.8	0.5	Py, Co
R19-10	477.0	486.5	9.5	0.5	0.4	0.6	Garnet, Co, Marcasite
R19-11	324.8	339.0	14.2	1.5	0.9	0.2	Co, Garnet, arsenate
R19-11	105.2	107.0	1.8	0.9	0.2	0.1	Garnet, Co
R19-12	345.0	353.0	8.0	1.1	0.6	0.3	Co, arsenate
R19-13	341.0	351.1	10.1	1.5	0.8	0.1	Co, Garnet, arsenate
R19-15	233.0	236.0	3.0	1.2	0.1	0.1	Arsenate, Cpy

* *Alteration not available*

Table 13-7: High-Grade Drill Core from Phase 2

Hole ID	Combined Length [ft]			As [%]	Co [%]	Cu [%]
	From [ft]	To [ft]	Length [ft]			
R19-05	513.5	515.7	2.2	1.5	1.0	2.5
R19-05	515.7	518.0	2.3	1.1	0.8	2.9
R19-05	518.0	520.1	2.1	0.6	0.5	2.0
R19-05	520.1	522.0	1.9	1.4	1.0	2.9
R19-06	564.1	566.1	2.0	1.4	1.1	3.9
R19-07	510.1	512.2	2.1	1.5	1.0	2.9

The total masses used to make up the Phase 3 bulk composite are tabulated below Table 13-8.

Table 13-8: Phase 3 Bulk Composite Mass Composite

Sample ID	Weight [kg]
Additional Drill Core Delivered	147
Phase 2 High-Grade Leftover Intervals	58.4
Phase 2 HH Comp	52.6
Phase 2 LL Comp	34.5
Phase 2 LH Comp	42.6
Bulk Comp 1	**335.1**

13.3.4.6 Phase 4 Bulk Sample

Approximately 3.1 tonnes of material were delivered separately to SGS for the purpose of generating additional cobalt concentrate. The samples' assay analysis summary results are tabulated in Table 13-2. The material was obtained by ICO from the original discovery surface outcrop material. Although it had elevated Arsenic: Cobalt ratios indicating it had been oxidised, the cobalt grade was still high. Furthermore, as the copper grade was almost negligible, a bulk flotation flowsheet (i.e. cobalt flotation only) was able to generate a cobalt concentrate which did not contain high grades of copper.

13.3.4.7 CAMMP Pre-Feasibility 1999/2000

The list of samples collected from PQ-core drilling utilised in the CAMMP pre-feasibility study are tabulated below in Table 13-9.

Table 13-9: CAMMP PQ Drill Core Samples (1999-2000)

Hole	Section	Horizon	Feet	% Co	% Cu	oz Au/ton	oz Ag/ton
MH00-02	1400 N	3023	6.5	0.104	0.308	0.016	0.06
MH00-03	200 N	3022	24.7	0.162	0.028	0.014	0.14
MH00-03	200 N	3023	7.1	0.851	0.037	0.038	0.12
MH00-04	200 N	3021	7.9	0.482	0.074	0.018	0.08
MH00-04	200 N	3022	5.5	2.130	0.050	0.032	0.13
MH00-04	200 N	3023	8.6	0.284	0.074	0.016	0.10
MH00-05	200 N	3021	16.2	0.364	0.360	0.018	0.14
MH00-05	200 N	3022	9.6	0.109	0.366	0.016	0.18
MH00-05	200 N	3023	23.2	0.834	0.389	0.018	0.14
MH00-06	200 N	3022	9.5	0.932	0.286	0.044	0.16
MH00-06	200 N	3023	6.5	0.494	1.630	0.012	0.16
MH00-07	200 N	3022	21.2	0.123	0.123	0.046	0.12
MH00-07	200 N	3023	18.3	1.010	0.285	0.048	0.12
Average			164.8	0.535	0.267	0.027	0.13

13.3.4.8 Feasibility Study 2007

The list of samples collected for SGS locked cycle testwork in 2005 and utilised in the 2007 Feasibility study by Samuels Engineering are tabulated below in Table 13-10, Table 13-11, and Table 13-12.

Table 13-10: Feasibility Study Drill Core Samples (2007) – Composite 1

Hole ID	Sample ID	From	To	Feet	Percent Siliceous	Percent Micaceous	Per cent Quartzite
R04-01	5907	429	431	2	0	50	50
R04-01	5909	431	433.2	2.2	0	96	4
R04-01	5923	561	563	2	10	57	33
R04-01	5924	563	566	3	9	15	76
R04-01	5925	566	568	2	12	19	69
R04-01	5926	568	570.1	2.1	7	27	66
R04-01	5928	570.1	572.8	2.7	3	17	80
R04-01	5929	572.8	574.9	2.1	21	32	47
R04-01	5930	574.9	577.2	2.3	16	49	35
R04-01	5949	633.5	637.1	3.6	85	15	0
R04-01	5963	809.5	811.5	2	23	0	77
R04-01	5965	811.5	813.8	2.3	0	1	99

Hole ID	Sample ID	From	To	Feet	Percent Siliceous	Percent Micaceous	Per cent Quartzite
R04-03	6018	919	921	2	2	77	21
R04-03	6020	921	923	2	3	77	20
R04-03	6021	923	925	2	9	12	79
R04-03	6022	925	927	2	1	8	91
R04-03	6044	1122	1125	3	1	29	70
R04-03	6067	1410.3	1414.2	3.9	23	32	45
R04-03	6068	1414.2	1417	2.8	4	4	92
R04-03	6069	1417	1419	2	10	1	89
R04-03	6070	1419	1421.5	2.5	10	4	86
R04-03	6071	1421.5	1425	3.5	19	11	70
R04-03	6072	1425	1430	5	10	15	75
R04-05	10106	955.3	957.2	1.9	28	8	64
R04-05	10107	957.2	959.1	1.9	5	26	69
R04-05	10110	964	966	2	2	27	71
R04-05	10112	966	967.6	1.6	0	100	0
R04-05	10113	967.6	969.3	1.7	0	90	10
R04-05	10114	969.3	971.3	2	13	36	51
R04-05	10129	1215.7	1217.5	1.8	9	0	91
R04-05	10130	1217.5	1220	2.5	21	19	60
R04-05	10132	1220	1222	2	4	2	94
R04-05	10133	1222	1224	2	4	2	94
R04-05	10141	1241.5	1243.5	2	0	10	90
R04-05	10142	1243.5	1245.5	2	0	16	84
R04-05	10143	1245.5	1248.5	3	0	10	90
R04-05	10144	1248.5	1251	2.5	3	0	97
R04-05	10145	1251	12554.3	3.3	2	5	93
R04-05	10150	1265.3	1267.4	2.1	25	55	20
R04-05	10152	1267.4	1270	2.6	55	20	25
R04-05	10153	1270	1272	2	0	5	95
R04-05	10160	1316.8	1319.2	2.4	10	30	60
R04-05	10162	1319.2	1320.9	1.7	25	10	65
Average				102	12.5	24.2	63.3

Table 13-11: Feasibility Study Drill Core Samples (2007) – Composite 2

Hole ID	Sample ID	From	To	Feet	Percent Siliceous	Percent Micaceous	Per cent Quartzite
R04-07	10009	281	283.7	2.7	0	65	35
R04-07	10010	283.7	286	2.3	0	25	75
R04-08	10173	664.8	666.8	2	25	70	5
R04-08	10179	703	706.1	3.1	0	3	97
R04-08	10180	706.1	709.4	3.3	0	80	20
R04-08	10212	1044.7	1046.7	2	0	100	0
R04-08	10214	1046.7	1048.7	2	0	60	40

Hole ID	Sample ID	From	To	Feet	Percent Siliceous	Percent Micaceous	Per cent Quartzite
R04-08	10233	1259	1261	2	15	0	85
R04-08	10234	1261	1263	2	4	0	96
R04-08	10235	1263	1266	3	10	0	90
R04-08	10239	1290.7	1293	2.3	4	0	96
R04-08	10241	1293	1295	2	20	5	75
R04-08	10250	1358.6	1362	3.4	20	70	10
R04-08	10252	1362	1365.5	3.5	40	30	30
R04-08	10260	1388.7	1390.2	1.5	40	60	0
R04-08	10262	1390.2	1391.5	1.3	20	50	30
R04-08	10263	1391.5	1393	1.5	0	10	90
R04-08	10264	1393	1395.8	2.8	0	3	97
R04-08	10266	1395.8	1397.8	2	15	0	85
R04-08	10267	1397.8	1399.7	1.9	65	0	35
R04-10	10034	622.9	624.9	2	85	5	10
R04-10	10036	624.9	627	2.1	80	0	20
R04-10	10037	627	630	3	60	15	25
R04-10	10038	630	632	2	65	30	5
R04-10	10039	632	634	2	90	0	10
R04-10	10040	634	637	3	85	0	15
R04-10	10041	637	640	3	60	30	10
R04-10	10042	640	643	3	80	20	0
R04-10	10043	643	645	2	80	20	0
R04-10	10044	645	648.1	3.1	80	20	0
R04-10	10045	648.1	652.1	4	10	20	70
R04-12	10306	934.7	937.2	2.5	0	10	90
R04-12	10307	937.2	940.4	3.2	40	50	10
R04-13	10562	712.9	717	4.1	90	10	0
R04-13	10564	717	719	2	90	10	0
R04-13	10565	719	720.6	1.6	80	20	0
R04-13	10567	720.6	722.6	2	20	80	0
R04-13	10568	722.6	724.9	2.3	95	5	0
R04-13	10570	724.9	728.8	3.9	75	25	0
R04-13	10571	728.8	732	3.2	40	60	0
R04-13	10572	732	735	3	85	15	0
R04-14	10049	404.5	407.3	2.8	15	75	10
R04-14	10051	407.3	412	4.7	10	10	80
Average				111.1	39.5	26.9	33.6

Table 13-12: Feasibility Study Drill Core Samples (2007) – Composite 3

Hole ID	Sample ID	From	To	Feet	Percent Siliceous	Percent Micaceous	Per cent Quartzite
R04-15	10387	945.8	947.8	2	20	40	40
R04-15	10388	947.8	950	2.2	20	70	10
R04-15	10395	1041	1044.5	3.5	5	95	0
R04-15	10397	1044.5	1047.5	3	0	30	70
R04-15	10454	1308.4	1310.8	2.4	15	85	0
R04-15	10455	1310.8	1313.3	2.5	60	40	0
R04-15	10456	1313.3	1315.8	2.5	5	95	0
R04-15	10457	1315.8	1317.6	1.8	30	70	0
R04-15	10458	1317.6	1319.7	2.1	40	60	0
R04-16	10065	473.4	476	2.6	0	100	0
R04-16	10066	476	478.6	2.6	0	85	15
R04-16	10074	807.3	809.3	2	0	100	0
R04-16	10075	809.3	812.3	3	4	92	4
R04-16	10076	812.3	814.4	2.1	3	90	7
R04-17	10544	1080.2	1082.6	2.4	10	70	20
R04-17	10546	1082.6	1085	2.4	15	85	0
R04-17	10547	1085	1087.1	2.1	40	60	0
R04-18	10089	571	573	2	40	20	40
R04-18	10090	573	575	2	0	75	25
R04-18	10091	575	579	4	17	83	0
R04-19	10608	971.2	973.5	2.3	3	5	92
R04-19	10609	973.5	975.5	2	50	0	50
R04-19	10617	1031.4	1033.7	2.3	30	20	50
R04-19	10618	1033.7	1036	2.3	15	35	50
R04-19	10655	1155.1	1157.3	2.2	20	70	10
R04-19	10657	1157.3	1159.6	2.3	40	20	40
R04-19	10658	1159.6	1161.6	2	55	45	0
R04-19	10660	1161.6	1164.8	3.2	60	40	0
R04-19	10661	1164.8	1167.2	2.4	20	80	0
R04-20	10708	434	436	2	4	86	10
R04-20	10709	436	438	2	3	97	0
R04-21	10669	488.3	490.9	2.6	5	15	80
R04-21	10688	674	676	2	10	0	90
R04-21	10721	910	912	2	0	7	93
R04-21	10673	999.5	1001.2	1.7	15	85	0
R04-21	10674	1001.2	1003	1.8	35	65	0
R04-21	10675	1003	1004.9	1.9	15	85	0
R04-21	10676	1004.9	1007	2.1	30	70	0
R04-21	10678	1007	1009	2	45	35	20
Average				90.3	19.5	60.1	20.4

13.3.5 Head Sample Mineralogy

Each composite for Phase 1 and 2, were submitted for semi-quantitative X-Ray diffraction analysis, with the results tabulated below in Table 13-13. The major copper and cobalt minerals are chalcopyrite and cobaltite, although trace quantities of cubanite and covellite were found in Comp 2 and the Oxide composites.

Major gangue minerals include quartz (44 – 52%), chlorite (14 – 29%), garnet (4 – 15%), and mica (6 – 21%). Low quantities of pyrite (0.3 – 2%) and Jarosite (0 – 2.3%) were also found to be present.

In general, Phase 1 & 2 composites were similar, although variations in the quantity of chlorite, garnet, and mica were noted. The Oxide composite presented similar mineralogy, but with more complexity.

No XRD analysis was conducted on the grind composites delivered to Grinding Solutions. The visual inspection, however, indicated that Grind Comp 1 and 2 contained garnet and high mica contents. A higher proportion of fines thought to be clays were observed in the oxidised sample relative to the other two grind composites.

No mineralogy analysis was conducted on Phase 3 material.

Table 13-13: Composite Semi-Quantitative X-Ray Diffractive Mineralogy

Mineral	Phase 1		Phase 2				
	Comp 1	Comp 2	HH Comp	HL Comp	LH Comp	LL Comp	OX Comp
	[%mass]	[%mass]	[%mass]	[%mass]	[%mass]	[%mass]	[%mass]
Quartz	47.4	47.5	46.9	43.7	46.5	47.8	52.1
Chlorite (Chamosite)	13.5	13.5	24.3	28.9	20.4	25.5	20.0
Garnet (Almandine)	14.6	10.2	5.6	4.2	3.6	4.1	3.8
Mica (Siderophyllite)	6.2	9.9	8.8	9.8	10.6	10.0	11.8
Muscovite (Mica)			6.3	6.5	10.3	7.5	
Plagioclase (Albite)			1.1	2.6	3.7	1.2	0.3
Chalcopyrite	4.7	4.9	2.9	1.3	1.9	1.0	0.4
Cobaltite	2.7	1.4	2.1	1.9	1.4	1.1	1.4
Orthoclase (Feldspar)	3.6	2.4	0.8	0.5	0.8	0.8	0.6
Pyrite	2.0	1.0	1.1	0.5	0.5	0.3	0.3
Ilmenite					0.6	0.7	
Actinolite (Amphibole)		5.4					
Anorthite (Feldspar)	2.6	1.2					
Zircon	2.2						
Cubanite		1.1					
Magnetite		1.0					0.5
Rutile	0.5	0.5					
Arsenopyrite							1.4
Covellite							0.6
Maghemite (yFe_2O_3)							2.5
Jarosite ($KFe_3(SO_4)_2(OH)_6$)							2.3
Hohmannite ($Fe_2(OH)_2(SO_4)_2.7H_2O$)							1.5
Rutile							0.3

Tabulated below in Table 13-14 is the semi-quantitative X-Ray Diffraction Analysis results of a grab sample from the Phase 4 feed material. This was oxidised surface outcrop material that did not contain chalcopyrite, but which still contained high quantities of cobalt. Approximately 80% of the cobalt was contained within cobaltite ($(Co,Fe)AsS$) and the remaining 20% contained within cobaltkorignigite ($(Co, Zn)(As{+5}O_3)(OH)\cdot H_2O$).

Table 13-14: Phase 4 Feed Mineralogical Analysis

Mineral	% Mass
Quartz	63.4
Chlorite	27.0
Cobaltite	4.0

Mineral	% Mass
Cobaltkorignigite	1.8
Mica	1.6
Potassium-Feldspar	1.2
Plagioclase	0.7
Xenotime	0.2

Table 13-15 shows the mineralogical modal analysis for the composite samples that were tested for the 2007 Feasibility study.

Table 13-15: Semi-Quantitative X-Ray Diffractive Mineralogy for 2007 Feasibility Composites

Mineral	Comp 1	Comp 2	Comp 3	Comp M	Comp Q	Comp S
Quartz & Feldspar	49	40	30	19	53	65
Chlorite (clinochlore+pennine)	14	13	33	34	9	9
Mica (biotite/muscovite/sericite)	15	13	16	21	14	12
Garnet (almandine)	7.6	14	5.3	9.6	6.4	4.4
Amphiboles (hornblende)	7.8	7.4	10.0	12	11	2.6
Carbonates (calcite/dolomite)	0.0	0.2	0.0	0.0	0.0	0.0
Chalcopyrite	2.9	5.1	0.9	0.8	2.2	2.1
Covellite	0.0	0.2	0.2	0.0	0.0	0.0
Cobaltite	1.9	3.5	2.1	2.2	1.1	2.1
Marcasite	0.4	2.6	1.1	0.0	1.5	0.4
Leucoxene	0.0	0.4	0.2	0.6	0.4	0.4
Magnetite	0.2	0.0	0.4	0.6	0.0	0.0
Fe oxy-hydroxide	0.2	1.3	0.5	0.2	0.2	1.7
Arsenopyrite	0.0	0.0	0.0	0.4	0.0	0.2
Other Cu / As (Carrollite??)	0	trace	0	0	0.0	0.2
Total	**100**	**100**	**100**	**100**	**100**	**100**

13.3.6 QEMSCAN Analysis

Each of the Phase 2 composites was submitted for Qemscan analysis after being stage crushed to 80% passing 150 µm. Each sample was then screened and analysed in different size fractions (+150 µm, -150+106 µm, -106+75 µm, -75+25 µm, and -25 µm).

The objective of the analysis was to determine the liberation and gangue associations of chalcopyrite/cobaltite within the ore body over a varied size range.

In Figure 13-10 depicted below, the liberation of chalcopyrite within all composites except Oxide Comp is good with at least 82% of the mineral being classified as liberated or free at 106 µm. The liberation of chalcopyrite in the oxide composite, however, is poor for size fractions larger than 25 µm due to a covellite surface coating of the chalcopyrite.

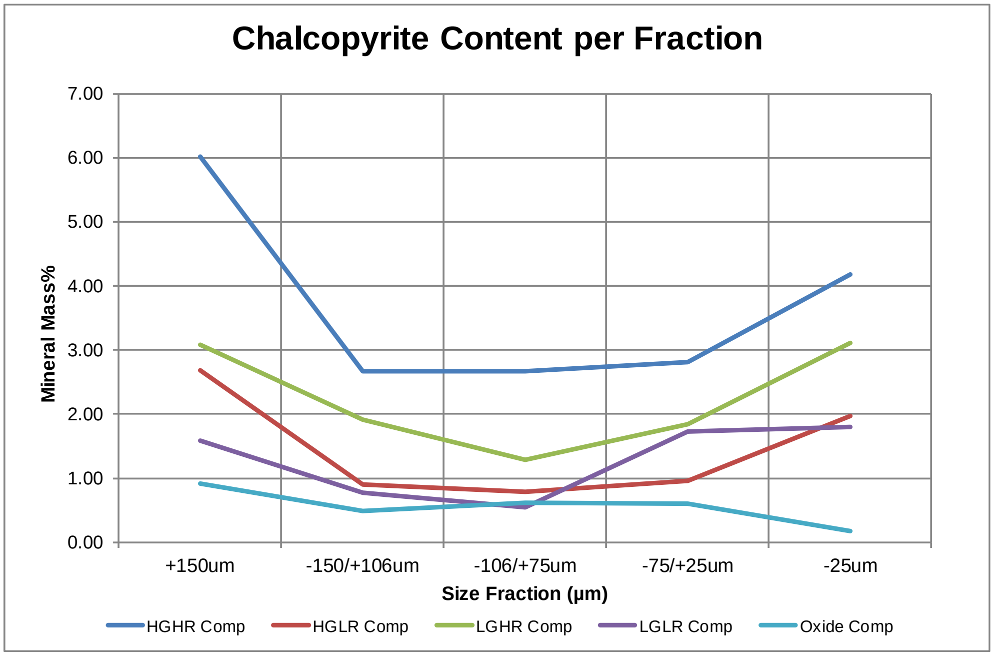


Figure 13-9: Chalcopyrite Content

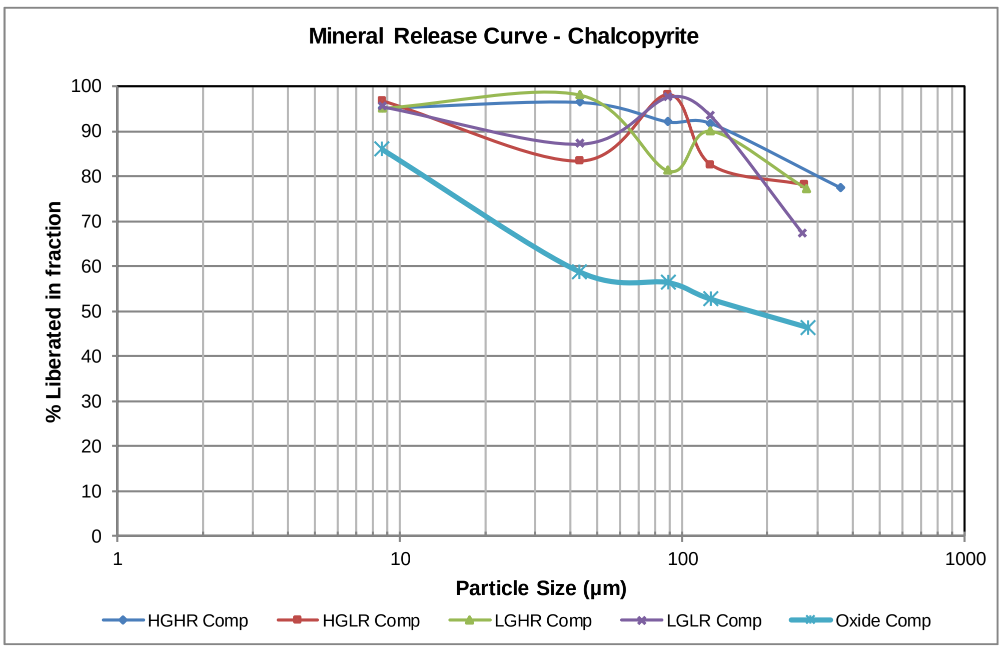


Figure 13-10: Chalcopyrite Mineral Release Curve

The mineral release and content for cobaltite are depicted below in Figure 13-12. The low grade and oxide composites followed a typical release distribution curve with a reduction in liberation for increased size. However, the high-grade composites displayed a bimodal liberation distribution with liberation beginning to increase again for coarser sizes. This demonstrates the importance of grind size selection for cobaltite in order to generate both good liberation while simultaneously not over-grinding already liberated material.

Based on this information, a grind size P80 of 75 µm was considered appropriate for flotation testwork. The main association of cobaltite was with silicates, with very little to no association with chalcopyrite.

The silicates were predominantly in the coarser size ranges which necessitated a change from laboratory scale rod milling to higher energy laboratory ball milling for batch flotation testwork in Phase 2. The presence of softer, flat, plate-like, phyllosilicates (chlorite and mica) also resulted in initial laboratory sieve analysis interpretation difficulties during testwork. As a result of this, various batch flotation tests were conducted to confirm optimised grind times for maximum flotation metal recovery.

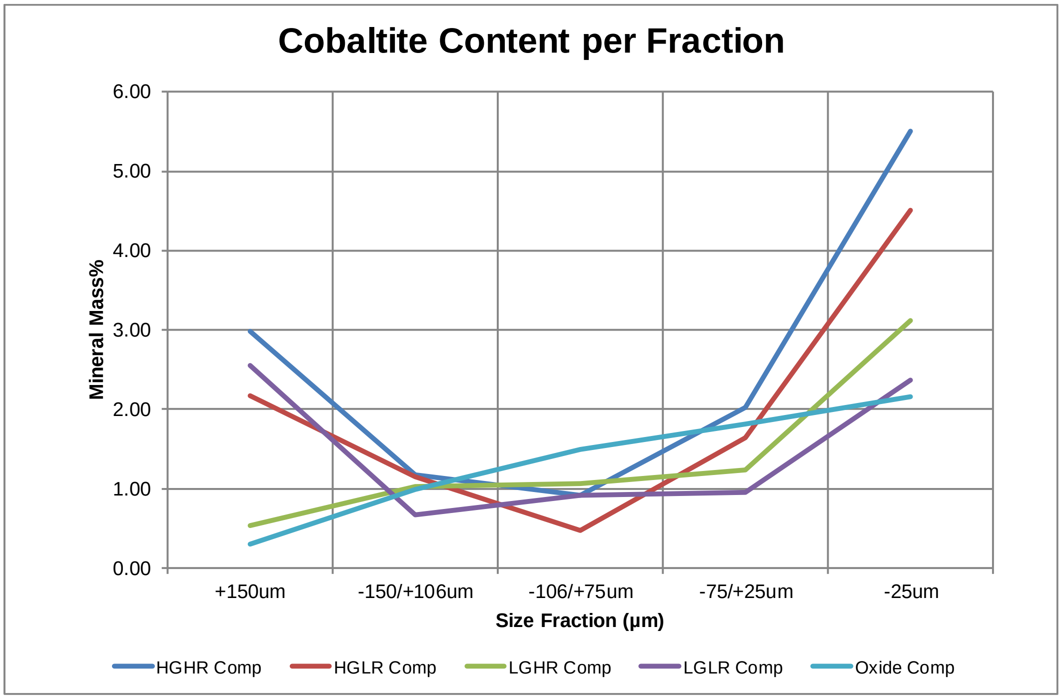


Figure 13-11: Cobaltite Content

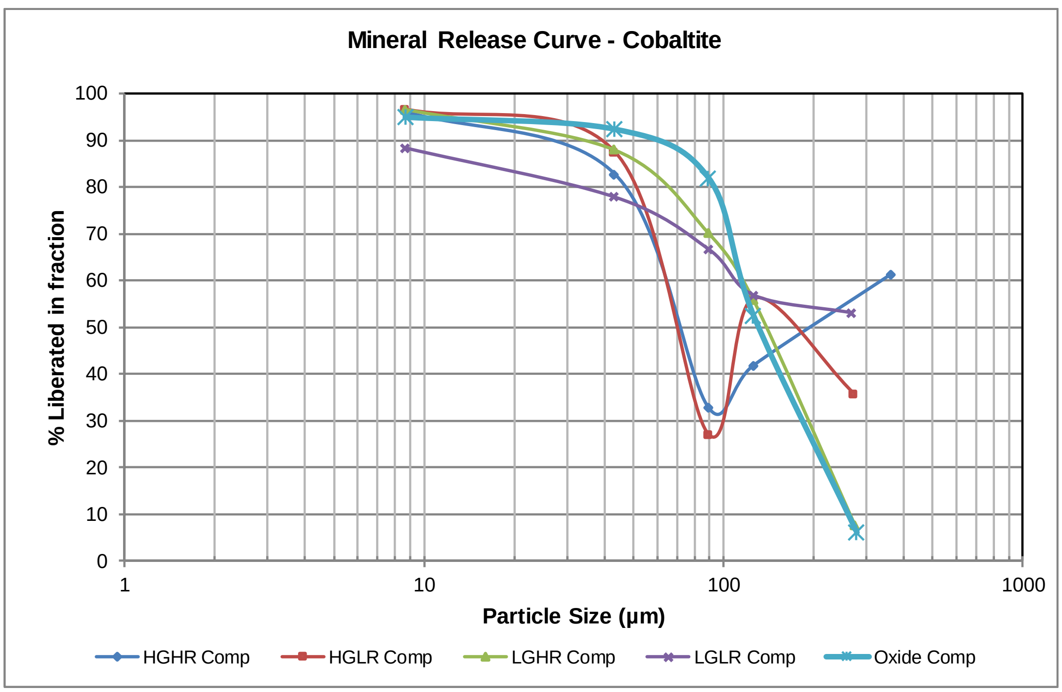


Figure 13-12: Cobaltite Content and Mineral Release Curve

13.3.7 Physical Characterization

Prior to the study commencing, two shipments of broken split drill core pieces were delivered to SGS in January and February 2019 to conduct SMC comminution tests (SGS Canada Inc, Feb 2019). This material originated from the same drill core composite as had been utilised at SGS in 2016/2017 (i.e. Phase 1 material, having been stored separately by ICO). Although DRA was not present during this work, the summarised results are tabulated below in Table 13-16.

Table 13-16: SMC Test Results

Parameter	Unit	Sample 1	Sample 2
S.G.	t/m³	3.07	2.96
A		63.8	66.5
b		1.21	1.44
A × b		77.2	95.8
Hardness Percentile		18	13
Classification		Soft	Soft
ta (Estimated)		0.65	0.84
DWi	kWh / m³	3.99	3.1
Mia	kWh / t	11.3	9.6
Mih	kWh / t	7.5	6.1
Mic	kWh / t	3.9	3.1
SCSE	kWh / t	7.9	7.1

Half of the two-grind composite material derived during Phase 2 of the feasibility study was submitted for physical characterisation at SGS, with the results tabulated below in Table 13-17.

Table 13-17: Phase 2 Physical Characterization - SGS

Parameter	Unit	Grind Comp 1	Grind Comp 2	Characterization
Specific Gravity	t/m³	2.93	3.03	
Bond Low Energy Impact (CWi)	KWh/t	6.0	7.7	Moderately Soft
	Hardness Percentile	23	36	
Bond Rodmill Grindability Test (RWi)	KWh/t @ 14 mesh grind	5.0	5.1	Very Soft
	Hardness Percentile	1	1	
Bond Ballmill Grindability Test (BWi)	KWh/t @ 150 mesh grind	14.2	15.7	Moderately Hard
	Hardness Percentile	48	65	
SAG Power Index (SPI)	CEET Crusher Index (Ci)	17.2	8.3	Soft
	SPI (minute)	19.6	20.4	
	Hardness Percentile	7	7	
Bond Abrasion Test (Ai)	g	14.2	15.7	Mildly Abrasive
	Abrasivity Percentile	14	29	

The remaining half of the material together with the R19-01 oxide drill core was submitted to Grinding Solutions Mineral Processing Services for characterization testing.

Table 13-18 summarizes test data derived at Grinding Solutions (Grinding Solutions Mineral Processing Services Grind, 2020). The grind mill test procedure involves milling different feed sizes, typically 100% <6.7 mm, 100% <9.5 mm and 100% <16 mm to a range of specific energy inputs. Two mill chambers of different dimensions are used with graded ball charges to ensure sufficient energy is available for breakage of the coarse proportion of feed particles. A standard Bond ball mill work index ("BBWi") test is carried out in parallel to provide a reference point. Detailed size analysis on each of the batch grind feed and products allows calculation of breakage rates using population balance modelling.

Overall, the biotite-chlorite and biotite-silicified samples produced very similar products size distributions for most tests, especially at 15 kWh/t. The oxidised sample was softer than the other two samples, as may be expected given its altered oxidation state.

Table 13-18: Phase 2 Grind Mill Test Results

Size	Energy	R19-01 Oxidised	Grind Comp 1 Biotite Chlorite	Grind Comp 2 Biotite Silicified
100% < 16mm	F80	10,696	14,211	11,802
	P80 at 7.5kWh / t	133.2		
	P80 at 9.5 kWh / t		138.6	133.2
	P80 at 15 kWh / t	134	126.4	126.7
100% < 9.5mm	F80	6,946	6,198	6,272
	P80 at 7.5kWh / t	135.7		
	P80 at 9.5 kWh / t		135.5	131.1
	P80 at 15 kWh / t	114.4	128.2	128
100% < 6.7mm	F80	4,264	4,029	4,133
	P80 at 7.5kWh / t	131.3		
	P80 at 9.5 kWh / t		144.8	137.3
	P80 at 15 kWh / t	113.8	125.4	128.3

The particle size distributions achieved at Grinding Solution consistently showed some irregularity (deviation from the linear line in log-log space) between 45 – 90 µm in the 7.5 and 9.5 kWh/t products, and between 90 – 250 µm in the feed samples. This behavior was consistent amongst all three samples, though much less pronounced in the 15kWh/t products. This is considered a mineralogical effect. Upon visual inspection of micrographs taken during the testwork, garnets are considered the likely cause as they were comparatively prevalent in the -425 +45 µm fraction.

Issues with viscosity were observed at Grinding Solutions during the 16 mm at 15 kWh/t grinds. This was most pronounced with the oxidised sample. Consequently, the solids percentage of the slurry was reduced slightly to mitigate this. Despite the small change in P80 from the low to the high energy grind, the issue was more pronounced in the latter. Assuming the viscosity increase is a mineralogical effect (possibly related to the discussion of abnormal particle size distributions above), this suggests that liberation of the mineral(s) responsible occurs during the later stages of grinding. It is possible that chlorite and clays are responsible for this effect. The current resource model of the project indicates that oxide material does not form a major part of the resource.

Table 13-19 summarizes key data from the bond work index tests at Grinding Solutions. Grindability of the oxidised, biotite-chlorite and biotite-silicified sample increased by approximately 1-1.5 kWh/t increments, which is in line with expectations. This behavior is driven both by the rate of fines generation and the grind product size.

Table 13-19: Phase 2 Bond Ball Mill Work Index Range

		R19-01 Oxidised	Grind Comp 1 Biotite Chlorite	Grind Comp 2 Biotite Silicified
Bond Ball Mill Work Index Range [kWh / t]	95% CI	7.97 - 8.31	9.53 - 9.87	10.31 - 11.0
	Last Cycle Only	8.26	9.82	10.92
	Last two cycles	8.14	9.7	10.86
	Last three cycles	7.91	9.48	10.64

From the various specific energy calculation methods, the estimated SAG specific energy requirement varies between 5-8 kWh/t, and the apparent Bond Ball Work Index varies between 10-12 kWh/t. Typically 15.5-20 kWh/t total energy is required to achieve the desired grind of 80% passing 75~84 µm (Morgan, 2020).

From the testwork, the Bond Ball Mill Work Index Population curve has been derived and is displayed below in Figure 13-13. The actual sample weighted average is 10.32 kWh/t.

This Bond Ball Mill Work Index test ("BBWi") required milling a 700 cm^3 sample of feed in a dedicated milling chamber and ball charge. The BBWi test procedure is iterative, with the target being to achieve a 250% ±3% circulating load at a given closing screen size by adjusting the number of revolutions. After each milling cycle, the full product was screened at the closing screen size (150 µm in this case) with the oversize supplemented with a fresh feed mass to bring it back to the mass of the original 700 cm^3 sample.

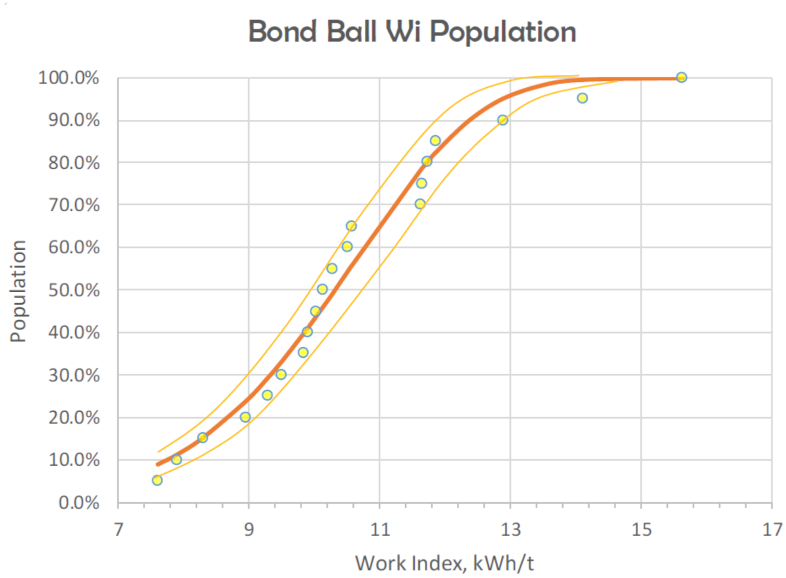


Figure 13-13: Idaho Cobalt Operation (ICO) Bond Ball Work Index Population

The Bond Work Index testwork from SGS and Grinding Solutions testwork have all been simulated to obtain their breakage rates which are displayed in Figure 13-14 below. The variation in hardness de-rate coefficient is estimated to be 0.7~1.45 about the weighted average for ICO ore. There is evidence in the grinding testwork conducted that micas and garnets 'build-up' because of differential breakage effects, This 'build-up' effect on grinding appears to influence the rates of breakage, especially below 450 µm, which may be the size at which the ore matrix essentially breaks down into discrete minerals.

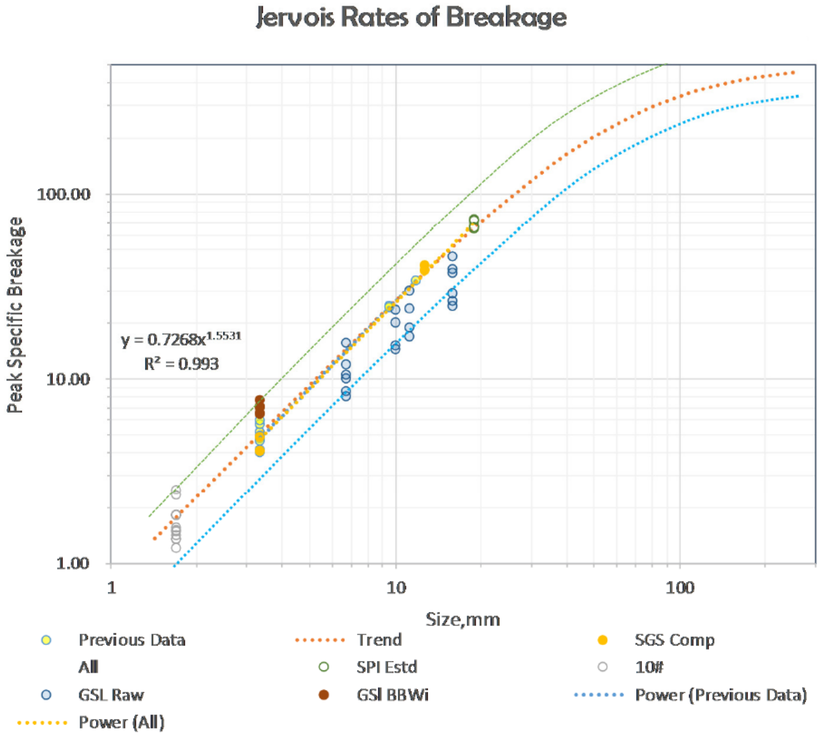


Figure 13-14: Idaho Cobalt Operation (ICO) Ore Rates of Breakage

13.3.8 Flotation Testwork

The objective of the 2020 FS flotation testwork phase 1, was to develop a sequential flotation flowsheet to a PEA or scoping study level of detail on available material at SGS prior to the arrival of the fresh Phase 2 drill core samples. Then Phase 2 successfully confirmed and optimised the initial testwork on the sequential split concentrate flowsheet. Two distinct saleable concentrates were to be produced while simultaneously minimizing losses of each metal to the other concentrate, with the targeted product grade detailed below:

- Copper concentrate 30 – 34% w/w Cu
- Cobalt Concentrate 10 – 15% w/w Co

Subsequent to all testwork being completed, the flotation section of the treatment plant reverted to the bulk concentrate flowsheet concept of the 2007 FS developed by Samuels Engineering Inc. As such, reference should be made to this study report for details of this bulk sulphide flotation testwork, although a summary of the results is provided below.

Where applicable, some important understandings of the material that were derived during Phase 1 and 2 flotation testwork are noted below

13.3.8.1 Feasibility Study 2007 Flotation Testwork Summary

In summary, one locked cycle test was conducted during the CAMMP pre-feasibility study and three for the 2007 feasibility study supervised by Samuels Engineering. These are summarised below and have been utilised to estimate the recoveries of the treatment plant.

For the CAMMP locked cycle test, a statistically valid test matrix was set up, and testing was carried out on a composite sample with an average head grade of 0.574% Co, 0.294% Cu, and 0.02 Troy ounces of gold per ton. The locked cycle test produced good recoveries and grades as tabulated below in Table 13-20.

Table 13-20: CAMMP Locked Cycle Test Results

	Cobalt	Copper	Gold
Concentrate Grade	14.40%	7.41%	0.396 oz Au/ton
Recovery	92.70%	92.80%	72.90%

In order to account for variations in metallurgical response due to the mineralogical composition, several horizons were set aside for flotation testing. The variability testing was completed after the circuit had been optimised, and the locked-cycle tests were completed on the composite sample. These tests indicated similar recoveries to the bulk composite, even using samples from areas that contained moderate oxidation. Comparative flotation test results are summarised in Table 13-21. All drill hole samples that were used in the composite were from section 200N except MH-000-02, which was from section 1400N.

The average of the test results does not include the results from testing of the composite sample.

Table 13-21: CAMMP Variability Test Results

Test	% Co	% Cu	oz Au/t	Co Recovery	Cu Recovery	Au Recovery	Remarks
Composite	0.57	0.29	0.020	92.0%	91.6%	73.0%	
4-3021	0.48	0.07	0.018	88.2%	92.4%	56.6%	cobaltite, trace oxidation
4-3022	2.13	0.05	0.032	92.5%	80.6%	86.0%	cobaltite, local silicification
2-3023	0.10	0.31	0.016	91.4%	92.2%	69.1%	moderate oxidation
3-3023	0.85	0.04	0.038	89.7%	77.0%	85.9%	cobaltite, tr oxidation, tr FeCO3
6-3023	0.49	1.63	0.012	96.9%	96.4%	67.6%	fine grained cobaltite
7-3023	1.01	0.28	0.048	94.3%	94.7%	89.5%	cobaltite, moderate silicification
Average	0.85	0.40	0.027	92.2%	88.9%	75.8%	

A six-cycle, locked cycle test was conducted on three composites (Comp 1 – 3) for the feasibility study locked cycle tests conducted at SGS in 2005. As well as product recycle, these locked cycle tests used water supplied by Formation Capital Corporation. Water was recirculated during the tests. The rougher and cleaner scavenger tailings from each test were filtered separately. The filter cake was coned and quartered, and opposite eighths of the filter were cut out and submitted for chemical analysis. The remaining three-quarters of the filter was stored in the freezer without drying. The results of the locked-cycle tests are tabulated in Table 13-22. Copper recoveries ranged from 96 to 97 percent. Cobalt recovery ranged from 91 to 95 percent, and the sulfur grade in the overall tailing was 0.05 to 0.08 percent.

Similarly, three batch variability tests were conducted on composites S, M, and Q.

The operating conditions of the locked cycle test are tabulated below in Table 13-22.

Table 13-22: Feasibility Study Locked Cycle Test Operating Conditions

Section	Criteria
Primary grind product size distribution, P80	65 to 82 microns
PAX collector addition to the grinding circuit	100 grams per tonne
PAX addition to the rougher flotation circuit	100 grams per tonne
Frother AF65	2.5 grams per tonne
Rougher flotation time	9 minutes
PAX collector addition to the 1st cleaner flot circuit	10 grams per tonne
1st cleaner flotation time	3.5 minutes
1st cleaner-scavenger flotation time	2 minutes

Table 13-23: Feasibility 2007 Bath and Locked Cycle Test Results

Comp	Test	Product	Weight %	Grade						Percent Distribution					
				% Cu	% Co	% S	g/t Au	g/t Ag	% As	Cu	Co	S	Au	Ag	As
1	F30	Cleaner 2 conc	3.8	10.60	14.00	25.0	7.020	---	---	96.5	93.4	95.5	---	---	---
1	LCT1	Projected cleaner conc	4.18	9.70	13.30	21.7	0.033	16.0	15.3	96.0	93.0	95.4	90.3	---	94.2
1	F30	Combined tails	96.15	0.01	0.04	0.0	6.750	---	---	3.3	6.3	4.1	---	---	---
1	LCT1	Projected combined tails	95.80	0.02	0.03	0.1	0.120	< 0.9	0.04	4.0	7.0	4.6	9.7	---	5.8
2	F31	Cleaner 2 conc	7.44	13.50	8.88	27.6	10.000	---	---	95.1	89.4	95.5	---	---	---
2	LCT	Projected cleaner conc	8.00	12.90	8.28	24.8	0.050	20.5	9.94	96.5	90.7	94.3	84.5	71.6	90.1
2	F31	Combined tails	92.43	0.05	0.08	0.1		---	---	4.5	10.1	4.1	---	---	---
2	LCT	Projected combined tails	92.00	0.04	0.04	0.1		0.7	0.10	3.5	9.3	5.7	15.5	28.4	9.9
3	F32	Cleaner 2 conc	5.86	6.66	17.20	19.3		---	---	94.8	93.3	94.6	---	---	---
3	LCT	Projected cleaner conc	5.86	7.09	17.90	20.0		11.2	21.0	97.1	95.1	96.4	92.0	---	95.2
3	F32	Combined tails	94.01	0.02	0.07	0.1		---	---	4.5	6.1	4.7	---	---	---
3	LCT	Projected combined tails	94.10	0.01	0.03	0.1		< 0.5	0.07	2.9	4.9	3.6	8.0	---	4.8

The flowsheet of the 2007 feasibility study locked cycle tests is depicted below in Figure 13-15.

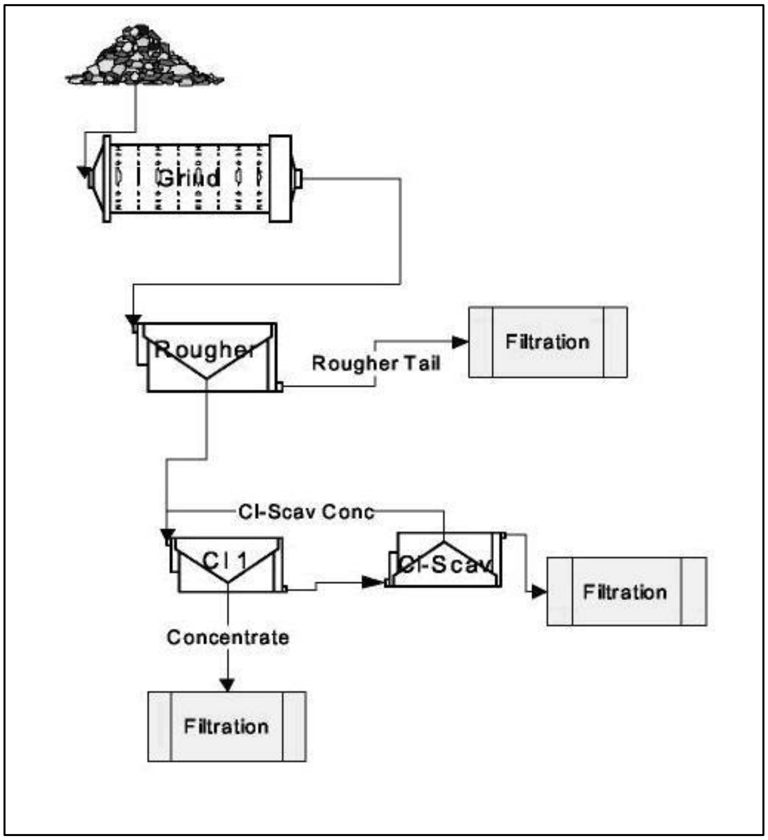


Figure 13-15: Locked Cycle Test Flowsheet – Feasibility Study 2007

13.3.9 Relevant Flotation Testwork – Phase 1

The following parameters were studied in the flotation test program for the 2020 feasibility study during Phase 1:

- Baseline test comparisons;
- It was demonstrated that by elevating the pH above 9, cobalt activation could be suppressed within the copper flotation circuit;
- A copper selective collector rather than PAX was shown to provide benefits;
- The effect of a regrind mill on copper and cobalt rougher concentrate was tested and selected for the copper circuit;
- Other factors such as selective pyrite flotation and the addition of the activator $CuSO_4$ were considered but not included within the flowsheet; and
- One locked cycle test was conducted at the end of Phase 1 on Comp 1 material. The locked cycle test products were then analysed via Qemscan analysis.

On conclusion of Phase 1, SMC comminution reject material was combined with the remaining drill core material that did not have excessive copper grades, to form a new composite (New Comp) and batch 10 kg flotation tests were conducted in order to generate additional product material for offtake agreement analysis.

The detailed results of these tests can be found within the relevant SGS report. However, the following additional understanding of the material relevant to bulk sulphide flotation is noted below:

- A cobalt promoter reagent, Aero 25, was tested but the results showed no benefit to the flowsheet;
- Heavy Liquid Separation (HLS) tests were conducted on SMC comminution reject material in order to assess whether a Dense Media Separation (DMS) process could potentially upgrade material prior to flotation. The material was stage crushed to -12.7 mm and screened at 0.85 mm before being separated at varied densities 2.70 – 3.30 t/m^3. The results are tabulated below in Table 13-24. HLS separation demonstrated very poor potential for the size fraction assessed, with approximately 13% of the mass being removed for a loss of 6.7% of the cobalt. This was due to the poor liberation of the cobaltite for the crushed size fraction assessed, and hence testing of this technology was not progressed.

Table 13-24: HLS Testwork Results

Fractional								
Product (-12.7+0.85mm)	Weight		Assay [%]			Distribution [%]		
	g	%	Cu	Co	S	Cu	Co	S
HLS Sink 3.30	439	7.4	5.74	0.49	11.3	45.6	27	48.6
HLS Sink 3.20	203	3.4	1.41	0.23	2.83	5.2	5.9	5.6
HLS Sink 3.10	587	9.9	0.57	0.13	1.46	6.1	9.6	8.4
HLS Sink 3.00	844	14.2	0.87	0.1	1.49	13.3	10.5	12.3
HLS Sink 2.90	1016	17.1	0.37	0.08	0.58	6.8	10	5.8
HLS Sink 2.80	894	15.0	0.26	0.1	0.45	4.2	10.8	3.9
HLS Sink 2.70	619	10.4	0.2	0.08	0.33	2.2	6.1	2
HLS Float 2.70	155	2.6	0.05	0.03	0.1	0.1	0.6	0.2
HLS -0.85mm	1200	20.1	0.76	0.13	1.12	16.5	19.6	13.2
Feed (Calc)	5957	100.0	0.93	0.13	1.71	100	100	100
Combined – Density Cut at 2.80 t/m^3								
Product	Weight		Assay [%]			Distribution [%]		
	g	%	Cu	Co	S	Cu	Co	S
Combined HLS Sink 2.80	3983	66.9	1.13	0.15	2.17	81.1	73.7	84.7
Combined HLS Float 2.80	774	13	0.17	0.07	0.28	2	6.7	2.2
HLS -850µm	1200	20.1	0.76	0.13	1.12	0.4	19.6	13.2
Feed (Calc)	5957	100	0.93	0.13	1.71	16.5	100	100

- In order to consider the possibility of gravity separation as a scavenging application, cobalt rougher tailings (final tailings) was treated with a Knelson Concentrator followed by further concentration on a Mozley Shaking Table. As tabulated below in Table 13-25, about ten times cobalt upgrade was achieved with the Knelson Concentrator alone, and a further ~five times upgrade with the Mozley Table. Cobalt stage recovery was low, however, with only about 10%-15% in each of these stages. Cobalt recovery in the gravity separation was only in the range of 1-2%. This shows that gravity separation has potential, provided higher feed grades or multiple passes are considered. As the project was developing fit-for-purpose solutions, the technology was not utilised in a scavenging application within the flowsheet.

Table 13-25: Gravity Concentration Testwork Results

Products	Weight		Assays %		Distribution %	
	g	%	Co	S	Co	S
B2 Mozley Conc	2	0.03	2.31	2.40	1.1	0.4
B2 Mozley Midds	42	0.5	0.11	0.50	1.0	1.7
B2 Mozley Tail	35	0.4	0.78	1.08	5.8	3.0
B2 Knelson Tail	8452	99.1	0.05	0.14	92.1	94.8
Head (calc.)	8,531	100	0.05	0.15	100	100
Head (Dir.)			0.05	0.14		
B2 Knelson Conc (Calc)	79	0.9	0.47	0.81	7.9	5.2

13.3.10 Relevant Flotation Testwork – Phase 2

Additional batch flotation tests were conducted on fresh drill core composites provided to SGS in order to optimise further the sequential split concentrate flotation circuit developed in Phase 1. Five composites (four plus one oxide) representing the expected range of copper/cobalt grades and ratios were used for this testwork. Halfway through the testwork phase, the two low-grade composites (LH and LL) were combined to form the average ("AG") composite. Differing operating parameters investigated for the sequential flotation circuit which are relevant to the bulk sulphide flotation flowsheet are noted below:

- Grind Times;
- Rod and ball milling;
- Effect of adjusting the energy input to the flotation cells;
- Reagent type and dosage optimisation;
- Impact of blending oxide ores;
- Effect of pulp density; and
- Effect of water type.

13.3.10.1 Optimised Grind Times

Three grind calibration tests were performed on 2 kg charges for LH, HL, and Oxide composites varying from 30 - 75 minutes of grind time within a rod mill. As tabulated below in Table 13-26, it was found that increasing the grind time did not change the milled particle size distribution significantly even after significant milling times had elapsed.

Table 13-26: Phase 2 Rod Mill Grind Calibration Tests

Size	LH Comp			HL Comp			Oxide Comp		
	45 min	60 min	75 min	60 min	75 min	90 min	45 min	60 min	75 min
212	100	100	100	100	100	100	99.8	99.9	99.9
150	99.6	99.8	99.7	99.2	99.4	99.7	98	98.6	99
106	92	97.6	97.3	92	92	95.4	81.9	86.5	87.4
75	70.4	84.2	79.5	70.7	72	76.4	60.5	65.4	61.4
53	53.5	65.5	61.3	54	55.4	59.7	45.5	48.8	45.2
38	40.2	49.2	46.8	40.3	41.3	45.8	34.1	36.5	34.1
K80	**88**	**70**	**76**	**88**	**87**	**81**	**103**	**96**	**97**

The size fractions of HL Comp after grinding for 90 minutes were saved and submitted for XRD analysis. The results are tabulated below in Table 13-27. Most minerals in the coarse fractions (+106 µm and -106 + 75 µm) were quartz and phyllosilicates (chlorite and mica), as well as some garnet. Cobaltite, chlorite, and mica seemed to be more concentrated in the -38 µm fraction. Chalcopyrite and pyrite were almost evenly distributed in the size fractions studied.

Table 13-27: HL Comp XRD by Size After 90 mins Rod Mill Grind

Mineral	+106 µm	-106 + 75 µm	-38 µm
	% mass	% mass	% mass
Quartz	62	68.4	28.9
Chlorite	13	8.2	30.2
Mica	15	10.4	24.8
Garnet	4.7	8.3	9
Cobaltite	0.8	1	2.2
Potassium Feldspar			0.1
Plagioclase	1.1	0.3	
Chalcopyrite	0.6	0.9	0.7
Magnetite	1	0.6	0.7
Pyrite	0.6	0.6	0.6
Calcite	0.8	0.8	1
Ilmenite		0.5	
Rutile	0.4		0.6
Covellite			0.4
Mahemite			0.6

The presence of chlorite and mica could result in skewed size distribution results due to their thin, plate-like shape. It was considered likely that the rod mill was over-grinding the cobaltite without sufficiently reducing the silicates. A ball mill was then tested for the grinding of Phase 2 material. Subsequent batch flotation tests focused on comparing rod mill to ball mill grinding, while simultaneously optimizing the grind time based upon comparable batch flotation test results.

13.3.10.2 Rod Mill Grind Times

Sequential flotation tests were conducted for varied rod mill grind times which are depicted in Figure 13-16 below. The final overall cobalt flotation recovery of these tests was similar. However, the flotation kinetics with 45 min grind time was faster than the 35 or 60-minute grind times. It is suspected that this is due to potentially poorer liberation and over-grinding of cobalt minerals, respectively.

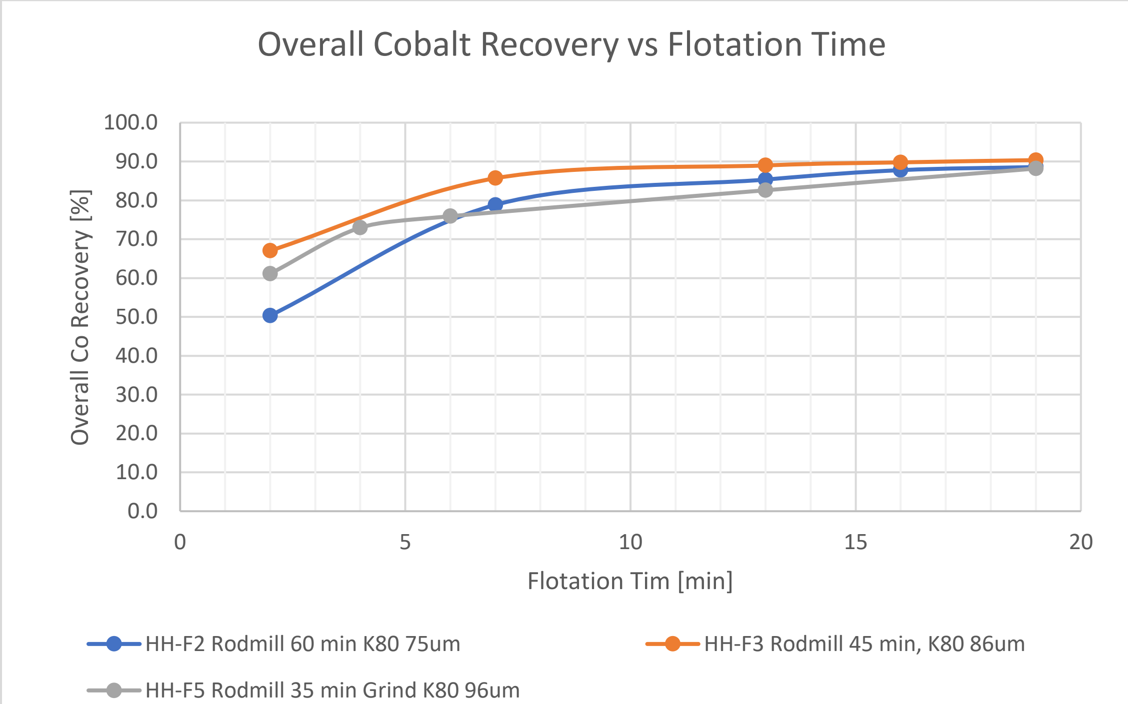


Figure 13-16: Co Overall Recovery Kinetics for Varied Rod Mill Grind Times

HH-F5 (35 min) first three points utilize cobalt cleaner after 2, 4 and 6 min.

13.3.10.3 Rod Mill Compared with Ball Mill

A comparative test was conducted between a rod mill and ball mill grind for the same time (45 min). As expected, the K80 of the ball mill product was finer than that produced by the rod mill (64 and 96 µm respectively). This indicates a better grinding efficiency to achieve the target particle size using a ball mill than a rod mill for this material, probably due to the high content of quartz and phyllosilicates in the ore.

The final overall cobalt recovery was similar between the two tests depicted in Figure 13-17. The results also indicated the cobalt recovery reached a plateau at about 13 minutes of cobalt flotation residence time subsequent to copper flotation.

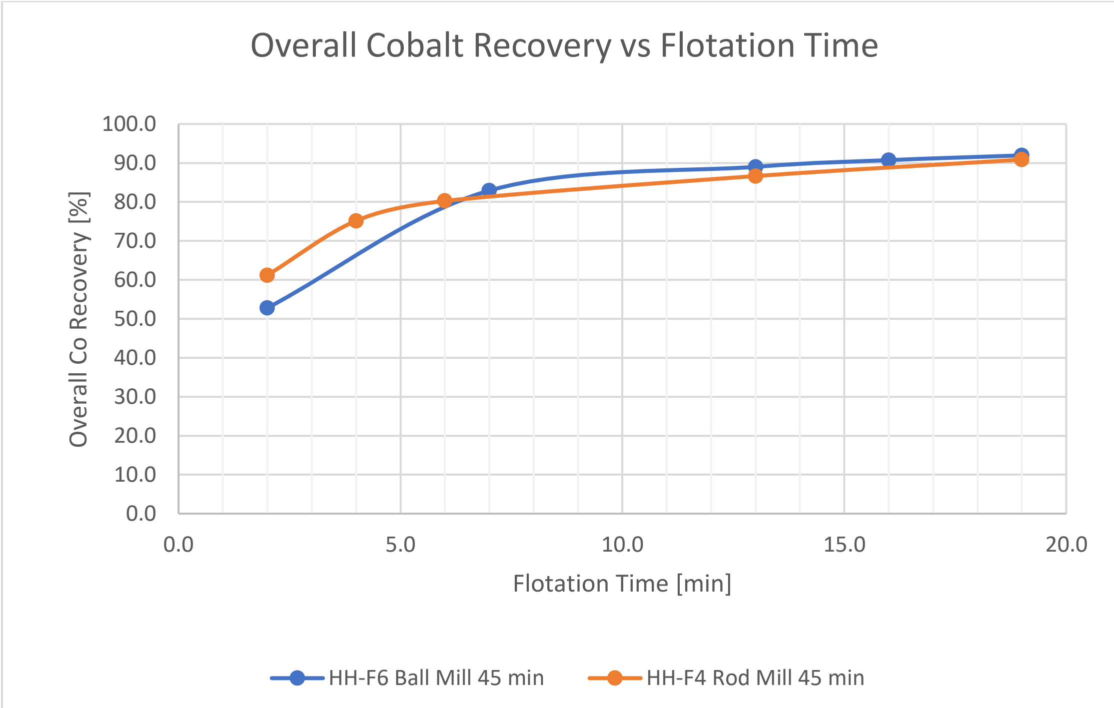


Figure 13-17: Rod Mill vs Ball Mill Grind Batch Flotation Comparison

13.3.10.4 Ball Mill Grind Tests

Further comparative tests were conducted in order to establish the optimal grind time when using ball mill grinding. By varying the grind time between 35 and 45 minutes (corresponding with a K80 variation of 64 to 76 µm respectively) for HH composite material, the overall performance was similar. The flotation performance of tests of the two-ball mill grind times is depicted below in Figure 13-18.

Similar results were achieved, indicating that a finer particle size did not improve the flotation recovery. This demonstrates that a flotation feed grind size of ~75 µm is adequate. Similar ball mill grind time tests were conducted on the other composites in order to obtain optimal grind times. AG composite required 30 minutes grind time, while 35 minutes was selected for HL composite.

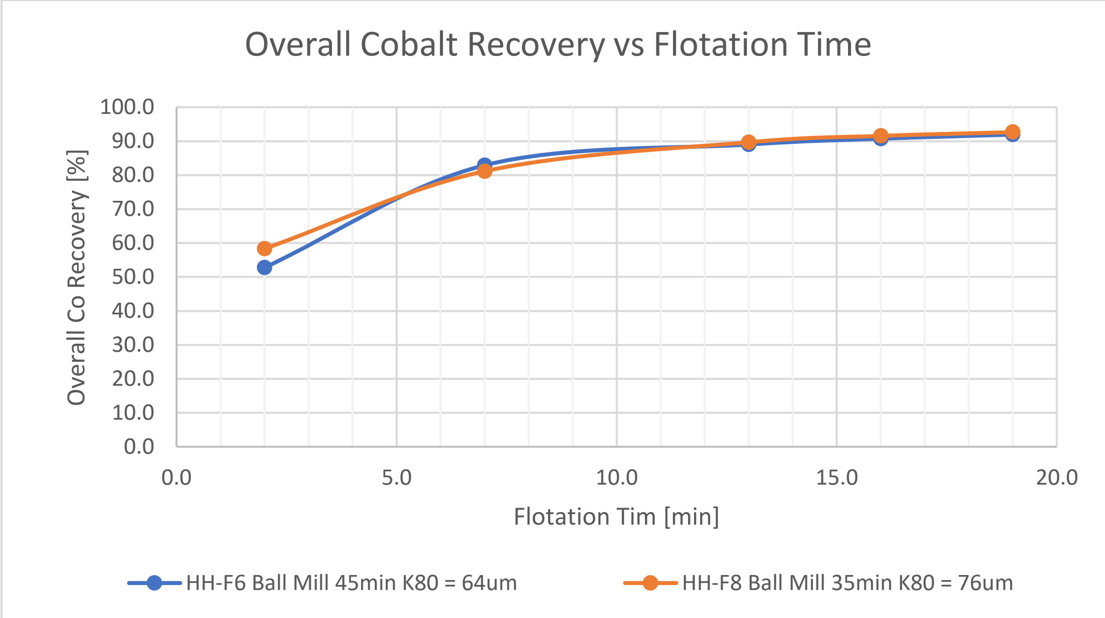


Figure 13-18: Ball Mill Grind Time Batch Flotation Comparison

13.3.10.5 Flotation Energy Optimization

All tests conducted had been based upon standard energy (speed) of the Denver flotation cells utilised. High speed (2000 rpm) against the vendor recommended standard speed (1700 rpm) was tested in the cobalt rougher flotation circuit. Low speed (1500 rpm) against standard speed (1700 rpm) was tested in the copper flotation circuit.

Depicted below in Figure 13-19 is the impact of increased impeller speed/energy in the cobalt rougher circuit. High speed/energy (2000 rpm) clearly demonstrated a better cobalt overall recovery as well as faster kinetics compared with standard energy/speed. The froth of the tests with high speed was noted to be visually clearer. It is thought that the high energy/speed may facilitate the interaction between the cobaltite particles (especially the fine particles) and collector, while simultaneously reducing the non-sulphide gangue minerals attachment (entrainment) within the froth bubbles.

High energy flotation within the cobalt rougher flotation cells was adopted within the testwork program and Phase 2 locked cycle tests. Standard energy, however, was used for the cobalt cleaning circuit.

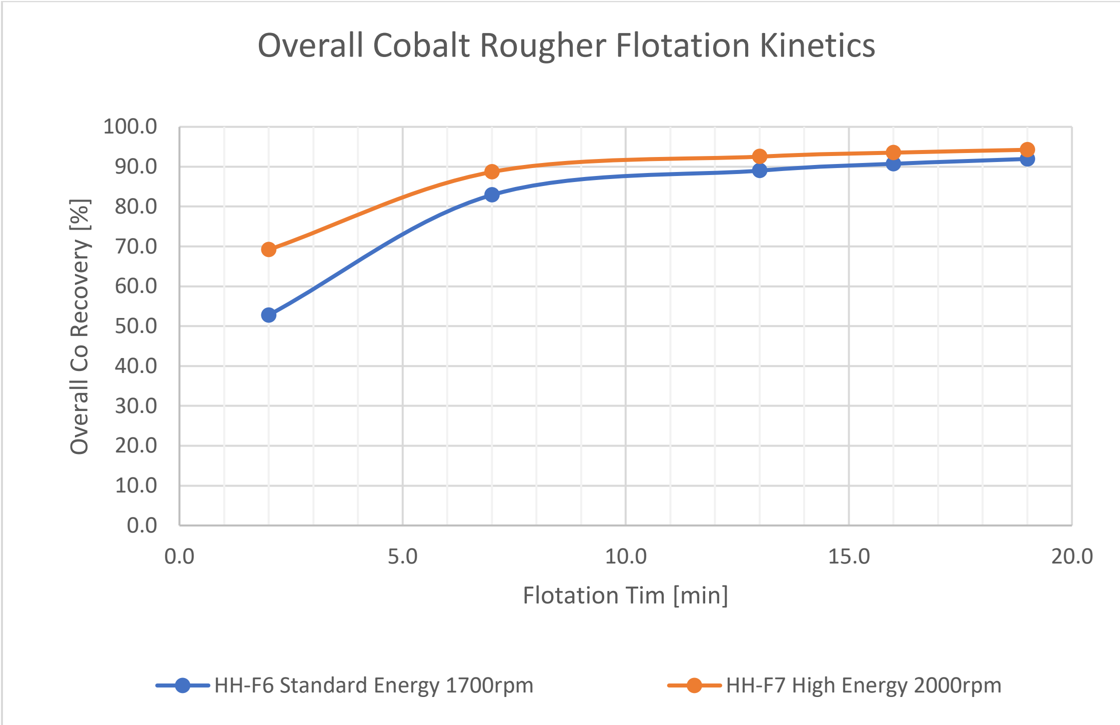


Figure 13-19: Variation in Cobalt Rougher Flotation Energy

13.3.10.6 Frother Reagent Selection

Methyl Isobutyl Carbinol ("MIBC") provided a superior flotation performance when compared against F507 frother.

13.3.10.7 Cobalt Collector Selection

Selective collector Sodium Isopropyl Xanthate ("SIPX") was evaluated against the generally non-selective Potassium Amyl Xanthate ("PAX") collector. The tests showed PAX provided greater cobalt recoveries when compared to SIPX.

13.3.10.8 PAX Collector Dosage

The dosages of the PAX collector was evaluated on three composite materials (HH, LH, and AG composites) with the results depicted in Figure 13-20.

Cobalt flotation performance with half the collector dosage for HH composite achieved higher cobalt grade and recovery initially (2 minutes) but the final cobalt recovery (16 minutes) was slightly lower than with standard PAX dosage (260 g/t).

For LH Comp, half the PAX dosage clearly resulted in improved cobalt grade and recovery curve when compared with the regular PAX dosage (260 g/t). It is possible that the differing results are due to the difference in composite head grades. A lower collector dosage might be enough for composites with low cobalt grade.

After blending the two low-grade composites LH and LL Comps to make the AG Comp, similar results were achieved when 30% less collector dosage was used.

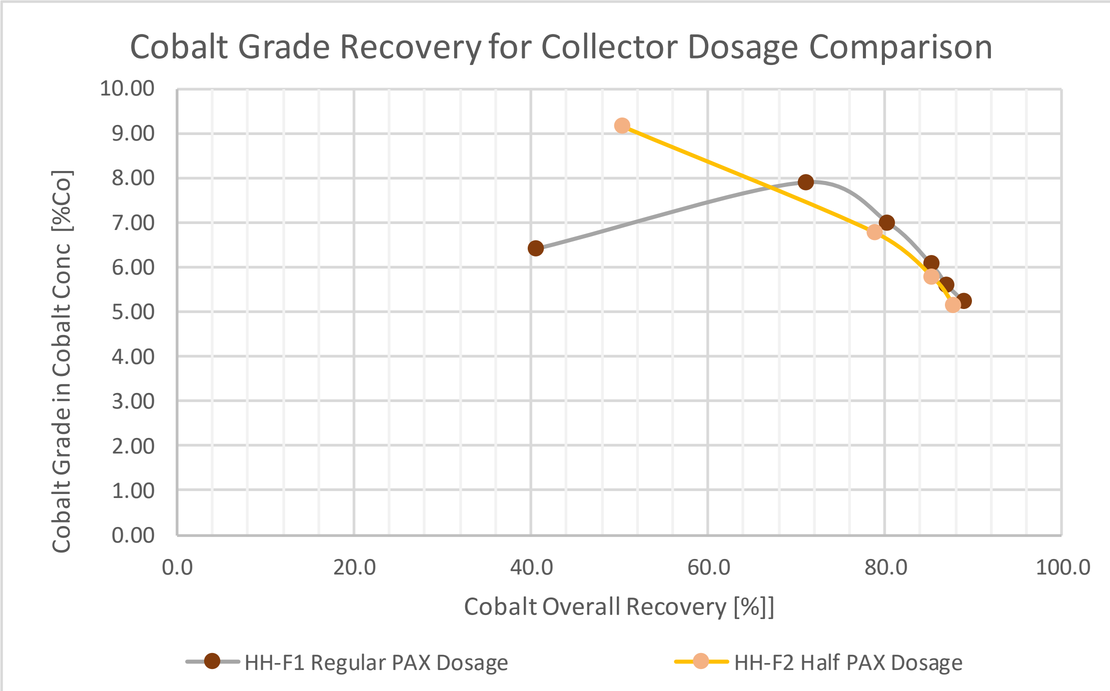


Figure 13-20: Collector Dosage Comparison

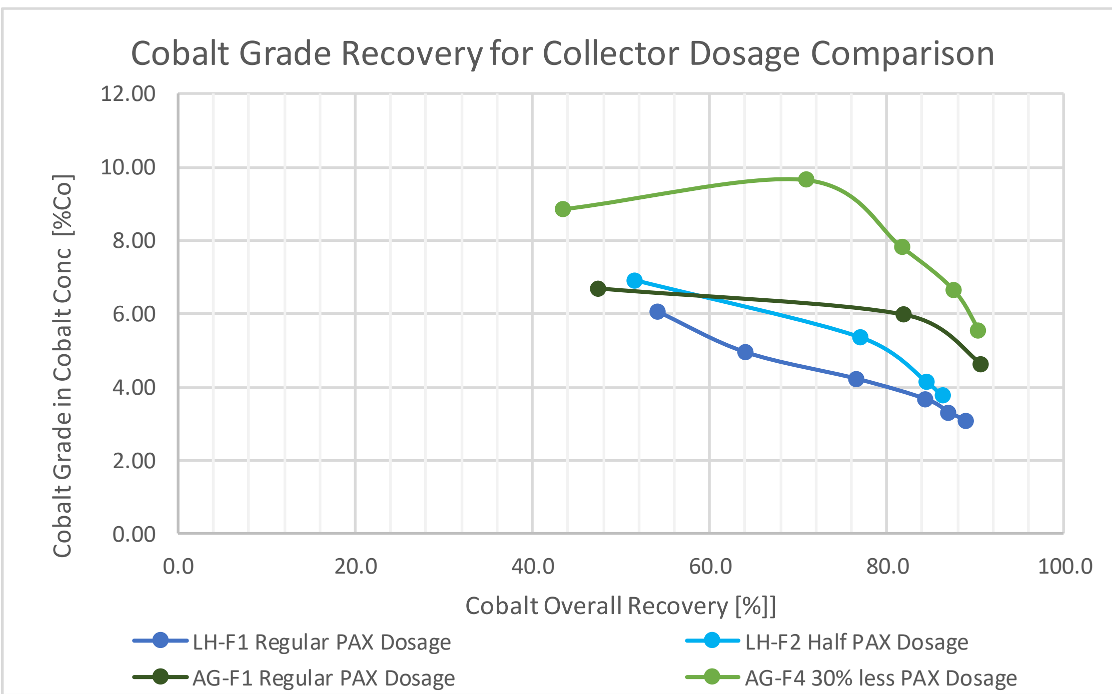


Figure 13-21: Collector Dosage Comparison

13.3.10.9 Oxidised Material

An oxidised composite consisting of representative oxidised material that could realistically be treated within the plant (i.e. was above cut-off grade) was evaluated. The oxide composite material tested in batch flotation was less oxidised than the material sent for comminution testing at Grinding Solutions.

The oxide composite for flotation was hard to grind and achieving a target size ~75 µm using a rod mill proved difficult. Consequently, the initial oxide test, where a grind size of ~100 µm was achieved, resulted in very poor overall cobalt recoveries. In addition, extremely slow copper kinetics were also experienced with a high copper loss to cobalt tailings which is likely related to the rimming of secondary copper mineral (covellite) on the chalcopyrite surfaces.

In the second oxide test utilizing a ball mill, a K80 of 68 µm was achieved, but viscosity problems (paste-like mill slurry product) like those experienced at Grinding Solutions was noted within the mill. A third oxide composite test utilised a ball mill grind at lower density (50% mass) and a higher collector dosage. This improved overall cobalt recovery to 60.5 – 74.1% after 16 – 19 minutes of flotation time respectively.

The projects geological model has indicated that oxide material is not prevalent within the ore body. Consequently, no more than 15% mass of plant feed will consist of oxide material. In order to evaluate whether the oxide material had any poisoning effect, mixtures of 10% and 30% Oxide: AG composite were evaluated. The selectivity of copper and cobalt in the copper circuit when blending with oxide material was like the standard AG material. In line with expectations, the recovery of copper and cobalt in the copper circuit with oxide mixed material were both lower than without oxide material. As a result of the slower chalcopyrite kinetics (covellite rimming), less copper was recovered in the copper rougher and much more copper reported to the cobalt circuit when treating an oxide mixture of material. The tests did not seem to show poisoning of the AG material.

Subsequent measurement of the residual xanthate within the filtrate water indicated a significant reduction in concentrations with an increasing oxide content. It is possible that an increased dosage, particularly in the 30% oxide test, could have improved cobalt recovery.

13.3.10.10 Pulp Density

Most batch flotation tests were conducted utilizing 2 kg charge at a standard pulp density (~33% mass) in the rougher circuits. Two tests were performed on AG and HL composite material to identify the effect of lower pulp density (28% mass) on the flotation by reducing the charge mass to 1.6 kg. Grind times in the ball mill were reduced to maintain similar grind sizes.

Lower pulp density results were similar when compared against standard densities for the copper flotation. Similar results were also obtained for the cobalt flotation but with marginally lowered grades and recoveries which were likely due to differences in grind sizes from the primary ball mill.

13.3.10.11 Mine Water

A batch test was conducted to compare mine water with SGS tap water. Mine water was obtained from a borehole located on the mining site. The results are depicted below in Figure 13-22 indicate very similar performance could be expected for the different water types.

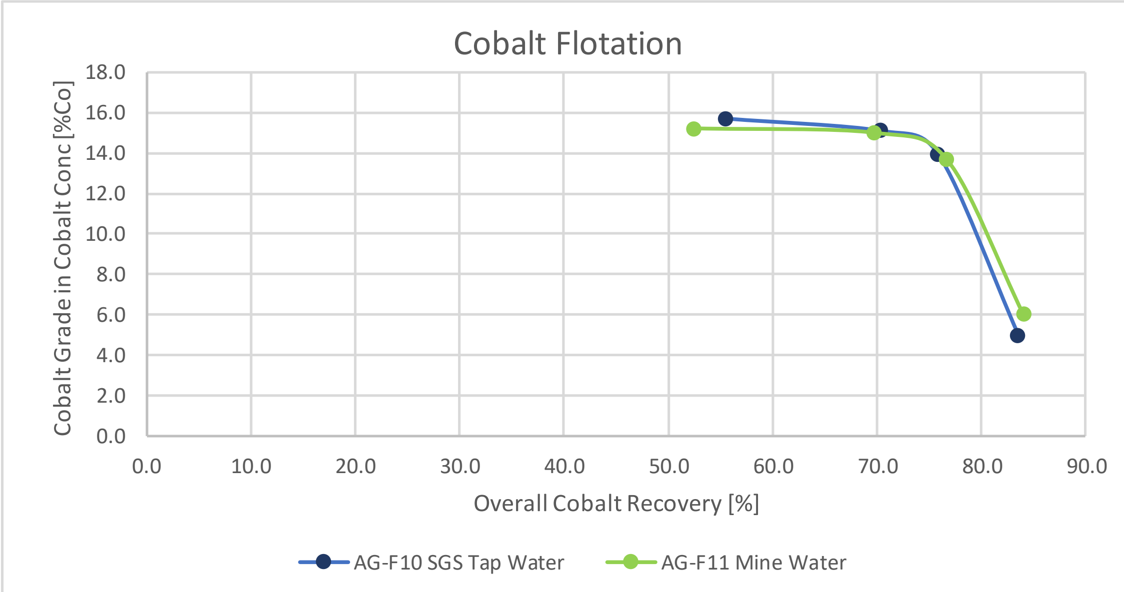


Figure 13-22: Mine Water Compared to SGS Tap Water

13.3.11 Solid Liquid Separation Tests – Phase 2

Solid-liquid separation testwork was conducted on final tailings of Phase 2 locked cycle tests 1 and 2 material. Vacuum filtration testwork was prioritised within the project schedule, and consequently, material derived from HL-F8 and AG-F9 batch flotation tests was used. The characteristics of this material are tabulated below in Table 13-28.

Table 13-28: Solid Liquid Separation Test Material Characteristics

Sample ID	Material Size				S.G. Dry Solids	pH
	d_{80} [1] [µm]	< 20 µm[1] [% vol]	< 1 µm[1] [% vol]	K_{80} [2] [µm]		
Thickening Test	91.0	27.5	1.1	73.0	2.9	7.6
Vacuum Filtration Test	82.0	28.8	0.9	78.0	2.9	7.8

Note 1: Determined using laser diffraction (Malvern)
Note 2: Determined using sieve sizing

13.3.11.1 Thickening

Flocculant scoping trials for thickening testwork showed the sample responded well to Magnafloc 10, a very high molecular weight, slightly anionic polyacrylamide flocculant. Static settling tests were conducted with the optimised results tabulated below in Table 13-29.

Table 13-29: Static Settlement Test Results

Dosage	Diluted Feed	Thickener Underflow	Unit Area	Settlement Rate	Overflow Description
g/t	% mass	% mass	m²/t/day	m³/m²/day	
30.0	10.00	63.00	0.16	716	Hazy

Subsequent dynamic raked thickener testwork where the mud bed height was maintained at 150 mm results are tabulated below in Table 13-30. After 30 minutes of no feeding and raking with a unit area of 0.16 m²/t/d, the underflow solids increased from 52.3% mass to 60.1% mass that had a yield stress of 43 Pa.

Table 13-30: Dynamic Thickener Test Results

Solids Loading	Unit Area	Net Rise Rate	Underflow Solids	Residence Time	Overflow TSS
t/m²/h	m²/t/d	m³/m²/d	% mass	hr.	mg/L
0.210	0.20	42.5	52.00	2.10	14
0.250	0.17	50.0	51.50	1.79	66
0.260	0.16	53.1	52.30	1.68	83
0.280	0.15	56.7	50.80	1.58	67
0.320	0.13	65.4	46.50	1.37	63

13.3.11.2 Vacuum Filtration

Tailings vacuum filtration tests were conducted before the thickening tests in order to confirm project flowsheet equipment selections. The slurry that was not flocculated was naturally settled to 58% mass solids concentration. Vacuum filtration tests at a pressure of -0.68 bar were conducted on cake thicknesses 16 – 37 mm. The results are tabulated below in Table 13-31.

Table 13-31: Vacuum Filtration Testwork Results

Form Time	Dry Time	Form/Dry	Cake Thickness	Throughput	Cake Moisture	Filtrate TSS	Cake
[s]	[s]	Ratio	[mm]	dry kg/m²/h	% mass	mg/l	Texture
74	8	9.46	36	2143	28.6	88	Wet
51	10	5	26	2077	28.5	145	Wet
52	31	1.67	26	1510	26.9	145	Sticky
54	90	0.6	26	877	23.3	113	Dry
51	255	0.2	26	416	19.6	130	Dry
26	260	0.1	16	263	16.1	156	Dry
81	587	0.14	37	263	18.5	95	Dry

13.3.12 Tails Compaction Tests – Phase 2

A standard proctor compaction test was conducted at SGS on the same material utilised for thickening testwork (i.e. cobalt tailings from locked cycle test 1 and 2). This resulted in the following conclusions:

- Maximum wet density of the compacted tails was 2.09 g/cm3
- Maximum dry density of the compacted tails was 1.685 g/cm3
- Optimum moisture content was 19.2% mass

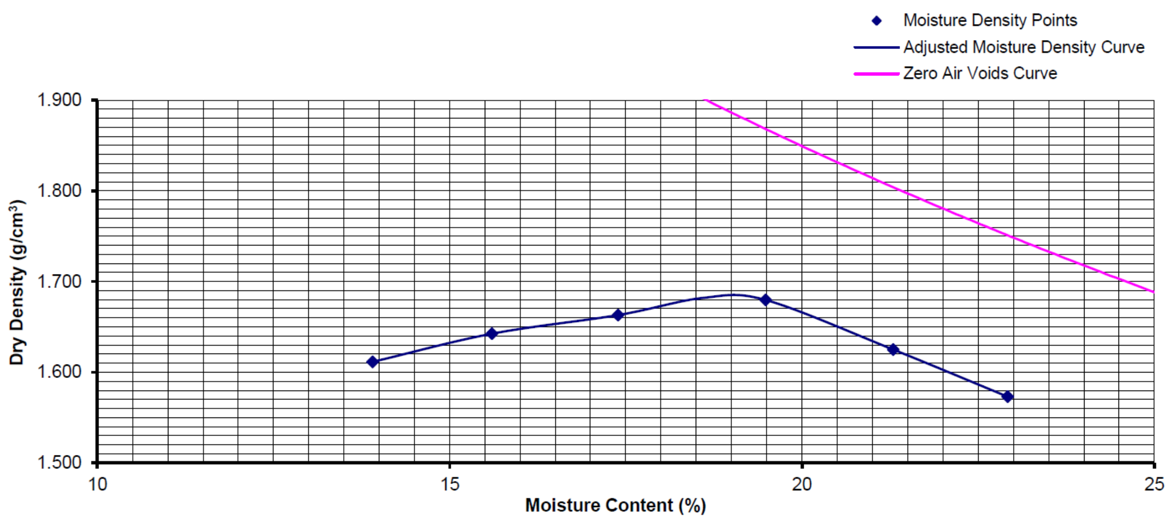


Figure 13-23: Compacted Tails Moisture Density Relationship

13.3.13 Concentrate Generation Flotation Testwork – Phase 3

In this phase of the project, the focus was to produce cobalt and copper concentrates in 12 kg batch flotation tests on the Phase 3 composite material (Bulk Comp 1) using a sequential flotation flowsheet similar to that derived in Phase 2. However, this still allowed additional testwork to be performed. Only testwork findings relevant to the bulk sulphide flotation are noted below:

- Primary grind times were adjusted with the aim of reducing the product size using a 12 kg rod mill. However, the flotation feed size K80 remained relatively constant (96 – 98 µm) for grind times 75 – 105 minutes. This is assumed to be for similar reasons to those experienced in Phase 2.
- The longer grind times resulted in increased cobalt losses to copper concentrate for similar copper recoveries. Within the cobalt flotation, results were similar. However, the shortest grind time (75 min) resulted in the highest cobalt recovery. This is likely due to the rod mill providing insufficient energy to liberate locked cobaltite from silicates, while simultaneously over-grinding already liberated material.
- The effectiveness of a higher pH within the primary grind using different modifiers (no pH modifier, lime, soda ash and sulphuric acid) was considered. No lime addition provided the maximum cobalt recovery for this material, but with a marginally elevated copper content in the cobalt concentrate. As a result, no pH modifier was used either in the primary grind or the cobalt flotation circuit.

13.3.14 Pilot Plant Concentrate Generation – Phase 4

Approximately 3.1 tonnes of surface material were delivered to SGS for the purpose of generating additional cobalt concentrate for offtake agreement analysis. As the copper feed grade was low, a combined flotation flowsheet depicted below in Figure 13-24 was adopted to maximise cobalt recoveries.

Approximately 20% of the cobalt in the pilot plant feed was contained within the non-sulphide korignigite mineral which was not recovered well with the PAX collector flotation. Grab sample XRD analysis of the cobalt product indicated the following mineralogy of the concentrate tabulated below in Table 13-32.

The ball mill was operated at 55% solids w/w and achieved a K80 grind of 73 - 77 µm for a net power consumption of 9.0 - 12.2 KWh/ metric tonne. The following flotation balance was achieved as tabulated below in Table 13-33.

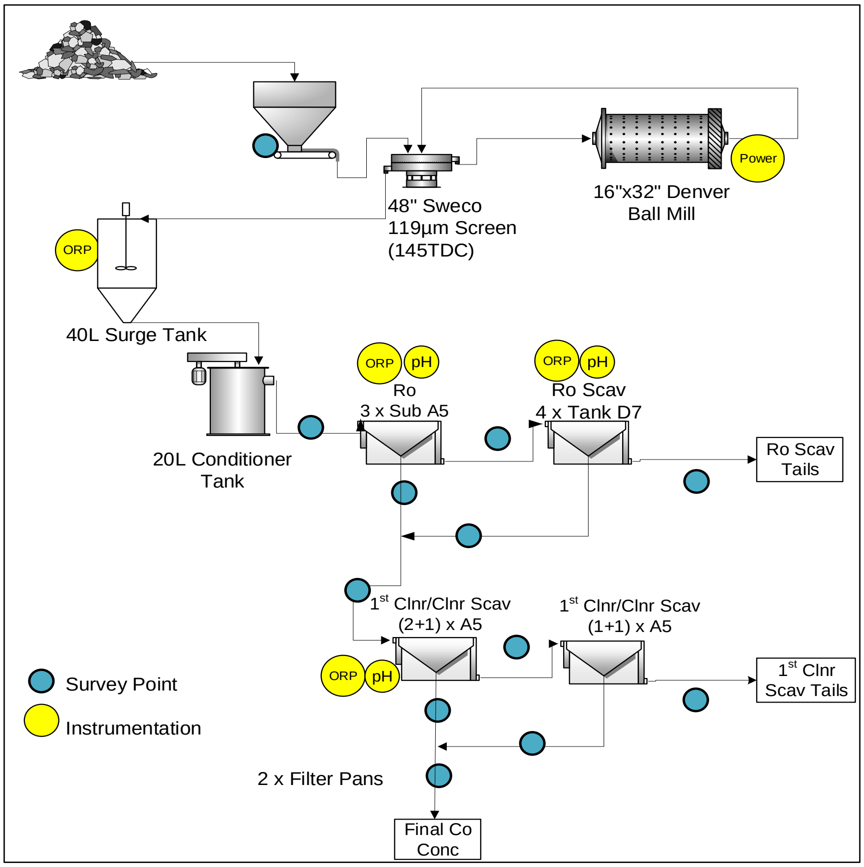


Figure 13-24: Concentrate Generation Pilot Plant Flowsheet – Phase 4

Table 13-32: Pilot Plant Feed and Concentrate Mineralogy – Phase 4

Mineral	Feed	Co Concentrate
	% Mass	% Mass
Quartz	63.4	15.8
Chlorite	27.0	
Cobaltite	4.0	74
Cobaltkorignigite	1.8	0.8
Mica	1.6	
Potassium-Feldspar	1.2	
Plagioclase	0.7	
Xenotime	0.2	
Chamosite		3.5
Biotite		2
Microcline		1.6
Anorthite		1.2
Chalcopyrite		1.1

Table 13-33: Pilot Plant Flotation Balance in Phase 4

Stream	Weight	Assay [%]				Recovery [%]			
	%	Co	Cu	As	S	Co	Cu	As	S
Ro Feed (Calculated grades)	100	1.36	0.03	2.92	0.7	100	100	100	100
Ro Conc	3.3	17	0.37	22.1	9.86	40.9	32.3	24.7	46.8
Ro Tail	96.7	0.86	0.03	2.31	0.37	59.1	67.7	75.3	53.2
Ro Scavenger Conc	8.4	5.87	0.07	9.4	3.3	36.4	15.9	26.9	39.8
Ro Scavenger Tail	88.3	0.35	0.03	1.59	0.11	22.7	51.8	48.4	13.5
1st Clnr Feed	11.7	9.09	0.18	13	5.12	77.3	48.2	51.6	86.5
1st Clnr Conc	5.5	17.5	0.25	21.8	10.2	70.4	36.8	41.8	80.7
1st Clnr Tail	6.2	1.52	0.07	4.67	0.65	6.9	11.5	9.8	5.8
1st Clnr Scavenger Conc	0.3	12.8	0.08	17	7.17	2.7	0.6	1.7	3.1
1st Clnr Scavenger Tail	5.9	0.97	0.07	3.96	0.32	4.2	10.8	8.1	2.7
Final Cobalt Conc	5.8	17.7	0.25	21.9	10	73.1	37.4	43.5	83.8

13.3.15 Gold Deportment and Recovery – Phase 6

Locked cycle tests conducted in Phase 2 showed that on average, approximately 91.7% of gold was recovered within the flotation concentrate as tabulated below in Table 13-34.

Table 13-34: Phase 2 Locked Cycle Gold Deportment

	LCT1-HL		LCT2-AG		LCT3-HH		Average	
	Au Grade [g/t]	Au Dist [%]	Au Grade [g/t]	Au Dist [%]	Au Grade [g/t]	Au Dist [%]	Au Grade [g/t]	Au Dist [%]
Feed	0.41	100	0.34	100	0.54	100	0.43	100.00
Cu Conc.	1.43	3.90	2.03	10.56	2.15	13.06	1.87	9.17
Co Conc.	10.6	87.71	9.05	83.18	9.31	76.75	9.65	82.55
Tails	0.04	8.39	0.02	6.25	0.06	10.20	0.04	8.28

Approximately 82.6% of the gold reported to the cobalt concentrate. Further testwork was then conducted in order to establish the reason for this and to establish whether alternative methods could be utilised to increase gold recovery to the copper concentrate. Only results relevant to bulk sulphide flotation are noted below.

13.3.15.1 Mineralogy

A mineralogy study was conducted using Phase 2, Locked Cycle 3 cobalt concentrate. The results are depicted below in Figure 13-25.

It was determined that the identified free gold is very rare, with a total of 17 gold occurrences being found. Of these occurrences, 15% were liberated by surface area, with the rest being either exposed or locked but typically associated with the cobaltite (sulphide). The gold was found to be fine-grained (<10 µm) with a variety of different species (native gold, electrum and various Au-Bi +/-Te phases).

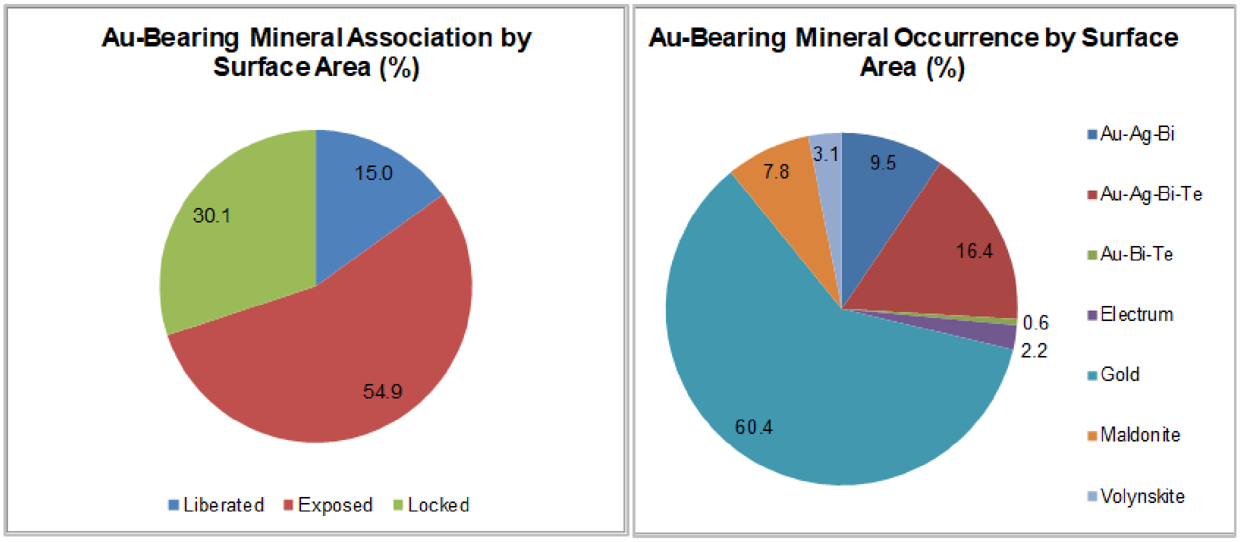


Figure 13-25: LCT-3 Gold Mineralogy Analysis

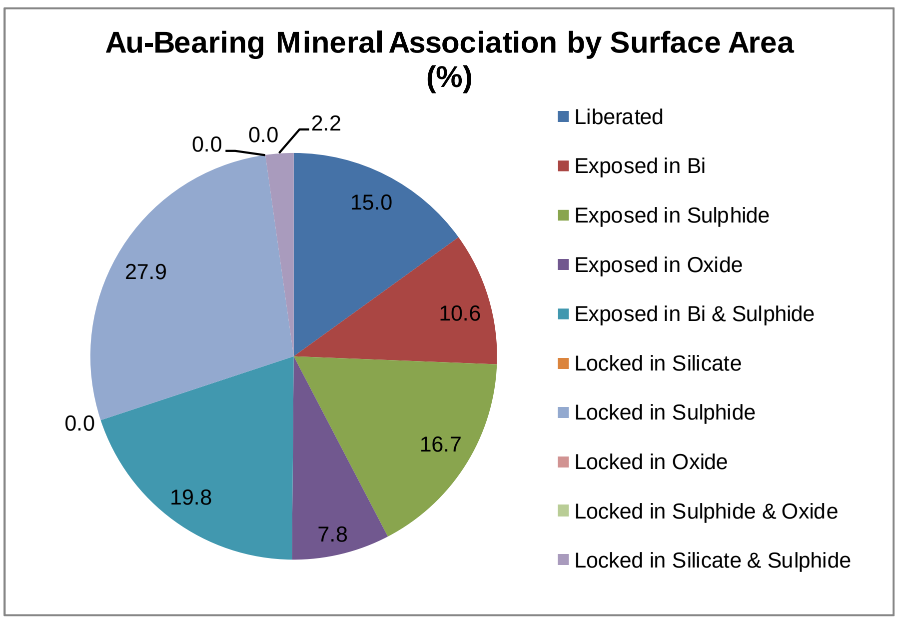


Figure 13-26: LCT-3 Gold Mineralogy Analysis

13.3.15.2 Gravity Separation

Gravity separation tests were also conducted in parallel with the mineralogy analysis to study the possibility of gravity recovery at various points within the flowsheet. The results of the gravity testwork confirmed the conclusions of the mineralogy analysis.

Mozley Shaking Table on Cobalt Concentrate

Testwork was conducted to consider the efficacy of gravity separation after the flotation process. A Mozley Shaking Table was used on cobalt concentrate obtained from Phase 2, Locked Cycle Test 3. As tabulated below in Table 13-35. The test only recovered 24.1% of the gold in 10.9% of the mass. The gold concentrate also contained 27% of the cobalt, which indicates gold could be associated with cobalt.

Table 13-35: Mozley Shaking Table Test on Cobalt Concentrate

Products	Weight		Assays, g/t or %					Distribution, %				
	g	%	Au	Co	Cu	As	S	Au	Co	Cu	As	S
Mozley Conc	10.6	10.9	19.7	31.2	0.0	35.1	20.7	24.1	27.1	0.3	25.4	10.4
Mozley Middlings	30.2	31.1	10.1	15.6	0.2	18.5	34.1	35.3	38.6	5.9	38.2	48.8
Mozley Tails	56.3	58.0	6.2	7.4	1.8	9.5	15.3	40.6	34.3	93.8	36.4	40.8
Feed Calc.	97.1	100	8.9	12.6	1.1	15.1	21.7	100.0	100.0	100.0	100.0	100.0
Feed direct			9.3	12.6	1.3	14.6	21.5					

Knelson and Mozley Table on Phase 4 Concentrate

Gravity testwork was also conducted on 5 kg of the pilot plant concentrate generated in Phase 4 of the project. The original feed material of this phase of the project was from an oxidised surface boulder outcrop that contained very low copper, but high cobalt grades. The Knelson concentrator was utilised at varied G forces in a primary and scavenging application followed by Mozley Shaking table concentration of Knelson concentrate.

Testwork results are tabulated below in Table 13-36 and are like those obtained from the locked cycle material. It was concluded that gravity separation of gold on the cobalt concentrate would not provide economic benefit for the project.

Table 13-36: Knelson Concentrator and Mozley Table Testwork Results

Products	Weight		Assays, g/t or %		Distribution, %	
	g	%	Au	Co	Au	Co
Mozley Conc	11.5	0.2	23.0	32.9	0.66	0.75
Mozley Tails	124.5	2.5	16.7	27.1	5.19	6.67
Knelson 2nd Pass Conc	171.7	3.5	16.3	29.9	6.99	10.1
Knelson 2nd Pass Tail	4579.2	93.7	7.62	9.11	87.2	82.4
Feed Calc.	4886.9	100.0	8.19	10.4	100	100
Feed direct				11.0		
Combined Products						
Knelson 1 Conc	136	2.78	17.2	27.6	5.9	7.4
Knelson 1 Tails	4750.9	97.2	7.93	9.86	94.1	92.6

Knelson Concentrator Prior to Flotation

Testwork was conducted in order to establish whether it was possible that some of the free gold could be recovered in the mill grinding circuit prior to flotation. The test flowsheet included a primary grind of material followed by a Knelson concentrator. Approximately 11 kg of Grind Comp 1 and Grind Comp 2 (phase 2 comminution composite material) utilised for this test.

These results indicate that there is marginal potential for gravity separation to be effective in the grinding circuit prior to flotation. It is also unknown how much of the free gold would report to the copper concentrate. The cost of running the gravity circuit may not be covered in the additional revenue from the gold added to the copper concentrate stream, and therefore this option was not progressed further.

Table 13-37: Pre-Flotation Gravity Gold Testwork Results

Products	Weight		Assays, g/t or %					Distribution, %				
	g	%	Au	Cu	Co	As	S	Au	Cu	Co	As	S
Mozley Conc	13.3	0.1	16.0					8.8				
Mozley Middlings	68.7	0.6	3.25	2.58	11.20	11.60	10.60	9.2	2.0	24.8	23.0	5.8
Mozley Tails	26.0	0.2	0.34	0.72	0.78	0.90	1.51	0.4	0.2	0.7	0.7	0.3
Knelson Tail	11013.6	99.0	0.18	0.77	0.21	0.24	1.08	81.7	97.7	74.5	76.3	93.9
Feed Calc.	11121.6	100	0.22	0.78	0.28	0.31	1.14	100	100	100	100	100
Feed direct			0.22	0.77	0.27	0.33	1.11					

13.3.15.3 Gold Diagnostic on Cobalt Concentrate

A gold diagnostic test was conducted on cobalt concentrate derived from the Phase 2 Locked Cycle Test 3. The purpose of the test was to confirm the deportment of gold within the various minerals. The results are tabulated below in Table 13-38. The results confirm that 68% of the gold within the cobalt concentrate is associated with cobaltite, while a further 28% is either free or unliberated, free gold.

Table 13-38: Gold Diagnostic Test Results on Cobalt Concentrate

Stage	Process	Description	Distribution (%) Au
1	*Intensive CN:*	Extraction of readily available gold	23.3
2	*Regrind to 10 µm & Intensive CN:*	Extraction of gold exposed by fine re-grind	4.9
3	*Hot Caustic Leach & Intensive CN:*	Extraction of gold possibly associated with arsenates, antimony compounds, bismuth compounds, or any other soluble compounds	1.6
4	*Hot HCl Leach & Intensive CN:*	Extraction of gold possibly associated with pyrrhotite, calcites, ferrites, dolomite, galena, and hematite	0.9
5	*Hot Aqua Regia Leach:*	Extraction of gold possibly associated with pyrite, arsenopyrite and other sulphides	68.3
	Remaining Material:	Gold locked in silicates or associated with fine sulphides locked in silicates	1.1
		Total %	**100.0**
	Overall Calculated Head (Calculated from Stage 1), g/t Au		9.16
	Direct Head, g/t Au		9.31

13.4 RECOVERY ESTIMATE

13.4.1 Introduction

A total of four locked cycle test results were conducted under the supervision of Samuels Engineering Inc and used within the 2007 feasibility study. The results of these tests are used to model the bulk sulphide flotation performances of the concentrator for this study. The performance predictions are therefore valid for the range of feed and product grades achieved together with the respective upgrade ratios within the tests, as tabulated in Table 13-39.

Table 13-39: Bulk Sulphide Locked Cycle Feed and Product Grade Range

	Copper [%]			Cobalt [%]		
	Feed Grade	Conc Grade	Upgrade Ratio	Feed Grade	Conc Grade	Upgrade Ratio
CAMMP (2001)	0.29	7.41	25.20	0.57	14.4	25.09
Comp 1 (SGS 2005)	0.44	9.7	22.05	0.59	13.3	22.54
Comp 2 (SGS 2005)	1.1	12.9	11.73	0.72	8.28	11.50
Comp 3 (SGS 2005)	0.45	7.09	15.76	1.2	17.9	14.92

The Life of Mine ("LOM") mill feed grade is depicted below in Figure 13-27. This shows that for cobalt feed grade ranges, the maximum testwork limit is not exceeded. Planned feed grades will be lower than the minimum tested grade in certain periods of the operation, particularly for cobalt. Planned feed grades of copper will exceed the tested range for certain periods of time.

The predicted concentrator recovery relationships have been extrapolated for the purpose of estimating the concentrator recoveries below and above the tested range.

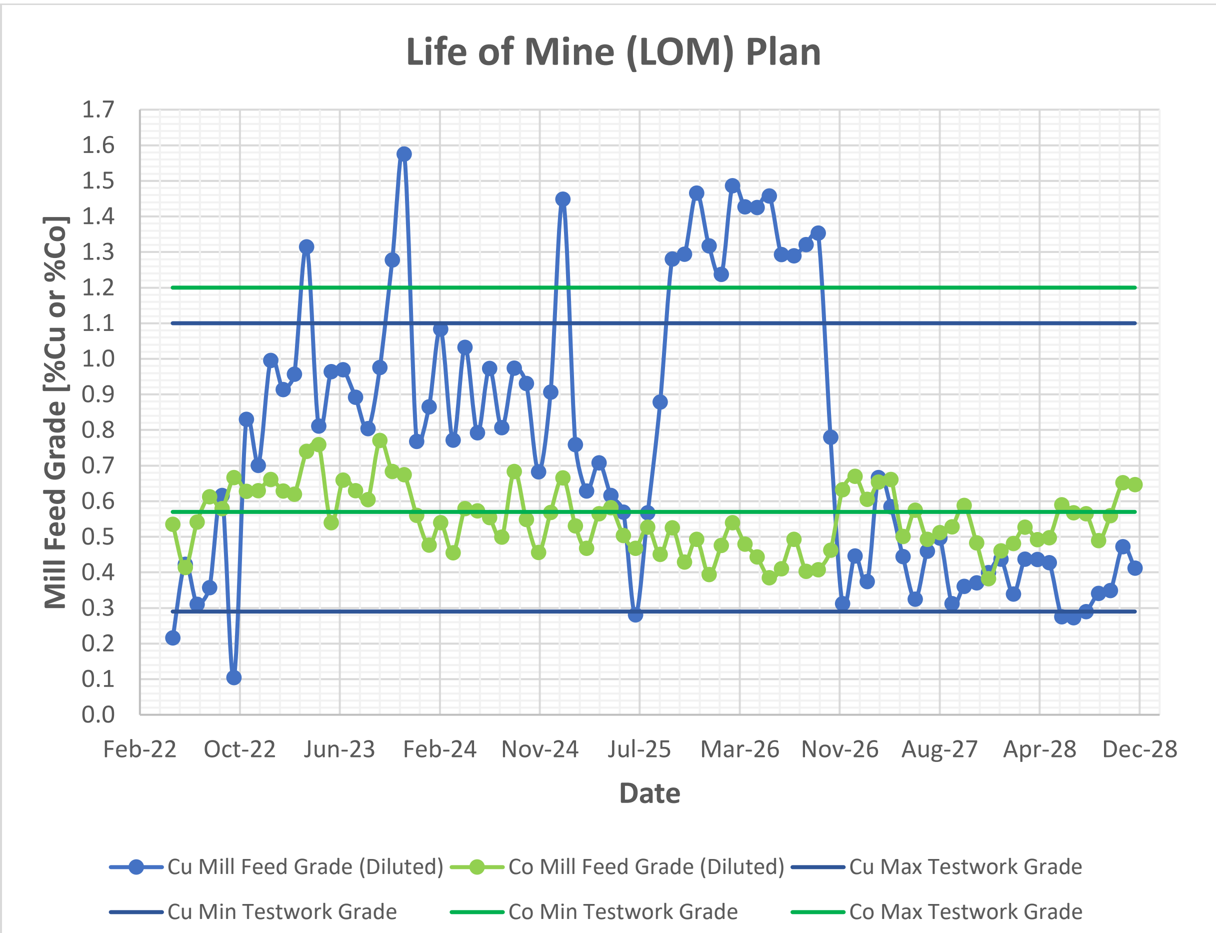


Figure 13-27: LoM Mill Feed Grade Plan

The flotation residence times selected for the flowsheet based upon the locked cycle test results are tabulated below in Table 13-40. Longer residence times have been installed for the cleaner and scavenger cleaner cells, as this equipment has already been purchased by the project.

Table 13-40: Selected Flotation Residence Times

Description	Laboratory Float Time (mins)		Plant Float Time (mins)	
	Target	Installed	Target	Installed[1]
Rougher	9	12.6	22.5 [2]	31.4
Cleaner	3.5	11.12	8.8	27.8
Scavenger Cleaner	2.0	3.1	5.0	7.75

Note 1: Excluding an additional 15% active air holding volume
Note 2: Scale-up factor of 2.5 × laboratory residence time

It should be noted that the modelled recoveries are steady-state recoveries and do not consider any transient operations. The modelled recovery profile includes a feed rate ramp up and initial recovery discount in the first months of operation to cater for start-up and commissioning.

It is also noted that flotation recovery for full-scale operations could be lower than that achieved in the laboratory locked cycle tests due to operational inefficiencies such as those listed below:

- Ore blend. No allowance has been made for a reduction in recoveries while treating oxide material which has been advised to form a very small part of the resource. ICO advises that the concentrator will never treat more than 15% of oxide material in its feed.
- Ore grade. The plant performance predictions are based on testwork samples within a feed grade range only. Where the Life of Mine (LOM) plan is outside of this range, the model has been extrapolated.
- Power: The laboratory flotation cell power (and air) inputs are extremely high (typically in the order of 10 kWh/m^3). This may tend to give higher recoveries due to the improved fines (<20 µm) recovery.
- Recycle water. Water sourced from the mine was utilised and recycled during the locked cycle tests. Although this is good practice, the only way to confirm the impact of the recycle water on the flotation process is to complete a full cycle flotation test in a pilot plant, where issues such as dissolved metal ion concentration in solution and oxidation-reduction potential in the mill can be identified and minimised.
- Operating conditions: Laboratory operation is undertaken under controlled, 'ideal' conditions. Operational disturbances on full-scale operations such as starting and stopping of the plant undoubtedly cause loss of recovery.
- Operational skills: The bench scale laboratory tests are supervised by 'expert' operators. In the actual plant recovery, losses may occur as a result of poor operational practices.

In order to address as many of these problems as possible, the plant design will allow a reasonable level of instrumentation and control within the flotation and milling circuit with the allowance for the installation of an online analyzer at strategic points within the flowsheet to allow for improved flotation control. Process operators need to be trained and supervised to reduce the occurrence of losses due to poor operational practices.

13.4.2 Flotation Performance Prediction

The locked cycle test data points show a strong relationship between the target Upgrade Ratio (Conc Grade / Feed Grade) and Mass Pull to Concentrate (mass concentrate / mass feed as depicted below in Figure 13-28.

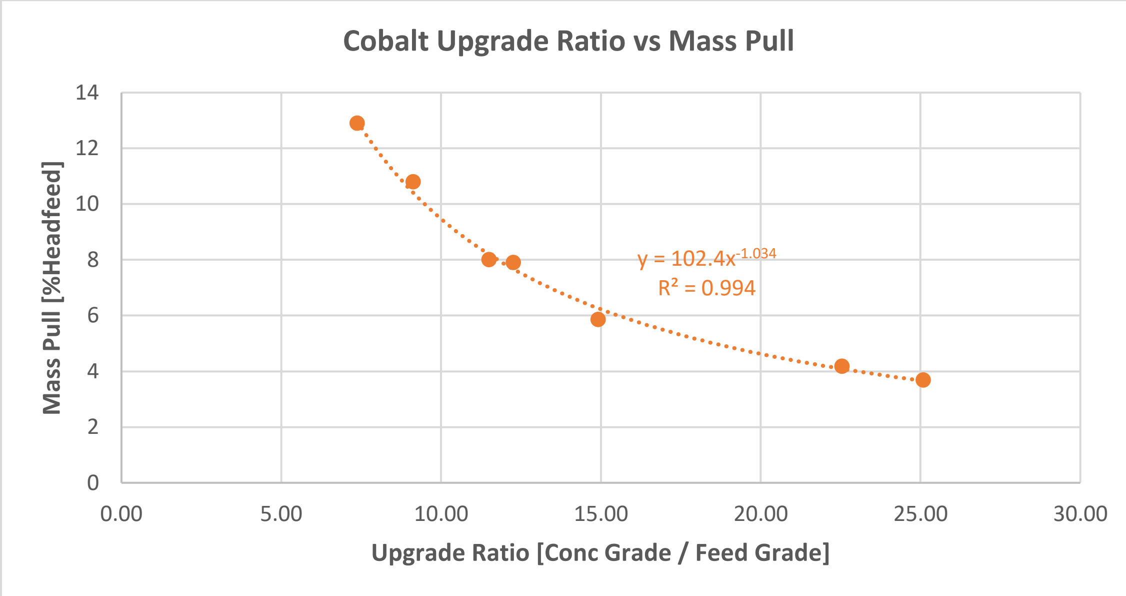


Figure 13-28: Cobalt Upgrade Ratio vs Mass Pull

The relationship between Copper and Cobalt metal recoveries compared against the mass pull is depicted below in Figure 13-29. Copper metal recovery provided are reasonably strong correlation; however, cobalt did not. On investigation, it was found that the two lower data points were derived from the same locked cycle test (Comp 2) which had a high copper:cobalt in feed ratio. Therefore, the relationship between cobalt metal recovery and the copper:cobalt in feed ratio was rather utilised for cobalt recovery modelling.

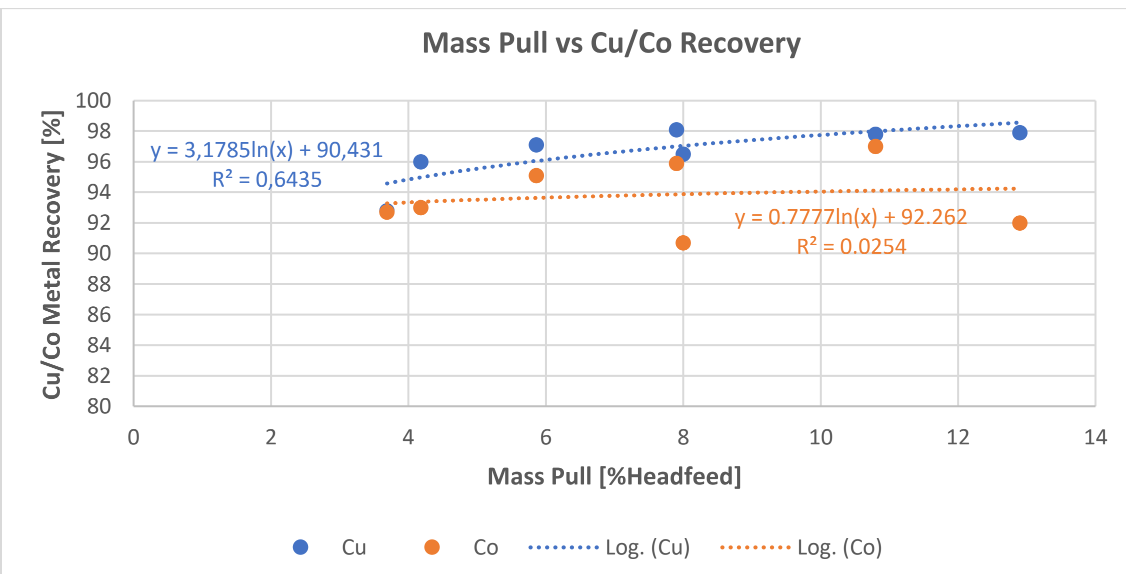


Figure 13-29: Mass Pull vs Cu / Co Metal Recovery

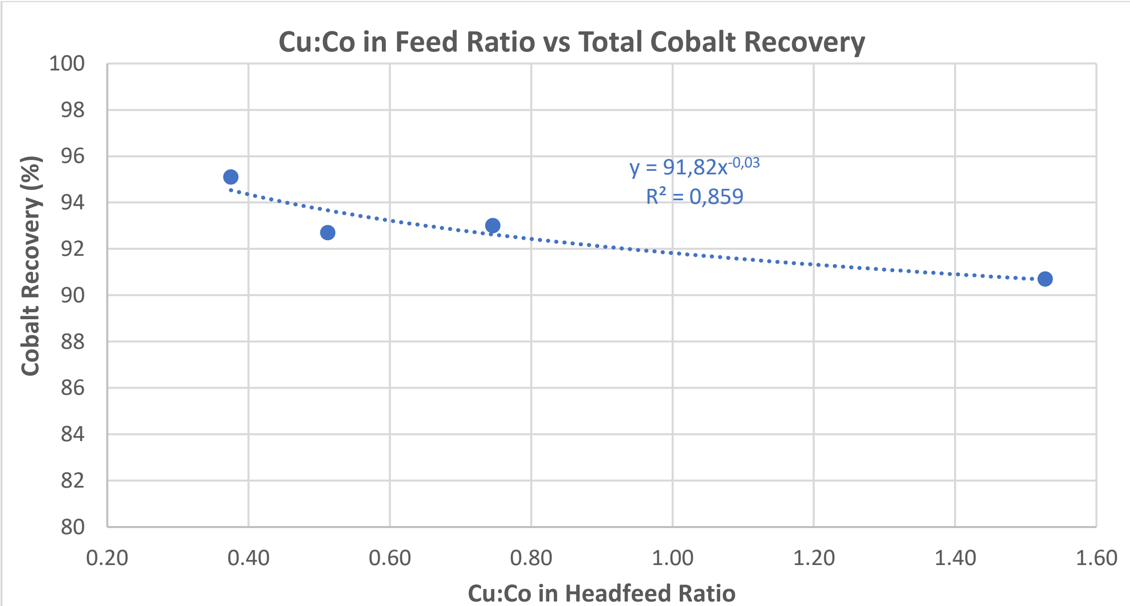


Figure 13-30: Cu:Co in Head Feed Ratio vs Cobalt Recovery

13.4.3 Gold Deportment Prediction

Gold recoveries for the locked cycle tests are tabulated in Table 13-41. Due to the low feed grades of gold analysed, the numerical average recovery to the concentrate has been used.

Table 13-41: Locked Cycle Gold Deportment

LCT Test	Feed	Product	Product
	Au [g/t]	Au [g / t]	Au Recovered [%]
CAMMP Comp	0.69	not available	72.9
Comp 1	0.35	7.02	90.3
Comp 2	0.69	6.75	84.5
Comp 3	0.67	10	92
		Average	84.93

13.5 DELETERIOUS ELEMENTS

The bulk copper/cobalt concentrate could be subject to penalty conditions, should significant grades of deleterious elements be present. At the time of report writing, no project offtake agreement has been concluded. Reference should be made to the detailed elemental analysis of the locked cycle test products for a potential list of these elements.

There is a strong correlation between arsenic and the cobalt grade within the concentrate product as depicted below in Figure 13-31.

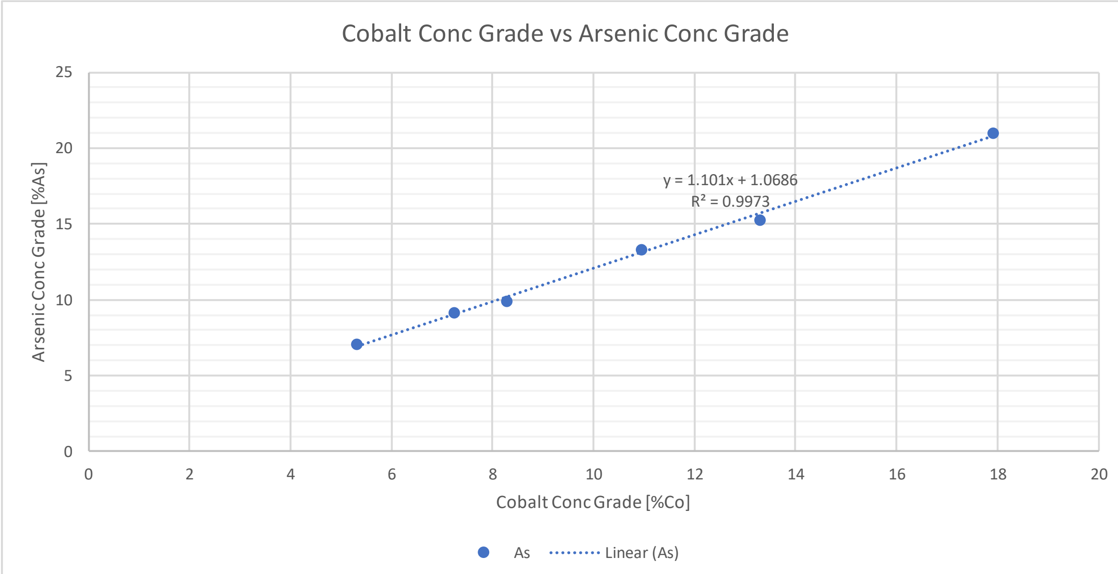


Figure 13-31: Relationship of Cobalt to Arsenic Grade in Concentrate

14 MINERAL RESOURCE ESTIMATES

The following resource estimation was completed by Scott Zelligan, P. Geo, with an effective date of January 20th, 2020.

14.1 DATA

Drill hole sample data (.xlsx files, .dm files), and wireframes (.dxf, and .dm files) for this resource estimate were supplied by Orix Geoscience, and imported into both CAE Datamine Studio© software (version 3.21) and GEOVIA Surpac™ software (version 6.3) and subsequently verified by standard internal Datamine and Surpac processes. These .xlsx files contain collar, survey, lithological, specific gravity, and assay data collated by Orix Geoscience and confirmed by the author. Assay data is from split diamond drill core or in selective cases from whole-core metallurgical samples. This estimate includes results from 111 drill holes completed on the Project to date.

14.2 INTERPRETATION

14.2.1 Geological Interpretation

The Ram Deposit is hosted by fault-bounded, meta-sedimentary sequence believed to be marine in origin. The rock types are dominantly quartzite and argillite. The sequence strikes north-northwest, and dips 50° to 60° to the west.

Given the transitional nature of the sediment package, special care and attention during current and historical logging programs was made to record the Biotite/Chloritic alteration which is associated with the cobaltite mineralization. This represented a unique challenge in modelling, because the host of mineralization is a combination of lithological unit and alteration. The main Ram horizon, called the main zone or "mmh" zone, varies between 15-20 ft and up to 60-80 ft in some places, with soft-sediment deformation and other structural features presenting as overlapping or split horizons in some areas of the deposit.

14.2.2 Wireframing

Wireframing of the mineralized zones as well as a fault system was performed in Seequent Leapfrog Geo© (v. 5) by Edwin Escarraga, M.Sc., P.Geo., of Orix Geoscience, who also was one of the primary core loggers on the latest drill program. The author was informed, provided feedback, and reviewed the wireframing throughout the process and agrees with the interpretations made with the current information available.

A full geological model, including lithological and structural modelling (based on the interpretation of a structural geologist), was created for the Ram deposit; for the purposes of this estimate the following wireframes were used:

- One wireframe representing the main zone or "mmh" zone (jer_ram_gm_mmh.dxf)
- One wireframe representing the black tail north zone or "btn" zone (jer_ram_gm_btn1.dxf)
- One wireframe representing the footwall zone or "fw" zone (jer_ram_gm_fw.dxf)
- Six wireframes representing the hanging wall zones
- "hw0" zone (jer_ram_gm_hw0.dxf)
- "hw1" zone (jer_ram_gm_hw1.dxf)
- "hw2" zone (jer_ram_gm_hw3.dxf)
- "hw3" zone (jer_ram_gm_hw3.dxf)
- "hw3a" zone (jer_ram_gm_hw3a.dxf)
- "hw4" zone (jer_ram_gm_hw4.dxf)
- One wireframe representing the primary faults (FLT System.dxf)

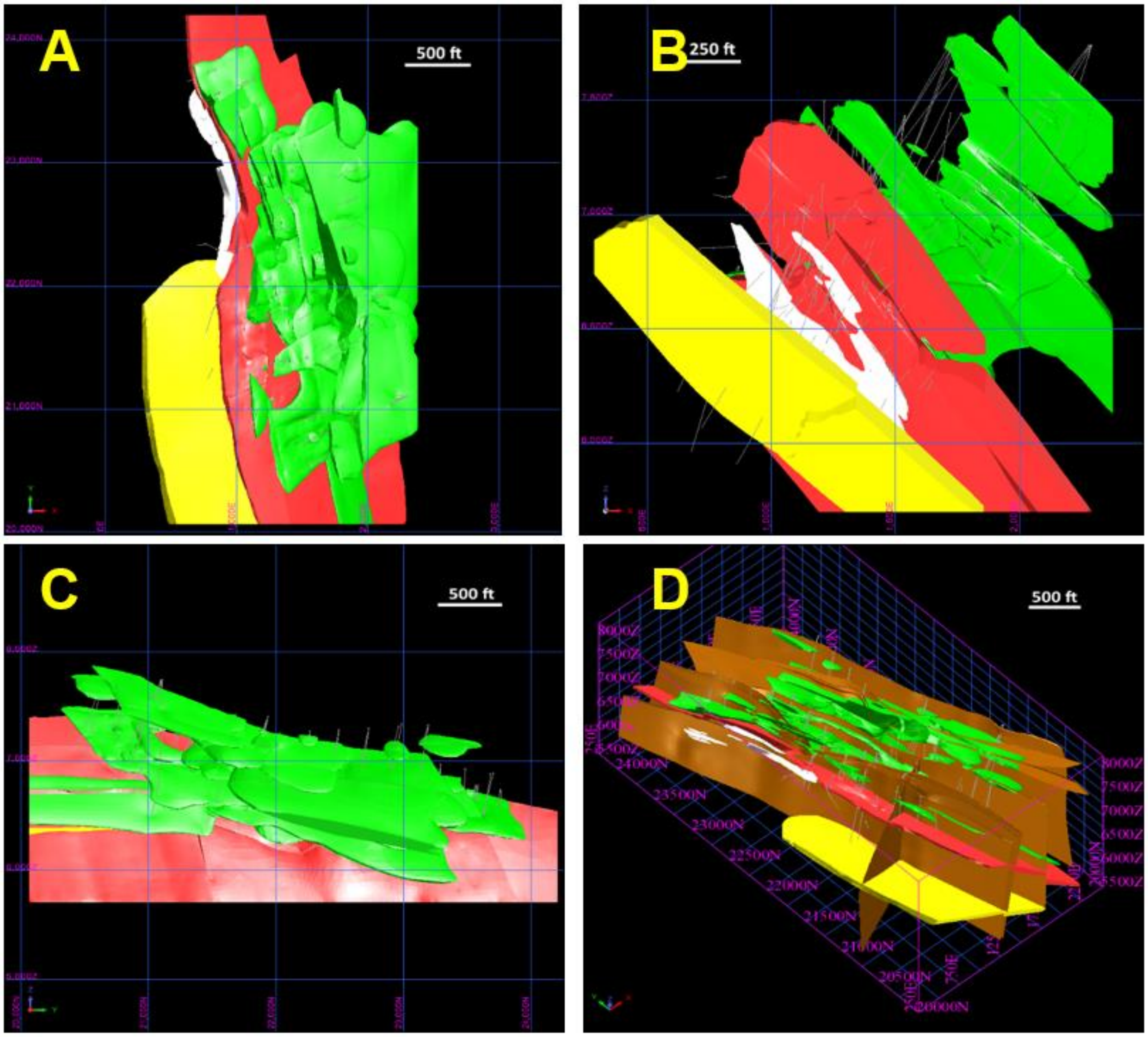


Figure 14-1: 3D Orthogonal View of Modelled Wireframes in Local rid

(drilling in grey, red – main zone "mmh", green – hangingwall zones "hw", white – footwall zones "fw", yellow – black tail north "btn" model): A – Plan View; B – Looking North; C – Looking West; D – Orthogonal view of lithologies with faults (brown)

All nine (9) of the lithology wireframes were shifted according to the primary fault model and clipped to topography. The primary fault model was used to label blocks for the consideration of Exploratory data analysis, including general statistics, histograms, probability plots, and contact profiles were used to test the validity of these models, and are discussed in the next sub-section.

14.3 EXPLORATORY DATA ANALYSIS

14.3.1 Raw Data Assays and Statistics

The uncomposited, uncapped data set contains 5,567 assayed intervals discussed in this section. Desurveying the assays into a drill hole file with lithological and selection zone resulted in a data set of 6,130 intervals. Of the 75,278 feet drilled, 60,669 feet is unassayed.

Summary statistics for the desurveyed raw assay data for each of the nine wireframed zones, as well as the “wall” rock, is shown in Table 14-1. These are length-weighted statistics.

Table 14-1: Summary Statistics, Raw Data

ZONE	FIELD	NSAMPLES	MIN	MAX	MEAN	VARIANCE	STANDDEV	STANDERR	CoV	SKEWNESS
btn	Co_per	61	0.002	0.115	**0.013**	0.00	0.02	0.00	1.62	3.214
fw	Co_per	302	0.002	5.060	**0.102**	0.09	0.31	0.01	3.01	10.709
hw0	Co_per	247	0.002	1.725	**0.053**	0.02	0.13	0.01	2.42	7.563
hw1	Co_per	857	0.002	4.170	**0.080**	0.08	0.28	0.01	3.55	7.759
hw2	Co_per	151	0.005	1.939	**0.059**	0.03	0.17	0.01	2.92	6.745
hw3	Co_per	106	0.005	4.000	**0.116**	0.17	0.42	0.03	3.58	7.055
hw3a	Co_per	8	0.010	0.030	**0.023**	0.00	0.01	0.00	0.30	-0.482
hw4	Co_per	8	0.020	0.510	**0.064**	0.01	0.12	0.02	1.83	3.423
mmh	Co_per	1446	0.002	10.650	**0.270**	0.35	0.59	0.01	2.19	5.632
wall	Co_per	2944	0.000	5.740	**0.053**	0.04	0.20	0.00	3.85	14.784
btn	Cu_per	61	0.000	1.585	**0.154**	0.06	0.25	0.02	1.63	3.322
fw	Cu_per	301	0.000	2.180	**0.121**	0.05	0.22	0.01	1.83	4.992
hw0	Cu_per	247	0.001	3.590	**0.126**	0.07	0.26	0.01	2.02	5.245
hw1	Cu_per	857	0.000	4.180	**0.084**	0.09	0.30	0.01	3.57	10.004
hw2	Cu_per	151	0.005	1.070	**0.068**	0.02	0.12	0.01	1.82	5.045
hw3	Cu_per	106	0.002	0.370	**0.064**	0.01	0.07	0.00	1.11	2.27
hw3a	Cu_per	8	0.020	0.160	**0.085**	0.00	0.06	0.01	0.65	0.152
hw4	Cu_per	8	0.005	0.630	**0.095**	0.04	0.21	0.04	2.17	2.211
mmh	Cu_per	1446	0.000	9.640	**0.481**	1.03	1.01	0.02	2.11	4.482
wall	Cu_per	2944	0.000	10.200	**0.151**	0.15	0.38	0.01	2.54	10.311
btn	Au_ozt	61	0.000	0.032	**0.001**	0.00	0.00	0.00	4.00	7.074
fw	Au_ozt	302	0.000	0.077	**0.002**	0.00	0.01	0.00	2.50	7.578
hw0	Au_ozt	247	0.000	0.053	**0.002**	0.00	0.01	0.00	2.50	6.24
hw1	Au_ozt	857	0.000	0.117	**0.002**	0.00	0.01	0.00	4.00	9.869
hw2	Au_ozt	151	0.000	0.034	**0.001**	0.00	0.00	0.00	3.00	7.116
hw3	Au_ozt	106	0.000	0.066	**0.003**	0.00	0.01	0.00	2.67	5.165
hw3a	Au_ozt	8	0.000	0.010	**0.002**	0.00	0.00	0.00	1.50	2.14
hw4	Au_ozt	8	0.000	0.012	**0.002**	0.00	0.00	0.00	1.50	2.149
mmh	Au_ozt	1425	0.000	0.573	**0.010**	0.00	0.02	0.00	2.40	10.268
wall	Au_ozt	2943	0.000	0.140	**0.002**	0.00	0.01	0.00	3.00	6.759

The summary statistics by zone show that for the main elements representing the mineralisation (Co and Cu) only the “mmh” zone has a mean distinct from the “wall” rock.

14.3.2 Histograms and Probability Plots

Log histograms and probability plots are displayed below for all zones (excepting hw3a and hw4 due to the low number of assays present) for Co and Cu. These plots are length weighted.

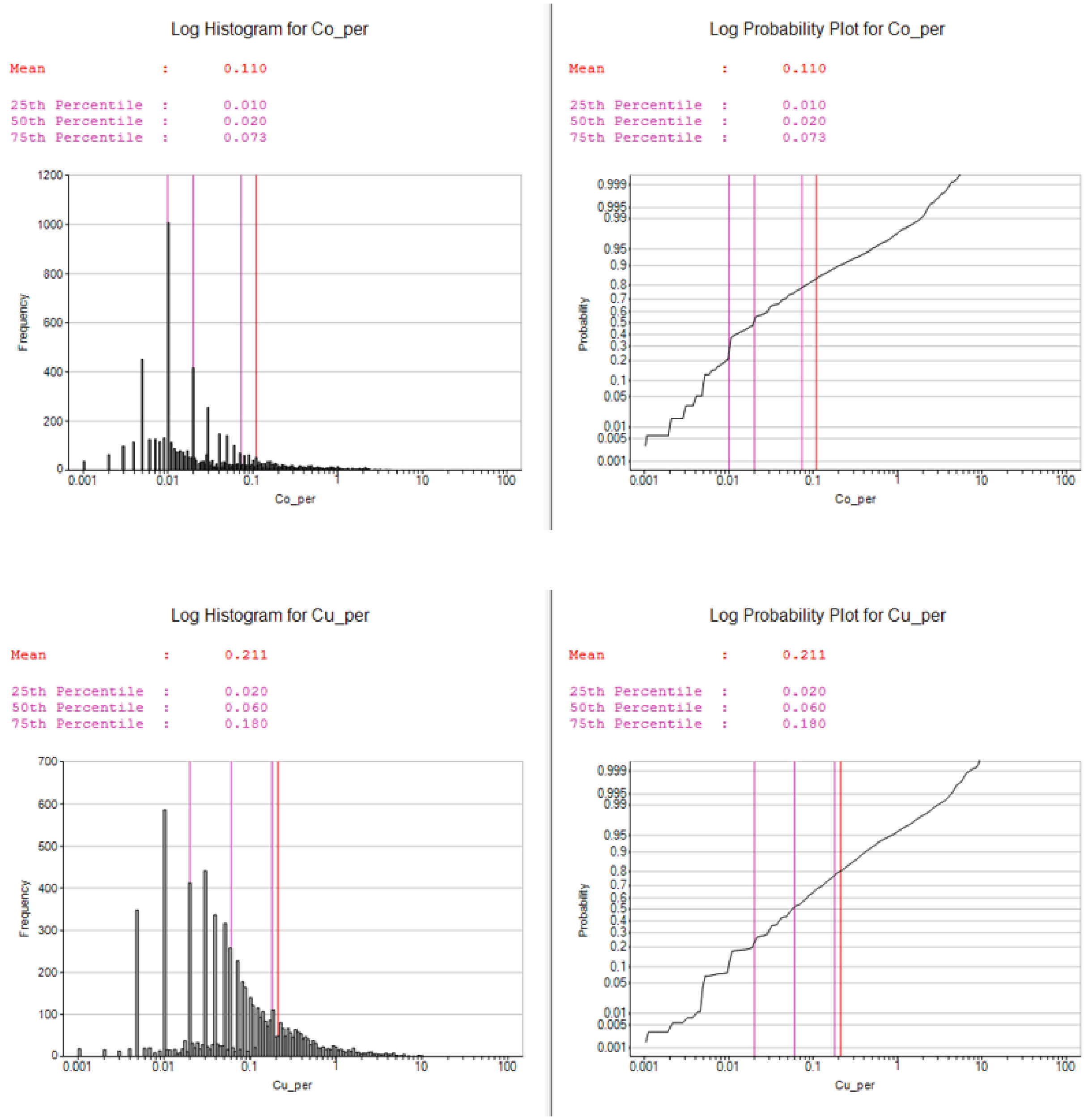


Figure 14-2: Log Histogram and Log Probability Plots for all Assays

e

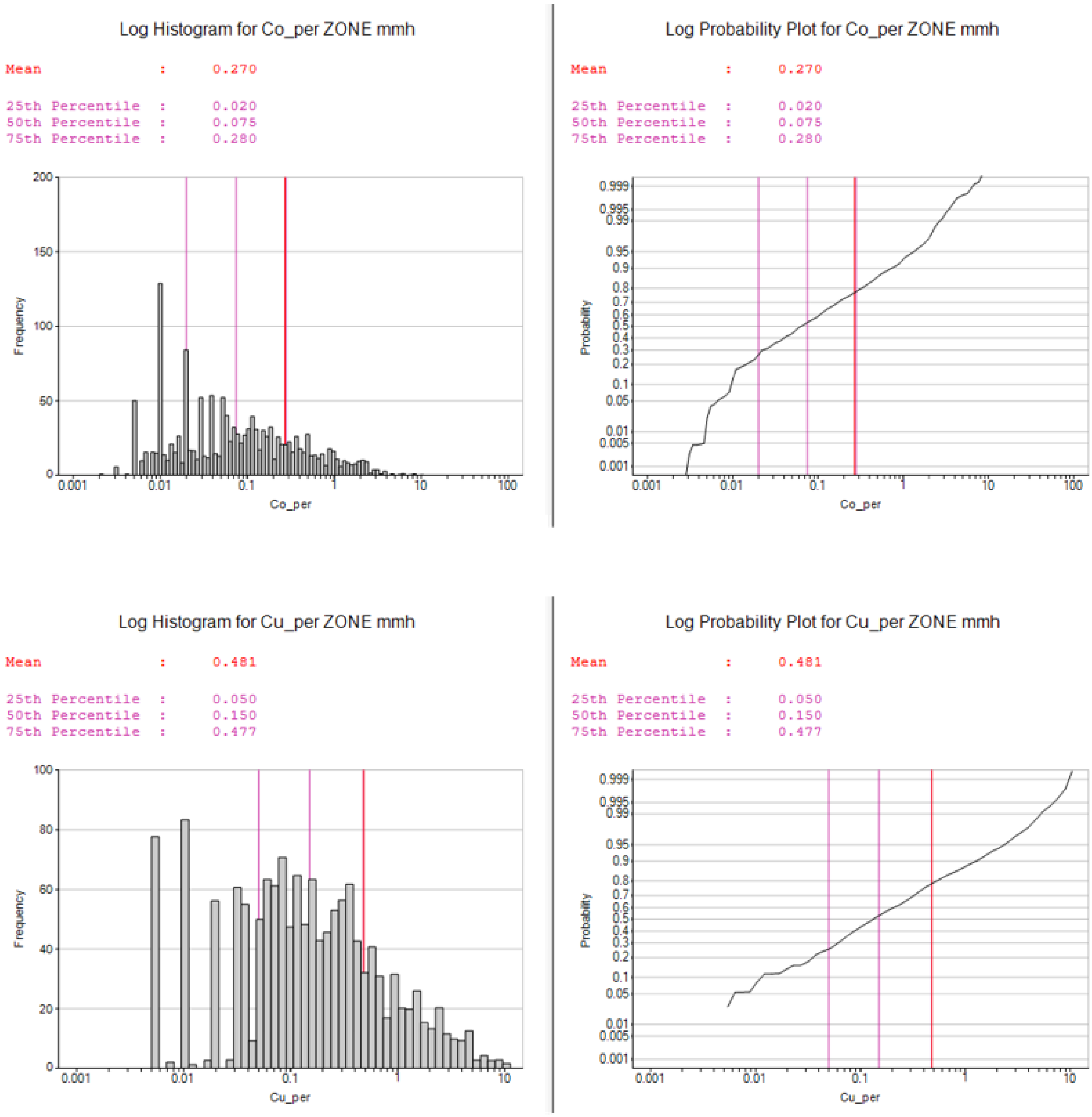


Figure 14-3: Log Histogram and Log Probability Plots for "mmh" Zone

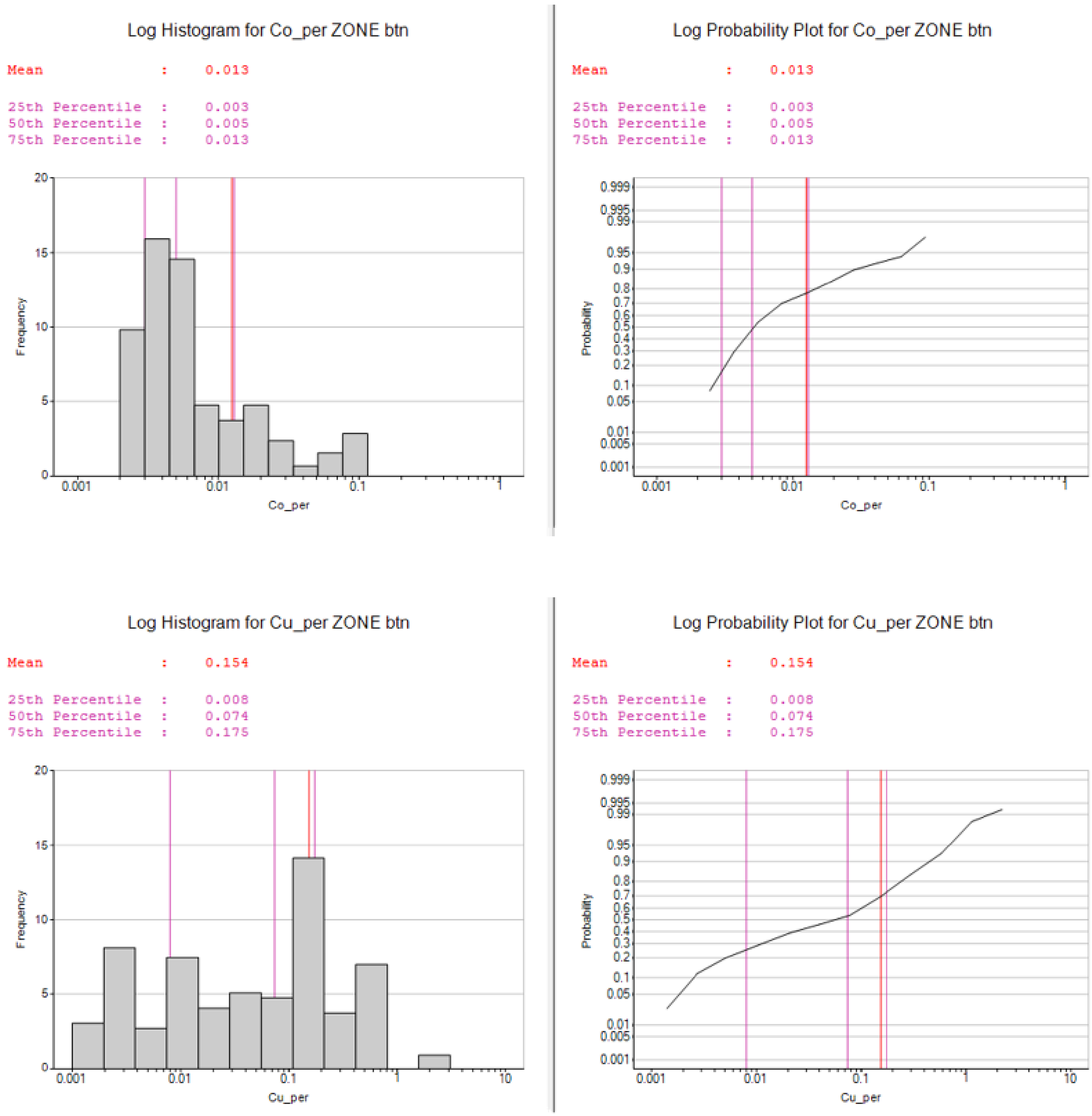


Figure 14-4: Log Histogram and Log Probability Plots for "btn" Zone

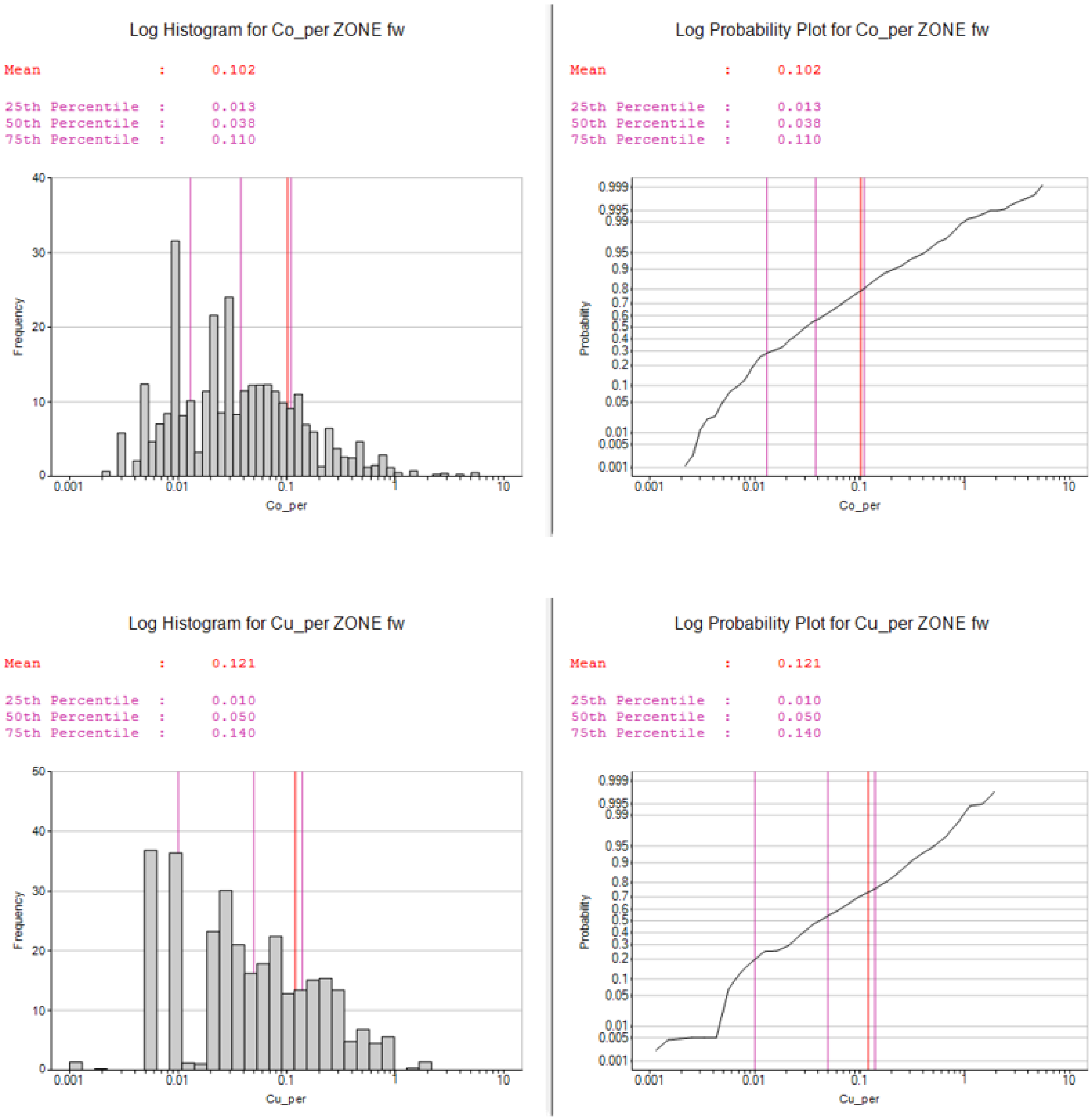


Figure 14-5: Log Histogram and Log Probability Plots for "fw" Zone

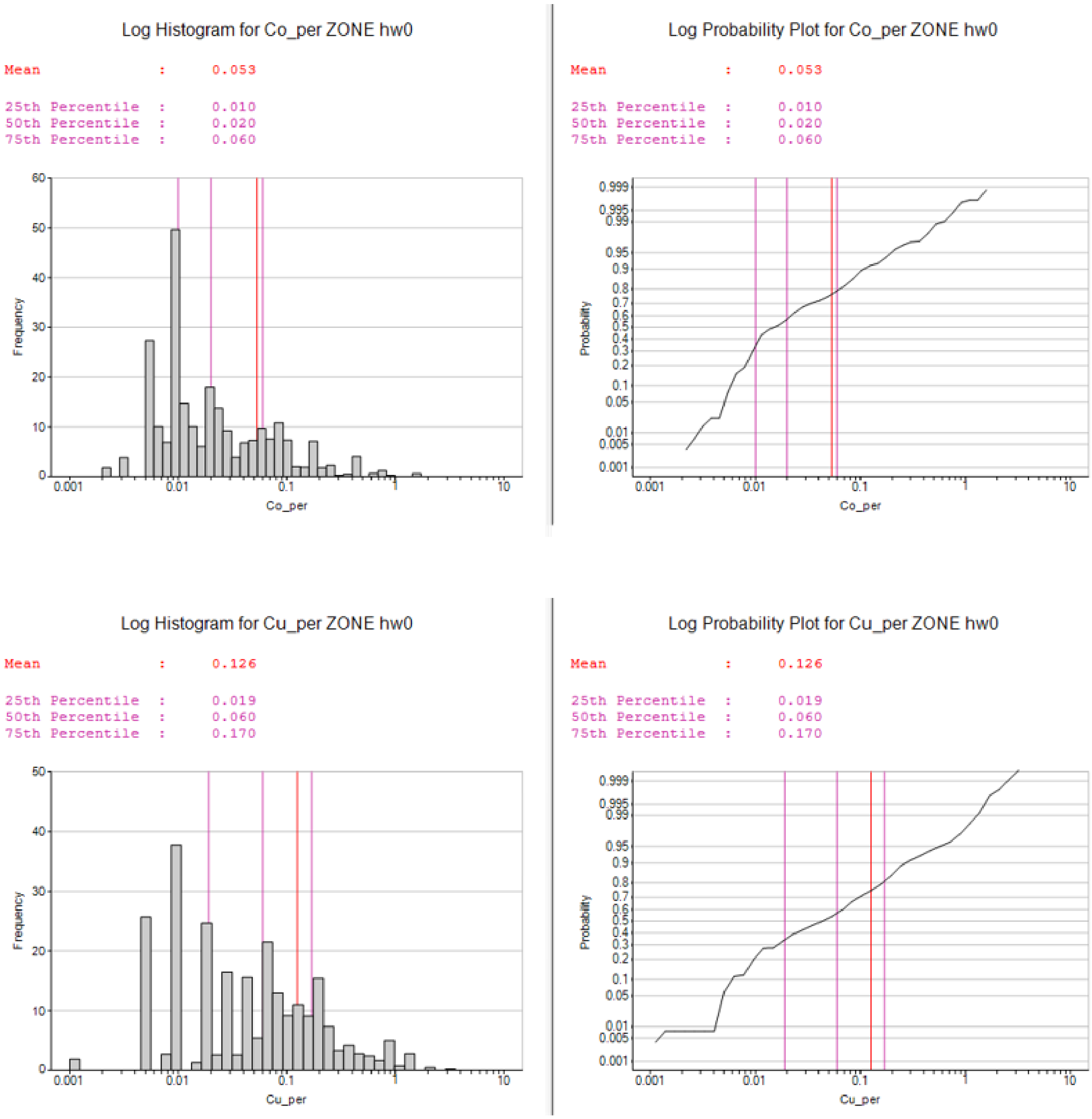


Figure 14-6: Log Histogram and Log Probability Plots for “hw0” Zone

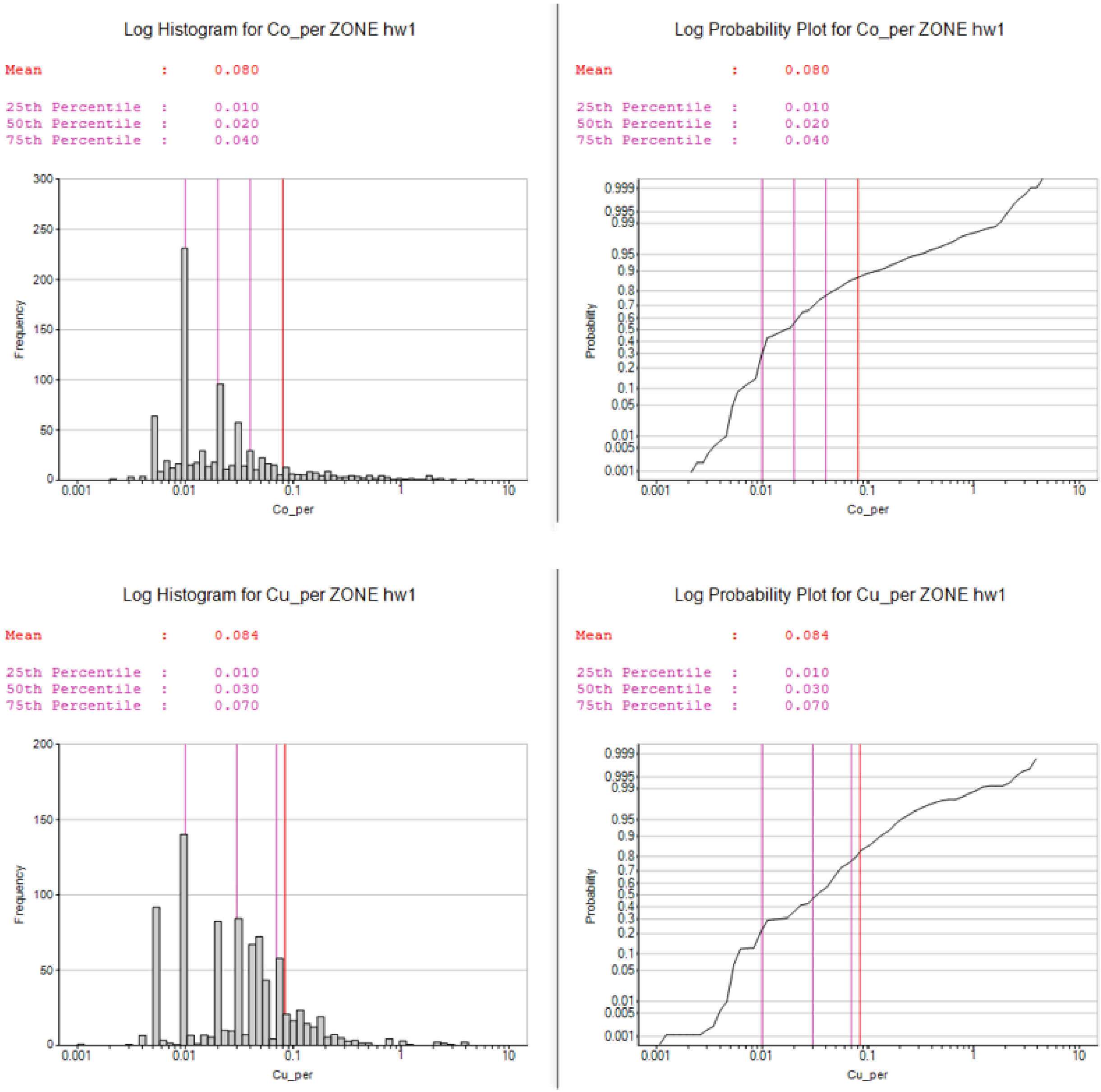


Figure 14-7: Log Histogram and Log Probability Plots for "hw1" Zone

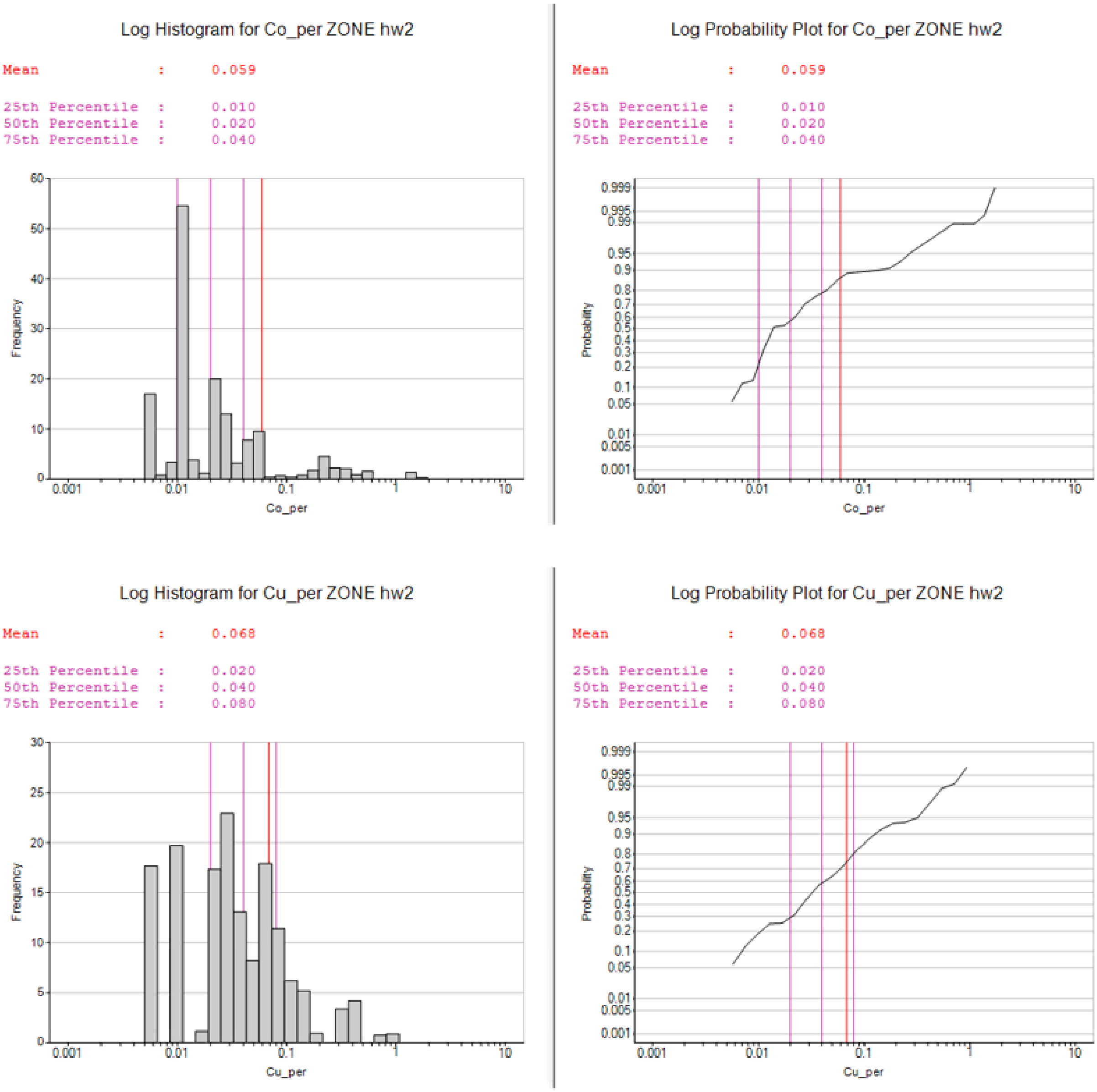


Figure 14-8: Log Histogram and Log Probability Plots for "hw2" Zone

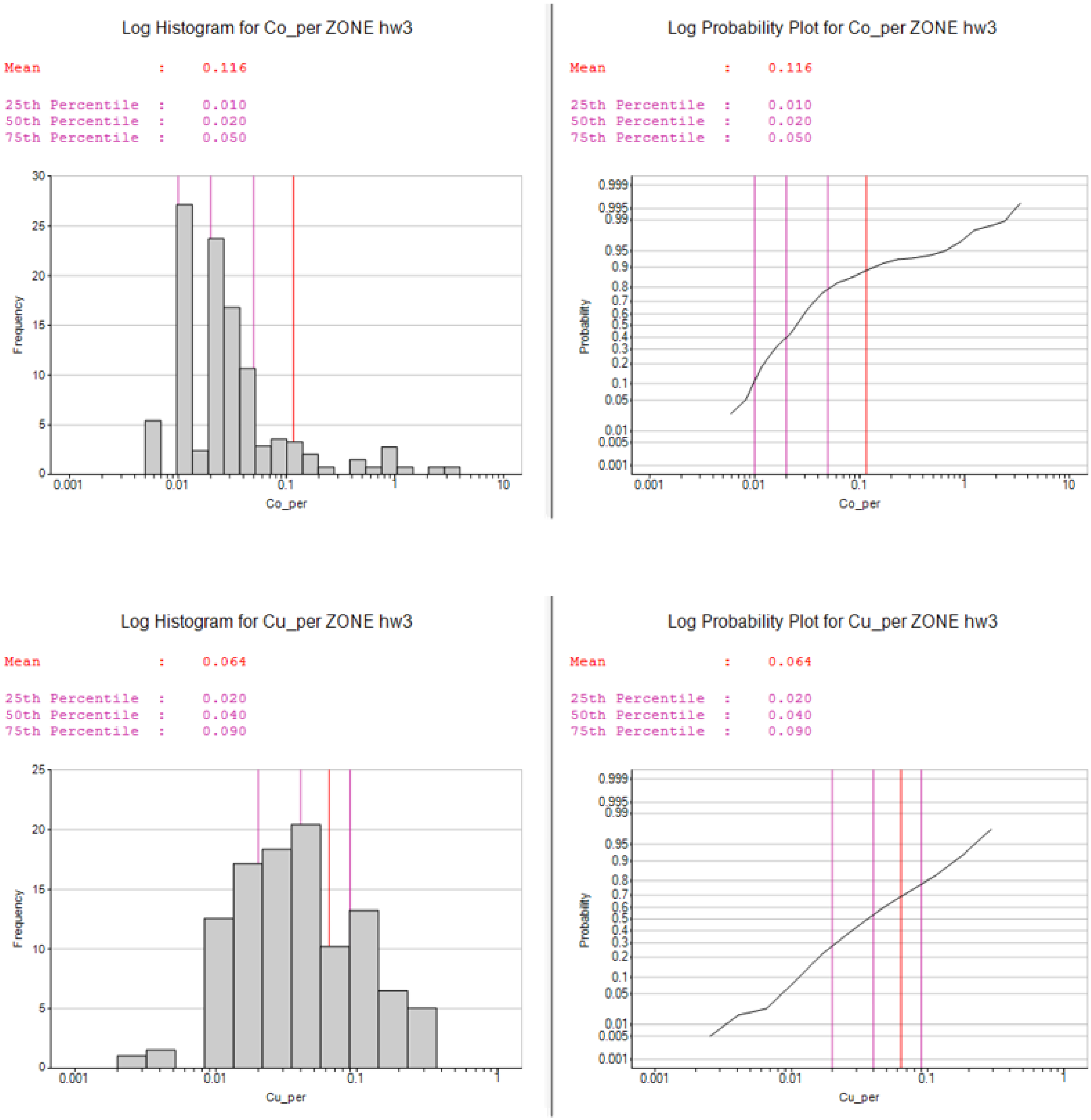


Figure 14-9: Log Histogram and Log Probability Plots for “hw3” Zone

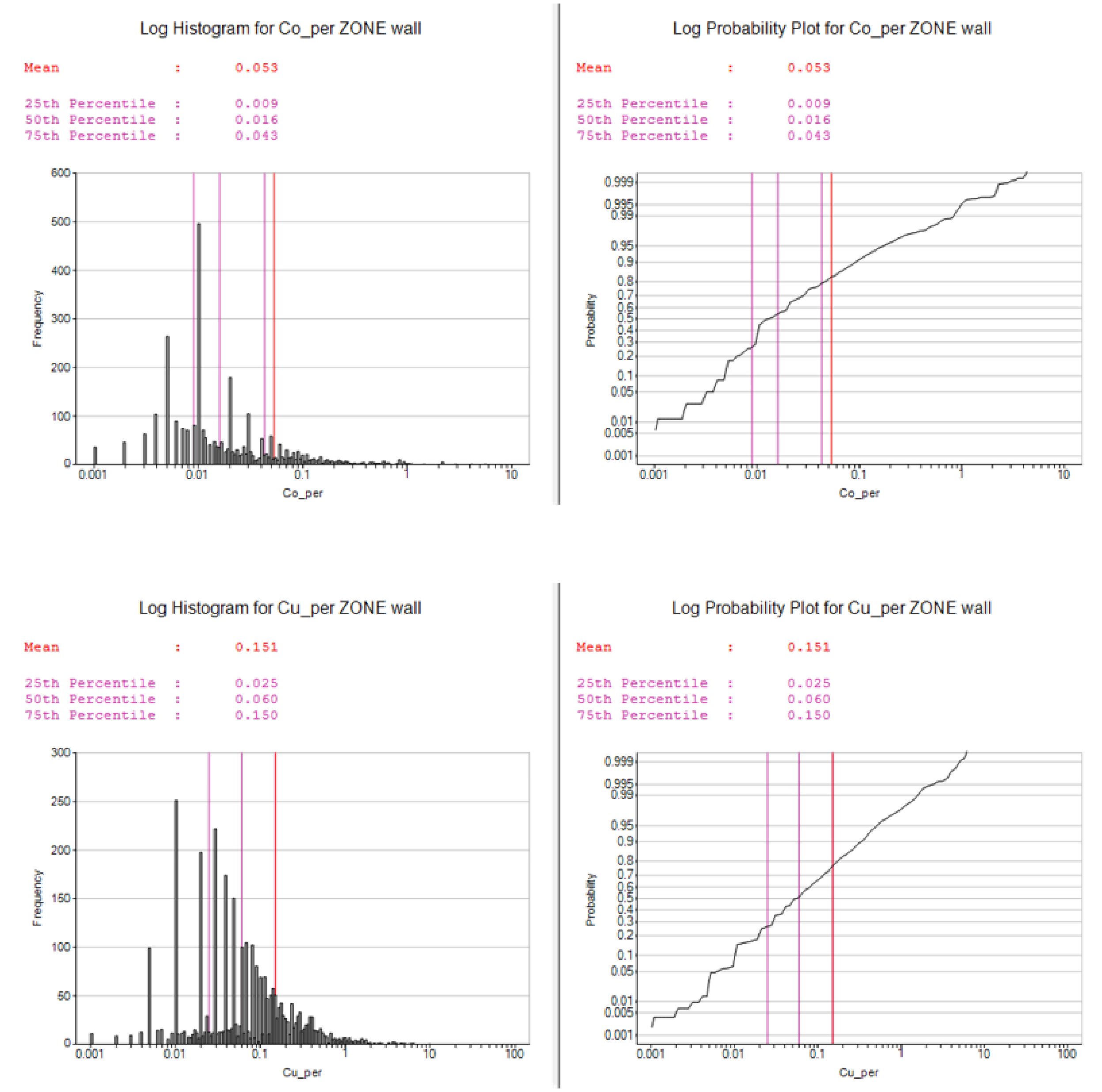


Figure 14-10: Log Histogram and Log Probability Plots for "wall" Rock Assays

Several things are brought to our attention when reviewing these plots. First, there is clearly an accuracy problem below 0.1% for both Co and Cu results, likely due to either assay techniques or reporting in older data generations. Most of the Co plots seem to show a break between a "lower" and "higher" grade population between 0.05% and 0.20%. This "higher" grade population does not appear to be well represented except in the "mmh" zone. The other zones are hardly distinct from the "wall" rock in either numerical statistics or population visualization.

14.3.3 Contact Profiles

Based on analysis of the above statistics and plots, a contact profile was only generated to test the validity of the "mmh" wireframe model and to determine the ideal method for treating its wireframe boundaries. Contact plots were developed between the samples within the "mmh" and the "wall" designations.

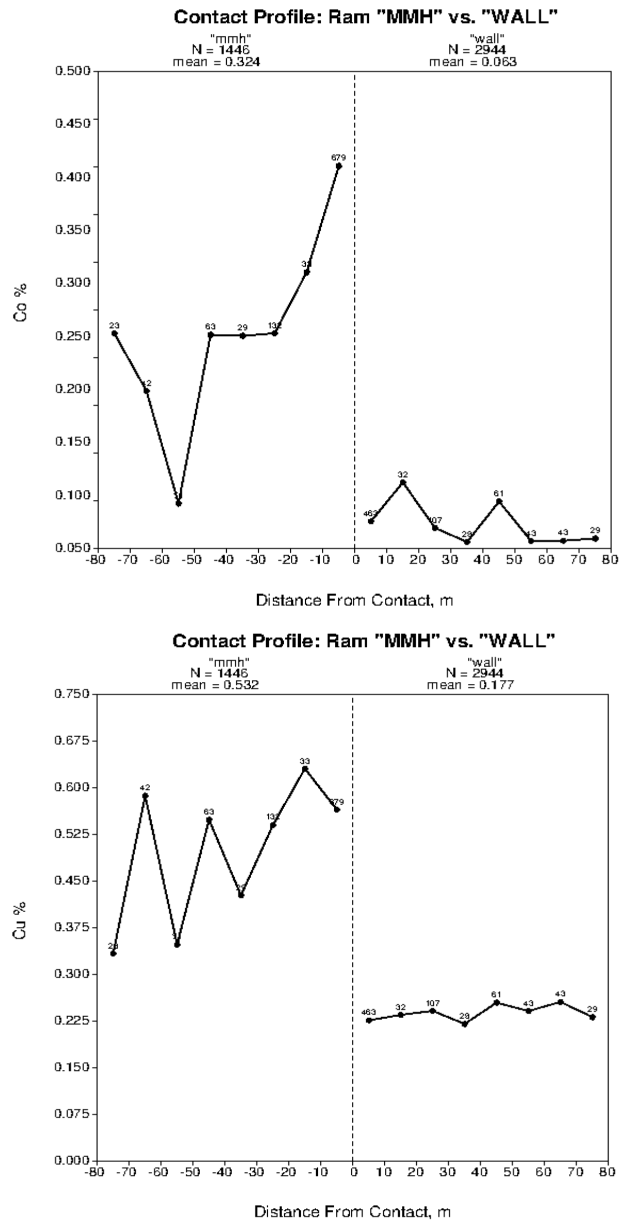


Figure 14-11: Contact Profiles for Co and Cu between "mmh" and "wall" Zones

The contact plots indicate that the although the statistics (for Co at least) do not indicate a single "clean" population within the "mmh" domain, that there is a hard boundary on the average between the "mmh" and the "wall" zones, even if the variability of the Co within the "mmh" zone is evident. The boundary for Cu between "mmh" and "wall" is even more clear, with Cu appearing to have less variability within the "mmh" zone.

14.3.4 Variography

Due to the results above, initial variography was only performed on the "mmh" zone. Downhole variograms seem to confirm the results of the contact plots; namely that variation across the "mmh" zone is high for Co (variance appears to reach the sill within one sample distance) but lower for Cu (range of 15 to 50 feet).

Variography performed within "mmh" parallel to the stratigraphy shows well developed continuity for Co in the nearly down-dip direction, while Cu has a much lower variation with distance.

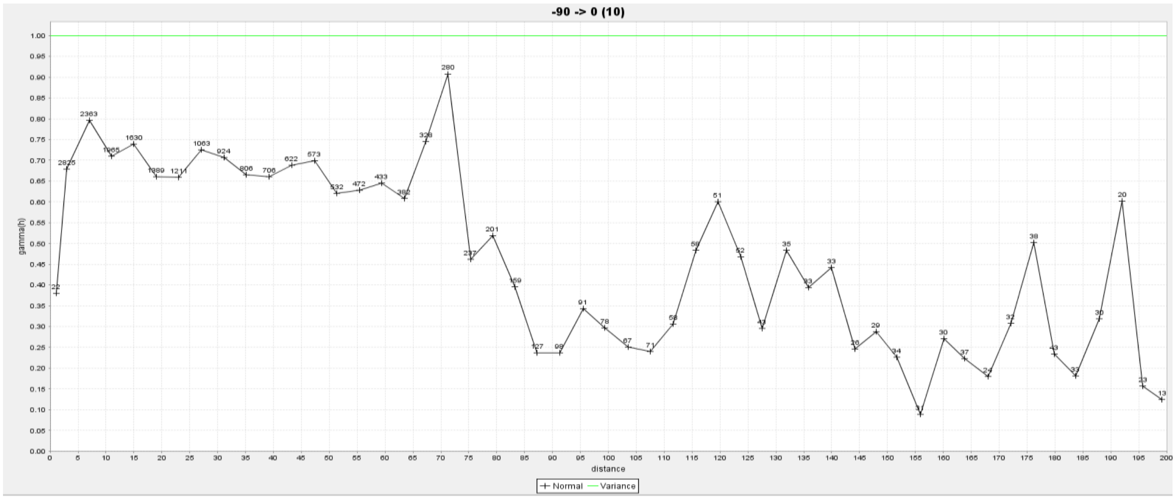


Figure 14-12: Downhole Variogram for Co within the "mmh" Zone

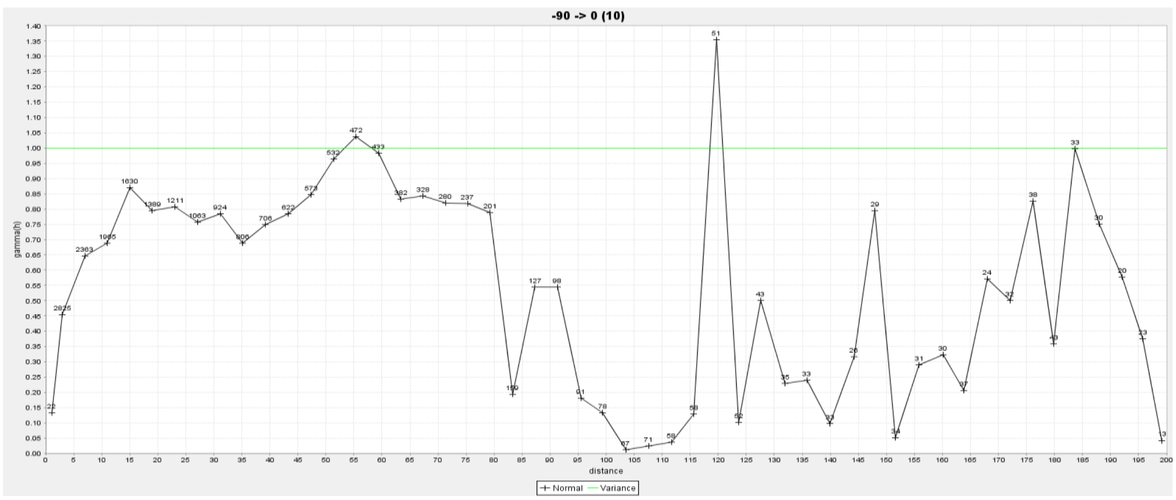


Figure 14-13: Downhole Variogram for Cu within the "mmh" Zone

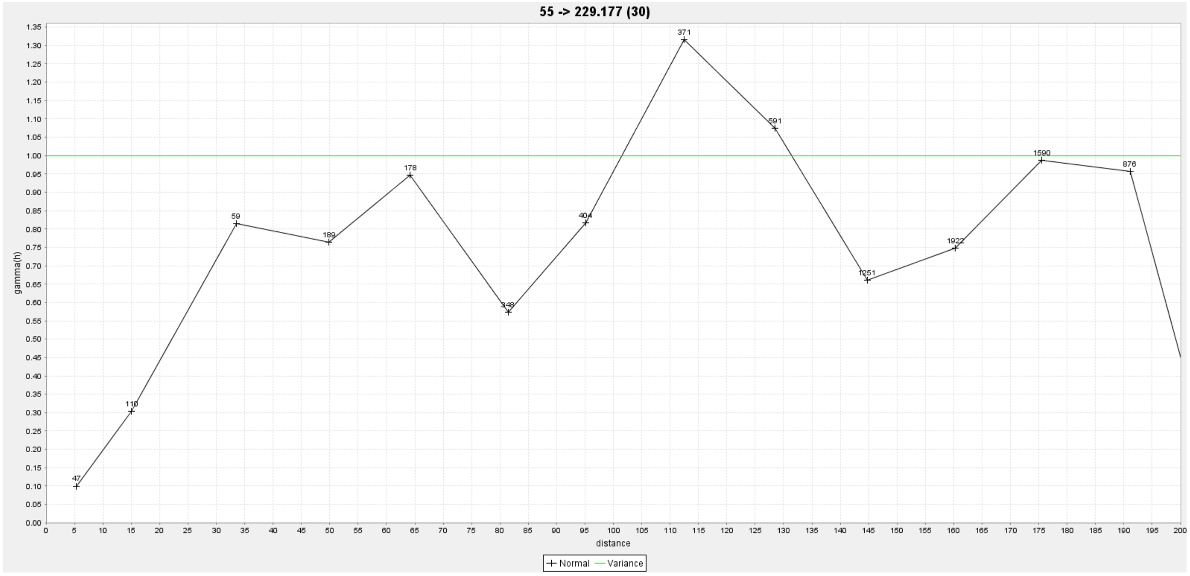


Figure 14-14: Directional Variogram parallel to the stratigraphy for Co within “mmh”

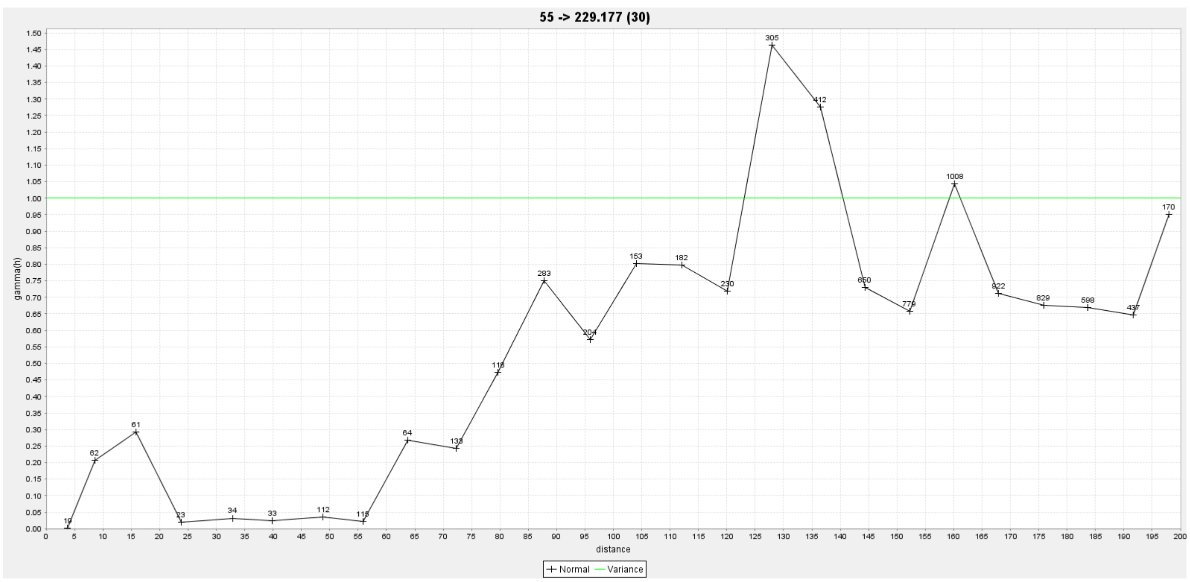


Figure 14-15: Directional Variogram parallel to the stratigraphy for Cu within “mmh”

14.4 COMPOSITING

Assay results from drilling were composited to 2 feet, as most samples were 2 feet or 3 feet. Rather than force samples to exactly 2 feet, the compositing process approximated as closely to 2 feet as possible within each drill hole interval without excluding any samples.

Due to the nature of the contact boundary, the “mmh” zone was composited separately from the other domains. In order to best review the results, the other eight zones were similarly composited. Compositing resulted in 7,751 “composite assay” intervals.

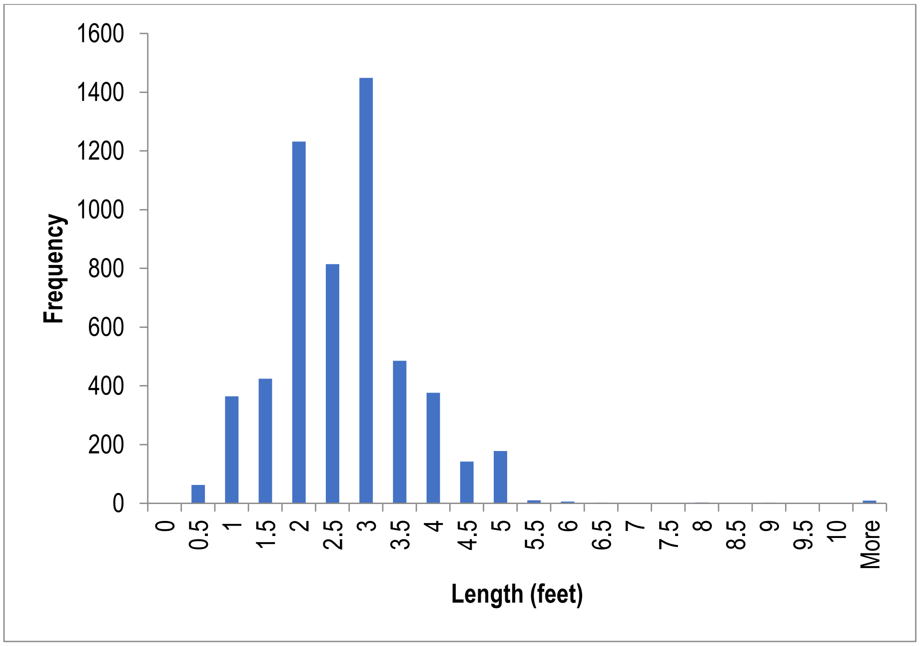


Figure 14-16: Histogram for Sample Length

14.4.1 Statistics after Compositing

Table 14-2: Summary Statistics, Composited Data

Zone	Field	Nsamples	Minimum	Maximum	Mean	Variance	Standdev	Standerr	Cov	Skewness
btn	Co_per	90	0.002	0.115	0.013	0.00	0.02	0.00	1.54	3.148
fw	Co_per	335	0.003	2.841	0.100	0.05	0.22	0.01	2.22	7.263
hw0	Co_per	293	0.002	1.193	0.051	0.01	0.11	0.00	2.06	6.444
hw1	Co_per	1073	0.002	3.313	0.079	0.06	0.24	0.01	3.08	7.234
hw2	Co_per	216	0.005	1.340	0.056	0.02	0.13	0.01	2.30	6.1
hw3	Co_per	153	0.005	3.993	0.108	0.15	0.38	0.02	3.54	8.04
hw3a	Co_per	10	0.010	0.030	0.023	0.00	0.01	0.00	0.30	-0.402
hw4	Co_per	13	0.020	0.398	0.065	0.01	0.10	0.02	1.54	2.859
mmh	Co_per	1676	0.003	7.103	0.269	0.26	0.51	0.01	1.89	4.667
wall	Co_per	3892	0.000	4.630	0.050	0.03	0.17	0.00	3.32	13.241
btn	Cu_per	90	0.001	1.585	0.154	0.05	0.23	0.02	1.51	3.19
fw	Cu_per	335	0.000	2.180	0.120	0.04	0.19	0.01	1.58	5.042
hw0	Cu_per	293	0.001	1.503	0.120	0.04	0.21	0.01	1.72	3.54
hw1	Cu_per	1073	0.000	4.140	0.082	0.06	0.25	0.01	3.09	9.418
hw2	Cu_per	216	0.005	0.799	0.067	0.01	0.10	0.01	1.52	4.275
hw3	Cu_per	153	0.002	0.370	0.065	0.00	0.07	0.00	1.00	2.222
hw3a	Cu_per	10	0.020	0.160	0.087	0.00	0.05	0.01	0.62	0.059
hw4	Cu_per	13	0.005	0.630	0.104	0.05	0.21	0.04	2.06	2.042
mmh	Cu_per	1676	0.000	9.640	0.478	0.83	0.91	0.02	1.90	4.259

Zone	Field	Nsamples	Minimum	Maximum	Mean	Variance	Standdev	Standerr	Cov	Skewness
wall	Cu_per	3892	0.000	6.663	0.146	0.10	0.32	0.00	2.16	8.969
btn	Au_ozt	90	0.000	0.032	0.001	0.00	0.00	0.00	4.00	7.274
fw	Au_ozt	335	0.000	0.043	0.002	0.00	0.00	0.00	2.00	4.92
hw0	Au_ozt	293	0.000	0.034	0.002	0.00	0.00	0.00	2.00	4.505
hw1	Au_ozt	1073	0.000	0.088	0.002	0.00	0.01	0.00	3.00	7.962
hw2	Au_ozt	216	0.000	0.020	0.001	0.00	0.00	0.00	2.00	4.628
hw3	Au_ozt	153	0.000	0.045	0.003	0.00	0.01	0.00	2.33	3.928
hw3a	Au_ozt	10	0.000	0.010	0.002	0.00	0.00	0.00	1.50	2.172
hw4	Au_ozt	13	0.000	0.009	0.002	0.00	0.00	0.00	1.50	1.715
mmh	Au_ozt	1655	0.000	0.397	0.010	0.00	0.02	0.00	2.10	8.355
wall	Au_ozt	3891	0.000	0.064	0.002	0.00	0.01	0.00	2.50	4.944

14.5 OUTLIER MANAGEMENT AND CAPPING STRATEGY

14.5.1 "MMH" Zone

Co grades were capped at 4% and Cu grades were capped at 4% (after compositing) in the "mmh" zone, based on the histogram/probability plot analysis, and visual review of grade distribution. This resulted in the capping of 2 Co composites (and 1% of the Co metal content) and 23 Cu composites (5% of Cu metal content). Au values did not warrant capping.

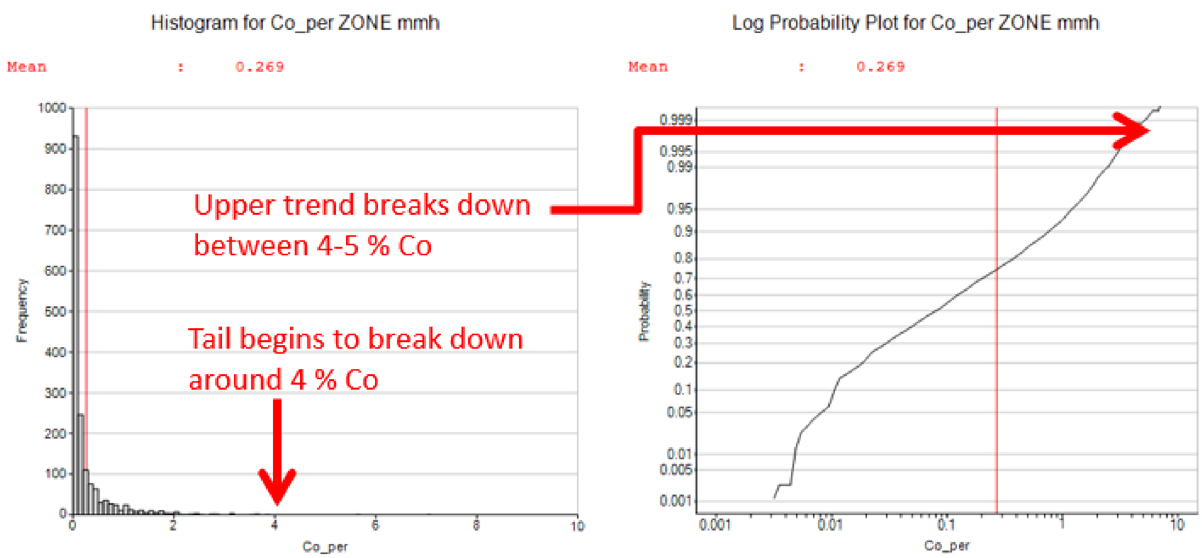


Figure 14-17: Drill hole Histogram and Probability Plot of "mmh" Co%

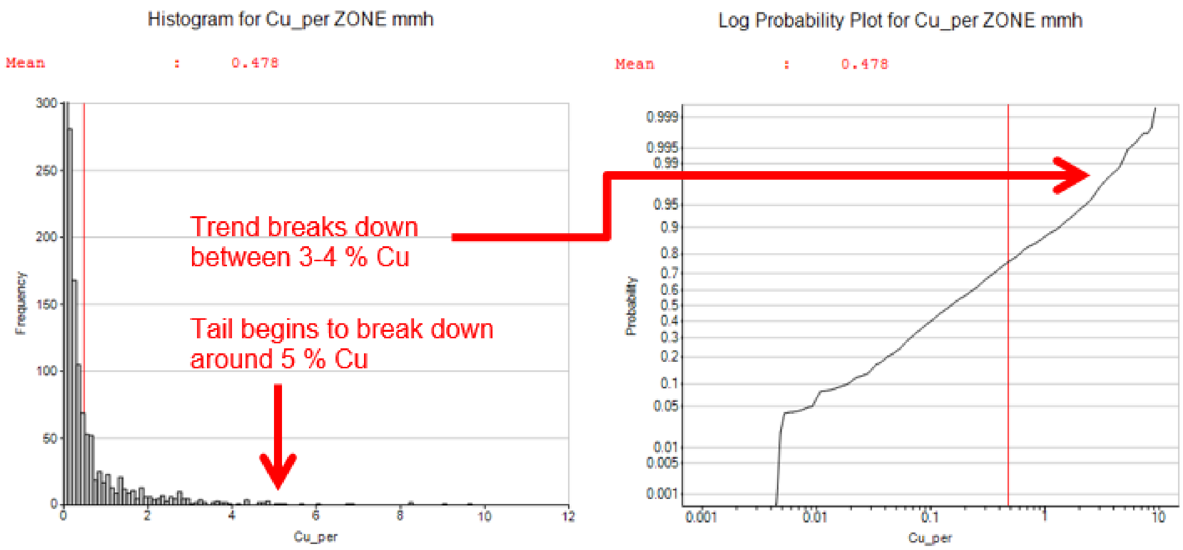


Figure 14-18: Drill hole Histogram and Probability Plot of "mmh" Cu%

14.5.2 All other zones

Co grades were capped at 0.7% Co and Cu grades were capped at 2% (after compositing) in the other domains, based on the histogram/probability plot analysis. This resulted in the capping of 80 Co composites (and 14% of the Co metal content) and 26 Cu results (4% of Cu metal content).

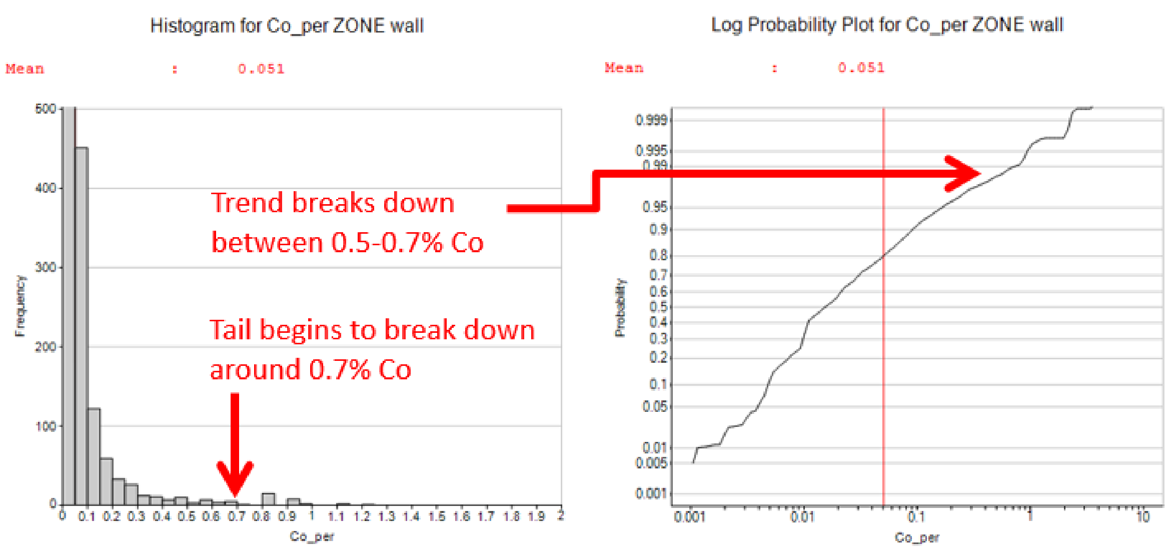


Figure 14-19: Drill hole Histogram and Probability Plot of Co%

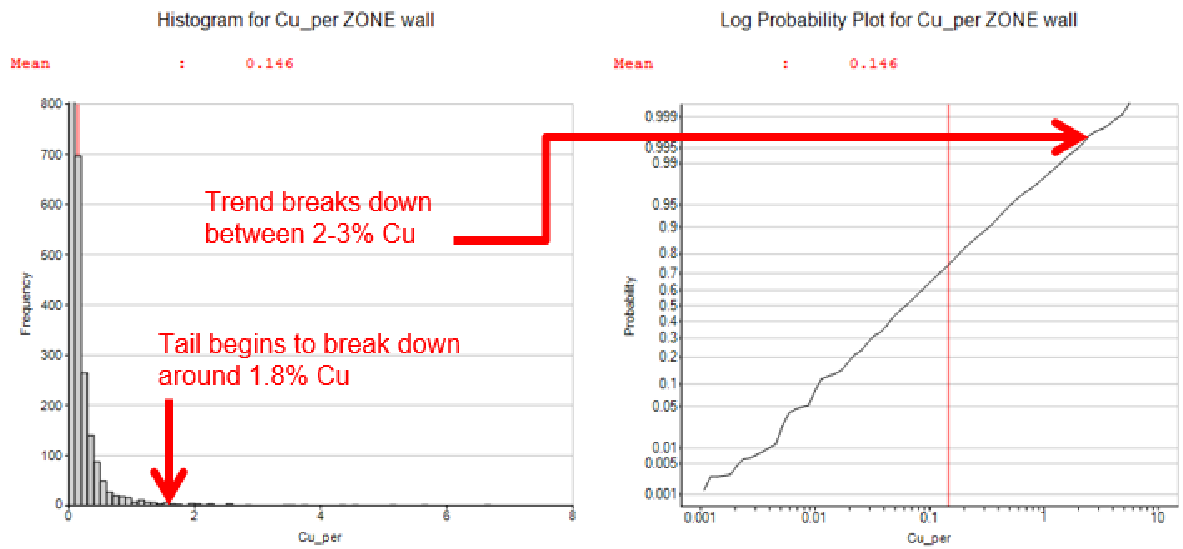


Figure 14-20: Drill hole Histogram and Probability Plot of Cu%

14.5.3 Statistics After Capping

Table 14-3: Summary Statistics, Capped and Composited Data

Zone	Field	Nsamples	Minimum	Maximum	Mean	Variance	Standdev	Standerr	Cov	Skewness
btn	Co_per	90	0.002	0.115	**0.013**	0.00	0.02	0.00	1.54	3.148
fw	Co_per	335	0.003	0.700	**0.089**	0.02	0.14	0.01	1.54	2.974
hw0	Co_per	293	0.002	0.700	**0.049**	0.01	0.09	0.00	1.80	4.476
hw1	Co_per	1073	0.002	0.700	**0.062**	0.02	0.13	0.00	2.10	3.779
hw2	Co_per	216	0.005	0.700	**0.053**	0.01	0.11	0.01	1.98	4.067
hw3	Co_per	153	0.005	0.700	**0.075**	0.02	0.14	0.01	1.89	3.167
hw3a	Co_per	10	0.010	0.030	**0.023**	0.00	0.01	0.00	0.30	-0.402
hw4	Co_per	13	0.020	0.398	**0.065**	0.01	0.10	0.02	1.54	2.859
mmh	Co_per	1676	0.003	4.000	**0.266**	0.23	0.48	0.01	1.80	3.587
wall	Co_per	3892	0.000	0.700	**0.044**	0.01	0.10	0.00	2.16	5.024
btn	Cu_per	90	0.001	1.585	**0.154**	0.05	0.23	0.02	1.51	3.19
fw	Cu_per	335	0.000	2.000	**0.119**	0.03	0.18	0.01	1.54	4.502
hw0	Cu_per	293	0.001	1.503	**0.120**	0.04	0.21	0.01	1.72	3.54
hw1	Cu_per	1073	0.000	2.000	**0.078**	0.04	0.21	0.00	2.65	7.518
hw2	Cu_per	216	0.005	0.799	**0.067**	0.01	0.10	0.01	1.52	4.275
hw3	Cu_per	153	0.002	0.370	**0.065**	0.00	0.07	0.00	1.00	2.222
hw3a	Cu_per	10	0.020	0.160	**0.087**	0.00	0.05	0.01	0.62	0.059
hw4	Cu_per	13	0.005	0.630	**0.104**	0.05	0.21	0.04	2.06	2.042
mmh	Cu_per	1676	0.000	4.000	**0.455**	0.59	0.77	0.01	1.69	2.844
wall	Cu_per	3892	0.000	2.000	**0.140**	0.06	0.24	0.00	1.74	4.539

14.6 DENSITY

14.6.1 Methodology

In this section both current specific gravity results and historical specific gravity results will be referred to. The historical results were measured using the wet-weight/dry-weight method on-site. The current results were measured by SGS Lakefield laboratory using the bulk density by water displacement, and in the cases where necessary wax coating was used.

14.6.2 Generalised Results

Table 14-4: Specific Gravity Results Summary

	# of Samples	Mean SG	# of Assayed Samples	Mean SG Assayed	Mean Co %	Mean Cu %
Historical SG	729	2.92	715	2.92	0.30	0.42
Current SG	99	3.08	98	3.09	0.46	0.60
Historical SG – “mmh” only	239	2.99	239	2.99	0.63	0.77
Current SG – “mmh” only	73	3.11	73	3.11	0.60	0.75
Historical SG – non-“mmh”	490	2.88	477	2.88	0.13	0.24
Current SG – non-“mmh”	26	3.01	25	3.01	0.05	0.17

14.6.3 Mineralised Zone

Historical SG correlations:

- Co: 0.31 (average Co grade for these samples was 0.30%)
- Cu: 0.28 (average Cu grade for these samples was 0.42%)
- Co+Cu: 0.37 (average of 0.72%)
- As: 0.29 (average As grade for these samples was 0.26%)
- S: 0.60 (average S grade for these samples was 2.1%)

Current program SG correlations (within the “mineralised” zones, 69 samples total):

- Co: 0.30 (average Co grade for these samples was 0.64%)
- Cu: 0.22 (average Cu grade for these samples was 0.79%)
- Co+Cu: 0.33 (average of 1.4%)
- As: 0.01 (average As grade for these samples was 0.68%, 66 samples)
- S: 0.27 (average S grade for these samples was 1.3%, 62 samples)

The current program cut specific SG samples from within assayed intervals, as opposed to specifically assaying the SG samples after measurement. This might account for the average specific gravity differences. Despite this, there is decent agreement from a correlation perspective.

Scatter plots of all SG results against Co+Cu suggest a formula:

$$\mathbf{y = 0.065x + 2.8861}$$

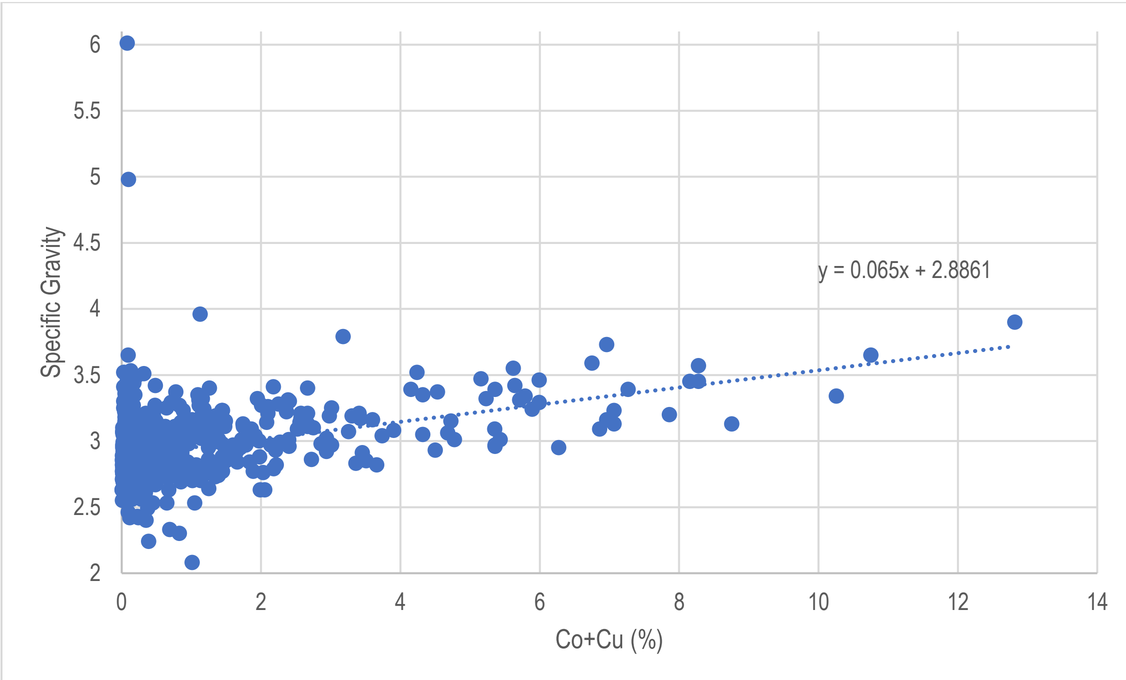


Figure 14-21: Co+Cu value vs. Specific Gravity (Historical and Current combined within mineralised zone)

Looking at just samples above 1% Co+Cu increases the correlation to 0.5, so the confidence in the above formula seems to increase with grade. Due to the capping discussed above, the maximum specific gravity possible within the model using the above formula will be:

y = 0.065(8) + 2.8861

y = 3.41

This is within 1.4 standard deviations of the mean (2.94, standard deviation of 0.26) for all assayed intervals of the historical and current SG measurements combined. It is likely, based on the evidence available, that the use of this formula will result in a conservative estimation of the actual specific gravity of the deposit.

14.7 VARIOGRAPHY

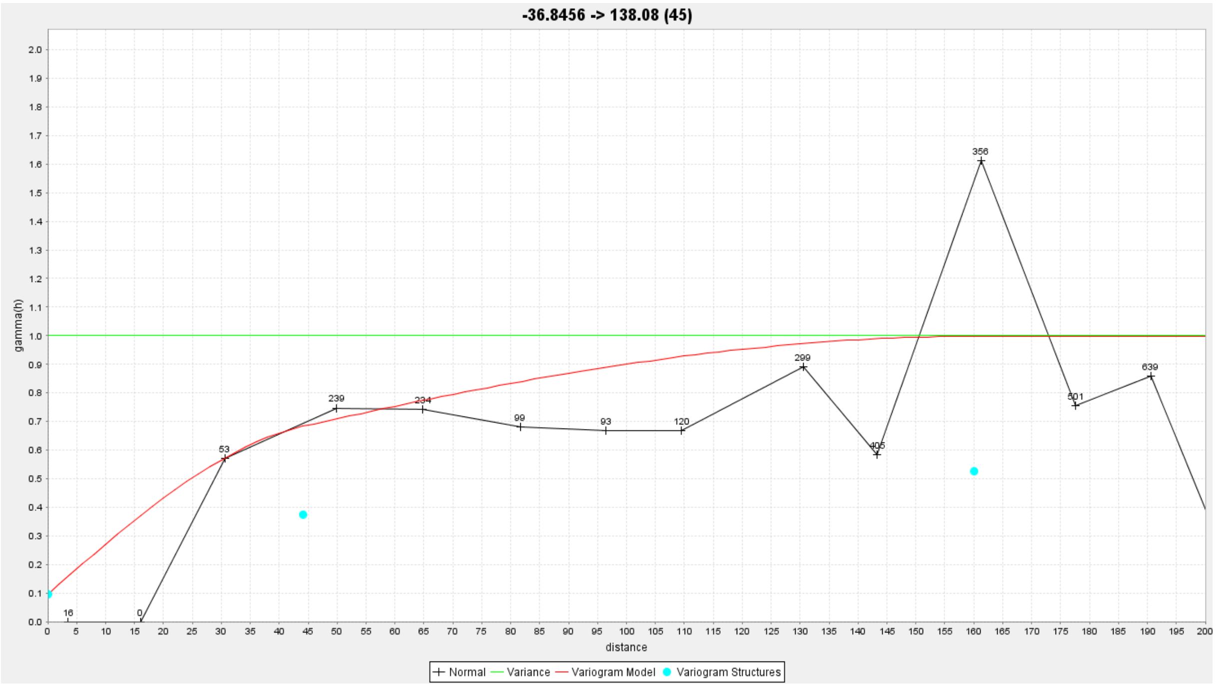


Figure 14-22: Co Variogram

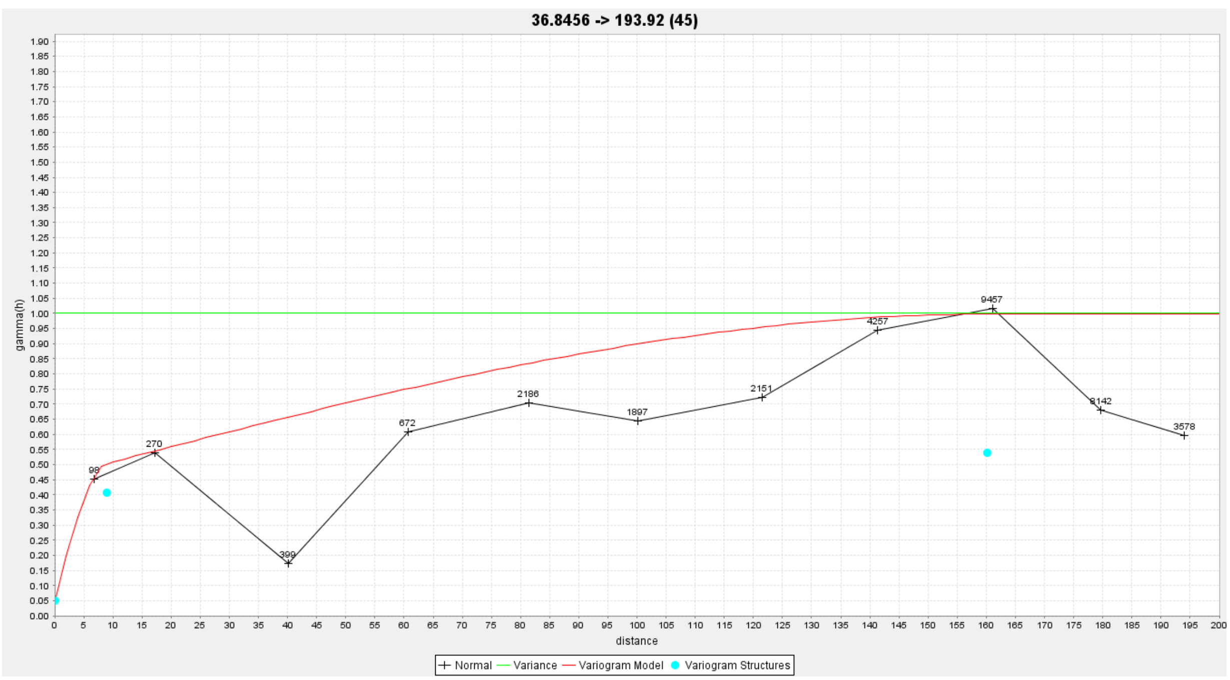


Figure 14-23: Cu Variogram

14.8 INTERPOLATION PLAN

Inverse-Distance-Squared ("ID^2") was chosen as the interpolation method. Variography was performed using various parameters and sample selections. Drill hole spacing was generally in a decent pattern (not necessarily a strict grid, but not too clustered) which lends itself nicely to a more accurate and precise estimation. In addition to the ID^2 method, grade estimations were also performed using Nearest Neighbour (NN) for comparative and validation purposes.

Due to the geometry of the deposit and the nature of the grade distribution, the estimation was additionally constrained by three (3) wireframe volumes to represent three volumes within the "mmh" zone with dip trends that vary from the mean. The "hw1" zone had two distinct strike/dip trends which were also constrained separately.

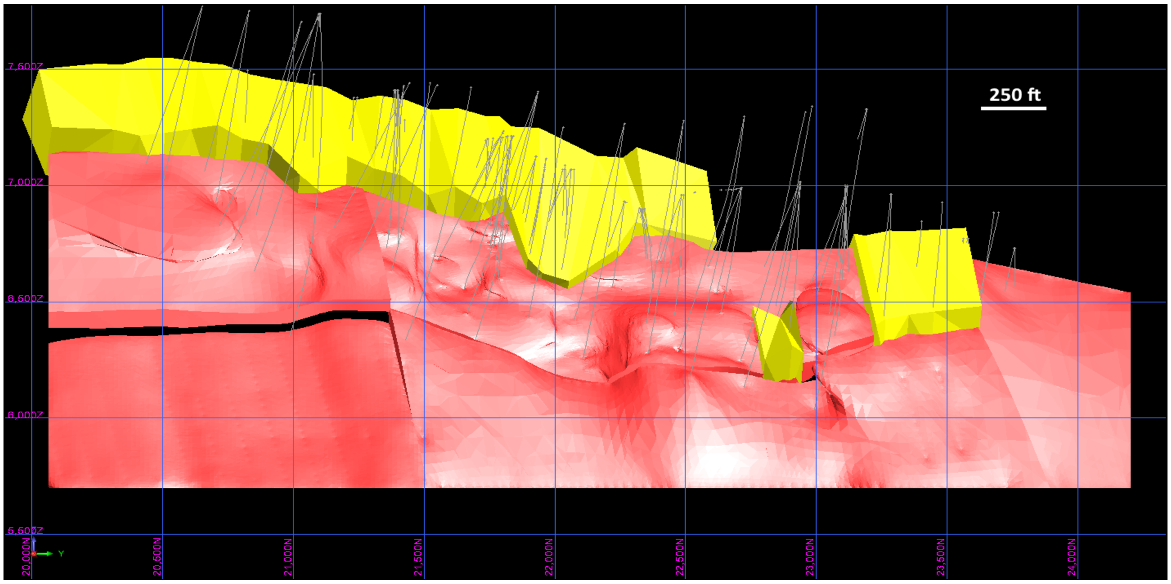


Figure 14-24: Long-section (looking West)

View in local grid displaying three alternate dip trend wireframes (yellow) in relation to "mmh" wireframe (pink)

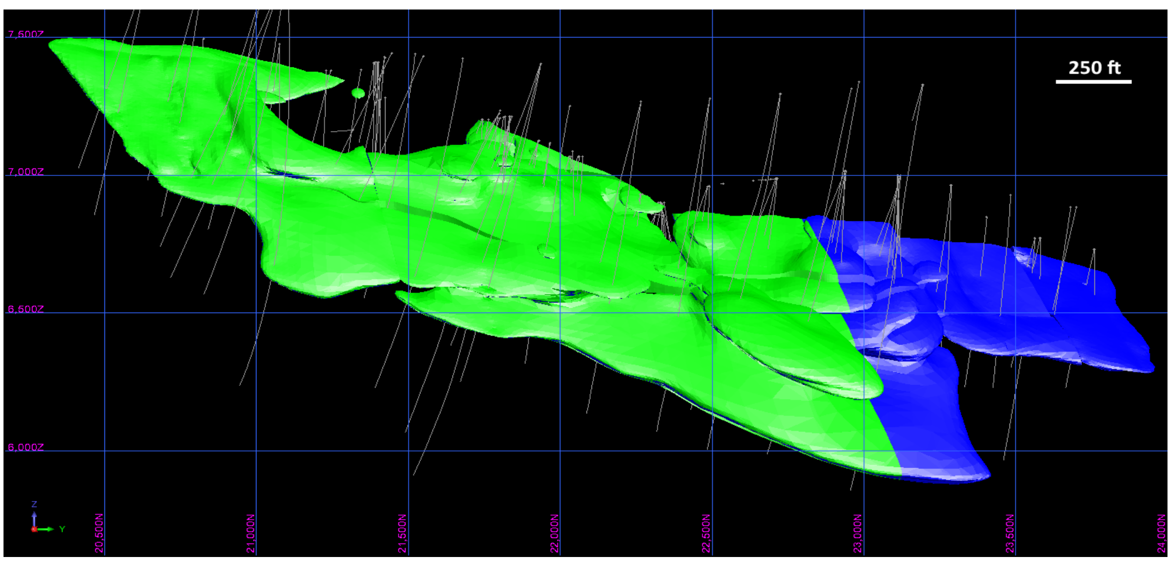


Figure 14-25: Long-section (looking West)

View in local grid displaying two strike/dip domains in relation to "hw1" wireframe (blue and green)

14.9 BLOCK MODEL PARAMETERS

The Block Model is a rotated model and was created with parent cells of 12 x 12 x 4 ft. The rotation is -14 around the Z axis (dominant strike of mineralization is 346), and -58 around the Y axis. Twenty-four (24) ID^2 interpolations were performed to populate the final grades into the block model. Table 14-5, Table 14-6, and Table 14-7 display the search parameters and estimation parameters used in the estimation. Variogram models were used to guide the search ellipse distances.

Table 14-5: Search Parameters – Co & Cu & Au NN Estimate

	Search Ellipse
Maximum Range	200 ft
Bearing	76
Dip (alternate dips 1, 2, 3)	-58 (-50, -54, -50)
Semi-major: Major	1
Minor: Major (Co/Cu/Au)	20/5/10

Table 14-6: Search Parameters – "mmh" Zone ID^2 Estimate

	Search Ellipse 1			Search Ellipse 2			Search Ellipse 3		
	Co	Cu	Au	Co	Cu	Au	Co	Cu	Au
Minimum Samples	6	6	6	4	4	4	2	4	4
Maximum Samples	16	16	16	12	16	16	10	10	10
Maximum per Drill hole	2	2	2	2	2	2	2	4	4
Maximum Range	160 ft			320 ft			240 ft		
Bearing	76			76			76		
Dip (alternate dips 1, 2, 3)	-58 (-50, -54, -50)			-58 (-50, -54, -50)			-58 (-50, -54, -50)		
Semi-major:Major	1			1			1		
Minor:Major	20	4	8	40	8	16	30	6	12

Table 14-7: Search Parameters – other zones ID^2 Estimate

	Search Ellipse		
	Co	Cu	Au
Minimum Samples	2	4	4
Maximum Samples	10	10	10
Maximum per Drill hole	2	4	4
Maximum Range	240 ft		
Bearing (hw1 alternate)	76 (57)		
Dips (hw0, hw1, hw1 alt, hw2, hw3, hw3a, hw4, btn, fw)	-42, -47, -43, -46, -45, -43, -46, -43.5, -52		
Semi-major: Major	1		
Minor: Major	10		

14.10 RESOURCE BLOCK MODEL

14.10.1 Configuration

The geometrical configuration of the block model is summarised in Table 14-8.

Table 14-8: Block Model Geometry

	X Coordinates	Y Coordinates	Z Coordinates
Minimum	700	19000	7000
Maximum	3208	24304	9200
Parent Block Size	12	12	4
Rotation	Bearing: -14	Dip: -58	Plunge: 0

14.10.2 Cell Attributes

The cell attributes of the block model are summarised in Table 14-9.

Table 14-9: Cell Attributes

Attribute	Type	Description
IJK	Integer	Location Code to identify Parent Block
XC	Real	X Coordinate of Block Centre in Rotated Grid
YC	Real	Y Coordinate of Block Centre in Rotated Grid
ZC	Real	Z Coordinate of Block Centre in Rotated Grid
XINC	Real	X Increment of Block
YINC	Real	Y Increment of Block
ZINC	Real	Z Increment of Block
XMORIG	Real	X Value of Minimum Corner (Block Model Origin)
YMORIG	Real	Y Value of Minimum Corner (Block Model Origin)
ZMORIG	Real	Z Value of Minimum Corner (Block Model Origin)
NX	Integer	Number of Parent Blocks along X-Axis
NY	Integer	Number of Parent Blocks along Y-Axis
NZ	Integer	Number of Parent Blocks along Z-Axis
zone	Character	Domain
fault	Character	Yes/No (to denote if block is intercepted by fault)
co_nn	Real	Nearest Neighbour Co Grade
co_id2	Real	ID^2 Co Grade
cu_nn	Real	Nearest Neighbour Cu Grade
cu_id2	Real	ID^2 Cu Grade
au_nn	Real	Nearest Neighbour Au Grade
au_id2	Real	ID^2 Au Grade
sg_id2	Real	SG calculated from ID^2 estimates of Co+Cu
denfacid	Real	Imperial Density calculated from sg_id2
rescat	Character	Resource Categorisation (inf=inferred, ind=indicated, mea=measured, null=not estimated)
cuco_rat	Real	Cu:Co ratio (cu_id2/co_id2)

14.10.3 Resource Categorisation

Mineral resource classification is the application of Measured, Indicated and Inferred categories, in order of decreasing geological confidence, to the resource block model. These are CIM definition standards (adopted by the CIM Council on May 10, 2014) for reporting on mineral resources and reserves, which are incorporated, by reference, in NI 43-101.

As per CIM (2014):

Measured Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

Indicated Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

Inferred Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

These categories are applied in consideration of, but not limited to, drill and sample spacing, QA/QC, deposit-type and mineralisation continuity, and/or prior mining experience.

Previous categorization, statistical continuities, and drill spacing were taken into account when considering the categorization of the Ram resources.

The main zone (mmh) has the highest confidence, due to the numbers of intersections, and the high continuity. For this reason, only estimated blocks within the main zone were considered for higher classification.

Measured resources were captured using a geometric shape (see Figure 14-26) which was intended to capture an average drill spacing of 160 ft or less within the main zone (mmh).

Indicated resources were categorized using the 2nd search pass. Confidence in this estimate is high due to the complete rebuilding of the drilling database and a ground up remodeling of the geology, based on a new lithological coding of the drill data, without consideration of grade. The results closely match those of the previous two estimates which used the previous approach of grade-continuity based domain wireframes. This was taken into consideration when classifying the indicated resources.

The inferred category was applied to all other estimated blocks.

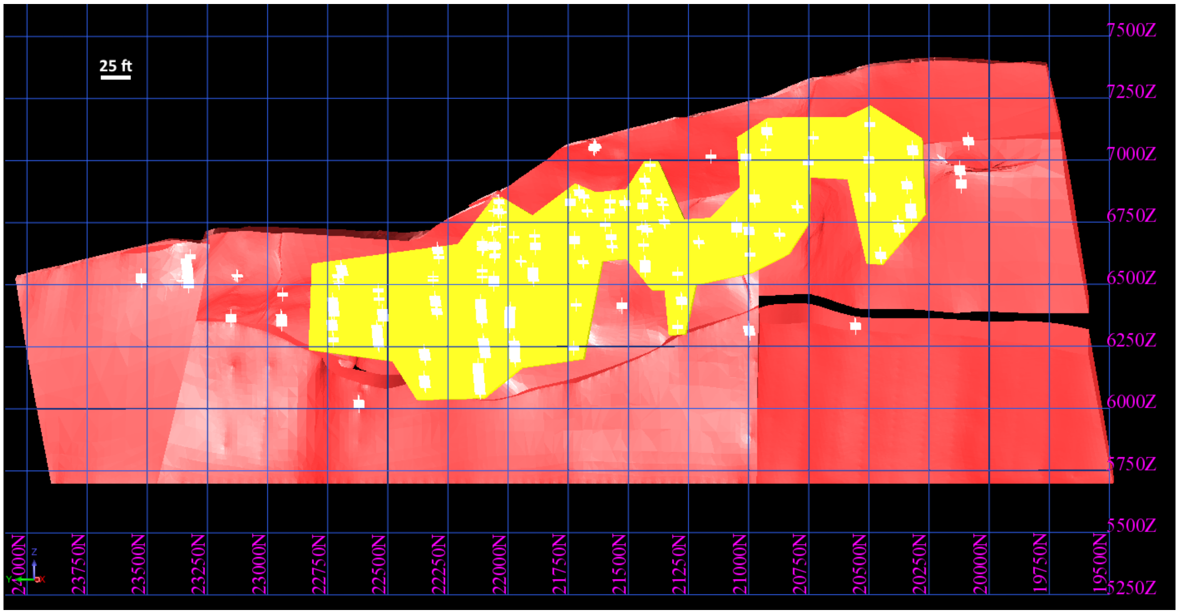


Figure 14-26: Measured category shape (yellow) for mmh zone (red) with composites (white crosses) in orthogonal 3D view in local grid

14.11 MODEL VALIDATION

14.11.1 Statistics

As in all estimates, the grade average between the estimate and the originating samples has lowered.

This is common in part because sampling is inevitably clustered around high grade areas, creating a bias in the input which is rectified geometrically in the estimation process. Compositing and capping also plays a role in this effect. Where this is not the case, as in some of the domains for Cu, this is likely due to the secondary role this metal has taken in the exploration planning and sampling (high-grade copper has not been sought out so is likely geometrically under-represented in sampling).

Table 14-10: Overall Estimated Block Statistics by Zone

Zone	Field	Nsamples	Min	Max	Mean	Var	Standdev	Standerr	Cov	Skewness
btn	co_id2	37085	0.001	0.107	**0.011**	0.00	0.02	0.00	1.49	3.214
fw	co_id2	53047	0.001	0.700	**0.047**	0.00	0.07	0.00	1.50	3.467
hw0	co_id2	38077	0.001	0.469	**0.031**	0.00	0.05	0.00	1.65	3.638
hw1	co_id2	209018	0.000	0.700	**0.034**	0.00	0.07	0.00	1.93	4.255
hw2	co_id2	49931	0.001	0.692	**0.022**	0.00	0.04	0.00	1.96	5.226
hw3	co_id2	45331	0.001	0.640	**0.032**	0.00	0.06	0.00	1.96	3.838
hw3a	co_id2	994	0.001	0.028	**0.006**	0.00	0.00	0.00	0.67	1.366
hw4	co_id2	1685	0.001	0.345	**0.021**	0.00	0.03	0.00	1.69	4.129
mmh	co_id2	239862	0.000	3.896	**0.255**	0.09	0.30	0.00	1.18	2.647
btn	cu_id2	37085	0.000	1.031	**0.141**	0.02	0.15	0.00	1.05	2.083
fw	cu_id2	53047	0.000	1.504	**0.082**	0.01	0.11	0.00	1.33	4.096
hw0	cu_id2	38077	0.000	1.104	**0.075**	0.01	0.10	0.00	1.36	2.284
hw1	cu_id2	209018	0.000	1.954	**0.057**	0.02	0.13	0.00	2.33	5.560
hw2	cu_id2	49931	0.000	0.592	**0.032**	0.00	0.05	0.00	1.61	3.652

Zone	Field	Nsamples	Min	Max	Mean	Var	Standdev	Standerr	Cov	Skewness
hw3	cu_id2	45331	0.000	0.291	**0.024**	0.00	0.03	0.00	1.28	2.334
hw3a	cu_id2	994	0.001	0.139	**0.025**	0.00	0.02	0.00	0.87	1.658
hw4	cu_id2	1685	0.000	0.610	**0.028**	0.01	0.07	0.00	2.65	3.482
mmh	cu_id2	239862	0.000	3.997	**0.510**	0.32	0.57	0.00	1.11	1.892
btn	au_id2	37085	0.000	0.023	**0.001**	0.00	0.00	0.00	1.75	2.958
fw	au_id2	53047	0.000	0.026	**0.001**	0.00	0.00	0.00	1.72	4.318
hw0	au_id2	38077	0.000	0.026	**0.002**	0.00	0.00	0.00	1.52	2.362
hw1	au_id2	209018	0.000	0.063	**0.001**	0.00	0.00	0.00	2.58	6.099
hw2	au_id2	49931	0.000	0.016	**0.000**	0.00	0.00	0.00	2.06	4.720
hw3	au_id2	45331	0.000	0.033	**0.001**	0.00	0.00	0.00	2.25	4.163
hw3a	au_id2	994	0.000	0.003	**0.001**	0.00	0.00	0.00	1.02	0.849
hw4	au_id2	1685	0.000	0.007	**0.001**	0.00	0.00	0.00	1.45	2.187
mmh	au_id2	239862	0.000	0.330	**0.010**	0.00	0.02	0.00	1.50	6.311

14.11.2 Population Distribution

Histograms are used to determine whether the population distribution has been accurately maintained in the estimation process. This ensures that the data has not been unnecessarily smoothed.

Histograms were reviewed for all zones and metals. The population distributions compare favourably, with the normal reduction of the mean and shifting of the population to slightly lower grades.

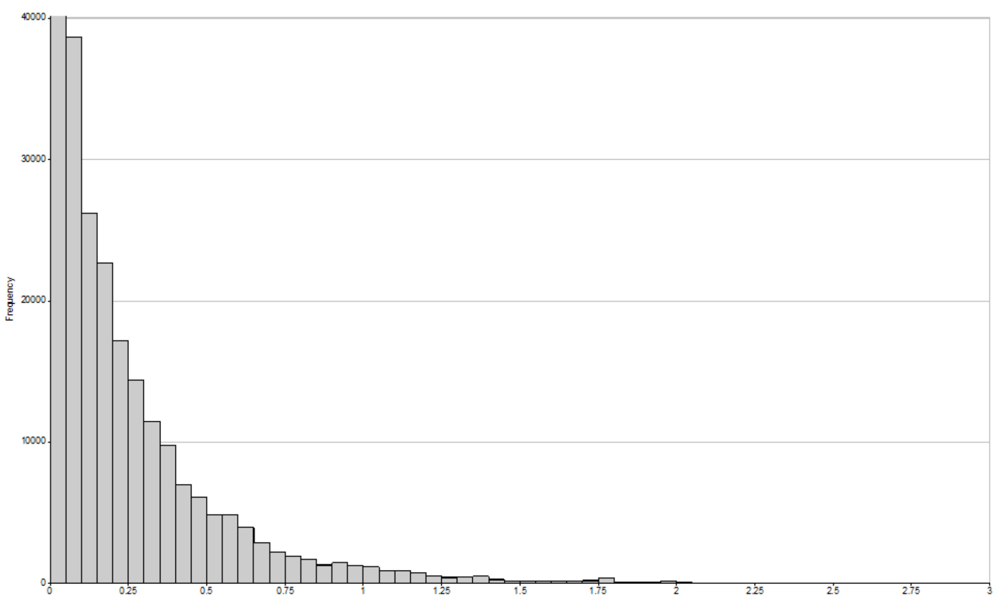


Figure 14-27: Co Histogram of "mmh" zone

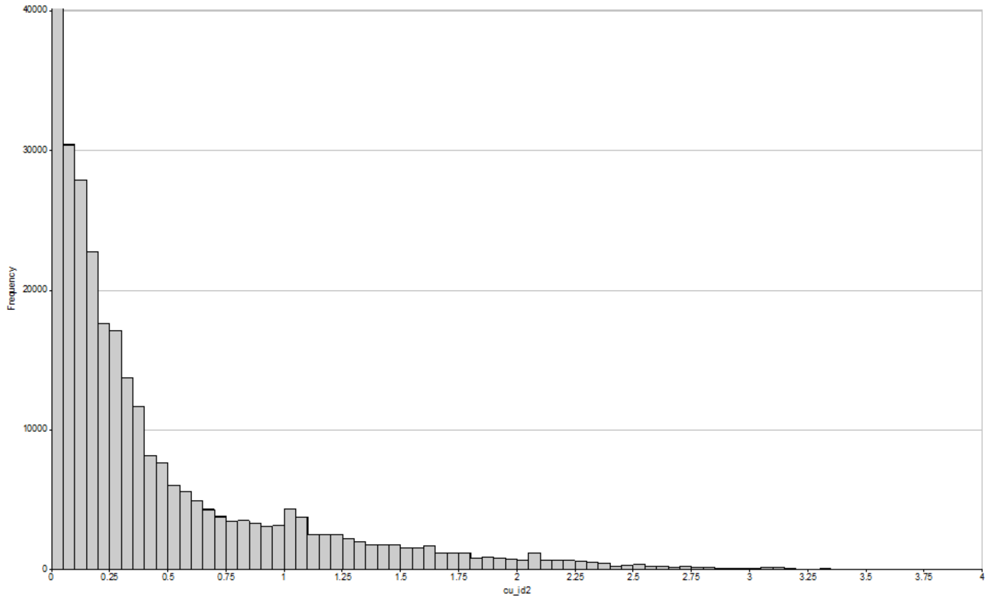


Figure 14-28: Cu Histogram of "mmh" zone

14.11.3 Sections and Plans

Sections and Plans confirm the correlation between drill results and estimated grades. Continuity seems logical and there are no glaring mismatches between drill hole grades and block model grades.

In the following sections and plans, the zones are demarcated as follows:

- "mmh" zone – red outline
- Hanging wall zones ("hw0" – "hw4") – green outline
- "fw" zone – grey outline
- "btn" zone – yellow outline

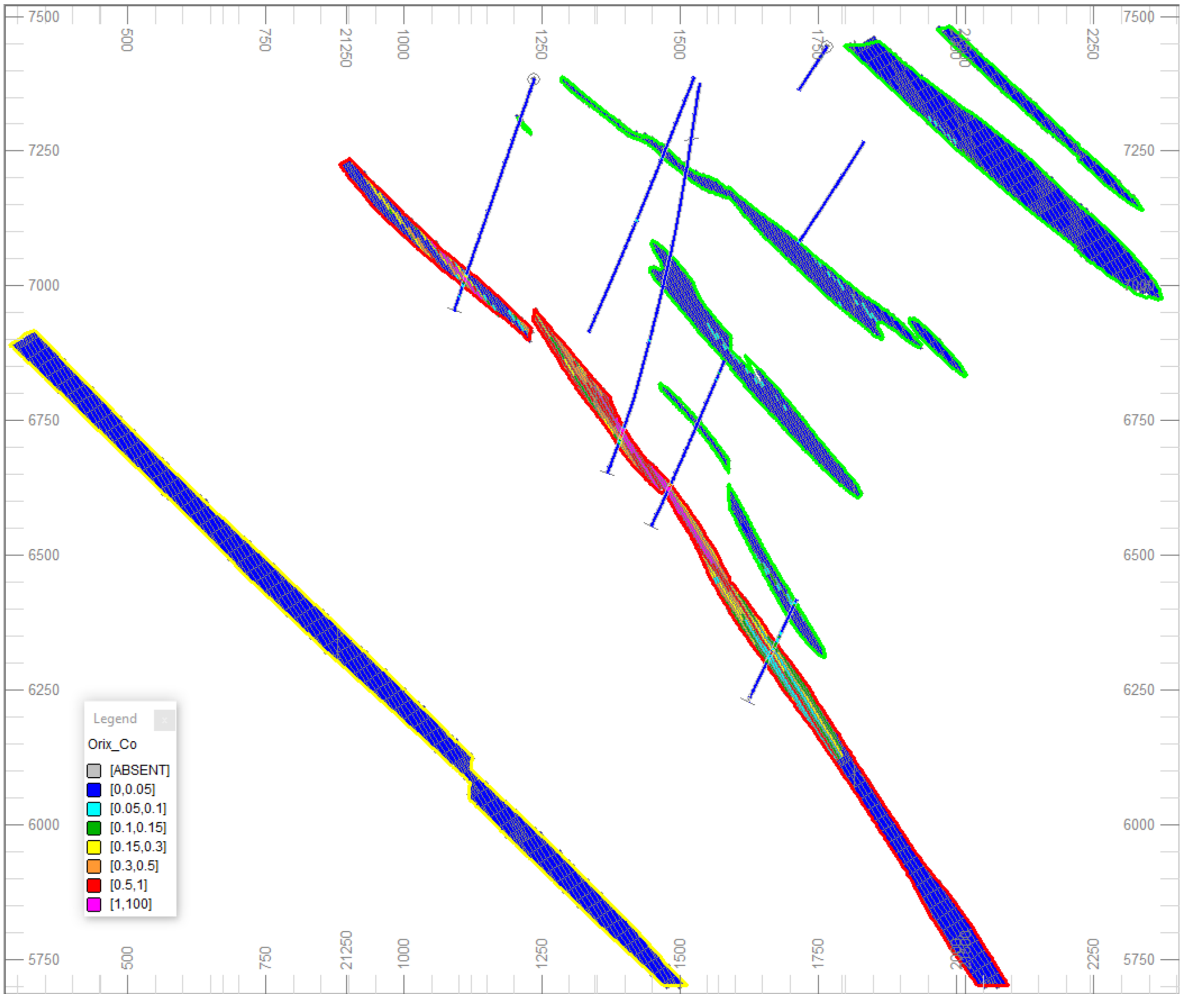


Figure 14-29: Typical Section looking NNW (346°, section burden 50 feet)

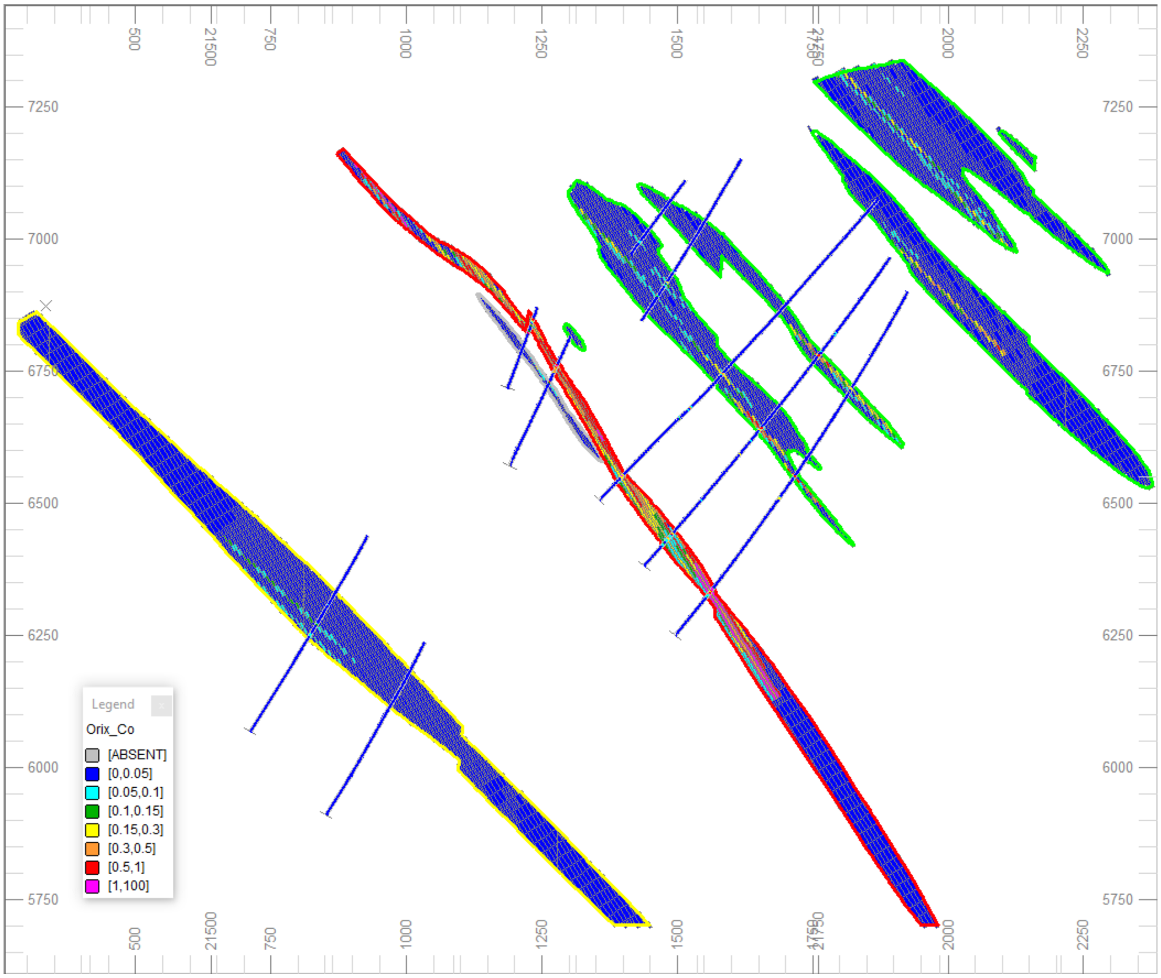


Figure 14-30: Typical Section looking NNW (346°, section burden 100 feet)

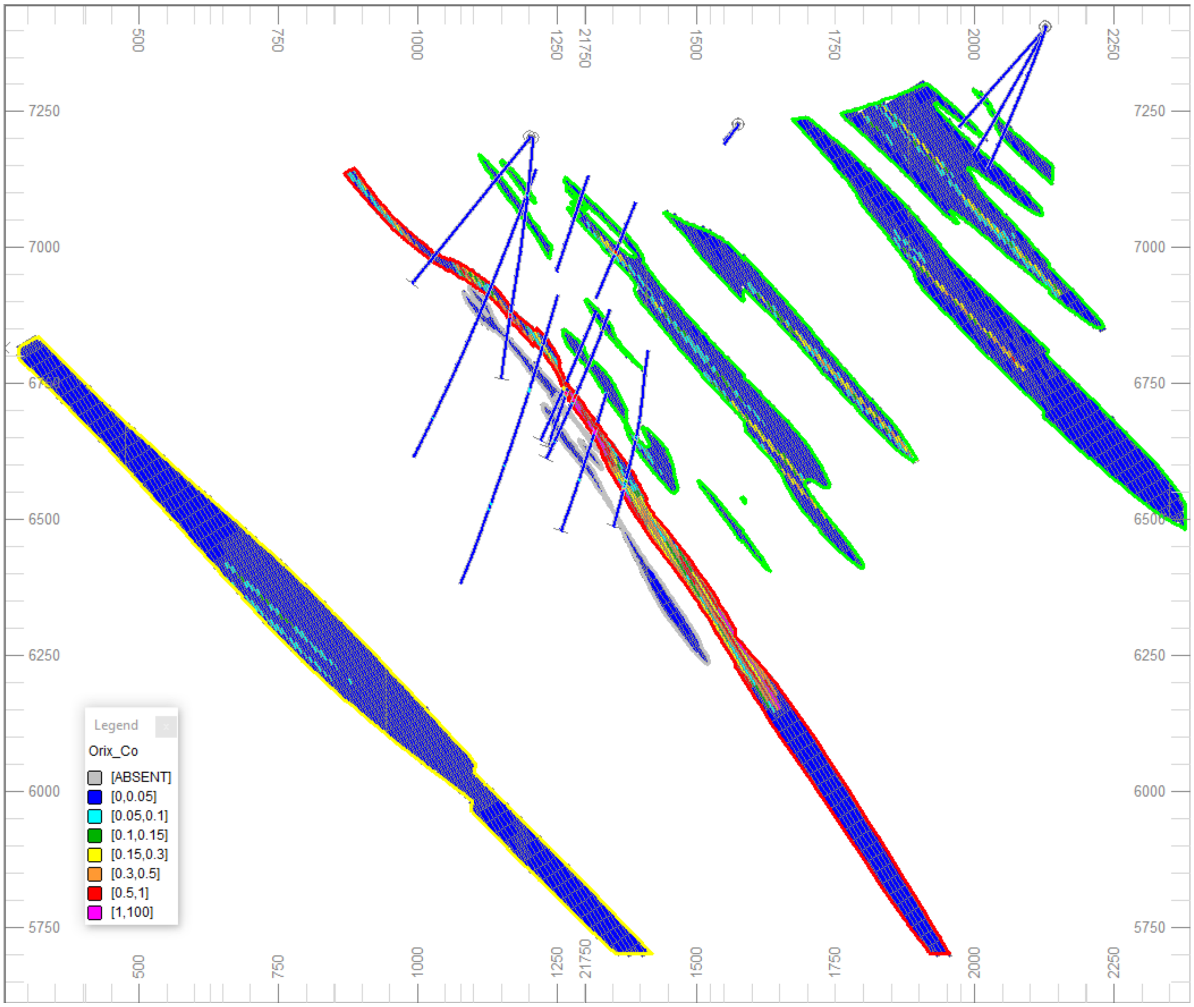


Figure 14-31: Typical Section looking NNW (346°, section burden 50 feet)

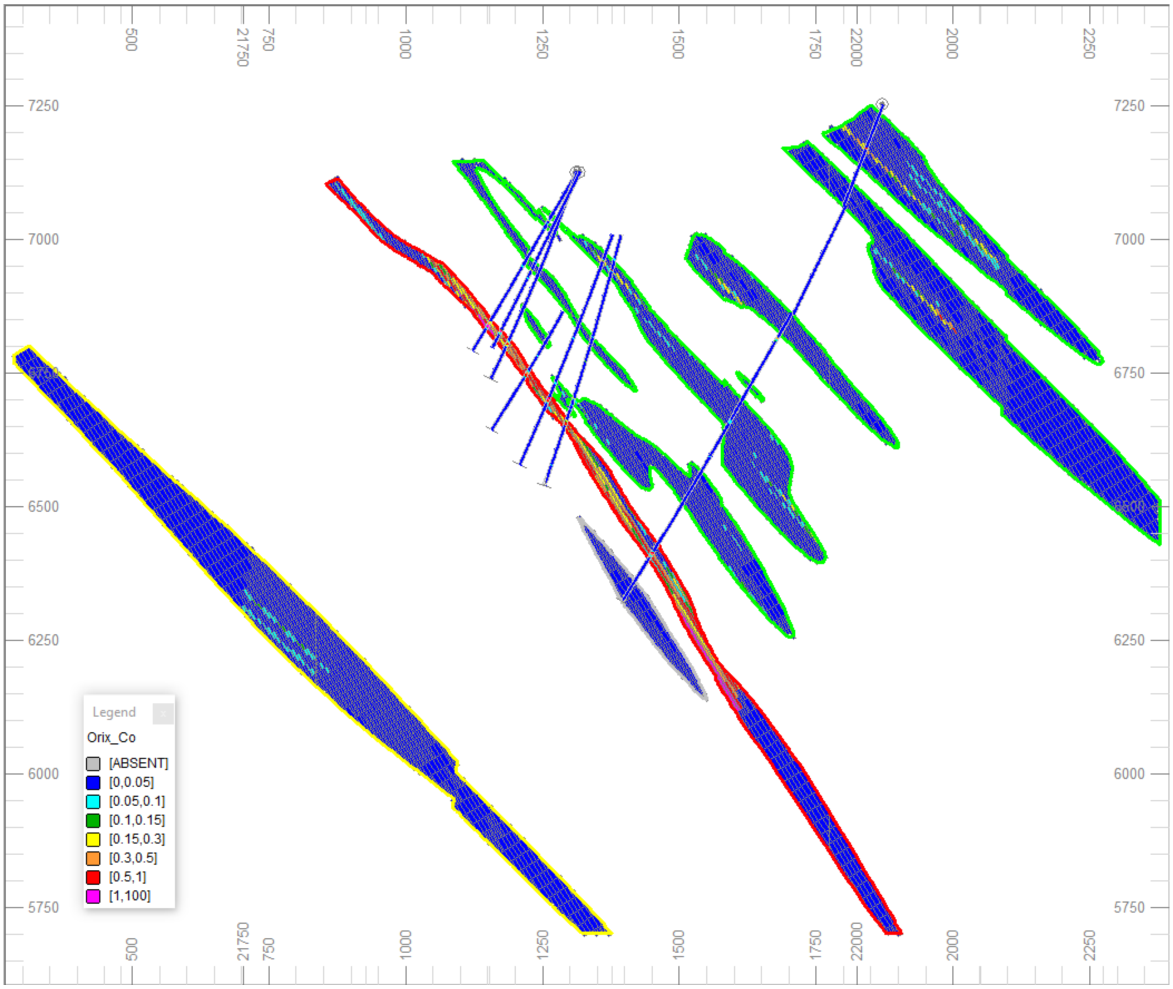


Figure 14-32: Typical Section looking NNW (346°, section burden 100 feet)

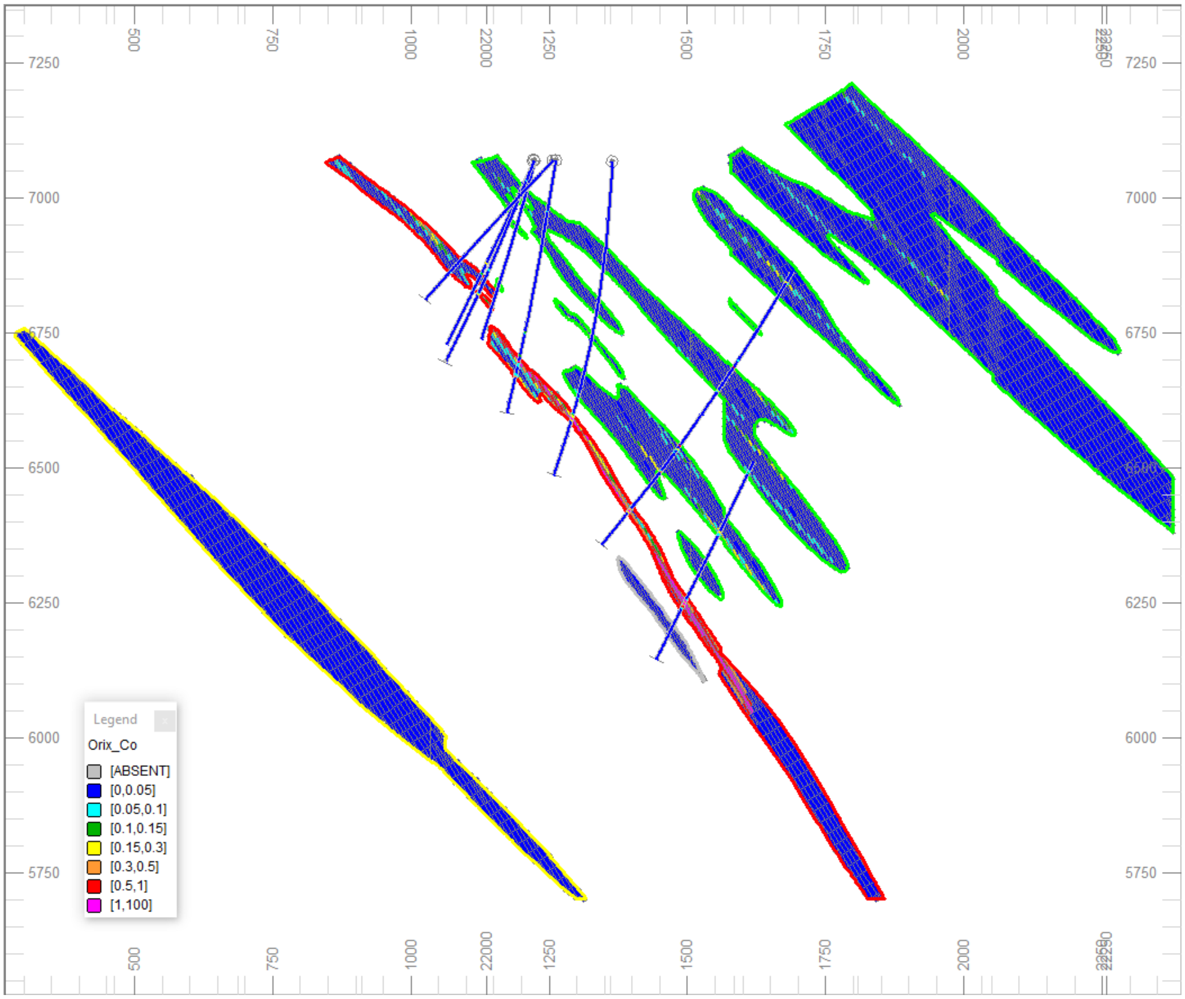


Figure 14-33: Typical Section looking NNW (346°, section burden 100 feet)

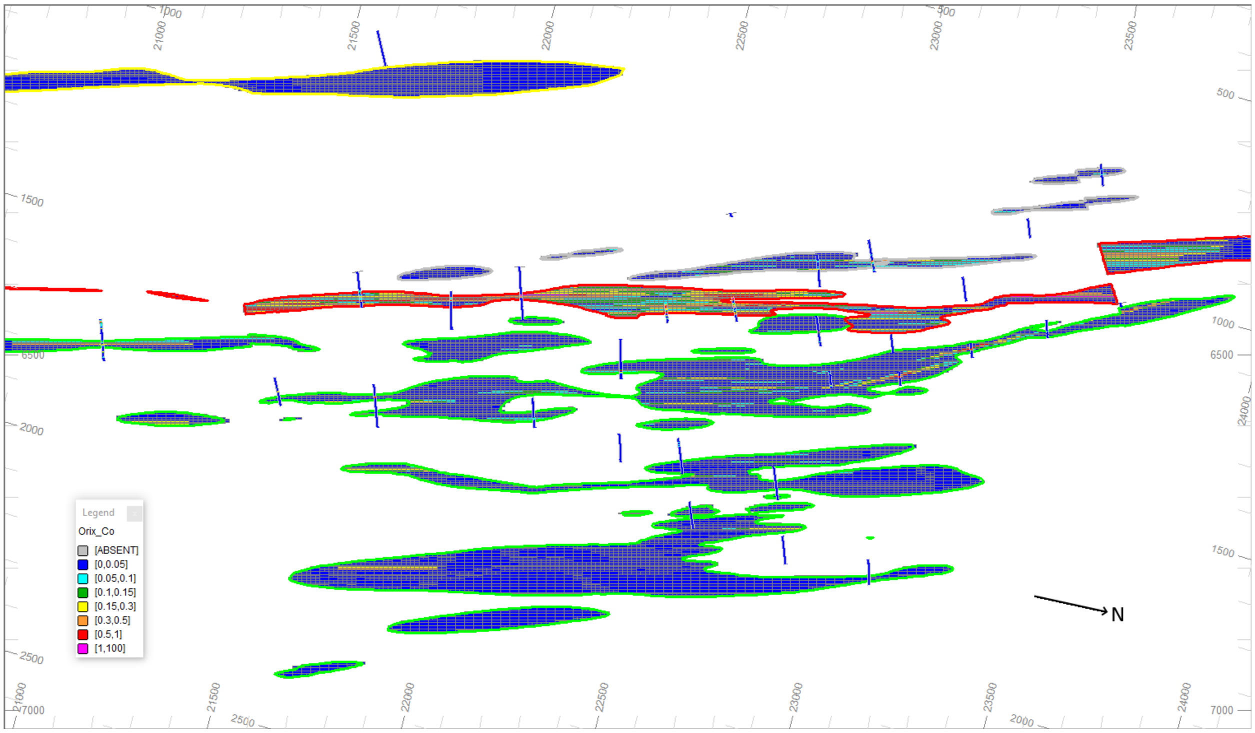


Figure 14-34: Typical Plan (Rotated to 256°, Inclined 32°, parallel to blocks)

14.11.4 Trend Analysis

Geographic trends are validated using swath plots. This can identify over-smoothing as well as high grade over-spreading. In this instance, the swath plots confirm the correlation between drill hole assays and estimated grades in all directions. The estimated grades track well in all directions with the sample data, with reduced peaks and valleys. The peaks or spikes observed correspond with areas of low numbers of samples.

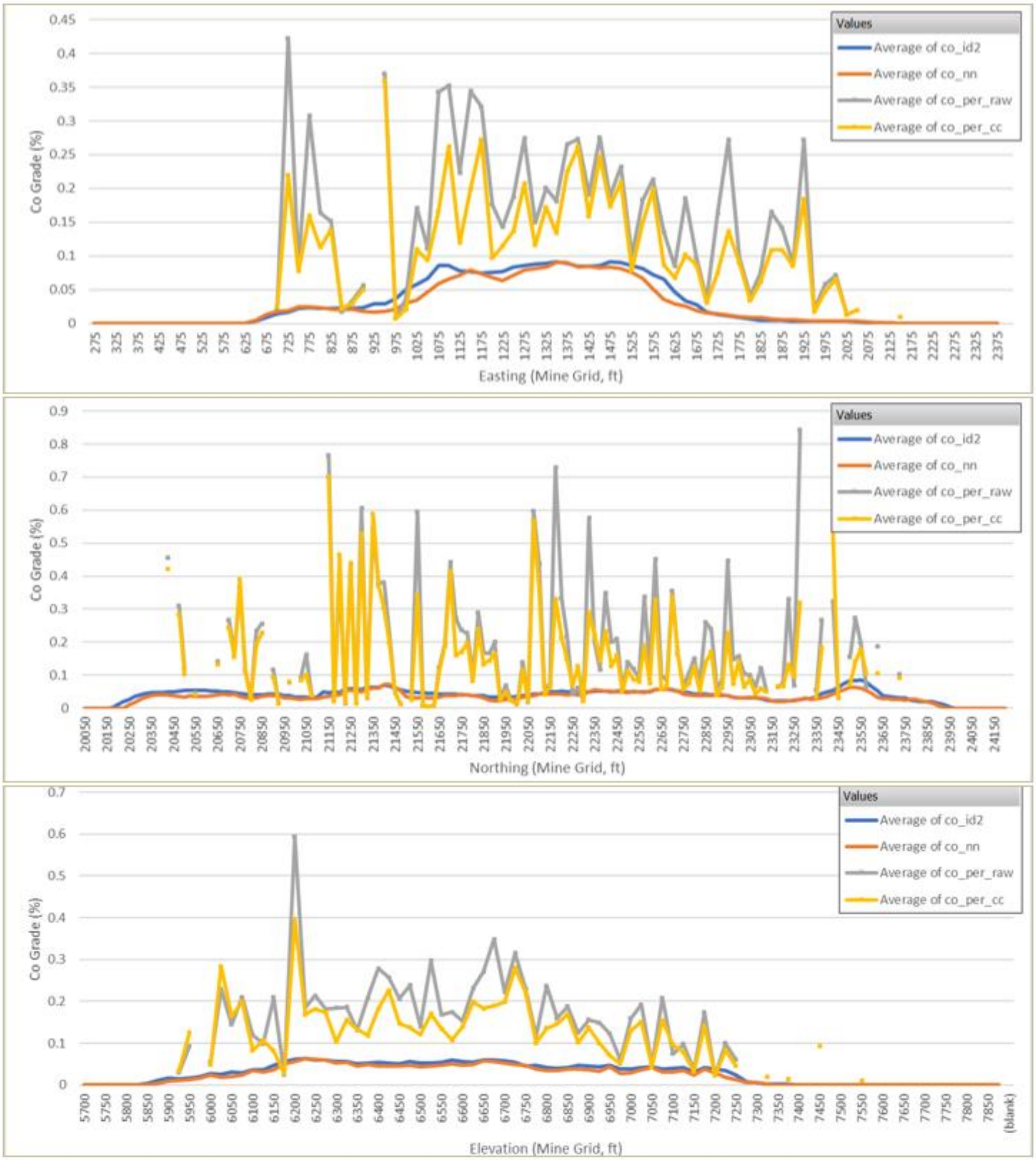


Figure 14-35: Co Swath Plots, All Zones

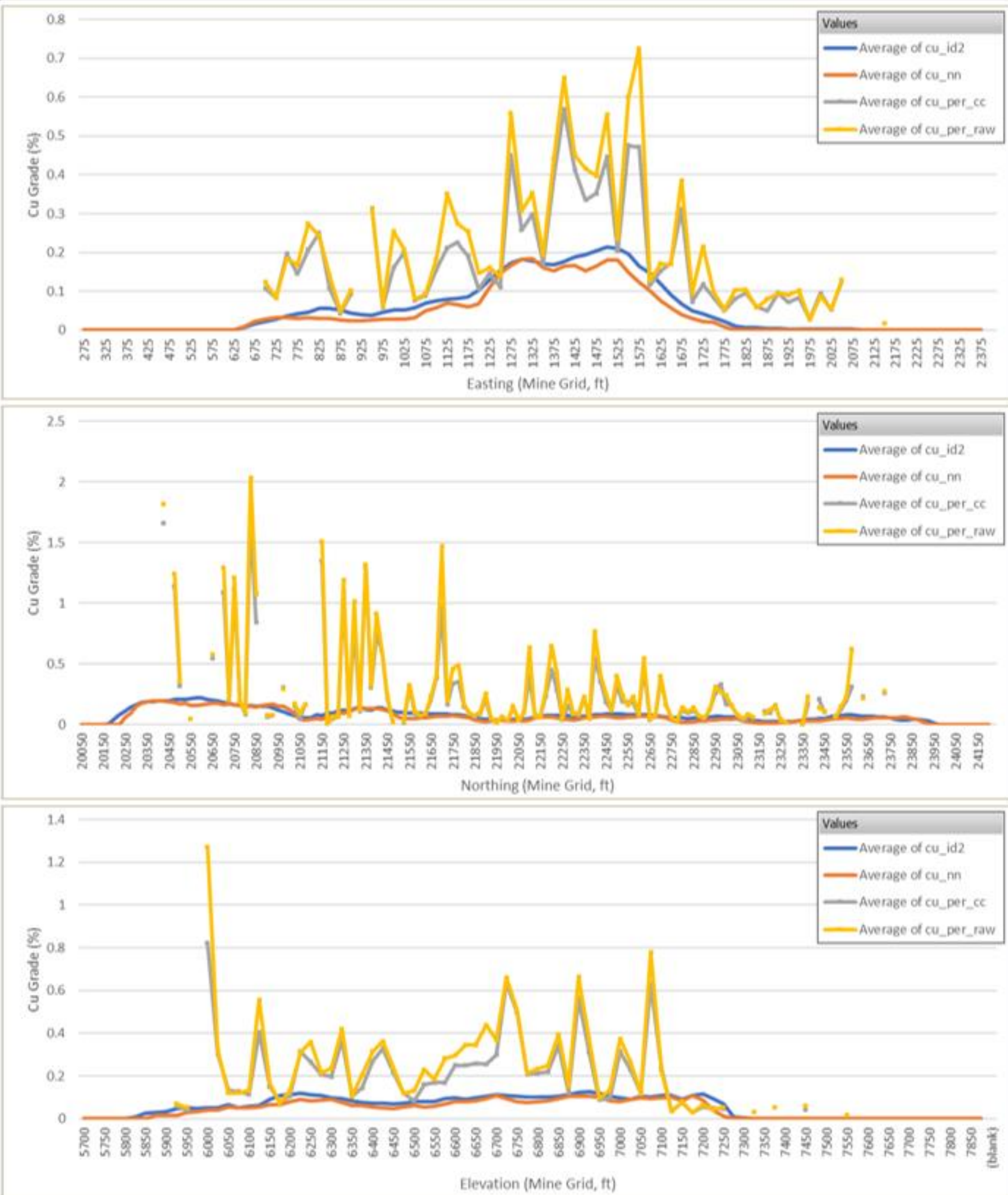


Figure 14-36: Cu Swath Plots, All Zones

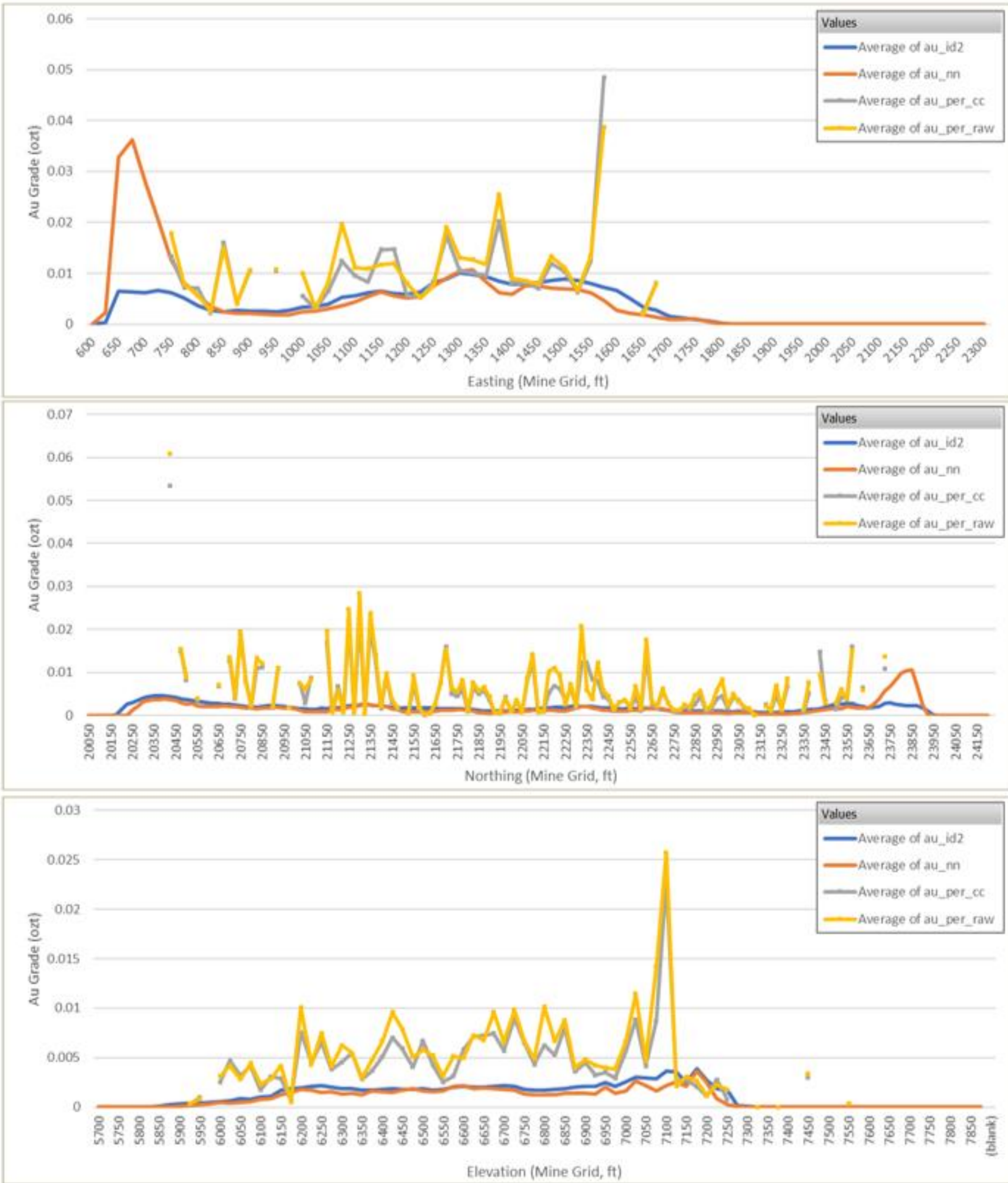


Figure 14-37: Au Swath Plots, All Zones

14.12 MINERAL RESOURCE TABULATION

The Ram mineral resource is tabulated at Co% cut-offs. A cut-off of 0.15 was chosen based on the results of metallurgical and rock-sorting studies as well as the currently proposed mining work (described elsewhere in this report). It is assumed the deposit will be mined underground using cut and fill, back slash stoping methods based on previously completed mining studies. Forecast Co prices were also considered, and the possibility of higher prices yielded the use of a cut-off slightly below previous studies.

14.12.1 Resource Table

Table 14-11 displays the grade-tonnage summary by Co% cut-off for the resources of the Ram deposit. Additional cut-offs are displayed to show the sensitivity to higher cut-off grades.

Table 14-11: Ram Deposit Resource Estimate

	Inferred				
Co Cut-off	Tons*	Co %	Co lbs	Cu %	Au ozt**
0.15	1,730,000	0.35	12,000,000	0.44	0.013
0.20	1,220,000	0.42	10,300,000	0.50	0.016
0.25	910,000	0.49	8,900,000	0.56	0.018
0.30	670,000	0.57	7,600,000	0.65	0.021
0.35	500,000	0.65	6,400,000	0.71	0.021
	Indicated				
Co Cut-off	Tons	Co %	Co lbs	Cu %	Au ozt
0.15	2,850,000	0.42	23,900,000	0.80	0.018
0.20	2,360,000	0.47	22,100,000	0.86	0.020
0.25	1,910,000	0.53	20,100,000	0.92	0.022
0.30	1,580,000	0.58	18,300,000	0.96	0.023
0.35	1,290,000	0.64	16,400,000	1.00	0.025
	Measured				
Co Cut-off	Tons	Co %	Co lbs	Cu %	Au ozt
0.15	2,920,000	0.45	26,200,000	0.59	0.013
0.20	2,340,000	0.52	24,200,000	0.63	0.015
0.25	1,950,000	0.58	22,500,000	0.67	0.016
0.30	1,640,000	0.63	20,800,000	0.71	0.017
0.35	1,400,000	0.68	19,200,000	0.74	0.018

**Imperial tons **troy ounces per imperial ton*

14.12.2 Grade – Tonnage Curves

Grade-tonnage curves demonstrate the sensitivity of the estimated grade and tonnage to higher cut-offs.

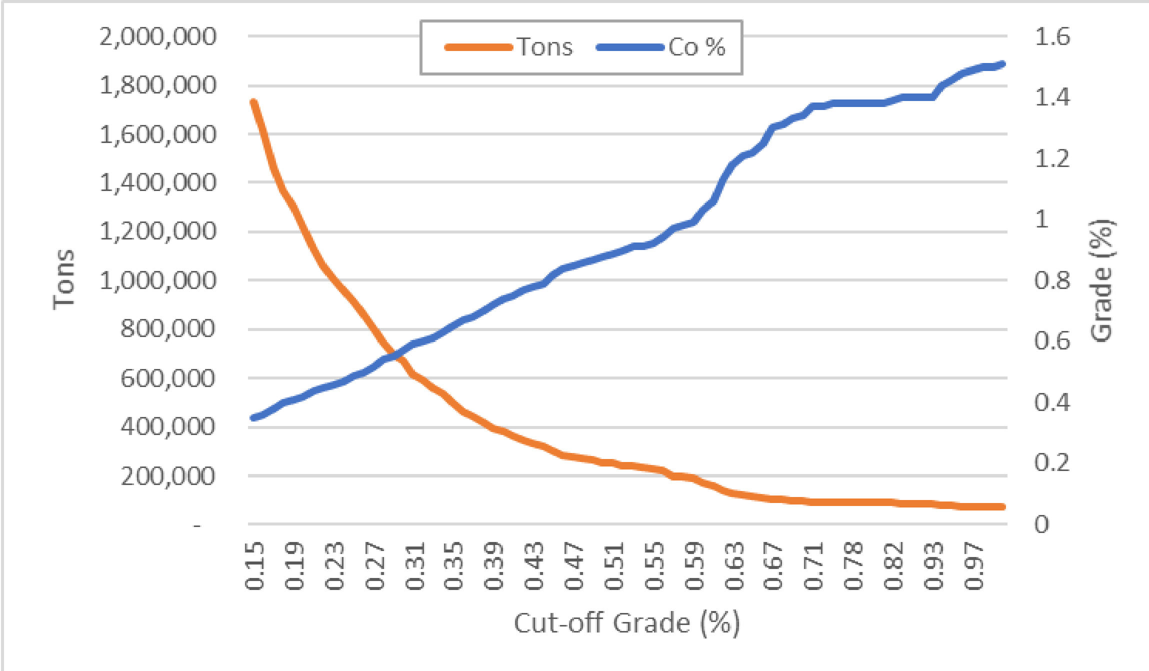


Figure 14-38: Grade (Co%) and Tonnage (t) Sensitivity to Cut-off Grade – Inferred

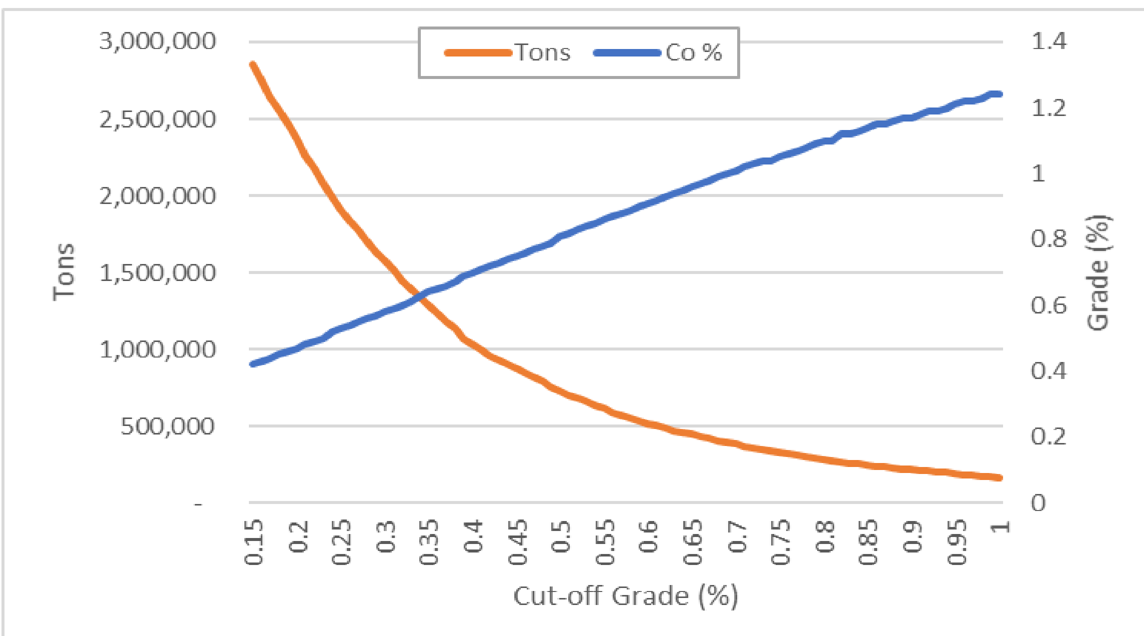


Figure 14-39: Grade (Co%) and Tonnage (t) Sensitivity to Cut-off Grade – Indicated

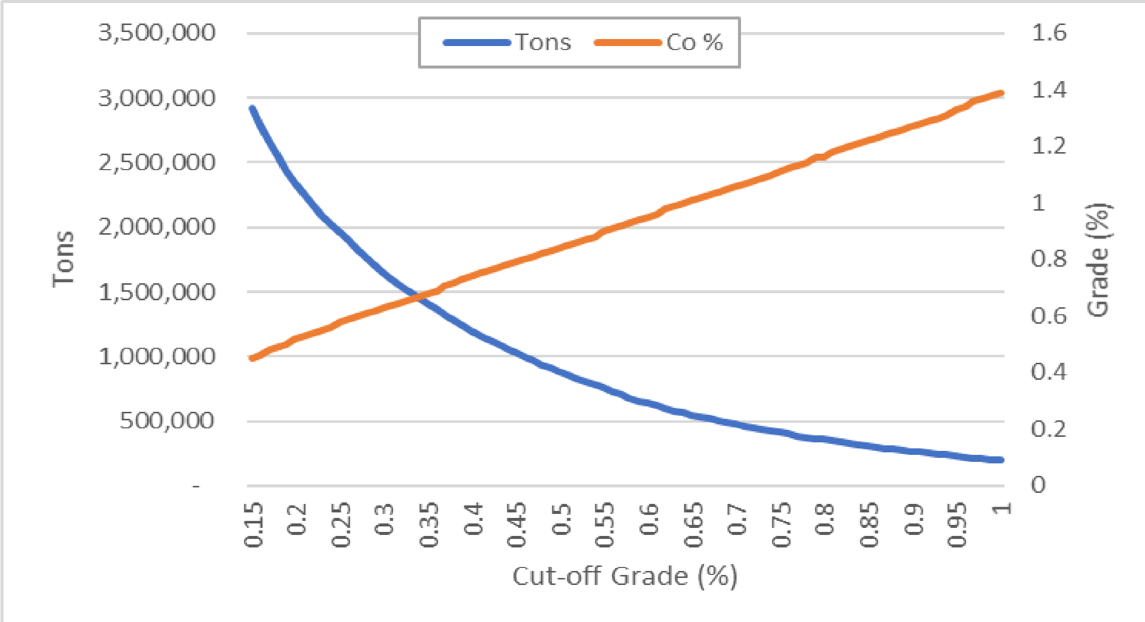


Figure 14-40: Grade (Co%) and Tonnage (t) Sensitivity to Cut-off Grade – Measured

14.13 COMPARISON TO PREVIOUS RESOURCE ESTIMATES

The previous resource estimates were completed in 2017 (Micon, 2017) and 2018. The 2018 resource estimate was issued but no report was filed, so comparisons beyond the table are difficult.

The 2017 model was completed using the previous zone interpretation, which primarily used grade-based continuity to model zones. This approach was reconsidered in favour of using a geology-based domaining. Table 14-12 shows a comparison of the Co values in the current estimation to the previous results at 0.2% Co cut-off.

Table 14-12: Comparison of current estimation results to previous results

Category	2017			2018			Current		
	Tons	Co %	Co lbs.	Tons	Co %	Co lbs.	Tons	Co %	Co lbs.
Inferred	1,543,000	0.51	15,594,000	1,820,000	0.46	16,700,000	1,202,000	0.42	10,159,000
Indicated	1,711,000	0.64	21,988,000	2,370,000	0.54	25,800,000	2,363,000	0.47	22,136,000
Measured	1,725,000	0.54	18,590,000	1,500,000	0.66	19,900,000	2,338,000	0.52	24,183,000

The author is confident in the results of the current estimation, which represents a "back-to-basics" approach of completely rebuilding the dataset and remodelling the estimation domains based solely on geology. The numbers generated hew closely to those generated in the last two estimates which employed a grade-based domain method, meaning two different approaches produced very similar results, which demonstrates the deposit's robustness.

14.14 RECOMMENDATIONS

The following two-phase program is recommended as a result of the geological modelling and resource estimation:

14.14.1 Phase 1 – Summer 2021

One thousand six hundred (1,600) metres of drilling is recommended to test up-dip positions on the Blacktail North zone. As the development of the underground mine is anticipated, further testing of the Hanging wall zones, or Main zone is not recommended at this time.

Cost of this program is anticipated at approximately US$340/metre x 1600 metres for a total of US$544,000.

14.14.2 Phase 2 – Winter 2021-2022

Phase 2 includes follow-up drilling based on the results of Phase 1 and definition drilling of the Hanging wall zones and Main zone from underground positions (in conjunction with grade control drilling).

For Blacktail North, an additional 1600 metres of drilling is recommended if Phase 1 produces promising results.

For the Hanging wall and Main zones, 1000 metres of definition drilling from underground (meaning shorter holes).

Cost of this program is anticipated at approximately US$340/metre x 1600 metres for a total of US$544,000 for the Blacktail North, and US$250/metre x 1000 metres for a total of US$250,000 for the Hanging wall and Main zones, for a total of US$794,000.

14.14.3 Additional Recommendations

Orix, Scott Zelligan, and CSA Global recommend the following be implemented for all future programs (including the above two phases):

- The use of four-acid digestion for the overlimit arsenic (“As”) values may be problematic because As, an element of interest, can volatise with this method leading to a potential underreporting. CSA Global recommends some of the OG62 arsenic overlimit pulps to be analysed with aqua regia digestion OG46 method to determine if there may have been any volatilization of As.
- The use of a regression formula to define densities for the resource calculation proved adequate. However, standardization of SG measurements potentially using whole core intervals, should be a considered for any oncoming drill program.
- Although the spread use of standard material produced in the late 1990s has yielded decent results, Jervois could re-examine and likely produce new CRMs for ongoing programs.
- QA/QC could be improved by the inclusion of field duplicates more frequently in the sequence, as well as the inclusion of true certified blanks as opposed to the red brick material used during the latest programs.
- Currently all historic and 2019 data is hosted in an excel format database. Ideally a commercial relational database should be used which has built-in error checking, audit documentation and QA/QC.
- Given the gradational nature of the sedimentary package, Orix recommends a detailed analysis of the existing geochemical data, to look for geochemical signatures that could highlight marker horizons. These marker horizons, particularly if found in the hanging wall zones, could prove very useful to determine true lateral continuity of some of those units, which could possibly give a positive impact in the re-definition of the resource categories.
- The current resource model presents the opportunity to test areas where plunges/ore shoots may be present. If said ore shoots do exist and continue at depth, they could represent prospective economic areas at depth.
- The Ram deposit lays less than 2 km north of the famous Blackbird group of deposits, mined by several operators in the 1900s. It is recommended that Jervois use the opportunity to explore the footwall of the Ram, in search of equivalent stratigraphic horizons that exist in the Blackbird mine.

15 MINERAL RESERVE ESTIMATES

For the ICO, the Measured and Indicated mineral resource from the main mineralised horizon was considered in the mine plan for conversion into a mineral reserve. Measured resources were, where above cut-off grade on a diluted basis in stope shapes, converted to Proven Reserves. Indicated resources were, where above cut-off grade on a diluted basis in stope shapes, converted to Probable Reserves.

Conversion of the mineral resource estimates to mineral reserve was inclusive of the modifying factors, diluting material and allowances for losses which are to be expected when the material is mined or extracted.

15.1 RESOURCE MODEL

The resource model described in Section 4 was used to determine the mineral resource considered in the mine plan. This resource was then converted to the mineral reserve.

Cobalt grades for the whole or complete parent block meeting the Cut-off Grade (“CoG”) and the criteria listed below, were used to determine the mineral reserve. The parent block dimensions are defined as 24 ft by 24 ft by 8 ft in the X, Y, and Z directions.

There was no additional or subsequent sub-blocking performed on these parent blocks of the resource model. This is because the parent blocks have been deemed to provide sufficient resolution for mine design, planning, and the necessary resolution to identify the interface between mineralised and waste material. The dimensions of the parent blocks also enable the definition of the stope outlines, and the estimation of dilution and material losses.

15.2 CUT-OFF GRADE (COG) CRITERIA AND ESTIMATE

The mineral reserve was based on the mineral resource model’s tonnages and grades, reported in blocks meeting or exceeding the CoG of cobalt. The CoG value used in the mine design was based on the operating costs, metallurgical recovery estimates, net payable value of the contained metal and market prices that resulted from this study.

The stope outlines and mineable tonnages and grades for short hole back stoping and sill mining methods were defined based on a CoG of 0.32% and 0.30% contained Co respectively to generate a mine plan further restricted by metal recoveries and net payable values.

Each stope block was assessed for total recovered and payable value on an equivalent cobalt content basis across a range of cobalt prices to determine the best value CoG. Only stope Blocks achieving Equivalent recoverable and payable cobalt grades above 0.24% were included in the reserve to support economic mining at the target cobalt price of US$25.00/lb used in the study. The commercial (payable) terms that underpin the reserve estimate are approximately 75% of the contained metal value in concentrate weighted across cobalt, copper and gold.

Parameters and values used to determine the CoG value are presented in Table 15-1.

Table 15-1: Cut-off Grade Criteria

Description	Unit	Values
Metal Prices		
Cobalt Price	US$/lb	25.00
Copper Price	US$/lb	3.00
Gold	US$/oz	1,450
Recoveries		

Description	Unit	Values
Cobalt -to Co Conc	%	86.2
Copper to Co Conc	%	8.3
Gold to Co Conc	%	82.6
Copper -to Cu Conc	%	89.4
Gold – to Cu Conc.	%	9.2
Cobalt in Co Conc	%	75.00
Copper in Co Conc	%	80.00
Gold in Co Conc	%	70.00
Copper in Cu Conc	%	87.16
Gold in Cu Conc	%	59.22
Costs		
Mining	US$/st	72.14
Milling	US$/st	19.93
General	US$/st	14.02

The output of the analysis of equivalent recovered and payable cobalt grade at varying prices is presented in Figure 15-1.

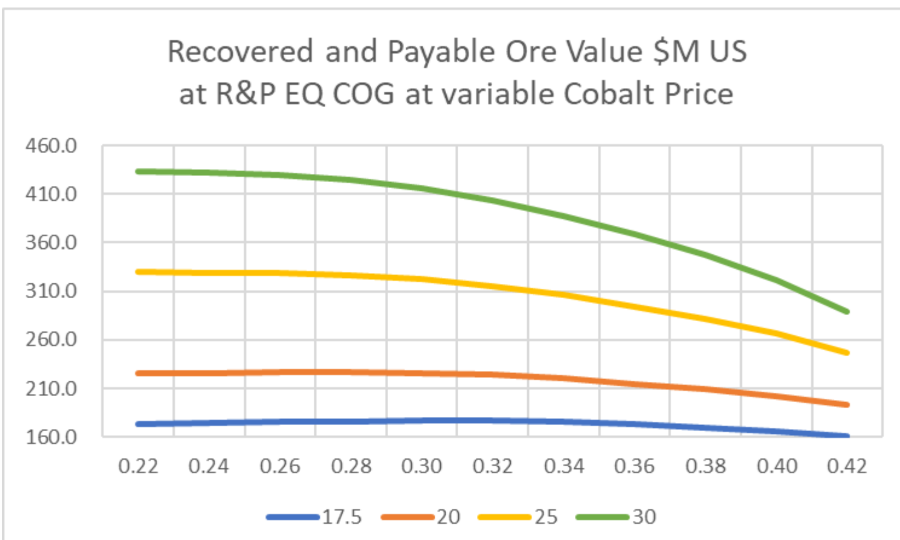


Figure 15-1: Cut-off Grade Determination

15.3 STOPE OUTLINE

The stope outlines were designed using the Deswik Stope Optimizer to represent the planned extraction of the mineralised zones together with any internal or adjoining waste rock which cannot be left in-situ. The tonnage and grade contained within these stope outlines are reported as whole blocks only with no grade being attributed to material not meeting a measured or indicated resource classification.

Stope blocks include 0.5 ft overbreak on the footwall and hanging wall (1' total rib dilution) and align to resource block orientations wherever possible.

The stope outlines were generated from 12 ft vertical level interval shells, honoring the cobalt CoG of 0.24% on a recovered and payable equivalent basis. Each stope block represents two production rounds.

15.4 DILUTION AND LOSSES

Two types of dilution values were applied in determining the mineral reserve, depending on the dip angle of the deposit, the configuration of the minimum mining width and the mining methods:

- Planned or internal dilution: including all the mineralised, low grade and waste material contained in the whole block and the stope outline.
- Unplanned or external dilution: accounting for additional zero grade waste material being included for the proposed mining methods due to the physical configuration of the horizons and mining widths.

The total planned volumetric dilution is approximately 16%, based on the difference of a 0.30% Co grade shell and designed ore mining shapes. This includes 0.5 ft overbreak included in the planned ore shapes. The unplanned dilution sources are:

- Additional possible stope overbreak in certain areas due to local geotechnical conditions
- Tunnel overbreak, as practical penalties for tunnel underbreak (re-drill and re-blast) are typically higher than slight overbreak

The weighted average unplanned dilution is calculated to be approximately 5%. The total planned and unplanned dilution together is approximately 22%.

15.5 MINING RECOVERY

Mineralised material losses arise because of the difficulty of loading and mining mineralised material from the excavated stopes. This includes losses due to fines and pillars left behind during mining.

Sill mats are constructed of high strength paste backfill material poured into lead stopes and as a cap in fill of back stopes to minimize the amount of mineralised material lost as pillars or sill pillars. Much consideration was done during the mine sequencing for the placement and location of the high strength backfill, to reduce the amount of mineralised material abandoned in the mine or left for extraction towards the latter years of the mine life.

Losses accounted for also include a 1-inch layer of fines in the sill drive of a long-hole stope, as well as a 1 ft skin pillar at the top of each column of stopes in a bottom-up sequence. The average recoveries are 95% of the diluted stope material.

15.6 MINERAL RESERVE ESTIMATE

Stope outlines were generated from two types of 12 ft vertical level interval shells, one being a minimum 15 ft width sill drift and the second being a minimum 6 ft width back stope for the two twelve ft level intervals immediately above the sills. Each stope shape represents production rounds. A base cut-off grade of 0.30% Co was used to create the sill shapes eligible for conversion to reserve and a cut-off grade of 0.32% was used for the back-stope shapes. These shapes were then further filtered to accept only those diluted shapes for which a recovered, and payable cobalt equivalent grade of 0.24% was achieved to provide value equal the cash operating cost estimate of US$120/st delivered to the mill at a floor price of US$25/lb. cobalt. Recoveries used in the calculation were derived from test work conducted as part of this study. Payable values were based on indicative terms from prospective off-takers. Table 15-2 summarises the mineral reserve estimate for the Idaho Cobalt Operation.

Table 15-2: Mineral Reserve for ICO at 0.24% Co Recovered and Payable Equivalent Cut-off Grade - Imperial

Category	Reserve (M short tons)	Co (%)	Co cont. (M lbs)	Cu (%)	Cu cont. (M lbs)	Au (oz / short ton)	Au cont. (oz)
Proven(1,2)	1.59	0.56	17.9	0.67	21.2	0.015	24,633
Probable(1,2)	1.16	0.53	12.3	0.96	22.3	0.023	26,758
Total	**2.75**	**0.55**	**30.1**	**0.80**	**43.6**	**0.019**	**51,391**

Table 15-3: Mineral Reserve for ICO at 0.24% Co Recovered and Payable Equivalent Cut-off Grade - Metric

Category	Reserve (M tonnes)	Co (%)	Co cont. (Tonnes)	Cu (%)	Cu cont. (Tonnes)	Au (g/tonne)	Au cont. (oz)
Proven(1,2)	1.44	0.56	8,100	0.67	9,600	0.53	24,633
Probable(1,2)	1.05	0.53	5,600	0.96	10,100	0.80	26,758
Total	**2.49**	**0.55**	**13,650**	**0.80**	**19,800**	**0.64**	**51,391**

16 MINING

16.1 MINING METHODS

The mining methods proposed for the ICO are overhand longitudinal short hole back stoping from 12 ft high sills spaced 36 ft vertically.

The selection of these mining methods for the deposit was determined primarily by the geometry of the mineralised horizons, including factors such as its continuity, dip and width, and the geotechnical parameters of the rock mass.

The ICO is composed of numerous parallel mineralised horizons, with thickness ranging from one foot to more than 12 ft, at an average dip of 55° (Orix 2020). Currently, only the main mineralised horizon ("MMH") contains the majority of the mineralisation is considered in the mine design, plan and mineral reserve as all of the measured and indicated resource lies within this zone of mineralisation.

The sills and backstops will be completely filled with waste rock and cementitious paste fill. Mining sequencing will be overhand with fully paste filled sills forming crowns to terminate the overhand back stoping in a final retreat blind back stope. The mining method significantly reduces the risk of variability in the orebody through detail mapping and sampling of the orebody from the sills to be developed under geologic control.

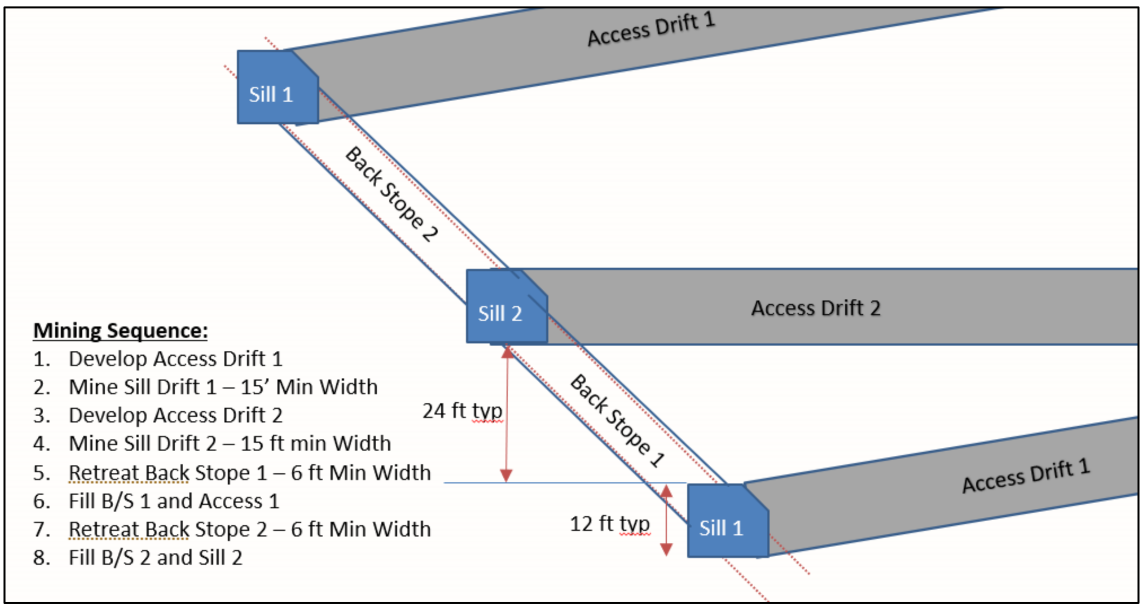


Figure 16-1: Typical Mining Sequence

The ratio of mineral reserve that will be extracted through short hole back stoping and sill mining methods is 62% and 38% respectively. In combination, these two mining methods provide a production capacity in the underground mine that is higher than the nominal mill capacity (1,200 st/d). The proposed mine working schedule is two 11 hours shifts, seven days a week to provide blast fume clearance between shifts. The mine operating cost estimates have been based on the life of mine schedule, created in Deswik supplied to contractors for tender.

Paste prepared from mill tailings will be utilised as backfill material in combination with waste rock fill arising from mine development. Development has been scheduled to maximise waste available for fill during stope fill cycles while providing sufficient development ahead of mining to ensure adequate workplaces should adverse conditions be encountered in a stope.

Excavated material, ore and excess waste, will be hauled by 30 mt payload ejector bed trucks to the portal area, and then loaded by a contractor into 30 mt articulated surface haul trucks for final transportation to the ROM pad and TWSF.

16.2 MINE DESIGN PARAMETERS

The following bullet points summarise the mine design parameters and criteria for the Ram deposit.

- Cut-off Grade of 0.24% Cobalt Equivalent on a recovered and payable basis derived from a population of stope blocks created in Deswik at 0.30% and 0.32% Co grade cut off
- Longitudinal Short-hole back stopes at a minimum width of 6 ft to be cast into the sill drift below and mucked before the subsequent blast to minimise ore loss.
- Sill stopes at a minimum width of 15 ft with a shanty back to allow mechanised mining with the selected fleet. Sills to be spaced at nominal 35 ft centres vertically
- Stope vertical level intervals set at 70 ft between sub-levels and stope blocks generated in Deswik at 24 ft along strike and 12 ft height.
- Only Measured and Indicated mineral resource from the MMH are considered in the mineral reserve estimate.
- Sill Production Advance Rates volumetrically capped at 2,100 ft³ per day per available heading
- Backstope Production Rates volumetrically capped at 5,000 ft³ per day per available stope.
- Paste Fill Rates capped at 7,500 ft³ per day.

16.3 GEOTECHNICAL CONSIDERATIONS

Dr. Woo Shin, Ph.D., P. Eng., an independent rock mechanics specialist was engaged to review previous work and data summarised in the Technical Report which largely drew on work in 2006 by Minefill. Data from the 2004 Ram drill program was largely utilised for rock mass classification. The findings were summarised in the March 2020 Technical Memorandum Re: Jervois Idaho Cobalt Operations Ground Control Recommendations. The 2020 report summarises:

- Geotechnical data review and summary
- Ground support standards development for main ramp and production drifts
- Ground support and developing sequence recommendation for production drifting for wide zones
- Recommendations of ground support QA/QC
- Longhole mine stope stability analysis and support recommendation in different ground conditions
- Backfill design recommendations

16.3.1 Geotechnical Assessments

Previous joint mapping and rock mass classification work is summarised in the sections below as well as commentary on potential seismicity.

16.3.2 Summary of Geotechnical Structural Mapping

A majority of the structural discontinuities were recorded from geotechnical structural mapping campaign carried out on the Ram deposit in 2005. Total 295 discontinuities were recorded during this mapping campaign, and summary of mapping results are presented in Table 16-1.

Table 16-1: Principal Structural Trends from Geotechnical Structural Mapping Campaign (Minefill, 2006)

Rock Type	Joint Set (Dip / Dip Direction)			
	(J_1)	(J_2)	(J_3)	(J_4)
All Data	49/062	39/038	82/311	58/343
Thin Bedded Siltite (TBS)		37/042	85/169	
Coarse Grained Quartzite (CGQ)	61/089			71/312
Medium to Fine Grained Quartzite (MFQ)	48/062	39/039		57/344

Geotechnical mapping data indicated that Joint Set 1 (J 1) could be expected to be a persistent problem in the back (or roof) of the upper levels of the decline because the structure trends parallel to the adit based on the previous mine plan. Depending on the actual persistence and spacing of these joints, the joint sets can be expected to form large wedges in the back of the adit; hence continuous ground support will be required. Joint Set 2 (J 2) is also parallel to some of the major crosscuts envisioned in the upper levels of the previous mine, particularly in stope development crosscuts. These structures will form wedges in the back of the cross cuts and will need artificial support to create a safe opening.

The Uniaxial Compressive Strength ("UCS") for the Ram deposit varies from 35 to 85 MPa on average with 85 to 285 MPa at peak strength. These values were estimated from Point Load Testing on drill cores available during year 2000 drilling campaign. An estimated 1,713 core samples were tested. A conversion factor with the value of 16 was used to convert the Point Load Index to UCS. Table 16-2 presents a summary of rock strength testing results.

Table 16-2: Uniaxial Compressive Strength for Major Rock Types

Rock Type (Code)	# of Tests	% of Sample	Avg. UCS (MPa)	Max. UCS (MPa)	Average ISRM Rating
Biotitic Tuffaceous Exhalative ("BTE")	23	1.3	40.4	121.7	Medium Strong
Coarse Grained Quartzite ("CGO")	27	1.6	82.3	201.9	Strong
Mafic Dykes and Sills ("MDS")	47	2.7	35.2	196.6	Medium Strong
Medium to Fine Grained Quartzite ("MFQ")	1421	83	57.4	275.8	Strong
Quartz Vein ("QTV")	25	1.5	75	285.7	Strong
Thin Bedded Quartzite ("TBQ")	169	9.9	39	150.6	Medium Strong

The quartzite rock types such as MFQ and CGQ, have the highest strengths with maximum values, generally, over 200 MPa with an average value ranging from 50 to 75 MPa, which is considered as very strong rock. While some of the other rock types such as the exhalates appeared to be generally medium strong with an average UCS ranging from 30 to 50 MPa.

Table 16-3: Rock Mass Rating Estimate for Ram Deposit from year 2000 drilling

Rock Type	RMR	RQD	Longest Stick	Count	Rating Description
BTE	57	37	0.099	17	Fair
CGQ	57	44	N/A	27	Fair
MDS	59	35	0.073	16	Fair
MFQ	56	36	0.51	477	Fair
QTV	52	25	0.068	11	Fair
TBQ	50	21	N/A	26	Fair

16.3.3 Summary of Rock Mass Classifications

The average Rock Mass Rating ("RMR") for the Ram deposit ranges from 50 to 59 which is equivalent to "Fair" rock quality. The RMR was determined by Minefill with the factors and parameters indicated above and by assuming "wet" ground conditions and similar joint set conditions which are planar with rough breaks and little or no clay infill from year 2000 drilling program (Holes R99-01 to R99-11) which included point load test on each core run (Minefill, 2006). Table 16-3 illustrates the Rock Mass Rating of the Ram Deposit.

The geotechnical data and parameters collected in 2004 drilling conformed to RMR and the Rock Tunneling Quality Index (Q) classification system proposed by Bieniawski and Barton. Both these classification systems are standard industry rock mass classification systems for mining and civil engineering. Minefill indicated that the Q system had not been used directly in the past to collect geotechnical data for the project. The Q values from the previous work were estimated by Minefill (2006 b) based on the following correlation between RMR and Q as proposed by Bieniawaski (1993):

$$RMR\ 76 = 9\ \mathrm{Log}\ e\ Q + 44$$

The average Q values for major rock mass and major fault zone for geotechnical study in this report were summarised in Table 16-4. For comparison purposes, the Equivalent RMR (Eq. RMR) values determined from the Q values recorded in the year 2004 are also presented in Table 16-4 with the RMR values collected in the year 2000 as a comparison.

Table 16-4: Summary of Rock Mass Classification for Major Deposit and Fault Zones

Description	Mineralized Zone and MFQ	Major Fault
Q	5 – 15 (Fair to Good)	0.1 – 1 (Poor)
Eq. RMR	58 – 68	23 - 44
RMR (previously reported in 2000)	56 – 63	-

16.3.4 Seismicity and Stress Conditions

As the planned maximum excavation depth is within 1,500' of surface, high in-situ stresses are not expected to be encountered. Stress relaxation above wide headings and induced stress redistributions around the long-hole stopes are considered in further sections.

16.4 GROUND SUPPORT STANDARDS FOR MAN ENTRY DRIFTS

16.4.1 Man Entry Opening Design Span

Man entry design span for main decline ramp and production drift have been reviewed based on the critical span curve presented by Ouchi et al. (2004) as shown in Figure 16-2. From this work, the available back span for man entry opening in fair to good ground condition was ranged from 26 ft to 49 ft. However, a maximum critical span in poor ground is limited less than 10 ft, which means immediate ground support such as pre-spraying of shotcrete will be required before installing primary ground support by pattern bolting with screen. The possible span of heading in extremely poor ground is less than 6 ft which lies on the boundary between unstable and potentially unstable back condition and, if wider than the critical span heading is required in this low rock mass quality ground, pre-ground support method, spilling and grouted pore-poling, may be required.

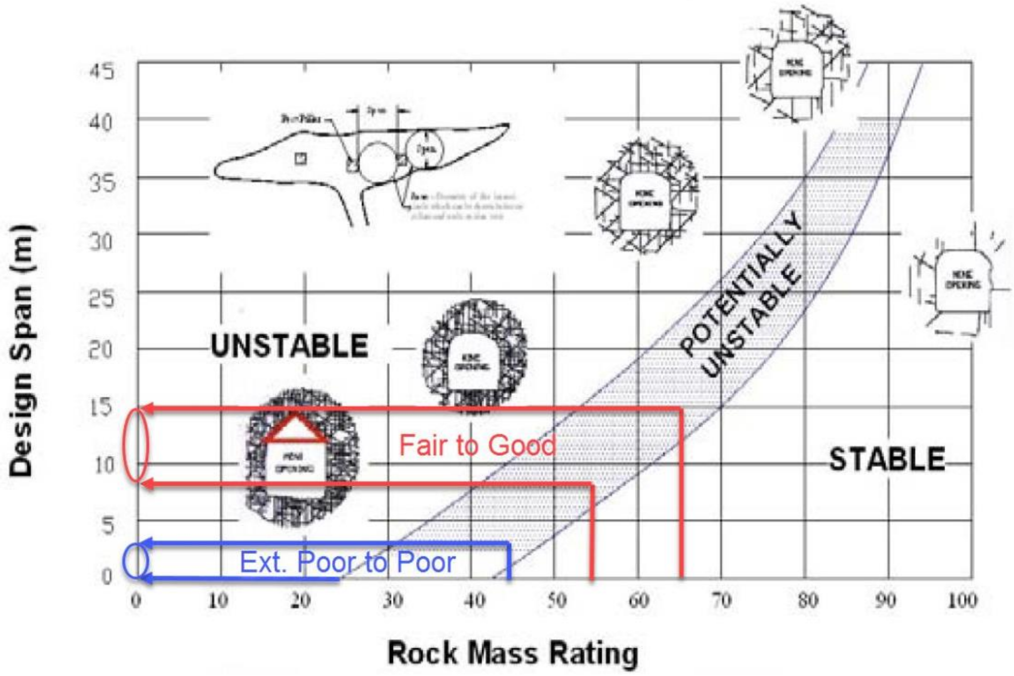


Figure 16-2: Design span recommendation from Ouchi critical span curve

16.4.2 Ground Support Requirements using Empirical Q Support Guideline

The ground support guideline for decline ramp with a dimension of 15 feet wide and 15 feet high in and production drift with a dimension of 12 feet to 29 feet wide, and 12 feet-high in different ground conditions are plotted in Figure 16-3. An Excavation Support Ratio ("ESR") value of 1.6 as a permanent mine opening is applied to the plot for the ramp and temporary mine opening ESR, 3.0, is applied for production drift ground support assessment. The value of ESR is intended use of the excavation and to the degree of security which is a demand of the support system installed to maintain the stability of the excavation for the planned stand-up time.

Table 16-5: The value of Excavation Support Ratio (ESR) by Barton et al. (1974)

Excavation Category		ESR
A	Temporary mine openings	3-5
B	Permanent mine openings, water tunnels for hydropower (excluding high-pressure penstocks), pilot tunnels, drifts. heading for excavation	1.6
C	Storage rooms, water treatment plants, minor road and railway tunnels, civil defence chambers, portal intersections	1.3
D	Power stations, major road and railway tunnels, civil defence chambers, Portal intersections	1.0
E	Underground nuclear power stations, railway stations, sports and public, facilities, factories	0.8

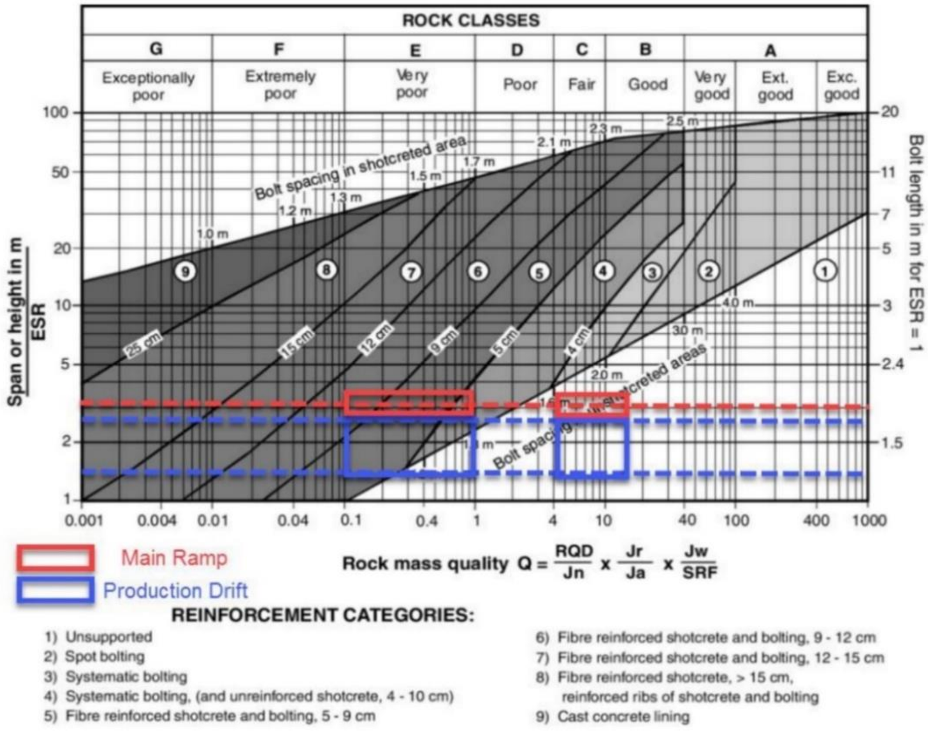


Figure 16-3: Estimated ground support requirements by Q classification (Barton et al., 1974)

Barton et al. (1974) also provided additional information on rock bolt length, maximum span of rock bolt. According to Barton et al., the length of rock bolts can be estimated from equivalent excavation width (B) and ESR value, and rock bolt spacing can be calculated using Q-value and ESR. Both empirical Q-support categories and rock bolt patterns for different ground conditions using empirical method are summarised in Table 16-6.

Table 16-6: The value of Excavation Support Ratio (ESR) by Barton et al. (1974)

Max. Width (B)	Q	Support Category	Rock bolt length (m) L = 2+0.15B/ESR	Rock bolt space (m) S = 2xESRxQ0.4
Ramp (15’)	15	(1)	2.4 (8’)	9.5
	5	(1)		6.0
	1	(4)		3.2
	0.1	(6)		1.3
Drift (12’)	15	(1)	2.1 (7’)	17.7
	5	(1)		11.4
	1	(1)		6.0
	0.1	(5)		2.4
Drift (29’)	15	(1)	2.4 (8’)	17.7
	5	(1)		11.4
	1	(4)		6.0
	0.1	(6)		2.4

16.4.3 Stand-up Time of Wide Ore Drift

According to feasibility, the selected mineable portion of the veins are mainly in the range of 6 feet to 29 feet wide. Stand-up time of unsupported span is one of the fundamental issues in mining with wide Drift. Stand-up time for unsupported 20 feet to 29 ft wide Drift in poor to fair ground was estimated as shown in Figure 6-3. The stand-up time graph indicated that if drift width is wider than 29 feet in fair ground, unsupported stand-up time will be longer than one week. However, if the drift width is wider than 20 feet in poor ground, stand-up time will be less than one day. In this case, short advancing blast or developing narrower Drift with retreat slashing need to be considered to maximise ground arching effect.

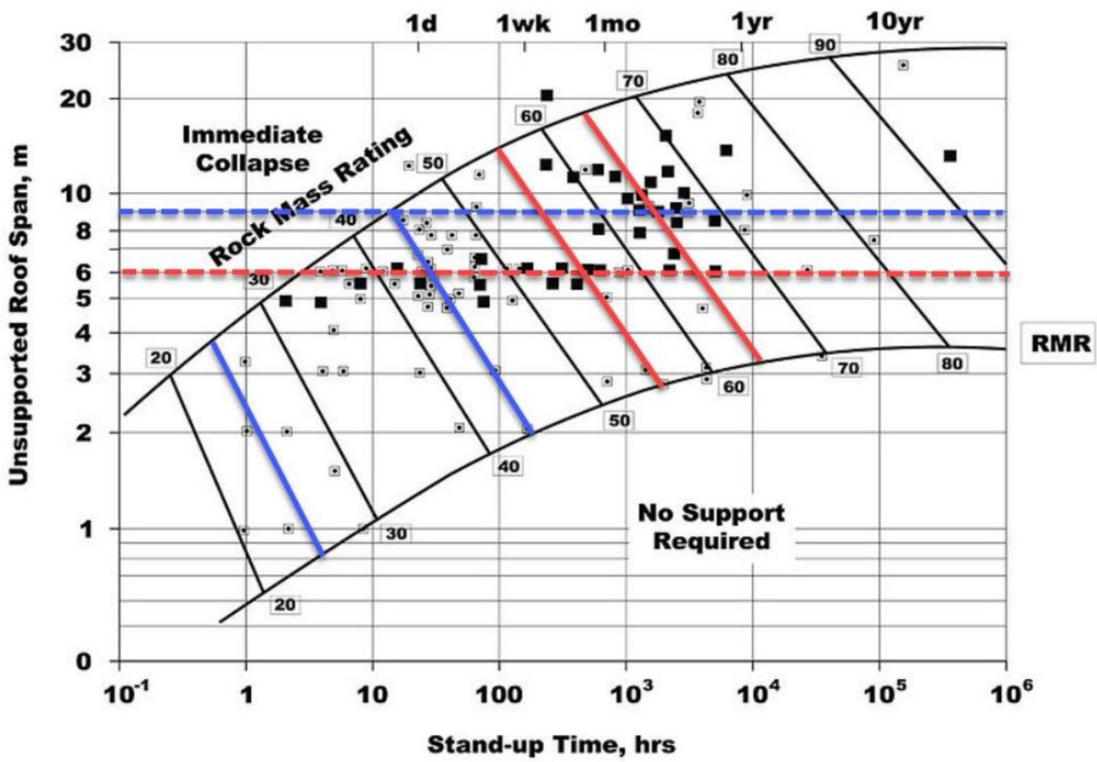


Figure 16-4: Stand-up time for 4 m and 7 m wide Drift in fair ground (Bieniawski, 1989)

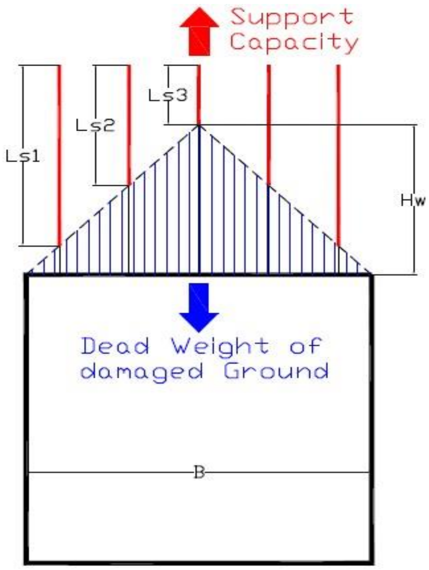
16.4.4 Ground Support Standards for Man Entry Openings

Main decline ramp and production drifts are developing through mainly Medium to Fine grained Quartzite ("MFQ") which is generally fair to good condition of ground, and some part of the production drift need to pass Fault zone. Systematic 8 feet long split set bolts or Swellex bolts with screen are required for the back and wall as a primary support elements for generating ground arching to prevent unconsolidated back and wall sloughing. Wide opening production Drift in fair to good ground may need 10 feet super Swellex bolts for holding possible wedge and openings in poor ground should spray shotcrete as a secondary ground support. Ground support standards for main decline ramp and production drifts are summarised in Table 16-7, and detailed support regimes for three different ground conditions are available in the 2020 Shin Technical Memorandum.

The Ground Support Standards form the basis for all ground support and are to be installed according to specification. In adverse geotechnical ground conditions (e.g. poor to good, presence of structural features, expected corrosion), the Ground Support Standards shall be reviewed, and additional support recommendations will be made by Geotechnical Engineer. Substituting Swellex bolt for split sets for production drift ground supports may permissible at the discretion of mining operations with approval by Mine Manager. The Ground Support Standards will be developed and updated as experience is gain upon development.

Table 16-7: Ground support standards for decline ramp

Type	Ground Condition		Ground Support Standard (Bolt space/Pattern)
Main Ramp (15'H x 15'W)			
R – FI	Fair to good (5 < Q < 15)	Back	8' Reg. Swellex (5' x 5')
		Wall	8' Reg. Swellex (5' x 5'), Mesh 5' from Back
R – P1	Poor to Fair (0.1 < Q < 5)	Back	8' Reg. Swellex (4' x 4'), Min. 2" SC as Req.
		Wall	8' Reg. Swellex (4' x 4'), Mesh 5' from Sill, Min. 2" SC as Req.
Main Drift (12'H x 12' – 15'W)			
D – FI	Fair to good (5 < Q < 15)	Back	8' Split Set (5' x 5')
		Wall	8' Split Set (5' x 5'), Mesh 5' from Back
D – PI	Poor to Fair (0.1 < Q < 5)	Back	8' Reg. Swellex (4' x 4'), Min. 2" SC as Req.
		Wall	8' Reg. Swellex (4' x 4'), Mesh 5' from Sill, Min. 2" SC as Req.
Wide Drift (12'H x 15' – 20'W)			
W – FI	Fair to good (5 < Q < 15)	Back	8' Split Set (5' x 5')
		Wall	8' Split Set (5' x 5'), Mesh 5' from Back
W – PI	Poor to Fair (0.1 < Q < 5)	Back	8' Reg. Swellex (4' x 4'), 10' Con. Super Swellex (8' x 8') as Req., Min. 2" SC as Req.
		Wall	8' Reg. Swellex (4' x 4'), Mesh 5' from Sill, Min. 2" SC as Req.
Wide Drift (12'H x 20' – 29'W)			
W – FII	Fair to good (5 < Q < 15)	Back	8' Split Set (5' x 5'), 10' Con. Super Swellex (8' x 8')
		Wall	8' Split Set (5' x 5'), Mesh 5' from Sill.
W – PII	Poor to Fair (0.1 < Q < 5)	Back	8' Reg. Swellex (4' x 4'), 10' Con. Super Swellex (8' x 8'), Min. 2" SC.
		Wall	8' Reg. Swellex (4' x 4'), Mesh 5' from Sill, Min. 2" SC.



Dead Weight
= 1/2 x grock mass (t/m3) x B x Hw x S

Support Capacity
= Bond strength x (2Ls1 + 2Ls2 + Ls3)

FoS = Support Capacity / Dead Weight

Required FoS
Ramp: FoS > 1.5
Temporary Drift: FoS > 1.3

B: Opening width
Hw: Depth of damaged zone
S: Bolt spacing

Figure 16-5: Factor of safety from Dead Weight analysis

16.4.5 Verification of Support Patterns by Dead Weight Analysis

The Factor of Safety for every support pattern associated with ground conditions for the main ramp and different width of production drifts were estimated using Dead Weight analysis. Outlines of Dead Weight analysis is illustrated in Figure 16-5. The factor of safety is the capacity of rock bolts installed at the back against the weight of damaged ground, which can be calculated by opening width and failure depth. 0.3B and 0.6B where B is opening width is assumed as a failure depth in fair and poor ground condition respectfully. The capacity of rock bolts should be estimated using the installed length beyond the damaged ground.

Factor of Safety ("FoS") with three different support patterns for 12 feet to 29 feet wide headings in poor to good conditions ground were estimated using Dead Weight analysis (Shin, 2020) and results of the analysis were summarised in Table 16-8. Minimum 1.3 of FoS is required for support standards. 8-foot-long regular Swellex bolts in 5 feet by 5 feet spacing is good for main ramp in fair to good ground as a permanent primary ground support. Secondary support with 10-foot super Swellex is required for wide production drift in poor to fair ground. Pre/post shotcrete may also require as an additional ground support in extremely poor fault zone because FoS with primary support with 10-foot super Swellex is not satisfied minimum 1.3 of FoS in this ground condition.

16.4.6 Quality Assurance/Control Plan for Ground Support

Quality assurance and quality control of ground support is to be carried out through ground support installation inspections, pull testing, shotcrete batch plant inspections, shotcrete strength testing (ASTM C1550/ASTM C1140), and long-term monitoring.

16.4.6.1 Rock Bolt QA/QC

The quality of ground support installation work shall be inspected by the Shift Boss during each shift. Operators will note any bolts that do not install correctly and report this to Shift Boss at the end of bolting the heading. The Geologist/Geotechnical Engineer shall also visually inspect the adequacy of bolting near the heading during mapping of each new round. Any non-conformances must be reported to the Shift Boss and to the Geotechnical Engineer. If the bolt(s) do not meet acceptable installation standards, new bolt(s) shall be installed within 1 foot of the original ground.

Routine pull testing will be implemented with a requirement that a minimum of 1% of the new rock bolt installed in the current year demonstrate a pull test capacity indicating 75% of the bolt's yield strength. Additional testing of ground support will be carried out after certain years of installation. At a minimum, 5% of all rock bolts in excavations after five years will be tested. If a systemic problem, such as excessive corrosion, is detected, a rehabilitation program including re-bolting will be initiated.

16.4.6.2 Shotcrete QA/QC

In the event that shotcrete is used to support or remediate poor ground, or for other Ground Control applications, quality assurance/control procedures will be implemented to verify adequate installation.

During spraying of the shotcrete, the Operator will note the ability of shotcrete to adhere to the excavation surface. Other problems such as balling fibre (if used) or poor mix quality shall be noted by Operator/Nozzleman. If the shotcrete mix is not suitable for spraying, then the batch is to be rejected. A record of the amount of additives used during spraying, total cubic feet sprayed, and operating times shall be kept by the Nozzleman. If the shotcrete breaks out during bolt installation, then the Operator will stop the bolt installation. Bolting must not recommence for a 30-minute duration, to give the shotcrete further time to cure.

Unconfined Compressive Strength ("UCS") samples shall be taken once for each heading where sprayed shotcrete by coring through shotcrete panels, or similar. Tests to determine representative UCS values at 1 day, 7 days, and 28 days shall be completed. Round Determinate Panel ("RDP") tests also shall be carried out, depending on the type of shotcrete used (e.g., normal vs fibre) and the intended service conditions. Energy absorbed by test panels shall be equal to or greater than 360 J.

16.4.7 Optimisation of Longhole Stope Dimension

16.4.7.1 Stress Change at the HW of Longhole Stope

To determine the proper dimension of longhole stopes and mining sequence it is requested that understand stress path change caused by development of longhole mine. Failure is a result of rock mass relaxation, and that is defined as a reduction in stress static parallel to wall excavation. Wedge failure occurs when the minor principal stress is below or equal to zero as shown in Figure 16-6 stress path A. The severity of sloughing (stress path B) also possible failure mode for the longhole stope, and the failure is related directly to the rock tensile strength. However, rock mass has a self-supporting capacity depending on the material properties and geological structures.

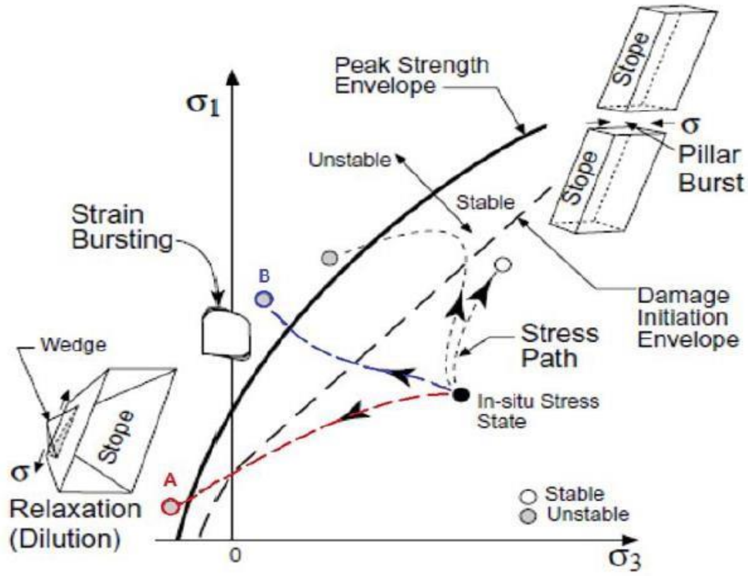


Figure 16-6: Possible stress path for a longhole stope (Martin et al., 1999)

16.4.7.2 Maximum Stope Strike Length

The widely used empirical tool for a maximum stope strike length is the stability graph method. The method is developed by Mathews et al. (1981) and defined by Potvin (1988). The stability graph method associates the stability number to the hydraulic radius of a stope. The graph helps to access the stability of an opening according to the stope hydraulic radius. The stability number (N) can be calculated by the following equation,

$$N' = Q' \times A \times B \times C$$

Where,
N: Stability number
Q': Modified NGI Q value with stress reduction factor
A: Stress factor – the ratio of intact rock strength to applied stress
B: Joint orientation factor – relative orientation of dominant structure with respect to the excavation surface
C: Gravity factor – the influence of gravity on the stability of the face being considered.
Q' values of 10, 5, and 0.5 were used in combination with A, B, and C inputs to calculate permissible stope strike length for 24 feet and 48 feet high HW with 55-degree dip (Table 16-9). The stope strike length for 10 feet, 20 feet and 30 feet wide stope back in three different category of ground conditions (fair to good, poor to fair, poor) were also recommended in Table 16-10. Input parameters for Stability Graph are summarised in Table 16-8.

Recommendation of minimum sill pillar between two drifts/stopes should be taken bigger thickness that is 33 feet or two times of wider opening's width. Minimum sill pillar between 12 feet wide openings in fair to good ground condition may be reduced by 24 feet, but horizontal overlap between two levels is not allowable. Specific precautions for horizontal overlap of sill levels will be determined based on mining conditions and specific sill pillar optimisation in the future.

Table 16-8: Stability Graph assumption for stope designs

Parameter	Value	Design Assumption
Q'	10 5 0.5	Fair to good ground Poor to Fair ground Extremely Poor to Poor ground
A	0.5 (wall) 0.1 (back)	Assume induced stresses concentrate above and adjacent to back. Walls are generally destressed
B	0.2 (wall) 0.2 (back)	Conservative assumption based on structural variability in all domain
C	4.0 (wall) 2.0 (back)	Defined based on critical discontinuity set assuming horizontal structure in back and structure parallel to wall

Table 16-9: Maximum Stope Strike Length Recommendation in different height of walls

Ground Condition	Stability Number (N')	Hydraulic Radius	Max. Stope Strike Length, m (')	
			H = 7.3 m (24')	H = 14.6 m (48')
Good (Q = 10)	4	6	> 100 m (300')	< 67 m (220')
Fair (Q = 5)	2	4	> 100 m (300')	< 18 m (60')
Poor (Q = 0.5)	0.2	2	< 9.2 m (30')	< 5.5 m (18')

Table 16-10: Maximum Stope Strike Length Recommendation in different width of backs

Ground Condition	Stability Number (N')	Hydraulic Radius	Max. Stope Strike Length, m (')		
			B = 3.0 m (10')	B = 6.0 m (20')	B = 9.0 m (30')
Good (Q = 10)	4	6	> 100 m (300')	> 100 m (300')	> 100 m (300')
Fair (Q = 5)	2	4	> 100 m (300')	> 100 m (300')	< 72 m (235')
Poor (Q = 0.5)	0.2	2	> 100 m (300')	< 12 m (40')	< 7 m (23')

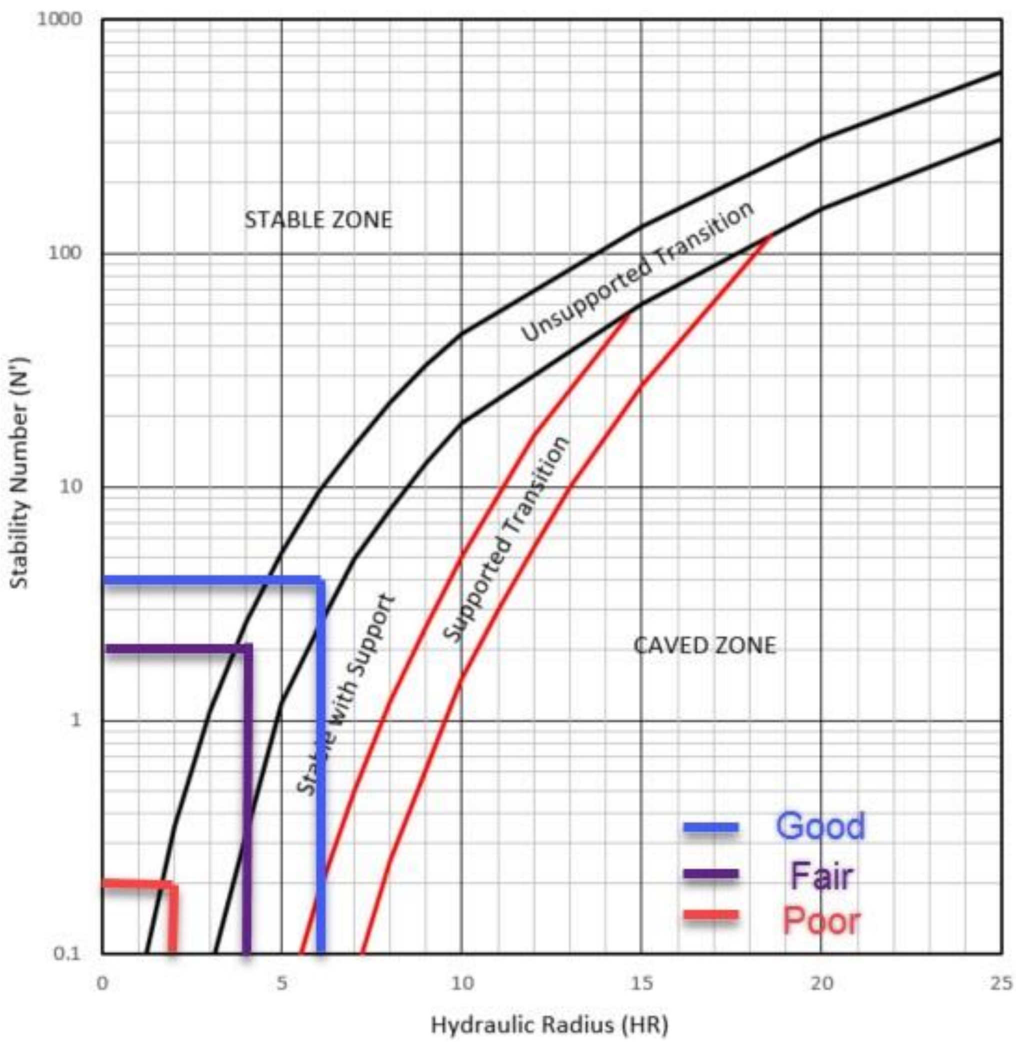


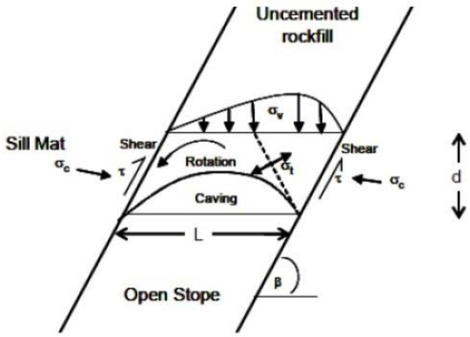
Figure 16-7: Stope Stability Plot for unsupported stopes (Potvin, 1988)

16.4.8 Backfill Strength for Underhand-Cut Stability

16.4.8.1 Failure Modes of Paste Fill and Applied Properties for Underhand-Cut Design

The methodology of span design under paste fill is complex because many different factors affect the overall stability, as shown in Figure 16-8. The failure modes and combination thereof should be analysed with respect to the cement paste properties, stope geometry, and other factors related to filling practice, such as cold joints and gaps above not tightly filled openings.

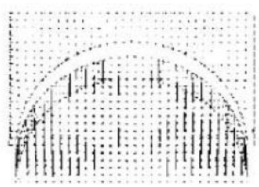
For the underhand-cut design, FoS against four different types of failure mode, were estimated from limit equilibrium analysis summarised by Mitchell (1991) as illustrated in Figure 16-9.



L : Span of the underhand-cut stope
γ : Unit weight of paste fill
σ_t : Tensile strength of the cement fill
d : Thickness of paste sill
σ_c : Horizontal confinement (assumed zero – conservative)
σ_v : Vertical stress above paste sill (uncemented rockfill)
τ : Shear strength along fill and wall contact
β : Stope wall dip angle

Figure 16-8: Schematic showing typical failure modes after Mitchell (1991)

Caving Failure



$$FS_{caving} = [\,(8 \times \sigma_t) / \pi\,] / [\,L \cdot \gamma\,]$$

Flexural Failure

$$FS_{flexural} = [2 \cdot (\sigma_t + \sigma_c) / (\sigma_v + d \cdot \gamma)] / [(L/d)^2]$$

Sliding Failure

$$FS_{sliding} = [2 \cdot (\tau / \sin^2\beta) \cdot (d/L)] / [\sigma_v + d \cdot \gamma]$$

Rotational Failure

$$FS_{rotation} = [(d^2 \cdot \sigma_t)/(L \cdot (L - d \cdot (\cot\beta \cdot \sin^2\beta)))] / [\sigma_v + d \cdot \gamma]$$

Figure 16-9: Limit equilibrium analysis of typical failure modes by Mitchell (1991)

16.4.8.2 Required UCS against Caving Failure

Caving failure would occur when the unsupported weight of backfilled sill material exceeds the tensile strength of the material. The caving is assumed to extend to a semi-circular arch shape defined by L/2 where L is the under-cut span. This failure is assumed to be related only to the self-weight of the material, independent of external loadings. Other than the sill drive geometry, the assumed tensile strength of the material is the critical factor to consider in this analysis. If tensile strength of backfill material is assumed 10% of compressive strength, required design strength ("UCS) can be calculated as follows:

$$UCS_{Caving} = (1.25 \cdot L \cdot \gamma \cdot \pi) \cdot FoS$$

Design strengths of backfill material, according to FoS, were summarised in Table 16-11. According to the analysis of caving failure, UCS of backfill material with 10 feet, 20 feet and 30 feet spans need to minimum 450 KPa, 890 KPa and 1300 KPa for FoS = 1.5.

16.4.9 Required UCS against Flexural Failure

Flexural failure would occur when the moments due to the bending of the sill mat under its self-weight, plus the vertical stresses applied to the sill paste exceed the moment capacity of the sill material. Following this analysis, the tensile strength of the material and thickness of the sill paste would provide the main resistance to flexural instability.

$$UCS_{Flexural} = \frac{(L/d)^2 \cdot (\sigma_y + d \cdot \gamma)}{2.2} \cdot FoS$$

Design strengths of backfill material, according to FoS, were summarised in Table 16-11. According to the analysis of flexural failure, backfill material with 10 feet, 20 feet and 30 feet spans need to minimum 60 KPa, 370 KPa and 830 KPa for FoS = 1.5.

16.4.10 Required UCS against Sliding Failure

Sliding or shear failure along the sill mat abutments would occur when the weight of the backfill material, in combination with the vertical loads emplaced on the sill mat, exceed the shear strength of the paste material. For the assessment of UCS against sliding, shear strength (τ) is defined by initial failure strength of unconfined compressive strength test. In general, initial failure occurs from 25% of the peak strength (σuc).

$$UCS_{Sliding} = \frac{(\sigma_y + d \cdot \gamma) \cdot sin^2\beta \cdot (L/d)}{0.5} \cdot FS$$

Design strengths of backfill material, according to FoS, were summarised in Table 16-10. According to the analysis of sliding failure, backfill material with 10 feet, 20 feet and 30 feet spans need to minimum 270 KPa, 540 KPa and 810 KPa for FoS = 1.5.

16.4.11 Required UCS against Rotational Failure

Rotational failure strongly depends on backfill thickness, d, as shown in below:

$$UCS_{Rotational} = \frac{(\sigma_y + d \cdot \gamma) \cdot L \cdot (L - d \cdot cot\beta \cdot sin^2\beta)}{d^2} \cdot FS$$

Design strengths of backfill material, according to FoS, were summarised in Table 16-10. According to the analysis of rotational failure, backfill material with 10 feet, 20 feet and 30 feet spans need to minimum 93 KPa, 370 KPa and 835 KPa for FoS = 1.5.

Table 16-11: Required UCS against different failure modes

Stope Span		10’	15’	20’	25’	30’
Required UCS (KPa)	Caving Failure	**450**	**670**	**890**	**1100**	**1300**
	Flexural Failure	60	240	370	610	830
	Sliding Failure	270	400	540	680	810
	Rotational Failure	95	225	370	615	835

16.5 RISKS

A relatively conservative design approach was applied for tunnel support including the use of inflatable friction bolts for all ground classified as "poor" even when split sets may be sufficient for such spans. Furthermore, a pull test program has been recommended to ensure ground support elements perform in accordance with design assumptions. In addition, quality control and assurance measured are prescribed for shotcrete application.

Nevertheless, it is possible the ground conditions encountered during mining differ from design assumptions in this report. As a result, ongoing joint mapping and assessment of ground conditions is recommended, in particular during the first few months of mining in concert with pull testing. Further observation and assessment is particularly recommended during stoping as well as exposing backfill material.

It is anticipated that if undesirable geotechnical conditions are encountered, revised ground support and sequencing methods may be required. If extremely poor-quality ground is encountered locally, that region may need to be extracted using cut and fill rather than stoping excavation. Similarly, if extremely poor ground is encountered in planned wide excavations, narrower excavations with "side-drifts" may need to be substituted.

While there may be requirements to alter the local ground support or mining method, it is not expected that ground conditions poor enough to preclude excavation would be required. Rather, varying geotechnical parameters may require local economic re-assessment to determine if increased ground support costs are warranted by the mineral value of that zone.

16.6 MINING CUT-OFF GRADE AND SPECIFICATIONS

The mining sill and backstope stope shapes are generated based on a CoG of 0.30% and 0.32% cobalt, respectively, which takes into account recoveries and cost estimates values for a resulting ore value of not less than 0.24% Co equivalent on a recovered and payable basis

Stope outlines were generated at 12 ft vertical interval, transformed into solid and sectioned by 12 ft H by 24 ft L generating individual mining stopes. Details on the CoG criteria is presented in Section 15.

16.7 SELECTIVITY, DILUTION AND RECOVERY

16.7.1 Mining Selectivity

Stope shapes were developed from the resource model blocks oriented on a best-fit basis to the orebody strike and dip to minimise grade smearing. The stope shapes cut the resource at mining geometries relevant for equipment selection. The Deswik stope optimiser generated shapes based on a minimum width criteria and capturing all ore grade material until the total stope block is at the reserve cut off grade limit. Variable minimum mining widths based on sill drift or back stope profiles of a minimum width of 15 ft and 6 ft respectively provide access to the mineralised resource while maintaining value sufficient for the economic recovery of the ore body.

The stope widths, based on the resource model, will be further refined during operations through infill drilling as well as face and back mapping with chip samples during operations. Operational budget has been provided for beat geology to map and sample every sill round as well as continuous infill drilling by a crew of 2 during the life of mine operations. Back stope widths and extraction plans will be informed by sill production drifting with the mapping of the 24 ft vertical back stope from above and below in addition to infill drilling to minimise dilution and capitalise on wider than expected zones.

These proposed mining methods will provide the mine with the flexibility necessary for economic operations.

16.7.2 Dilution

Planned dilution accounts for all the material which is contained within blocks having centroids that lie within the design stope boundaries and anticipated 0.5 ft overbreak, which are determined by the selectivity of the mining methods and the continuity of the orebody. The total value of the planned dilution is approximately 16%. This value was estimated from a comparison of the undiluted grades with diluted block grades for those blocks lying within the stope boundaries.

Unplanned dilution, however, arises primarily due to imprecision of the mining operation. The sources of unplanned dilution include waste rock extracted from the walls of the cut, the percentage depending on the variance on strike, dip, and width of the ore from the model. This was assigned a total of 5.0% additional dilution.

The average total of planned and unplanned dilution for the ICO project is approximately 22%.

16.7.3 Mining Recovery

The mining recovery was estimated based on the difficulty of mining, loading or recovery of the blasted material from the mining stopes. Losses can occur as material left in the floor of stopes during mucking or which was retained in back stopes due to the relatively low dip angle of the deposit. Loss mitigation was planned into the operation by utilising a top fill paste method and creation of a paste cap on fill to provide efficient recovery of ore in overhand mining. In addition, the relatively short vertical length of the backstops combined with the conservative extraction rates will allow blasting to be planned so ore clears the back stope and is cast into the sill below for optimum recovery. The average estimated recoveries from mining operation are 95%.

16.8 MINE DESIGN

The mine design was developed to support a mine production rate of 1,200 st/d feed to a milling operation for the proposed mining methods. High-grade stopes were given priority during the mine production scheduling. The Central ramp system was prioritised as it contains over half of the total tons of ore and highest net value ore. The South Ramp system was assigned a second priority based on equivalent recovered and payable value due to its high copper and gold value and the North ramp zone lowest priority. A small pod of above reserve cut off grade resource exists further to the north but was not included in the reserve mine plan in consideration of incremental development costs to access.

16.8.1 Underground Excavation Dimensions

The main underground development was designed to a cross-section area of 15 ft H x 15 ft W to allow 30T haulage fleet to operate with minimal impact on ventilation safely. The stope access and vent raise connections drifts are 14 ft H x 14 ft W. Main ventilation raises will be excavated by drop raise methods (drilling and blasting) a 9x9 profile. Raises will be fitted with ladder ways and landings to comply with escapeway requirements.

The current mine development design represents a 42% reduction of waste development compared to previous published feasibility level designs by use of direct stope access from main haulage ramps rather than extensive lateral development. Table 16-12 summarises the estimated Life of Mine ("LoM") development length and the proposed excavation dimensions.

Table 16-12: Estimated Life of Mine (LoM) development length and the proposed excavation dimensions

Estimated Development Footage	Dimension (H ft x W ft)	Total LoM (ft)
Ramps including Decline, Re-muck & Safety Bays	15 x 15	20,872
Access Drive	14 x 14	19,815
Ventilation Drift	14 x 14	2,282
Ventilation Raise	9 x 9	2,368
Sub-Total Hor. Dev.		42,969
Sub-Total Vert. Dev.		2,368
Total		45,337

16.8.2 Mine Access

The main decline and a system of ramps provide access to the underground workings and production areas. There are two portals into the mine: one as the main decline giving access into the mine production heading, and the other acting as the ventilation tunnel.

The service tunnel provides access to main underground services and storage areas such as the mechanics shops, ventilation intake raise, main sump, and explosive. The service areas are located in lateral drifts connecting the main decline to the service tunnel. Ventilation and fire doors or bulkheads are placed to prevent short-circuiting of the ventilation system and for fire control. Both portals are located at approximately 7,080 ft elevation.

The decline is designed at a maximum -15% grade with reduced gradients between levels for access and ventilation drift intersections. Muck bays of 50 ft L are located at approximately every 400 ft along the ramp system. These bays will be converted into safety bays, drill bays, staging sumps or vehicle passing bays during operation. Figure 16-10 shows the mine development layout.

16.8.3 Underground Mine Layout

Currently, all the underground development and access into the stope is located in the hanging wall, enabling a better location for additional exploration drilling to be carried out and facilitating definition drilling.

Access into the production stopes will be from hanging wall and will be through two access drives connecting the decline or the ramp to the ore zone every 70 ft sublevel., Figure 16-11 shows the stope layout.

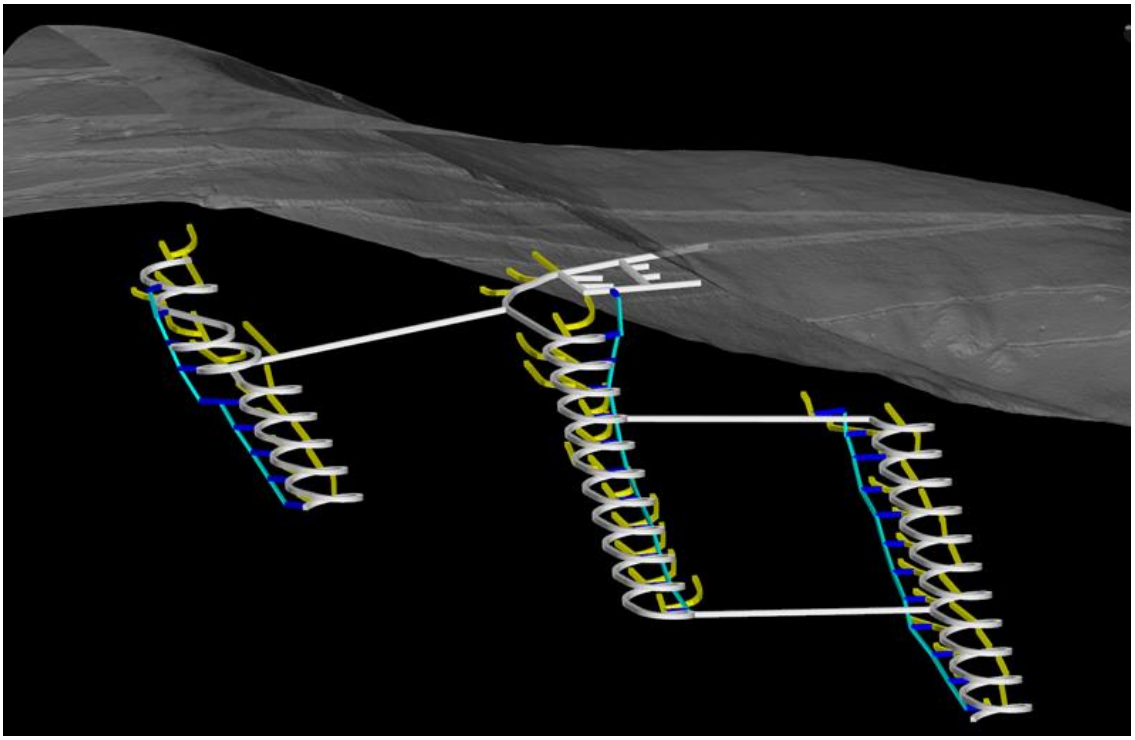


Figure 16-10: ICO Mine Development Layout

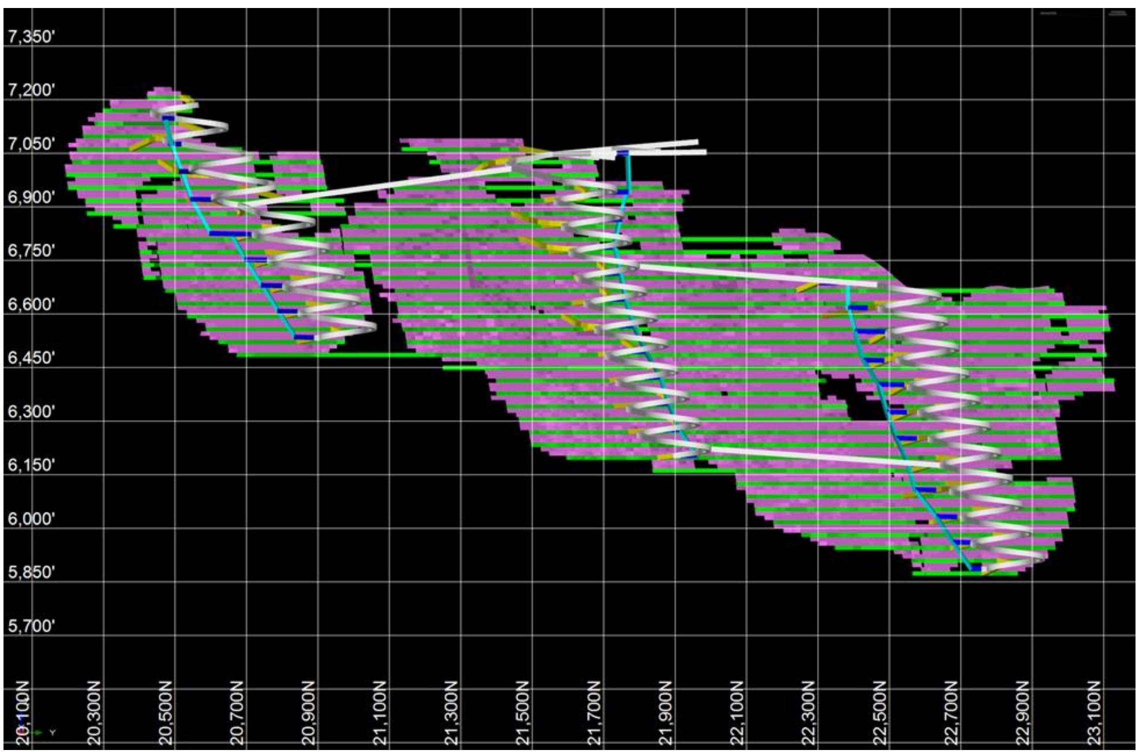


Figure 16-11: ICO Stope Layout

16.9 MINE DEVELOPMENT AND PRODUCTION SCHEDULE

Mine development and production schedules were generated in Deswik utilising resource limitations identified in Table 16-13 for an overall rate of ore production matched to the nominal mill capacity of 1,200 st/d.

Table 16-13: Mine schedule resource constraints

Activity Task	Rate
Ramp/Acc Max (ft/d)	14
Sill Max (ft³/d)	2,100
BS Max (ft³/d)	5,000
URF Fill (ft³/d)	7,500
Raise Vrt (ft/d)	6
Resource	
Max Lateral (ft/d)	70
Max Ore Hauling (ft³/d)	20,000

Table 16-14: Deswik Schedule Time Variant Ore Ton Target Limits

Every	Period	For	Limit
1	Month	7	0
1	Month	2	1000
1	Month	2	11000
1	Month	1	18500
1	Month	3	25000
1	Month	80	36500

In the project cash flow model, the variance between mining and milling is reflected as a notional mill-feed stockpile.

The mine development and production from pre-production through operations is based on a proposal by Small Mine Development ("SMD"), an underground mining contractor. The mining sequence commences with the extraction of sill drifts from the bottom of the sequence. The first sill acts as a sill pillar where a sill mat of high strength paste fill will be constructed and placed. The second sill provides the working area to mine the back stope below the initial sill of the sublevel above. All fill above the first sill will be a combination of waste rock from development and paste fill for elimination of voids and to provide a net neutralising potential for groundwater protection.

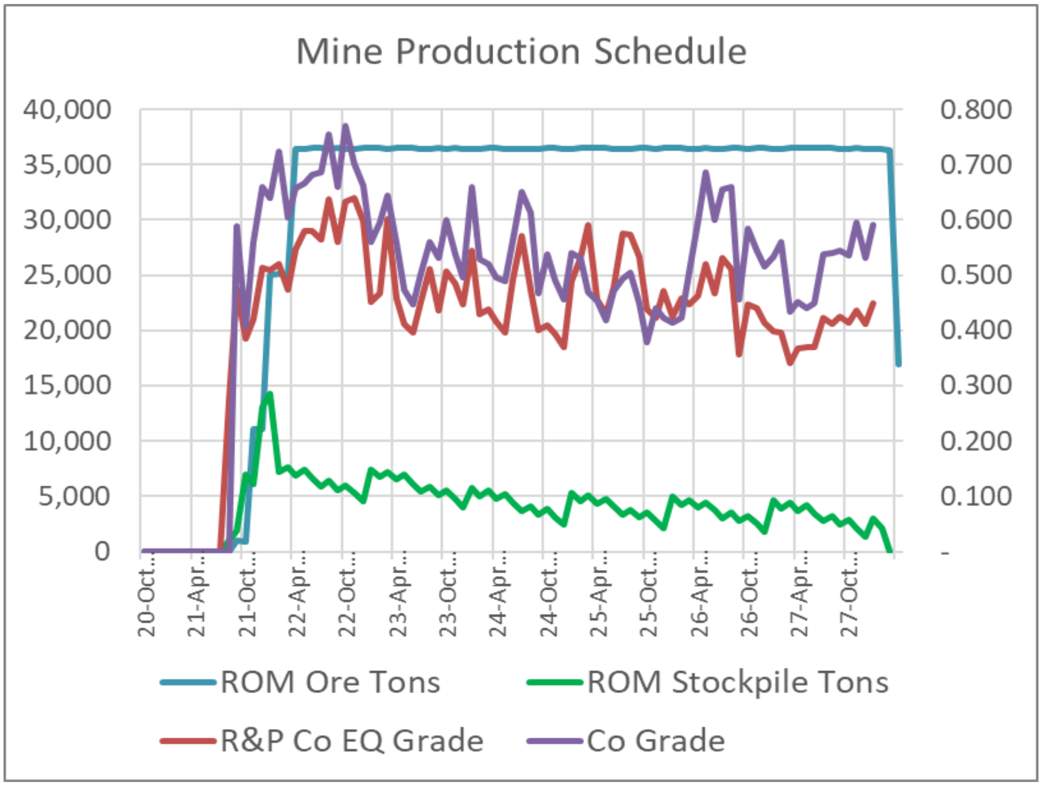


Figure 16-12: Mine Production and Stockpile Schedule by Month

16.10 MINE DEVELOPMENT

The mine development commences during the pre-production Y -1, focusing on the excavation of the primary access to the central ramp system. Ramp sub-levels at nominally 70 ft vertical intervals will be connected to the intake portal vial a series of raises excavated by drop raise drill and blast methods between sublevels and fitted with prefabricated escapeway ladder and landing systems. The mine ventilation raises will serve as secondary escape-ways.

The advance rate for single development heading with mechanised mining equipment is approximately 12.0 ft/round.

Mucking bays will be situated at approximately 400 ft along the main ramp drives. These bays will be utilised as safety bays, drill bays, vehicle passing areas, sumps and storage areas during operations. The mine development and layout includes a dewatering and backfill sump at each stope access.

Access to the stopes from the main ramp is nominally 330 ft in length at -3% for the first sill and +10% for the second sill in each sublevel. Stopes will have minor downward grade towards the access facilitating drainage out of the stoping areas.

The LoM mine development footages and tonnage summaries are presented in Table 16-15.

Over the life of the mine, underground development generates approximately 1,051,000 st of waste of which about 747,000 st will be used as backfill to reduce transportation and paste fill costs. The remaining 304,000 st of waste material will be transported to the tailings and waste management facility.

16.10.1 Production Schedule

Mining will commence with an initial extraction of sills in high-grade zones of the Central Ramp, followed by the mining of back stopes in a bottom-up sequence.

A sill mat of strength paste fill placed into the lead sills enable higher recovery of the mineralised horizons and safer working area during the extraction of stopes beneath the initial sills. An allowance of 28 days backfill curing days for mining beneath sills is planned.

Mechanised mining methods will be applied in areas with widths ranging from 15-29 ft. Longitudinal back stoping will be used to extract stope blocks above the sills, drilling and mucking from the sill below at minimum 6 ft widths.

The stopes are designed at 70 ft H by 500 ft L sublevels each side of the ramp access. Sills will be mined by horizontal lifts of 12 ft H, with an advance rate of 12 ft per round. Two sills will be driven on each sub-level. A backstope lift of 24 ft will be taken above each sill. Each set of sill and back stope will be filled as a single fill cell, two per sub-level. Back stope drilling, production blasting and mucking are carried out from the sills.

Mine ore production starts in Year 1 with 2,000 st in the 2nd quarter, 40,000 st in the 3rd quarter and 86,000 st in the 4th quarter. The mine ramps up to a full production rate of 1,200 st/d (437,500 st/yr) in the 1st quarter of Year 2 to the end of mine life. Table 16-15 presents the mine production schedule Figure 16-13 presents a schematic of the mine and stope layout.

The number of active headings and stopes vary depending on the mining horizons, widths of the horizons and mining methods. A minimum of 8 active production faces is required to meet the daily production target.

Table 16-15: Mining Development & Production Schedule

	LOM	2021	2022	2023	2024	2025	2026	2027	2028
Ore ST	2,741,520	-	129,252	439,293	438,539	439,317	438,633	438,400	418,085
Co lbs	30,133,351	-	1,597,119	5,836,025	4,745,583	4,545,920	4,039,378	4,860,635	4,508,691
Co grade	0.55%	-	0.62%	0.66%	0.54%	0.52%	0.46%	0.55%	0.54%
Cu lbs	43,600,305	-	1,368,080	9,114,625	7,737,686	7,689,779	10,747,609	3,828,843	3,113,684
Cu grade	0.80%		0.53%	1.04%	0.88%	0.88%	1.23%	0.44%	0.37%
Au oz	51,418	-	2,512	9,516	7,022	8,645	12,079	6,053	5,591
Au grade oz/st	0.0188	-	0.0194	0.0217	0.0160	0.0197	0.0275	0.0138	0.0134
Paste Placed ST	1,145,510	-	16,353	93,218	123,020	223,995	245,456	199,169	244,300
Dev Feet	42,969	3,701	10,543	15,231	9,479	4,015	-	-	-
% Dev Ft		9%	25%	35%	22%	9%	0%	0%	0%
Tails to TWSF	1,917,138	-	112,928	360,169	343,428	281,703	268,827	296,599	253,484
Waste to TWSF	218,789	59,083	215,022	152,153	91,328	(67,882)	(63,674)	(55,898)	(111,343)
Cum total TWSF	10,110,316	59,083	387,034	899,356	1,334,112	1,547,933	1,753,085	1,993,786	2,135,927
Total Ft Dev Drift 15x15	20,873	2,103	5,492	7,013	4,608	1,656	-	-	-
Total Ft Dev Drift 14x14	21,853	775	5,245	7,752	4,687	2,337	413	388	255
Total Ft Dev Raise	2,368	-	644	765	628	331	-	-	-
Total Raises	34	-	9	11	9	5	-	-	-
Total Sill Tons Including non	1,131,294	-	105,882	170,830	196,325	131,977	191,779	212,308	122,193
Total B/S Tons	1,702,041	-	31,827	263,619	259,175	324,212	257,321	245,889	319,997
Waste Haul to/from TWSF	816,384	59,083	215,022	152,153	91,328	67,882	63,674	55,898	111,343
Ore Haul to Mill	2,752,746	-	131,177	440,431	439,957	442,236	439,927	439,865	419,153
Paste Backfill Tons Tails	1,145,510	-	16,353	93,218	123,020	223,995	245,456	199,169	244,300
Cement Use Paste (tons)	45,820	-	654	3,729	4,921	8,960	9,818	7,967	9,772
Total Tons Mined	3,792,311	59,083	376,326	767,084	670,608	546,959	457,654	466,229	448,368

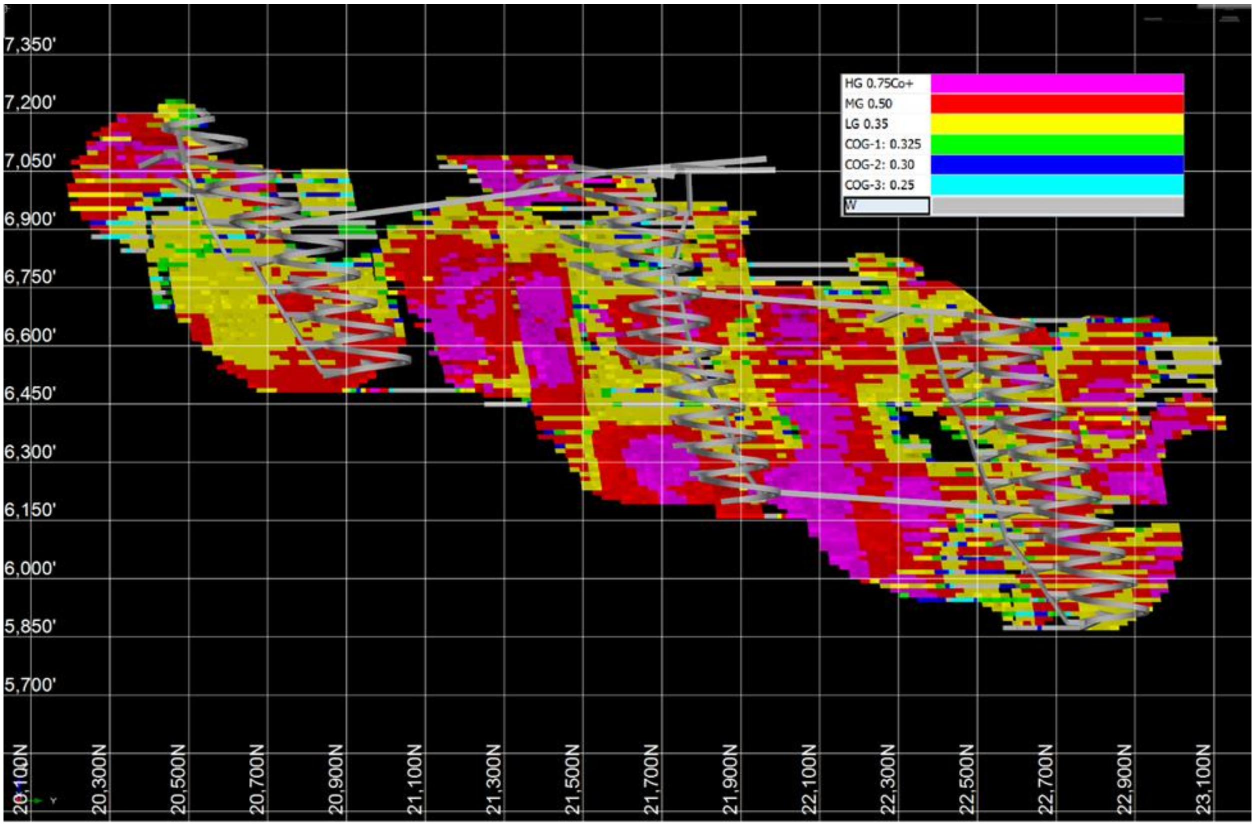


Figure 16-13: Mine and Stope Layout with Co Grades

16.11 MANPOWER REQUIREMENTS

SMD supplied the workforce and mine labour requirement during the life of mine development and production. ICO will also have limited mine staff during this period working with and supervising the work carried out by the contractor. Specifically, ICO will staff a Mine Manager, Tech Services Manager who will rotate to provide contractor oversight as well as Geology staff for updating of the resource model with detailed mapping, assay and infill drilling information. The workforce requirements were estimated based on productivities, capacities and availabilities of the equipment. The mine staff and labour for ICO is listed in Table 16-16.

Table 16-16: Mine Staffing Plan

SMD Personnel – ICO Project	
SMD Supervision	
Superintendent	1
Assistant Superintendent	2
Safety Superintendent	2
Maintenance Superintendent	1
Electrical Lead	1
Total Supervision	7
Rotating Crew Manpower	
Shifter	1
Miners	2
Operators - Mucker / Truck	8
Long Hole Drill	1
Shift Mechanic	1
Luber/Nipper	1
Electrician	1
Backfill	1
Crew Nipper	1
Total Crew Manpower	17
Total Rotating Crew Manpower (4 Crews)	68
Day Shift (7 days)	
Yard Man Nipper	1
Mine Surveyor (Optional)	1
Mine Engineer (Optional)	1
Total Day Shift Crew	6
Total SMD Manpower	81

16.12 EQUIPMENT SELECTION

The mining contractor will supply all the required mining equipment and workforce during the mine development and production operations. The mining equipment proposed by SMD is summarised in Table 16-17.

Table 16-17: Mining Equipment List

Equipment List	Quantity
Emulsion Truck with Boom	1
Two Boom Jumbo	2
Mechanized Bolter with Screen Handler	3
LHD, 4 cubic yard / with remote	4
U/G Articulated Truck, 30 tonne with ejector bodies	6
Shotcrete Spray Truck	1
Shotcrete Remix Truck	1
U/G Fuel/Lube Truck	1
Dozer (105 hp class) with jammer	1
U/G Grader	1
U/G Water Truck	1
All Terrain Forklift	4
Scissor Lift	1
Underground Crew Transport	6
Pneumatic Jackleg Drill	4
Self-Propelled Diesel Powered Manlift	1
Surface Front End Loader, 4 cubic yard	1
Air Compressor	1
Auxiliary Ventilation Fans	12
Face Dewatering Pumps	12

16.13 UTILITIES, SERVICES FOR UNDERGROUND

16.13.1 Temporary Mine Area Building

SMD will set up a temporary surface structure on the portal area during the pre-production stage of the mine development to provide equipment maintenance and service until the underground mechanic shop and infrastructure are completed. Office space will be provided in the miners dry building adjacent to the site administration building.

16.13.2 Explosive Storage

The main explosive storage is located on the surface with temporary underground storage located in between the connecting drift between the main decline and service tunnel storing no more than 24 hrs consumption and so a day box in regulatory standards. The explosives vendor will deliver to and supply the surface storage magazines. The primary blasting agent will be ANFO based emulsion in totes delivered by a bulk transport that refills totes on site.

Blasting supplies and explosives will be transported daily from surface to the underground storage facility and distributed to the underground development and production area by certified individuals.

16.13.3 Underground Communication System

The communication system will be via telephone and leaky feeder radio systems. The leaky feeder system will also be used to control the fans and pumps and blast initiation as well as provide gas monitoring. All mobile mining equipment are equipped with two-way radio systems.

16.14 VENTILATION

The East Portal via the Central Fresh Air Raise will mainly supply fresh air into the development and production areas. Ram deposit ventilation network will be composed of a series of ventilation drifts connecting underground development to the ventilation shafts and raises. Ventilation along lateral development and in production areas will be supplied and controlled by a combination of regulators, ducting and auxiliary fans.

There are three main ventilation shafts into the Ram deposit: the main Central Zone connected to the East Portal Access, and two secondary shafts situated in the South and North Zones. The baseline study envisions no vertical shafts directly breaking to surface. An alternate design, Stage II Raise Option, consists of the South Ventilation Shaft breaking to surface at 7285 elevation.

The ventilation shaft in Stage II Raise Option may be either a 10' raise bore. Ventilation raises in between levels will be excavated with the conventional drilling and blasting with minimum dimensions 12' x 12'. All the mine ventilation shafts and raises at the Ram deposit also serve as secondary emergency escape ways.

The ventilation plan consists of three stages:

- Stage I: mining activities focused on the Central Zone
- Stage II: mining activities focused on the South Zone with continuing development and mining in the Central Zone
- Stage II: mining activities mainly limited to the North Zone, no activity in the South Zone

For initial development, air will be supplied by 2x Howden 4800-VAX-3150 150hp portal fans pushing fresh air down low resistance (K~14 x 10-10) G+ plastic vent tubing running down the main, west, portal entrance ramp. Once the secondary East Portal and Central Raise System are established in Stage I, auxiliary fans on each level will pull a total of ~140 kcfm air down the East Portal and raise system. Negative pressure is not expected to exceed -0.3" w.g. No primary surface fan will be required. Return air will exhaust up the main ramp and out the West Portal. Stage I can be seen in Figure 16-14.

During Stage II, the previously utilised 2x Howden 4800-VAX-3150 fans will be reused to supply ~115 kcfm fresh air from the West Portal through G+ ducting to the South raise system. From there, auxiliary fans on each level will supply fresh air to each level in a push-pull system with raise relative pressure close to zero. The Central Raise System will continue supplying ~50 kcfm to the central zone to support some ore mining as well as development ramp drives to the North Zone. Both options can be seen in Figure 16-15.

In an alternate scenario, the auxiliary fans in Stage II will draw air directly from the South Raise breakthrough to surface. In this case, the portal fans will not be required, and the South ventilation system will function in a purely pull fashion, powered by the level fans. In either case, return air will exhaust out the connection drift through the West Portal.

Once mining activities switch to primarily ore extraction from the North Zone, ventilation to the South shall be ceased and the access ramp barricaded. Fresh air will continue to be supplied via the Central Raise system and delivered to the North System via the Lower Connection. From there, fresh air shall be directed to the North Raise System and supplied by auxiliary fans to each level as required. Total airflow at this point is expected to be 140-145kcfm as continuing ramp development activities will not be needed. Figure 16-16 shows the schematic during Stage III Mining. Approximate mining requirements assuming NIOSH Tier III engine specifications are:

- 25 kcfm for mucking
- 15 kfm for bolting/drilling
- 5 kcfm for man-entry

Power and annual cost requirements are shown in Table 16-18 below, assuming US$0.055/kWh:

Table 16-18: Ventilation Summary by Stage

	Stage I	Stage II RSE to Surface	Stage II Portal Fans	Stage III
Total Fresh Air (kCFM)	139	164	167	143
Total HP	275	234	474	264
Max Primary Loop Pressure (relative in w.g.)	1.8	3.3	2.1	0.3
Min Primary Loop Pressure (relative in w.g.)	-0.3	-0.2	-0.3	-0.6
Annual Cost @ 0.05 US$/kWh	$ 98,542	$ 84,031	$ 170,518	$ 91,737
Clear-out time min. (Co<20 ppm @West Exhaust Portal)	50	50	65	45

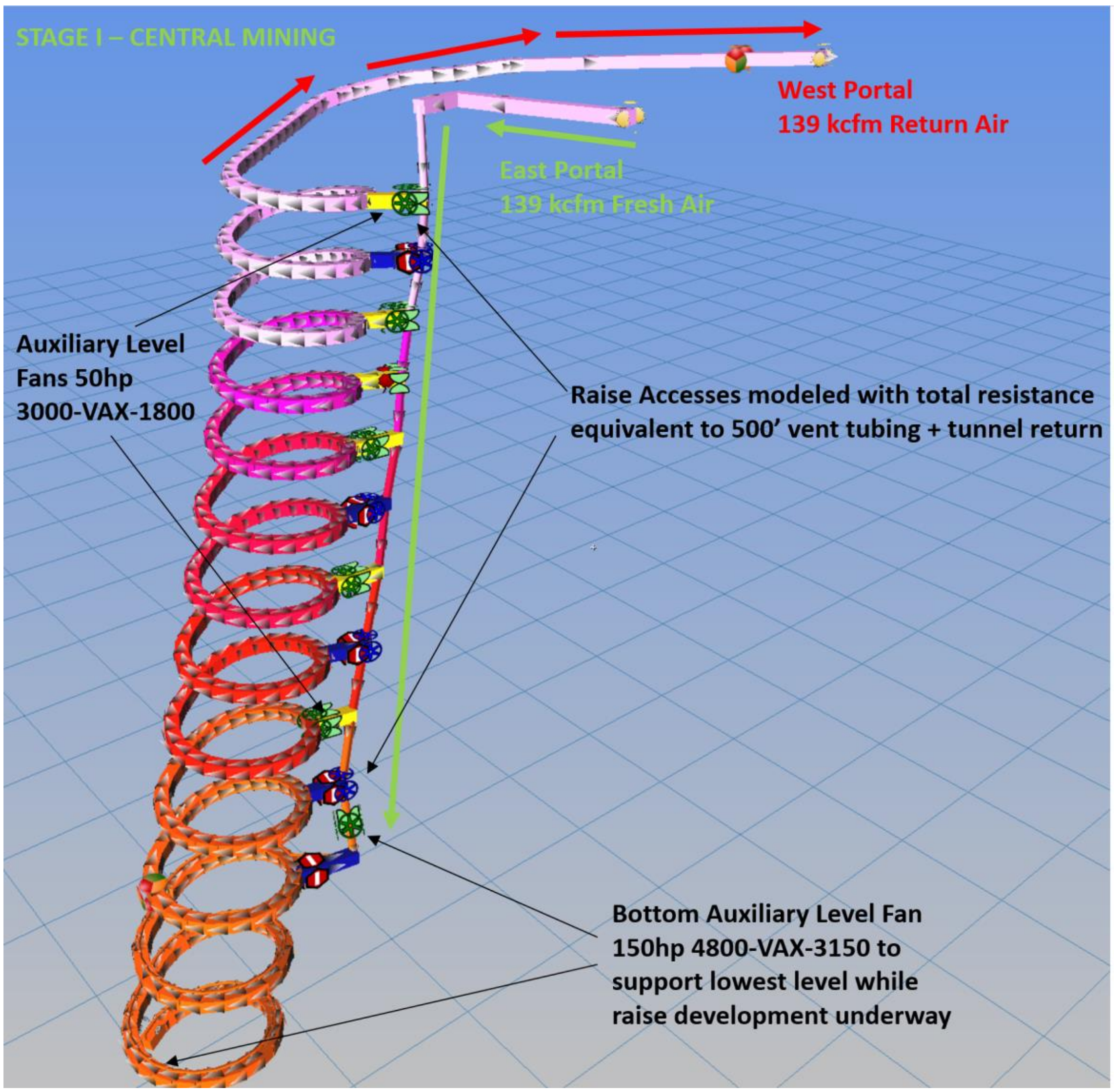


Figure 16-14: Stage I Schematic

Contaminant monitoring was also simulated to evaluate blast fume clear-out times. A 500' tunnel was modelled for each Stage at the bottom of each spiral ramp with a 300 lb. charge of 94/6 ANFO. A conservative dispersion factor of 1.5 was used (the default moderate factors are ~5) to produce realistic plume tails. Several dynamic simulation modeling points were selected with the clear-out time defined as when the tail of the last plume at the West Exhaust Portal reaches [CO] <= 20 ppm. Clear-out times range from 45 to 65 min depending on the mining stage and location of the blast. This should be viewed as an upper bound due to the conservative dispersion factor. Higher dispersion factors may accelerate the clear out by 5-10 min. In addition, all fans are prescribed to be equipped with variable frequency drives, and as a result, airflows may be temporarily ramped up after blasting to accelerate clear-out further if needed. An example of a dynamic monitoring graph can be seen in Figure 16-17 below.

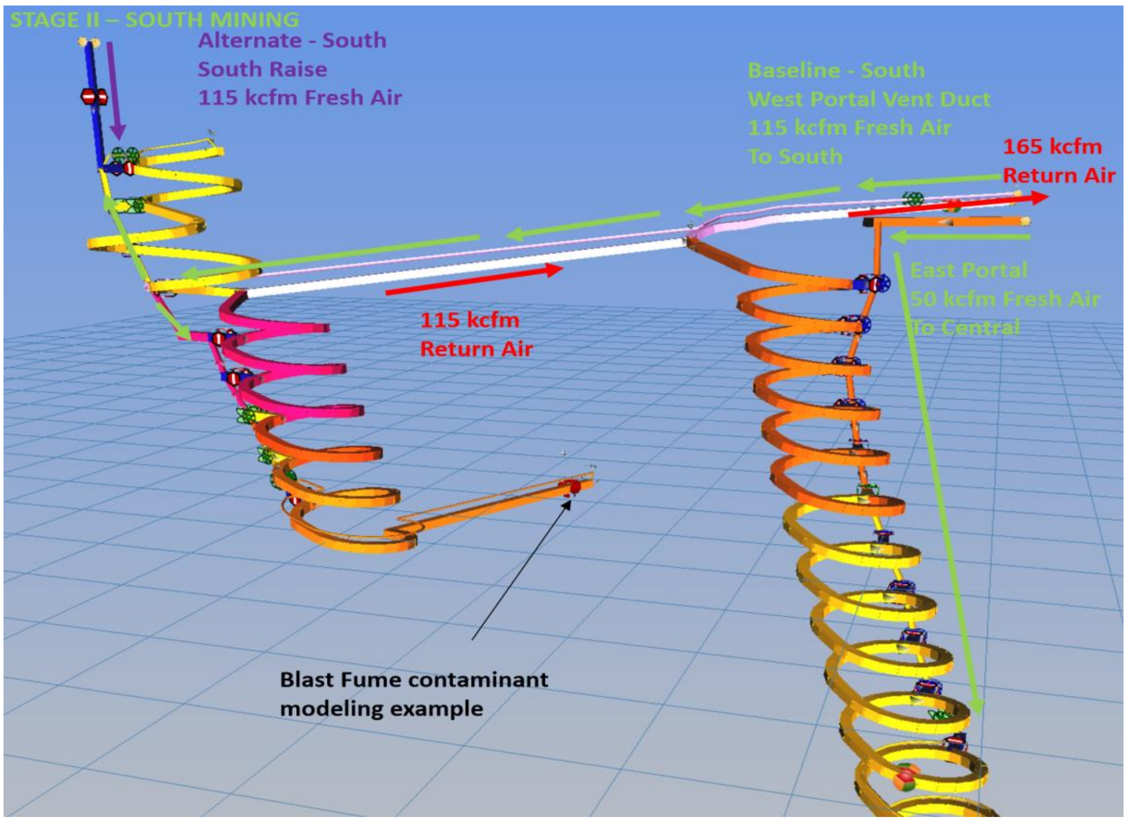


Figure 16-15: Stage II Schematic

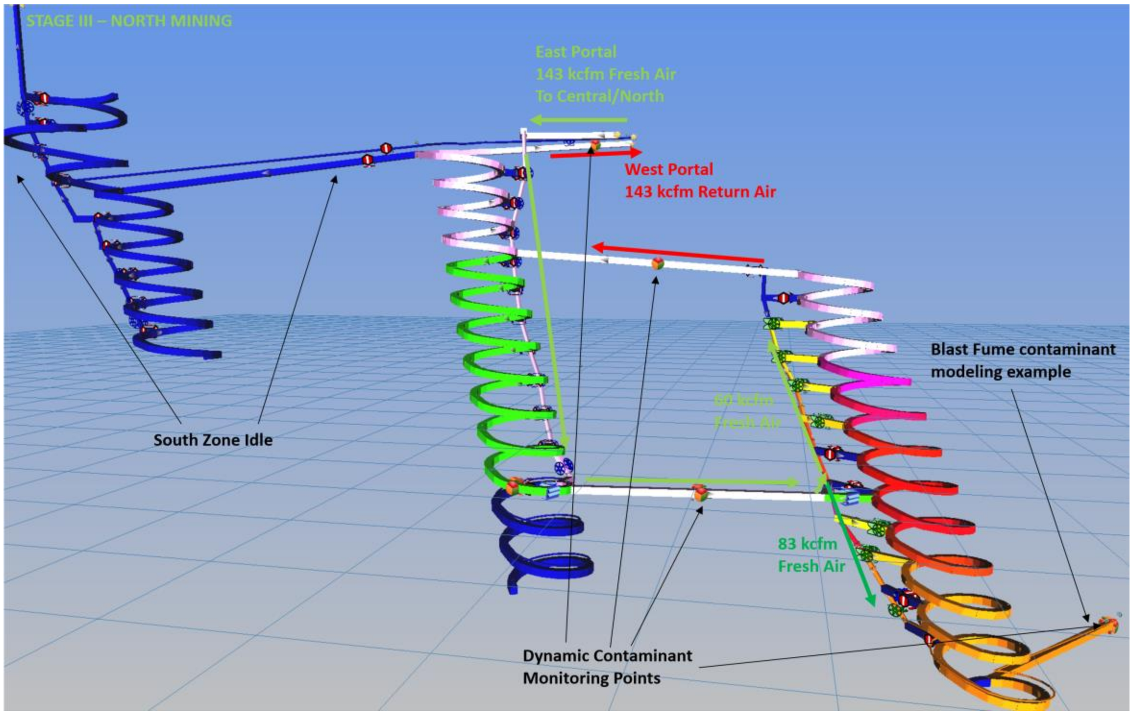


Figure 16-16: Stage III Mining

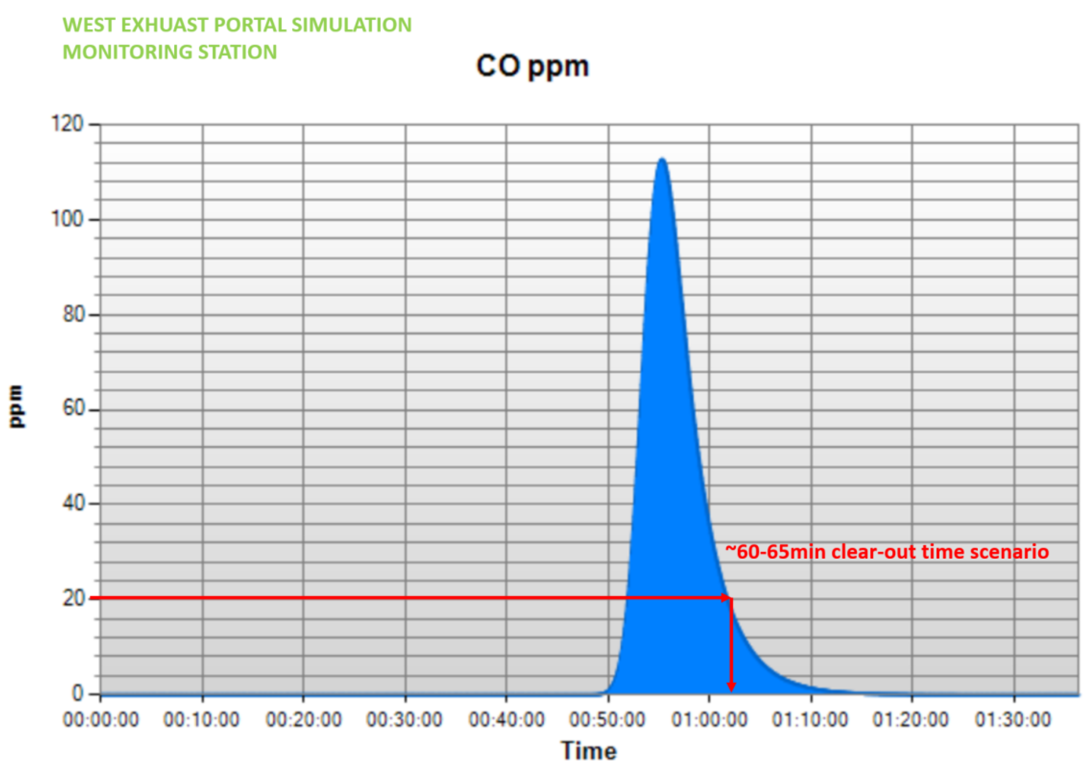


Figure 16-17: West Exhaust Portal Dynamic Monitoring Graph

A robust and conservative ventilation model was used for the preliminary assessment. Relative pressures are generally low. Further scenarios were conducted showing the specified system can handle in excess of 200 kcfm if required without any equipment changes. In addition, only two types of fans are prescribed as previously discussed: "larger" 150 hp 4800 series and "smaller" 50 hp 3000 series Howden or equivalent. The larger fans are to be used at the portal entrance during initial mining and for the baseline south ventilation Stage II, as well as for the lowest most spiral heading while the drop raise development catches up to the mining horizon. Finally, an additional spiral loop was modelled for each spiral to ensure specified equipment does not need changing in case deeper mining is required. One additional level per spiral can be accommodated without the need for raise extension to the bottommost level.

As an additional safety precaution, fire modelling was simulated for the model. The worst-case scenario was a fire originating in either the lower Central/South Spiral ramp connection or in the East Portal Ramp. In either of these scenarios, a fire from 2,000 lbs of diesel lasting 45 min would fill all downstream airways with up to 1000 ppm CO. As per regulations, no flammable materials will be stored in these primary airways. In addition, although equipment access in the lower Central/North bypass shall be restricted to special circumstances following adequate Job Hazard Analysis and precautions once that ramp becomes a main fresh supply airway from the Central to the North zones. For Stages I and II a fire anywhere outside of these scenarios results only in exhaust way contamination with the fresh air supply excavations remaining safe.

As a risk, it is worth noting a fire in the Central/South connection would melt through the fresh-air supply vent ducting in the baseline Stage I scenario. In this case, it is likely the power supply to the South Zone fans would also be terminated, and those fans would be inoperable. Nevertheless, it is recommended that all South Zone Auxiliary fans are connected to "kill switches" triggered from both underground and surface in the event of a fire to prevent fire contaminants from being drawn through the burnt vent tubing. With these precautions, the South Zone can be effectively sealed off from the fire.

While not modelled in the primary simulation, it is understood one ore access from the south zone to the central zone will remain unfilled until the end of mine life. This tunnel will serve as a secondary egress in the event the primary connection drift is blocked by a fire. In the alternate Stage II plan, the fresh air raise to surface will serve as the primary, secondary egress. An evaluation of the cost/ benefits of a fresh air raise to support the south ramp system is advised.

16.15 BACKFILL SYSTEM

The principal method of backfill at ICO is paste fill with in combination with waste material generated from the mine development. Summary of the annual backfill placement is shown in Figure 16-18. Beginning in Year 6, waste will need to be hauled from the TWSF to the portal as a source of fill material as mine development of the known reserve will have been completed. Waste backhaul costs have been included in the operating cost estimates. Waste back haul offers a savings over 100% paste fill and ensures stopes are filled soon after mining is completed.

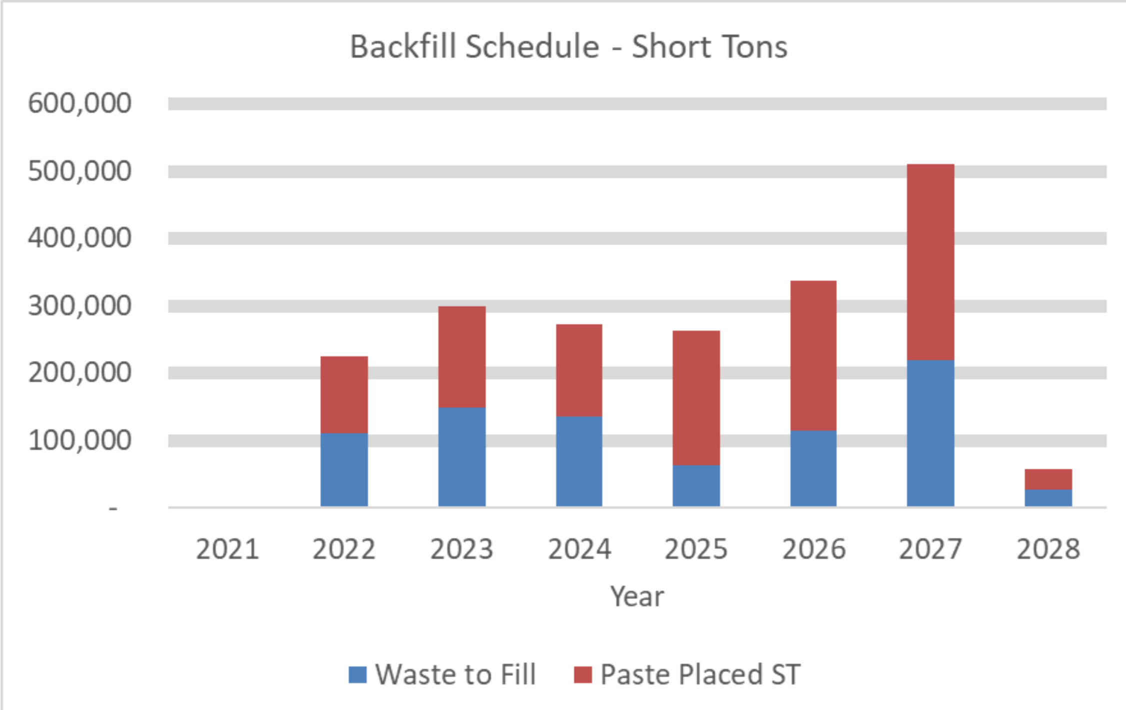


Figure 16-18: Backfill Schedule and Material Source

The paste fill prepared in the backfill plant located at the processing plant is routed through an overland pipeline along the pump back system pipeline route corridor to approximately elevation 7,445 ft. From there, the backfill material is directed within a cased boreholes into the Main Ramp. Paste fill is then routed into the mines through piping in the raise system to the stope accesses. Currently, a single main delivery line and two cased boreholes, (one standby) are proposed for the backfill system at ICO. The mine development schedule provides ample additional mining horizons should backfill be delayed by the anticipated rare problems with the surface pipeline portion of the paste delivery system.

The paste backfill design criteria, hydraulic design, pump recommendations and control philosophy developed by Paterson and Cooke ("P&C") in 2012 and verified in studies conducted in 2017 and 2019. The paste fill pump, mixer, filtering system and a majority of the backfill plant system had been purchased in 2012 during the original project execution, and there is a close similarity of the current mine layout to the original plan.

16.15.1 Backfill Reticulation and Pumping System

The backfill reticulation and distribution philosophy for the current study is similar to the proposed route proposed in the 2017 FS where a pipeline is routed overland on contour from the paste plant to a borehole located on the powerline access road with two entry boreholes into the mine workings, one active and one standby.

The following summarises the proposed paste fill distribution specified by P&C in 2012, with slight modification to match with the current mine design:

- Schedule 120, 6-inch pipes from the surface to the borehole, and down to approximately elevation 6,740 ft. The higher-pressure rating and thicker pipe walls provide additional "safety margin against wear considering the critical location, difficulty of replacement and potential slack flow". This pipe rating was incorporated into the design, included in all the haulage levels up to elevation 6,700 as per recommendation. The current mine design also includes this pipe rating for pipe installed in the raises.

- Schedule 80 pressure rated; 6-inch pipes are required for the backfill reticulation system in levels 6,700 to 6,386. Low SDR HDPE pipe is acceptable alternative based on pressure rating.
- HDPE, SDR 9, 6-inch pipes will be installed throughout the mine below level 6386.
- HDPE 4-inch DR9 on XC and AR is used for in-stope piping as well as in the crosscuts and attack ramps.
- Burst disks will be located at pipe rating changes along the pipeline route to discharge paste safely should overpressure occur.

The backfill plant has been designed to operate continuously during pours with monitoring of pressure in the surface plant and use of a remote camera to monitor the fill process when unattended. At least one staff member will be dedicated to the underground paste placement process during fill cycles (P&C, 2011).

16.15.2 Backfill Material Testing

There have been three backfill material testing campaign being performed on the tailings from ICO:

- 2008 material was tested with Holcim Type I Ordinary Portland Cement.
- 2017 material was tested with Ash Grove Type I-II, II-IV cement and also blended with Dura-Slag.
- 2017 material was tested with Ash Grove Type I-II cement to verify the 2017 results as recommended in the 2017 NI 43-101 technical report on the project by Micon Intl.

The backfill material testing includes material characterisation and determination of the rheology of the tailings. The following sections present conclusions and observations made on the results from both of these material testing campaigns.

Table 16-19: Mineralogy of Flotation test work

Mineral	Type	2008 Tailings [1]	2017 Tailings	2019 Tailings [2]
Quartz	Tectosilicate	10 - 12 %	42 - 45 %	47.3 %
Garnet	Orthosilicate	2 %	5 %	3.8 %
Mica	Phyllosilicate	18 - 19 %	24 %	-
Chlorite		63 - 68 %	18 - 22 %	11.2 %
Annite		-	-	30.4 %
Siderite	Iron Carbonate	-	5 - 7 %	-
Actinolite	Amphibole			7.0 %
Hematite	Iron Oxide	-	<2 %	0.3 %
Other	-	<5 %	-	-

Particle size distribution of the tailings from the three test regimes were also compared in Table 16-20 and Figure 16-19.

Table 16-20: Particle Size Characteristics of Tailings

Parameter	2008 Tailings	2017 Tailings	2019 Tailings
d_{90} particle size (µm)	138	115	126
d_{50} particle size (µm)	52	40	45
% passing 20 µm (%m)	29	29	26

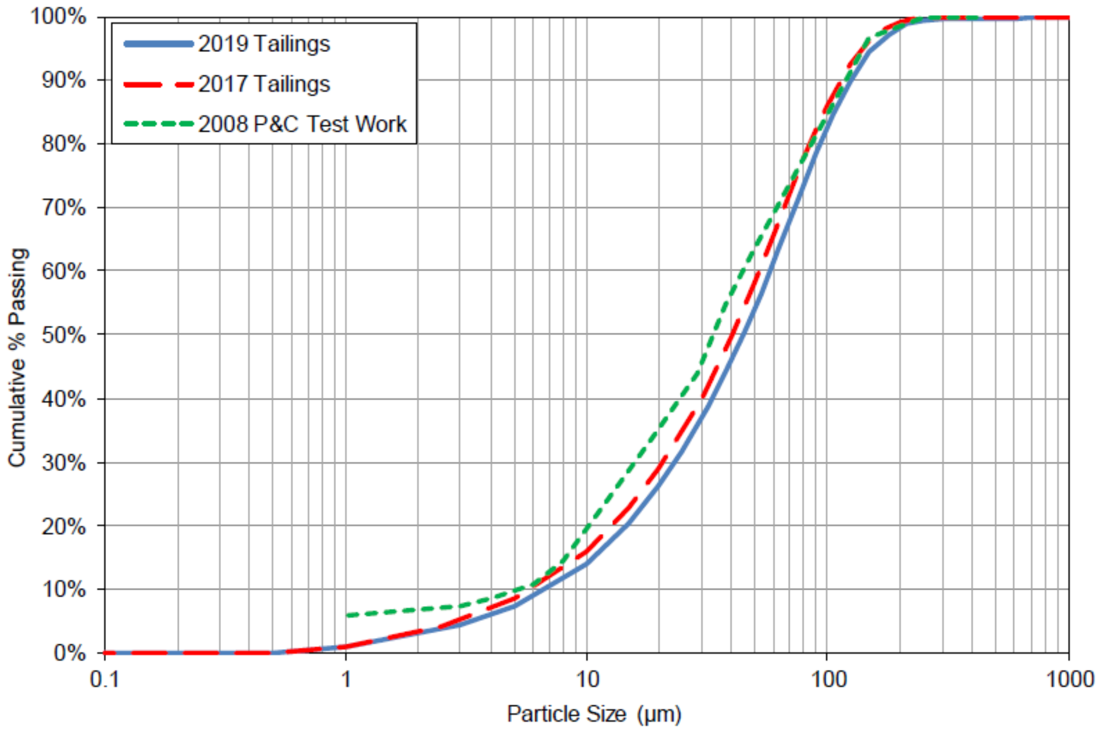


Figure 16-19: Particle Size Distribution of Tailings

16.15.3 Strength Testing Results

Table 16-21 summarises the strength testing results from 2008 where material was tested based on Holcim Type I Ordinary Portland Cement at 28 curing days. The test matrix in 2017, Table 16-22, was more comprehensive with the testing of material with a blend of binder and up to 28 curing days. Testing in 2019, Table 16-23, from fresh drill core confirmed the 2017 results. The 2008 results were likely skewed by source material including surface pit samples which may have been oxidised.

Table 16-21: 2008 UCS Testing Results

Mix #	Slump (inches)	Est. Mass Conc. (%m)	Binder Content (%)	W:C Ratio	28 Day UCS (kPa)
1	5	70.7	4	10.4	160
2	6	69.8	2	21.7	82
3	6	69.5	4	11.0	150
4	6	70.0	8	5.4	368
5	7	68.4	4	11. 6	117
6	8	66.5	2	25.2	81
7	8	66.8	4	12.4	125
8	8	67.9	8	5.9	387
9	9.25	65.9	4	12.9	154
10	10.25	65.3	4	13.3	104
11	-	66.0	10	5.2	551
12	-	68.0	4	11.7	160
13	-	68.0	10	4.7	723
14	-	68.0	12	3.9	1010
15	-	70.0	10	4.2	808

Table 16-22: 2017 UCS Testing Results

Mix #	As Cast %m Solids	Calculated as Cast Backfill Density (kg/m^3)	Binder Content (%)	Binder Type	W:B Ratio	28 Day UCS (kPa)
1	70.5	1901	9	50% Type I-II, 50% DuraSlag	4.6	1794
2	74.0	1990	6	50% Type I-II, 50% DuraSlag	5.9	1093
3	74.3	1998	8	50% Type I-II, 50% DuraSlag	4.3	2050
4	74.4	2000	4	50% Type I-II, 50% DuraSlag	8.6	597
5	74.4	2000	6	Type I-II	5.7	504
6	74.3	1998	6	Type II-V	5.8	547
7	71.4	1923	4	50% Type I-II, 50% DuraSlag	10.0	408
8	71.3	1920	3	50% Type I-II, 50% DuraSlag	13.4	247

Table 16-23: 2019 UCS Testing Results

Mix	As Cast %m Solids	Calculated as Cast Backfill Density (kg/m³)	Binder Content (%)	Binder Type	W:B Ratio	28 Day UCS (kPa)
1	72.2	1869	10	Type I, II, V	4.0	1,032
2	71.7	1858	6.5	Type I, II, V	6.1	562
3	71.7	1858	5	Type I, II, V	8.0	366
4	71.9	1863	4	Type I, II, V	10.0	281

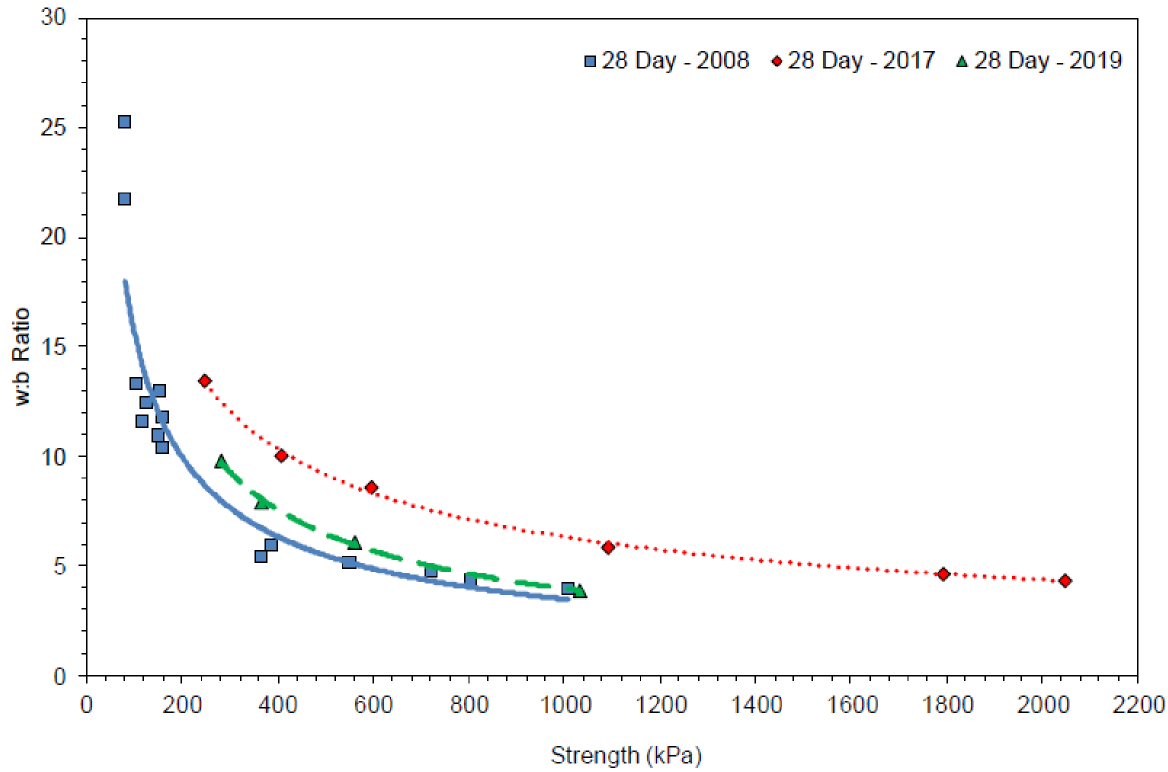


Figure 16-20: Comparison of 28 Day UCS Test Results

16.15.4 Backfill Material Testing Conclusion and Observations

The following are comparisons, conclusions and observations were made by P&C (2019).

Test results from 2008, 2017, and 2019 tailings were compared, and the following noted:

- Tailings segregation was observed during the 2008, 2017, and 2019 test campaigns.
- 2017 and 2019 tailings have higher quartz and lower chlorite than 2008 tailings which could potentially result in higher strength backfill.

- 2008 tailings have approximately 25% more ultra-fines (5-10 µm) and approximately 60% more clay materials (<5 µm) than the 2017 and 2019 tailings. The decrease in ultra-fines in the 2017 and 2019 tailings composite sample would improve paste strength and rheology.
- 2008 tailings displayed different flow behaviour, where 2017 and 2019 tailings have lower yield stress than 2008 tailings. This improvement is likely due to the reduction in ultra-fines in the 2017 and 2019 tailings samples.
- Recipes using 2017 tailings and 50:50 Type I-II and DuraSlag blend produced backfill with higher strength at the same w:b ratio than recipes using the 2008 tailings and Holcim Type I OPC or the 2019 tailings and Type I, II, V cement.
- There is a very good correlation between the various data sets that suggests that the early strength is not fully dependent on the different type of binders but rather on the water to cement ratio.
- The difference in UCS between 28 and 120 days cured was 8% with Type II-V cement and less than 2% with Type I-II cement (i.e. strength loss).

16.15.5 Design Criteria

The strength of the backfill required for the mining methods proposed at ICO was estimated based on the stability of free-standing backfill formula by Mitchell (1983).

However, assumptions were made for the binder or cement additions for the current study because of the availability of the latest test results. These assumptions were made based on 28 days curing time binder blends and the information from 2019. Due to market conditions in 2019 cost and availability of Dura-Slag was considered a high risk and design was advanced based on commonly available cement.

The assumptions made are presented in Table 16-24, where the initial stope sill mats' have the highest cement or binder content to support the extraction of the stope underneath it during the subsequent mining cycles. High strength caps, the last 10-20% of the paste fill cycle, are prepared with additional binder enabling transiting of mining equipment while working in the stope and the cores of the stopes have the lowest cement content to achieve the minimum free-standing target strength. Mine operating costs assumed a blended binder content of 5% for fills other than the initial sill cut representing 20% of the stope utilising a high strength cap.

Table 16-24: Summary of Estimate Binder Addition

Description	Estimated Binder Addition
Sill Mat	10.0%
High Strength Cap	8.0%
Core Stope Fill	4.0%

16.16 MINE DEWATERING

The groundwater inflow estimate was based on a preliminary estimate documented by Telesto Solutions, Inc. (Telesto) in 2006 for the development of the ICO Ram deposit. The estimate ranges from 33 to 66 gpm which Telesto considered to be over-estimated and in the opinion that a flow rate of 43 gpm is a more accurate estimate for the ICO deposit at full excavation.

The mine could be dry during the development and pre-production stages where water will be recycled and reuse for the initial development until water wells are established for the mine. ICO currently has two wells for potable and process makeup water as well as nine groundwater intercept wells for potential groundwater contamination mitigation that can provide 150 gpm to the mine during dry periods. By permit, the groundwater capture system will be installed prior to mine development.

A series of 15 hp submersible pumps located at sumps along the main ramp system in former muck bays will dewater to the main sumps located near the portals. The main sumps will decant clear water for reuse in mining. Excess water, anticipated seasonally during spring, will then be pumped to the surface water ponds by a 100 hp pump.

Dewatering from the mine development, production areas and operating levels will be accomplished by 6 hp submersible pumps each having the capacity to deliver up to 150 gpm of water through 4-inch HDPE diameter pipes to the ramp sumps during steady-state operation.

16.17 COMPRESSED AIR

Compressed air will be supplied by means of 200 hp rotary screw, air-cooled air compressor capable of delivering 1,075 scfm @ 125 psig maximum discharge pressure. Compressed air will be distributed via 4 and 6-inch HDPE lines.

16.18 POWER REQUIREMENTS AND DISTRIBUTION

The mine electrical power demand is approximately 1.1 MW to 1.3 MW with approximately half of the power demand is from mine dewatering, ventilation and air compressors. Variable frequency drives installed on the fans, the strategic location of the dewatering pumps and regulating the air compressor to on-demand basis can reduce the power consumption.

16.19 RECOMMENDATIONS

The following summarises the recommendations observed during the preparation of the current feasibility:

- Geotechnical - Classification and characterisation of the rock mass in relation to it spatial location during initial development and access to ore zones will assist in stope dimension and overall mine design validation in coordination with the geotechnical borehole and study conducted in 2017 at the recommendation of the prior NI 43-101 technical report.
- Ventilation - A mine ventilation study utilising best-practice modelling software was conducted on the proposed mine design. The study identified the potential for a relatively short, bored ventilation raise to surface above the south ramp system would reduce ventilation costs. It is recommended the cost and permitting requirements for a ventilation raise be examined prior to the development of the South Ramp system (Y3).
- Electrification – Additional optimisation of the mine design, plan and especially the use of battery/electric haulage equipment to enable automation of mine systems should be examined early in the mine life to reduce costs and the operational carbon footprint. Preliminary trade-off studies indicate additional power supply to the site will be required to support electrification. Studies should be advanced prior to ore production. Unused re-mucks along the ramp system may function as battery bays to reduce excavation requirements in this scenario.

17 PROCESSING AND RECOVERY METHODS

17.1 INTRODUCTION

The process plant metallurgical design is based upon data and design criteria provided by Jervois, DRA Americas ("DRA"), vendor data, testwork and regulatory/permitting requirements. These inputs formed the basis for the entire process plant design, including process flowsheet and mass-water balances.

The crushing and grinding circuit design is based upon the design throughput requirements and ore competency and hardness characteristics obtained by testwork. The SAG/Ball mill sizing is based on achieving the grind size required for optimal flotation performance and based on the outcomes of the metallurgical testwork. Equipment sizing calculations have been completed using energy based populated balance modelling techniques.

The design and configuration of the bulk sulphide flotation circuit are based upon the locked cycle test results conducted for the 2007 feasibility study under the direction of Samuel Engineering Inc. These results also provided the basis for recovery and grade calculations.

Concentrate and tailings products are thickened and then dewatered using a conventional plate and frame pressure and vacuum disc filtration, respectively. The filtration circuit design is based on common design practices for concentrate and metallurgical testwork.

For other equipment items, the peak production rates were used for sizing. Where equipment sizing has been influenced by volumetric flowrates, a 15% safety margin above the peak instantaneous slurry flowrates has been applied.

17.2 PROCESS FLOWSHEET DESCRIPTION

The concentrator plant is designed to produce a single concentrate product containing copper and cobalt which are derived via a bulk sulphide flotation circuit flowsheet. A fit-for-purpose design philosophy has been adopted within the project design, which is described below. The simplified blockflow of the concentrator process is depicted below in Figure 17-1.

17.2.1 Ore Receiving

Ore is loaded from a Run of Mine ("ROM") stockpile on the ramp by Front End Loader ("FEL") and discharged into the ROM Tip Bin. An angled ROM Static Grizzly protects the ore receiving section from oversize which is placed to the side of the ramp and periodically removed. A fixed speed ROM Vibrating Grizzly Feeder extracts material and discharges the oversize, into the Primary Jaw Crusher. After being recombined with grizzly undersize, all material is then conveyed to the Fine Ore Bin. The Fine Ore Bin provides enough buffer storage time for the operator to load the plant feed and then attend to other duties, while a controlled feed is maintained into the SAG mill.

17.2.2 Milling

Ore at a controlled feed rate, together with a controlled quantity of process water, is discharged into the SAG Mill. SAG Mill discharge is trommel screened in order to remove oversize scats. Trommel undersize gravitates into the Mill Product Sump. Combined SAG and Ball mill product slurries are then pumped at a controlled rate to the Mill Product Cyclones.

The Mill Product Cyclones size the mill product slurry in a closed-circuit loop with the Ball Mill. Cyclone underflow gravitates into the Ball Mill Feed Chute while the product overflow is discharged onto the Vibrating Trash Screen which guards the downstream flotation process against tramp oversize such as plastic, woodchips and coarse rock. SAG and

Ball Mill trommel oversize scats, as well as Trash Screen oversize, are discharged into containment areas which are periodically cleared by the operator.

Provision has been made for the addition of sulfuric acid and potassium amyl xanthate (“PAX”) into the mill product sump via tote and dosing pump systems.

17.2.3 Rougher Flotation

Mill final product gravitates in a pipe launder to the concentrator building where the slurry is sampled prior to being discharged into the agitated Flotation Feed Surge Tank. The surge tank provides a buffer capacity to smooth out production surges prior to the flotation circuit. Additional PAX collector is added at a controlled rate into the surge tank, while frother (AF65) is added just prior to the rougher flotation cells.

The slurry is then pumped at a controlled rate to the Rougher Flotation Cells. Copper and cobalt-rich froth product are collected and pumped to the Cleaner flotation cells. The Rougher Flotation tails slurry is combined with Cleaner Scavenger flotation tails and pumped to the Thickener Feed Box.

Online slurry analysers located on both the rougher and thickener feed pipelines, provide online analysis assay indications to the operator. This allows the operator to respond quickly to any variations in plant feed grade or flotation performance.

17.2.4 Cleaner and Cleaner Scavenger Flotation

Rougher concentrate is pumped to the Cleaner flotation cells. Froth product is laundered to the Concentrate Thickener, while the tailings gravitate into the Cleaner Scavenger flotation cells.

Cleaner Scavenger froth product is laundered back to the Cleaner Feed Pump, while the tailings are pumped to the Tails Thickener.

Collector (PAX) reagent is dosed in a controlled manner into the Cleaner Flotation Feed Sump.

17.2.5 Concentrate

Cleaner flotation froth product is combined with a controlled quantity of flocculant and thickened in a concentrate thickener from where the underflow is pumped to an agitated Concentrate Surge Tank. The slurry is then filtered within a plate and frame filter press. The filtrate is gravitated back to the Concentrate Thickener, while the cake is discharged to the dedicated Concentrate Bagging System. Each concentrate bag is weighed, sampled and sealed for transportation by the bagging system, prior to it being manually removed and stored in the concentrate area. The bags will be loaded periodically within a 20-foot container and trucked from the plant.

Concentrate thickener overflow is reticulated and re-used within the flotation circuit or pumped to the tails thickener process water tank.

17.2.6 Tails Thickening

Flotation tails are pumped via a sampler to a Tails Thickener, where flocculant is added at a controlled rate. Flocculated material agglomerates and settles to the base of the thickener from where it is pumped to the agitated Tails Surge Tank. It is then pumped at a controlled rate to the Tails Vacuum Disc Filter.

Tails thickener overflow clarified solution is discharged into the Process Water Tank, which provides enough buffer volume for plant start-up conditions and normal operations. The process water is then reticulated throughout the plant by the Process Water Pump.

17.2.7 Tails Filtration

The slurry is pumped to the Tails Filter at a controlled rate. The filtrate is returned back to the Tails Thickener, while the solids filter cake is discharged onto the reversible Tails Conveyor.

The reversible Filter Cake Conveyor can discharge either to the Tails Stockpile – Dry Stacking or alternatively, to the paste backfill plant.

In the event the paste backfill plant is not available, the material from the Tails Stockpile is manually removed by Front End Loader and trucked to the Tails Storage Facility.

When the paste backfill plant is available, the Filter Cake Conveyor direction is reversed, and the cake discharged into a Backfill Mixer along with a controlled quantity of cement. Process water is also added to the Backfill Mixer in order to generate the design slurry density before the paste is pumped by the Paste Backfill Pump to the mine backfill site.

17.2.8 Water Reticulation

The Water Management Ponds receive water from the following sources:

- Mine Dewatering
- Snow Melt and Precipitation site drainage
- Pond Leak Detection
- Excess concentrator process water and uncontrolled concentrator spillage events.

The water is stored within the ponds and periodically used as Tailings Waste Storage Facility (“TWSF”) dust suppression water or make-up to the concentrator. Any water discharged to the environment will first be treated by the Water Treatment Plant (“WTP”).

Plant clean water make-up is generated from a borehole well field that provides make-up water that feeds the Fire Water Tank first followed by the Potable and Clean Water tanks.

17.2.9 Reagents

Flocculant

Dry flocculant is delivered to the concentrator. Periodically a hydrated batch of flocculant slurry is generated by a flocculant make-up system which is dosed at controlled rates to the concentrate and tails thickeners in order to increase the solids rate of sedimentation.

Collector - Xanthate (PAX)

Dry Potassium Amyl Xanthate (“PAX”) is delivered to the concentrator. Periodically, a hydrated batch of PAX is generated by a PAX make-up system which is then dosed at controlled rates to the selected process within the flotation circuit via dedicated dosing pumps.

Sulfuric Acid

Dilute sulfuric acid is dosed at a controlled rate to the ball mill in order to reduce the pH of the flotation circuit should the natural pH become elevated above testwork conditions. The acid is contained within a tote and dosed via a dedicated dosing pump.

Frother (AF65)

Aeroflot AF65 is used as a frother reagent to aid the flotation processes. It is delivered in tote containers directly to the site.

Cement

Dry cement powder is delivered to site by truck, before being mixed into the concentrators' backfill tails pumping system at a controlled rate.

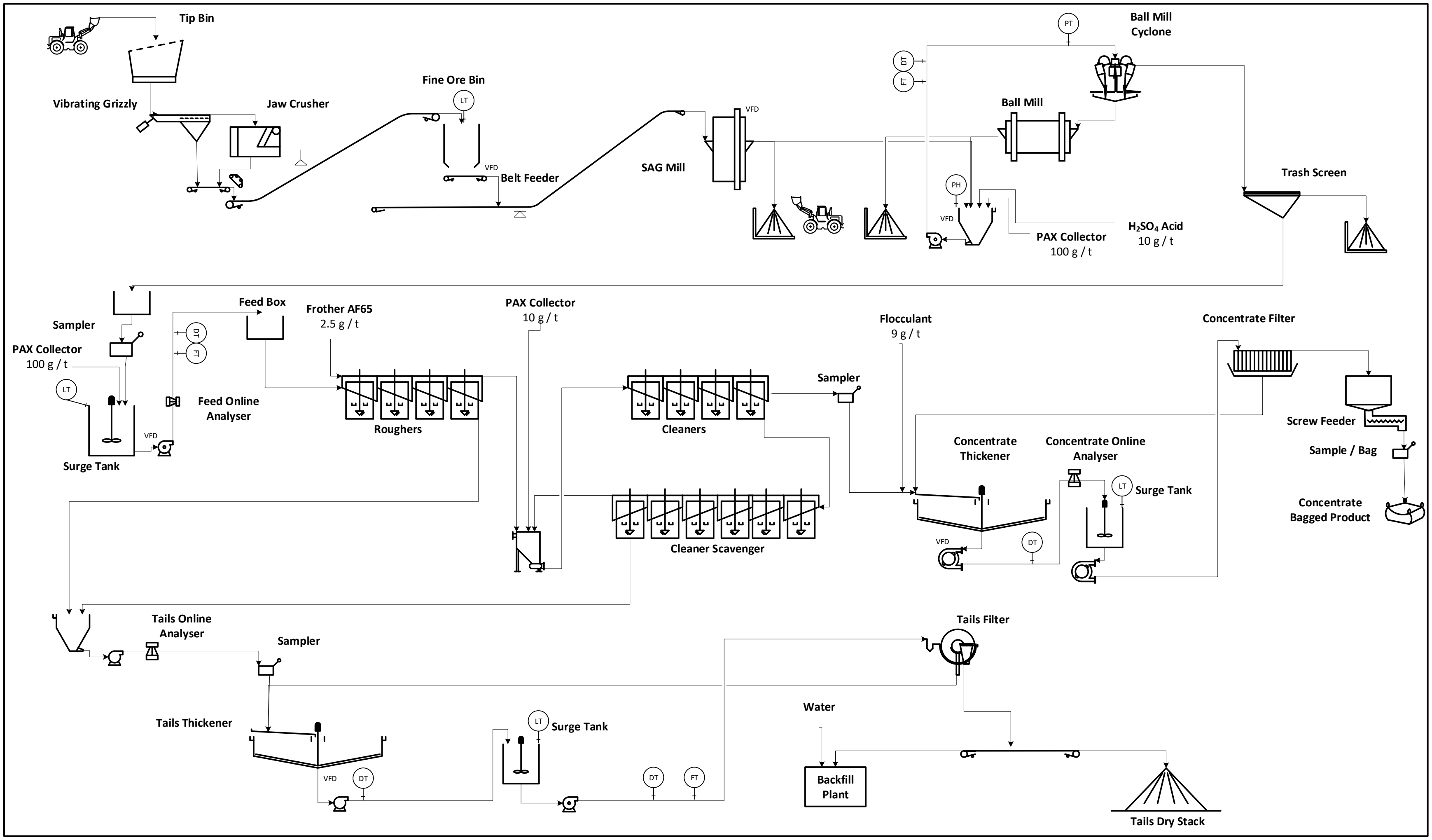


Figure 17-1: Simplified Overall Process Flow Diagram

17.3 PLANT DESIGN

The processing plant is designed to process a nominal 1,200 short (1,089 metric) tonnes per day of Run of Mine ("ROM") ore which is trucked to the plant feed stockpile from an underground mine. The plant will produce a single concentrate product using a bulk sulphide flotation flowsheet. Plant tailings will be filtered and either trucked to a dry stack tailings facility or repulped and pumped by the paste backfill plant for deposition in the underground workings.

The summarised process design criteria are tabulated below in Table 17-1.

Table 17-1: Summarised Process Design Criteria

Description		Unit	Value	Source
Plant Operating Schedule	Availability	%	92	Design
	Daily Treatment Rate	dry metric tpd	1,089	Client
	Hourly Treatment Rate	dry metric tph	49.3	Calculation
Material Type	Sulfide in Feed Max	% Feed	100	Client
	Oxide in Feed Max	% Feed	15	Client
Primary Jaw Crusher	Installed Power	KW	75	Existing / Vendor
	Feed Size F80	mm	75	Client
	Closed Side Setting (CSS)	mm	75	DRA
	Fine Ore Bin Volume	m^3	282	Existing
Ore Hardness	CEET Crusher Index (CEET Ci)		8.3 - 17.2	Phase 2 Testwork
	Bond Impact Work Index (CWi)	KWh/t	2.6 – 10.5	Phase 2 Testwork
	Abrasion Index (Ai)	g	0.056 - 0.138	Phase 2 Testwork
	Bond Ball Mill Work Index (BWi)	KWh/t	14.2 - 15.7	Phase 2 Testwork
	Bond Rod Mill Work Index (RWi)	KWh/t	5.0 - 5.1	Phase 2 Testwork
SAG Milling	Installed Power	KW	735	DRA / Vendor
	Mill Diameter	m	4.62	DRA / Vendor
	Mill Length	m	2.5	DRA / Vendor
	Trommel Screen Aperture	mm	9.5	DRA / Vendor
	Mill Loading	%	15.8 - 21.5	DRA / Vendor
	SAG Mill Grind Product (P80)	mm	0.425	DRA / Vendor
Ball Milling	Mill Feed	%New Feed	259.6	DRA / Vendor
	Installed Power	KW	551	Existing
	Mill Diameter	m	2.9	Existing
	Mill Length	m	4.88	Existing
	Mill Loading	%	34 - 35	DRA
	Mill Target Grind Size	µm	75 - 85	Phase 2 Testwork
	Number of Cyclones	No	3+1	DRA / Vendor
Flotation	Surge Tank Residence Time	Min	15	DRA

Description		Unit	Value	Source
	Rougher Laboratory Flotation Time Required	Min	9	2007 Feasibility, Samuels Eng.
	Rougher Installed Residence Time Required	Min	22.5[1]	DRA / Vendor
	Number Rougher Cells	No	4 Tank Cells	DRA / Vendor
	Rougher Cell Volume	m^3/cell	15	DRA / Vendor
	Flotation Circuit pH		Natural	2007 Feasibility, Samuels Eng.
	Cleaner Laboratory Flotation Time Required	Min	3.5	2007 Feasibility, Samuels Eng.
	Copper Cleaner Installed Residence Time	Min	8.8[2]	DRA / Vendor
	Number Cleaner Cells	No	4	Existing / DRA
	Cleaner Cell Volume	m^3/cell	5.1	Vendor
	Scavenger Cleaner Laboratory Flotation Time Required	Min	2	2007 Feasibility, Samuels Eng.
	Copper Scavenger Cleaner Installed Residence Time	Min	5[3]	DRA / Vendor
	Number Scavenger Cleaner Cells	No	6 Denver Cells	Existing / DRA
	Scavenger Cleaner Cell Volume	m^3/cell	0.71	Vendor
Thickening Filtration	Concentrate Solids Loading (Installed)	tph/m^2	0.29	DRA / Vendor Pocock Testwork
	Concentrate Installed Diameter	m	3.6	Vendor
	Tails Solids Loading (Installed)	tph/m^2	0.26	DRA / Vendor Phase 2 Testwork
	Tails Installed Diameter	m	15	Vendor
	Installed Concentrate Filtration Area	m^2	80	Existing
	Concentrate Filter	Type	Plate & Frame	Existing
	Installed Tails Filtration Area	m^2	120	Phase 2 Testwork / Vendor
	Tails Filter	Type	Vacuum Disc	DRA / Vendor
	Tails Moisture	% mass	18	Client

The nominal and design feed grades were determined from the 50th and 85th percentile of the original Life of Mine ("LOM") plan. The summarised process mass balances design criteria are tabulated below in Table 17-2.

Table 17-2: Summarized Mass Balance Design Criteria

Description		Unit	Minimum	Average	Maximum
Feed Grade	Copper	% Cu	0.49	0.76	0.98
	Cobalt	% Co	0.48	0.57	0.65
	Copper: Cobalt Ratio		0.82	1.33	1.64
Product Grade	Copper Product	% Cu	10.64	13.74	15.47
	Cobalt Product	% Co	10.00	10.00	10.00
Metal Recovery	Copper to Concentrate	% Cu in Feed	95.13	95.73	96.18
	Cobalt Recovery to Concentrate	% Co in Feed	91.73	91.03	90.71
	Gold Recovery	% Au in Feed		84.9	
Product Mass	Concentrate	% Feed	4.39	5.30	6.10
	Concentrate	dry metric tph	2.16	2.61	3.01
	Plant Tailings	dry metric tph	47.14	46.69	46.3

17.4 PRODUCTION SUMMARY

An integrated mine and process plant concentrator plan was developed with the following considerations for the concentrator:

- The process plant nominal throughput rate is 1,200 short (1,089 metric) tonnes per day;
- The basis for the development of the concentrate recoveries and grades were the locked cycle tests carried out during the 2007 feasibility study, in consideration of the final flow sheet;
- The process plant throughput is ramped up over six months, and metal recovery has a recovery ramp-up period of four months; and
- Life of mine mill feed grades of 0.80% Cu and 0.55% Co.

Life of mine metal recovery is estimated at:

- Copper to Concentrate 95.47% at a grade of 14.85% w/w Cu; and
- Cobalt to Concentrate 91.07% at a grade of 10% w/w Co.

Life of mine total gold recovery to concentrate is estimated at 84.9%. The concentrator Life of Mine production plan is tabulated below in Table 17-3.

Table 17-3 Jervois Life of Mine Production Plan

Year	Feed				Concentrate							Tailings		
	Tonnes	Grade		Au	Tonnes	Grade		Au	Recovery			Tonnes	Grade	
	(s tonne)	% Cu	% Co	oz	(s tonne)	% Cu	% Co	oz	Cu Rec	Co Rec	Au Rec	(s tonne)	% Cu	% Co
2022	129,200	0.56	0.62	2,476	7,150	9.48	10.05	2,007	93.42	89.65	84.93	122,050	0.04	0.07
2023	438,000	1.04	0.66	9,516	26,745	16.35	9.87	8,082	96.20	90.66	84.93	411,255	0.04	0.07
2024	439,200	0.88	0.54	7,022	21,692	17.08	9.92	5,964	95.53	90.50	84.93	417,508	0.04	0.05
2025	438,000	0.88	0.52	8,645	20,647	17.72	9.95	7,342	95.38	90.66	84.93	417,353	0.04	0.05
2026	438,000	1.22	0.46	12,079	18,307	27.80	9.87	10,258	95.01	89.52	84.93	419,693	0.07	0.05
2027	438,000	0.44	0.55	6,053	22,151	8.27	10.14	5,140	95.62	92.54	84.93	415,849	0.02	0.04
2028	421,120	0.37	0.54	5,591	20,738	7.22	10.19	4,748	95.52	92.93	84.93	400,382	0.02	0.04
Total	**2,741,520**	**0.80**	**0.55**	**51,381**	**137,430**	**15.16**	**10.0**	**43,540**	**95.47**	**91.07**	**84.93**	**2,604,090**	**0.04**	**0.05**

17.5 ENERGY, WATER AND PROCESS MATERIAL REQUIREMENTS

The plant material and water balances have been developed based on the process flowsheets and design criteria.

17.5.1 Reagents and Consumables

The following reagents are used throughout the process plant:

- Aeroflot (AF65) – frother;
- Potassium Amyl Xanthate (PAX) – flotation collector;
- Sulfuric Acid – pH control in ball milling; and
- BASF Magnafloc 10 Flocculant – for increased settling rates in thickeners.

Reagent mixing will be completed in a designated area within the plant within dedicated bunded areas. The layout and general arrangement of the reagent area account for the need to prevent contact of incompatible reagent types.

Reagents are made up or diluted with fresh water (where necessary); flocculant powder and xanthate are made up using clean fresh or raw water. AF65 frother and sulfuric acid are delivered in 1,000 L bulk containers and added to the circuit without dilution.

Grinding media is supplied in 200 L steel drums.

17.5.2 Air

Compressed instrument air for the flotation concentrator building distribution will be provided by the centralised compressor plant and filtration/drying system. Air receivers will be positioned throughout the process plant to buffer and control fluctuations within the system.

An independent system will provide compressed instrument air for the mill building area.

A dedicated, independent, low-pressure air blower and distribution system will supply the process air required for the flotation circuit.

17.5.3 Water

The use of external make-up water has been minimised as part of the process plant design. Process water is recovered within the circuit using thickeners and filtration unit operations. The concentrate thickener allows for the recirculation of process water within the flotation circuit to an extent. Only a marginal offtake is required from the tails thickener process water for concentrator make-up, allowing most of the water to be recirculated back to the mill circuit. Make-up water from the Water Management Ponds is pumped, when required, up to the tails thickener process water tank when plant make-up is required.

Clean (fresh) water will be withdrawn from local boreholes which will supply the fire water tank, clean water tank and potable water tank. The tanks have sufficient volume to meet the instantaneous process demands of the plant. The clean water tank feed nozzle will be placed mid-way up the fire water tank wall to ensure a reserve volume is always available for fire suppression.

The discharge of paste backfill from the plant allows for a continuous bleed of water. This will prevent a potential buildup of salts within the process which could impact on flotation performance. As further mitigation, if required, the Water Treatment Plant can be additionally used to mitigate this risk.

18 PROJECT INFRASTRUCTURE

18.1 WORK COMPLETED TO DATE

The Idaho Cobalt Operations ("ICO") has been progressed to an advanced pre-construction stage which includes the following project infrastructure:

- The access road from Highway 93 to the mine site has been completed
- The security gatehouse has been installed along the access road to the mine site
- General site preparation and construction of the plant terraces including stripping and grading
- ROM haul road from the portal bench to the ROM pad has been completed
- The portal bench has been partially constructed and will be completed ahead of the commencement of mining development
- Bulk earthworks required for the Phase 1A and 1B Tailings Waste Storage Facility ("TWSF") was completed during the 2011 construction phase. The remaining earthworks construction, as well as replacement of the existing liner, installed for TWSF-1A, and installation of the 1B liner will be completed during project implementation
- Civil construction and foundations for the Concentrator building were partly completed during the previous construction phase in 2018
- The water management ponds have been completed, and installation of return water pumps will be completed during project implementation
- The incoming power supply line was completed during the last phase of construction. Tie-ins to the supply line and the site distribution system have also been completed including power supply to the Pumpback system
- Construction of the Water Treatment Plant was largely completed during the previous phase of construction in 2018 and commissioning of the treatment plant will form part of the scope to complete environmental systems to enable mining development
- The Pumpback equipment has been supplied and is currently in storage in Salmon. The intention is to install and commission the Pumpback system as part of the scope to complete environmental systems to enable mining development
- Administration building
- Maintenance building
- Fuel storage

During previous phases, the majority of mechanical and electrical equipment as well as the milling and concentrator structures were procured and is in storage at the Jervois' warehouse in Salmon Idaho.

The Salmon Depot is currently used for storage of the purchased equipment. In future, this site will be used as a mustering point for construction and operations employees who will be bussed to site. It will also serve as temporary storage of shipments bound for the mine site.

Of the existing equipment, the following will be used under the current design:

- Mobile crushing plant consisting of a primary jaw crusher, secondary cone crusher and sizing screen
- Secondary ball mill including ancillaries and main drive motor
- The Fine Ore Bin and belt feeder
- Flotation Equipment – Box Cells
- Concentrate Filter including ancillaries
- Paste Pump and Mixer

- Various Slurry Pumps
- Backup Generator
- Pond Reclaim Pumps
- Pumpback equipment

As part of the Feasibility Study M3 completed a condition assessment of the existing equipment to determine the suitability and extent of refurbishment required as part of project implementation. The complete report is included in Appendix 09-01 for reference.

Figure 18-1: ICO Overall Development – TWSF and Water Management Ponds

Figure 18-2: ICO Plant Infrastructure – Admin and Maintenance Buildings, Water Treatment Plant

18.2 SITE LAYOUT

The ICO is located in the Salmon River Mountains of central Idaho. The site is generally characterised by relatively steep v-shaped valleys and flat-topped mountain tops. The mine portal is located in one of these valleys. Adjacent to this valley is a flat-topped mountain which is referred to as the Big Flat. The processing facilities and the TWSF are all located on the Big Flat. The Big Flat is a gently sloping area at an elevation of approximately 8,000 feet, which is approximately 1,000 feet higher than the mine portal. The ROM haul road connects the Big Flat and portal bench. Facilities located on the Big Flat include:

- Coarse ore storage on the ROM pad
- Waste (filtered tailings and mill scats) storage outside the mill building
- Crushing facilities located in the mill building
- Primary and secondary milling and classification
- Flotation circuit located in the concentrator/process building
- Concentrate filtration and storage
- Tailings thickening and filtration
- Paste/Backfill preparation plant located in the concentrator/process building.
- Process water treatment and storage
- Reagents mixing and storage
- Water treatment plant
- TWSF and water management ponds
- Mine office building and admin facility
- Fuel and lube storage and dispensing facilities
- Mine equipment maintenance facility
- Miners dry and cafeteria
- Camp
- Electrical substations and emergency power

- Firewater system

Figure 18-3: ICO Process Plant Layout

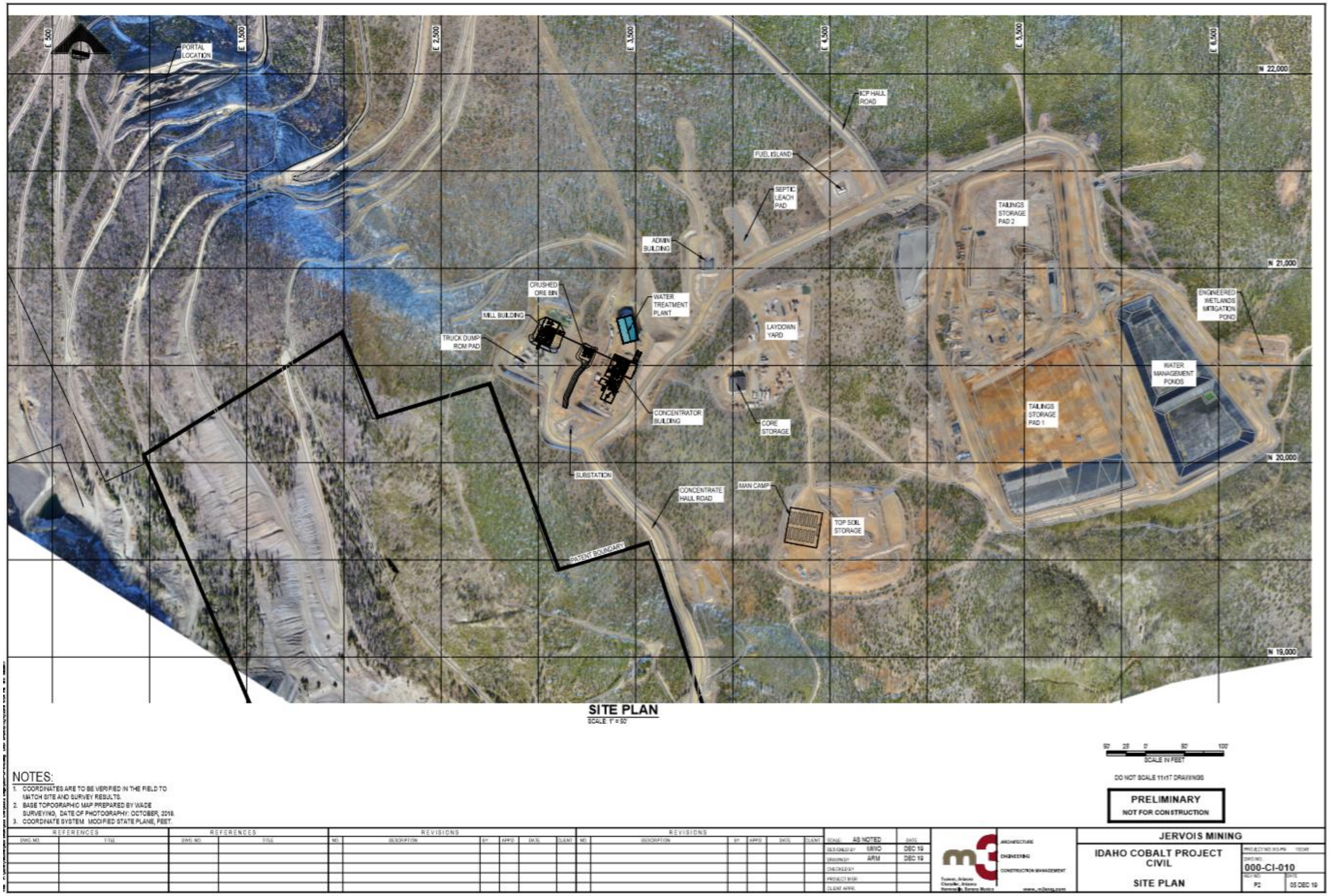


Figure 18-4: ICO Site Layout

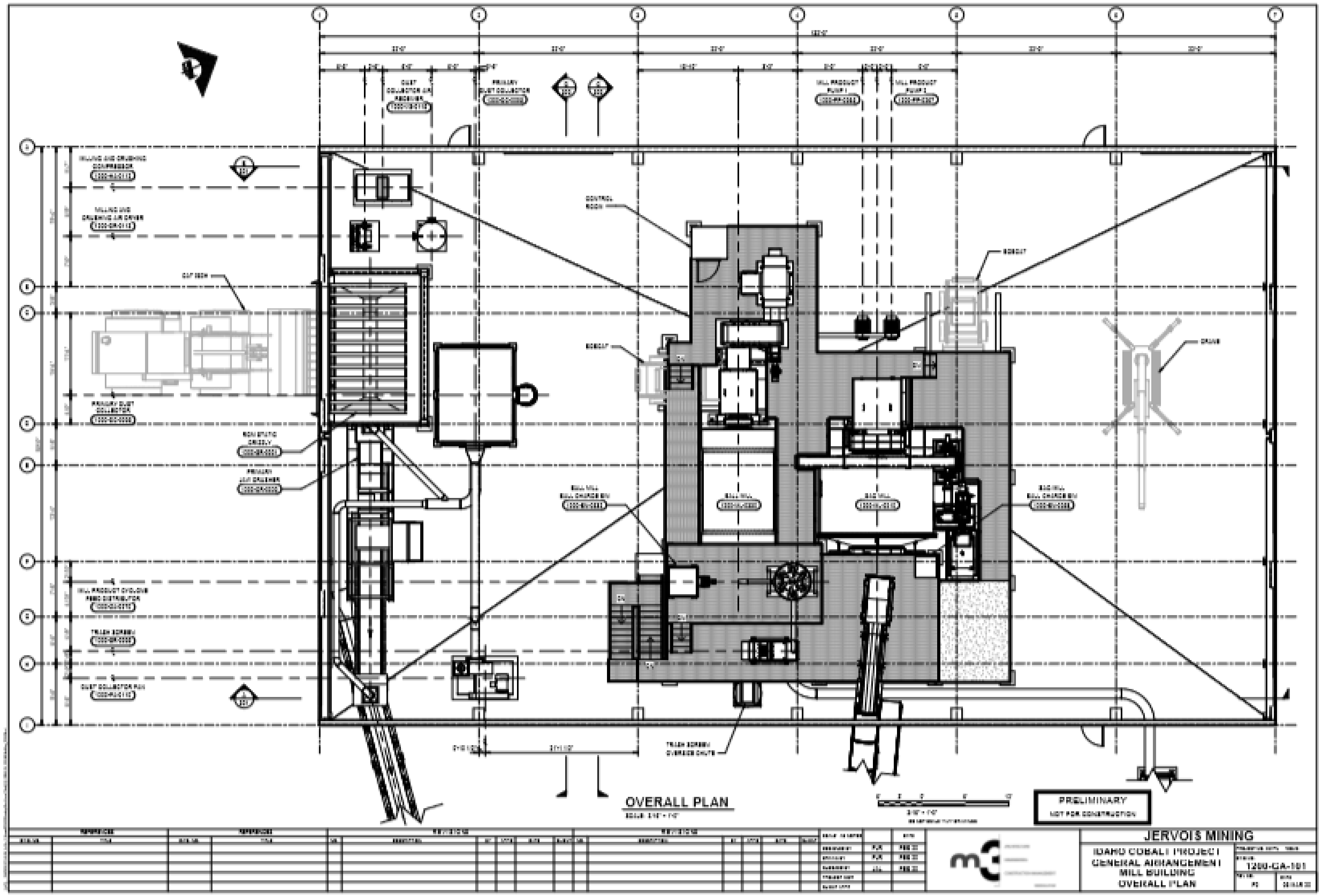


Figure 18-5: ICO Milling General Arrangement

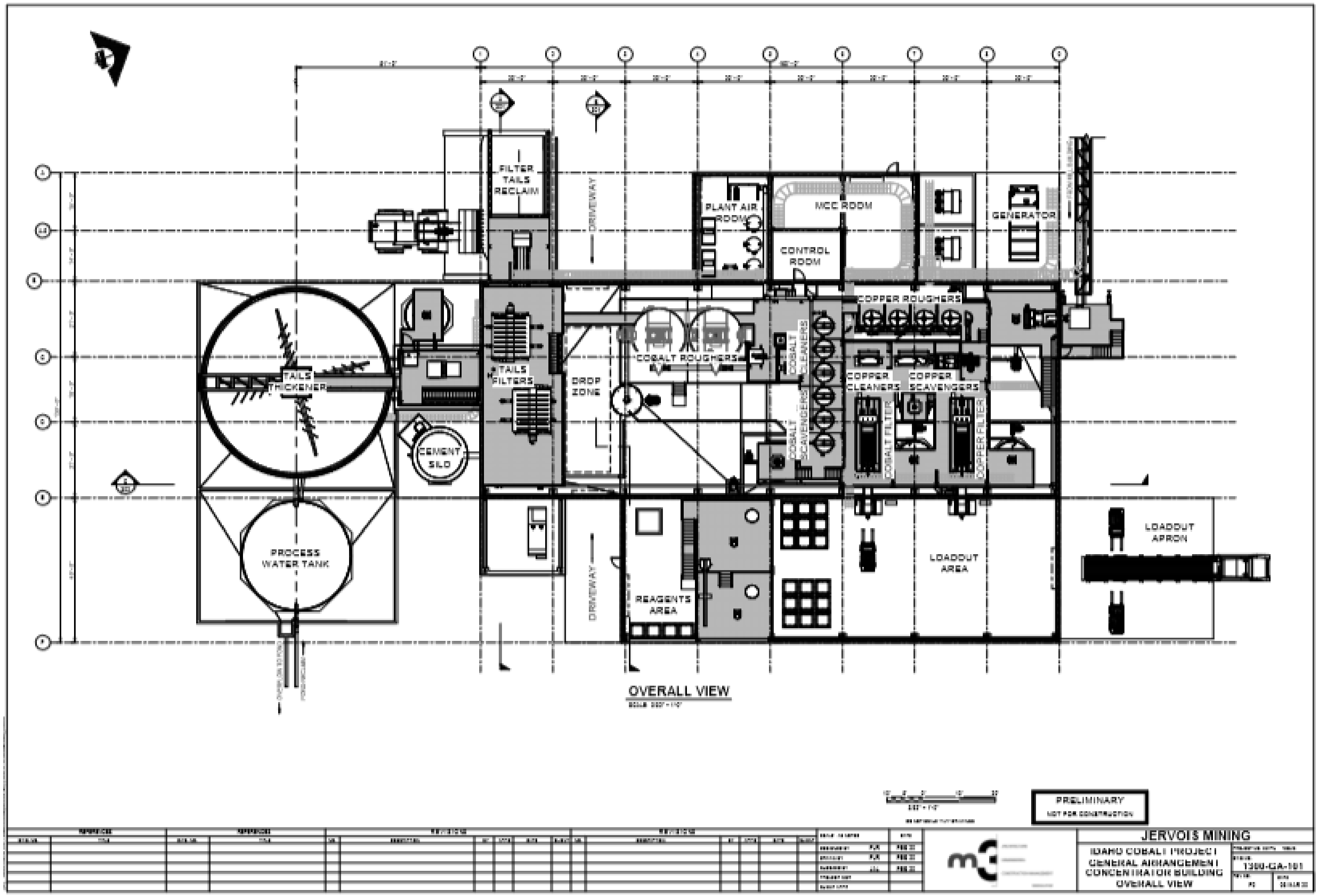


Figure 18-6: ICO Concentrator General Arrangement

18.3 ACCESS AND SECURITY

Site access through public land is provided via a combination of previously constructed roads and upgrades of existing roads. All roads are constructed or upgraded in accordance with United States Forest Service ("SFS") guidelines for road construction. The specifications for road design are covered in the Feasibility Design of the Tailings/Waste Rock Facility, and Roads for the Idaho Cobalt Project (Telesto, 2007c) and Transportation Baseline Report and Transportation Plan (TTE, 2006).

18.3.1 Mine Site Access Roads

Vehicle access to the ICO is via a series of well maintained, public access roads which are open year-round. Access to the road is approximately 6 miles south of Salmon Idaho on Highway 93 which also services the Blackbird mine (no longer in operation). The total distance from the Salmon depot to the ICO is approximately 48 miles.

This route will also be used for transportation of concentrate, equipment, reagents, and other freight.

18.3.2 Design Features

Specific design elements for the road design include:

- A minimum road width of 15 feet
- Two-per cent canter toward the ditch on the uphill side of the road in steep terrain or two per cent out slope without ditches in the Big Flat area
- Cut slopes of 0.8H:1V, fill slopes of 1.5H:1V.
- Maximum grade of 12 per cent on access roads, the maximum grade of 10 per cent on the portal and haul roads
- Design primary roads for 40-ton trucks (80-ton gross weight)
- Subbase thickness of 12 inches and aggregate surface of six inches
- Switchback turn radius that can accommodate an AASHTO WB-40 truck
- Secondary and tertiary road design for light-duty vehicles
- Stormwater control ditches and Best Management Practices ("BMPs")

18.3.3 Site Roads

All new site roads are constructed in conformance with Mine Safety and Health Administration ("MSHA") regulations, as appropriate. Existing haul roads are improved as necessary. Stormwater ditches and sediment control measures on all roads are constructed in accordance with BMPs for Mining in Idaho guidelines (Idaho Department of Lands [IDL], 1991) to control stormwater runoff.

These are more fully described in the Storm Water Management Plan for the Idaho Cobalt Project (Telesto, 2006c).

Site roads are classified as primary, secondary and tertiary roads. Primary roads are all main access roads and roads over which there will be ore or waste rock haulage. Secondary roads are roads that will see daily, year-round use in the operations but are not ore haul routes or main access routes. Tertiary roads are roads that will see only seasonal use or intermittent use, such as roads to water monitoring locations.

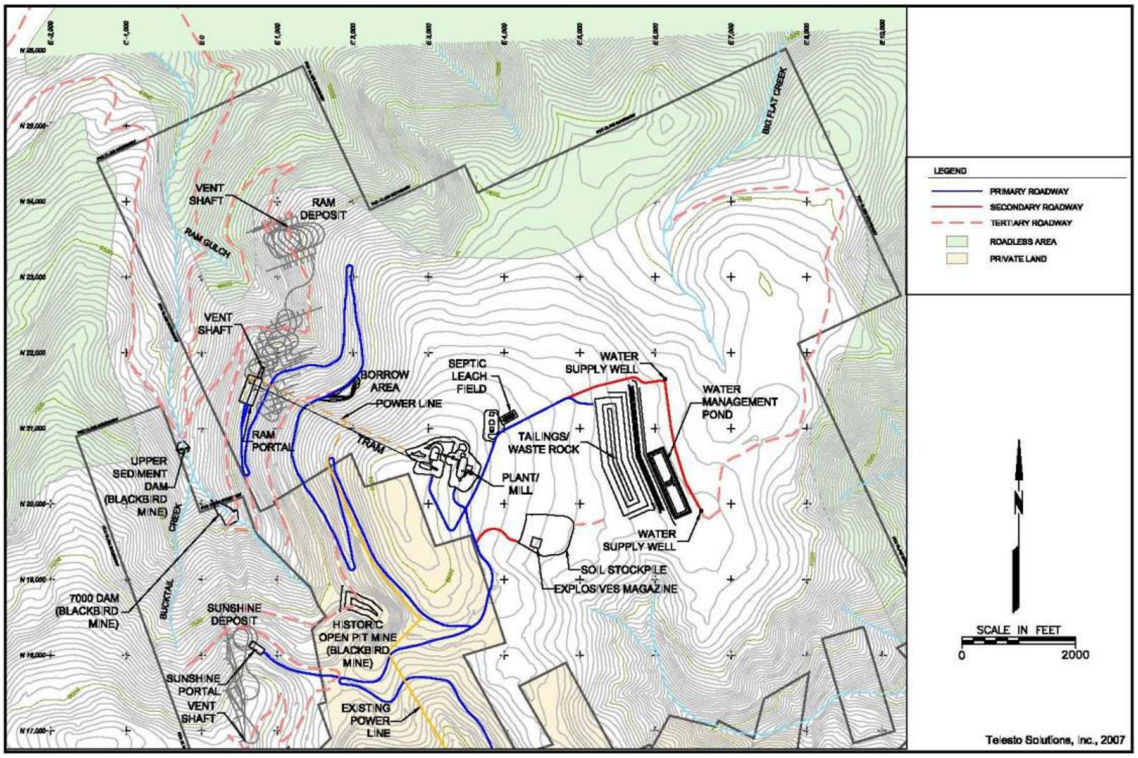


Figure 18-7: ICO Facilities and access roads

18.3.4 Mine and Concentrator Security

Access to the site is through the existing Blackbird property and is controlled by a guardhouse located near the intersection of the Blackbird property boundary with the Forest Service road. The guardhouse will be manned on day shift five days per week. All visitors to the site must sign in, receive ICO-specific site hazard awareness training, and directions to the appropriate site facilities, which are located approximately 4.3 miles further along the road. Site personnel will receive notification of the impending visit.

The mine and concentrator are located in a very sparsely populated area of Idaho 1.5 hours away from the nearest city.

Employees are bused or transported in company vehicles. No private vehicles are allowed on the mine site. This policy reduces the opportunity for theft by the employees. The requirement for site security is considered to be minimal because of the location, the very low traffic through the area, and the low value of the product produced.

Warehousing of materials and supplies is primarily inside the mill building or inside the process building. Both facilities will be permanently staffed. Security fencing at the site is limited to specific areas, which are fenced to prevent personnel access to hazardous areas, such as the substation transformer yard. In addition, some areas such as the water management ponds are fenced to prevent wildlife access. Fencing for the ponds protects wildlife and also protects the synthetic liner from wildlife damage.

18.4 BUILDINGS

The process facility and ancillary buildings include the following:

- Mill Building (purchased and stored at the Salmon Depot)

- Concentrator Building (purchased and stored at the Salmon Depot)
- Control Room (enclosure within the Concentrator Building)
- Sample Prep Room and Laboratory (enclosure within the Concentrator Building)
- Administration Building (already constructed)
- Miners Dry/Change House
- Security gatehouse (already constructed)

The two main facilities at the mine site are two separate Sprung® structures, which house the comminution (crushing and milling), and the concentrator (flotation, tailings and concentrate filtration and reagents and utilities). Additionally, there is a small guardhouse that is located near the intersection of the Blackbird property boundary and the access road.

The mining contractor is responsible for warehousing of the necessary mining supplies. The mine warehouse consists of a room excavated within the mine near the portal. Explosives storage is also provided in the same manner. The mining contractor is also responsible for maintenance of its equipment, and the maintenance functions are carried out inside the mine or on the portal bench. The mining contractor's office facilities are self-provided and are located on the portal bench near the mine portal.

The septic system is sized to handle the effluent from both change facilities and the concentrator. A separate septic system will be provided for the camp.

A fuel and lube storage and dispensing facility are located near the concentrator. Fuel and lubricants are distributed around the site, as required, by a fuel and lube truck.

18.5 CAMP

The scope of work includes the supply and construction of a prefabricated camp for use during construction which will transition to an operations camp for the mining contractor as well as the Jervois operations team.

The required camp size was calculated based on the required manning estimates for the various mining, concentrator and owner labor requirements, in consideration of the various leave rotations.

Estimated number of rooms required	88 rooms
Contingency	7 rooms
Estimated camp capacity	95 rooms

The envisaged camp will consist of prefabricated accommodation units as well as kitchen and dining facilities arranged on the current topsoil storage area. The scope also includes utilities and services required for camp operation.

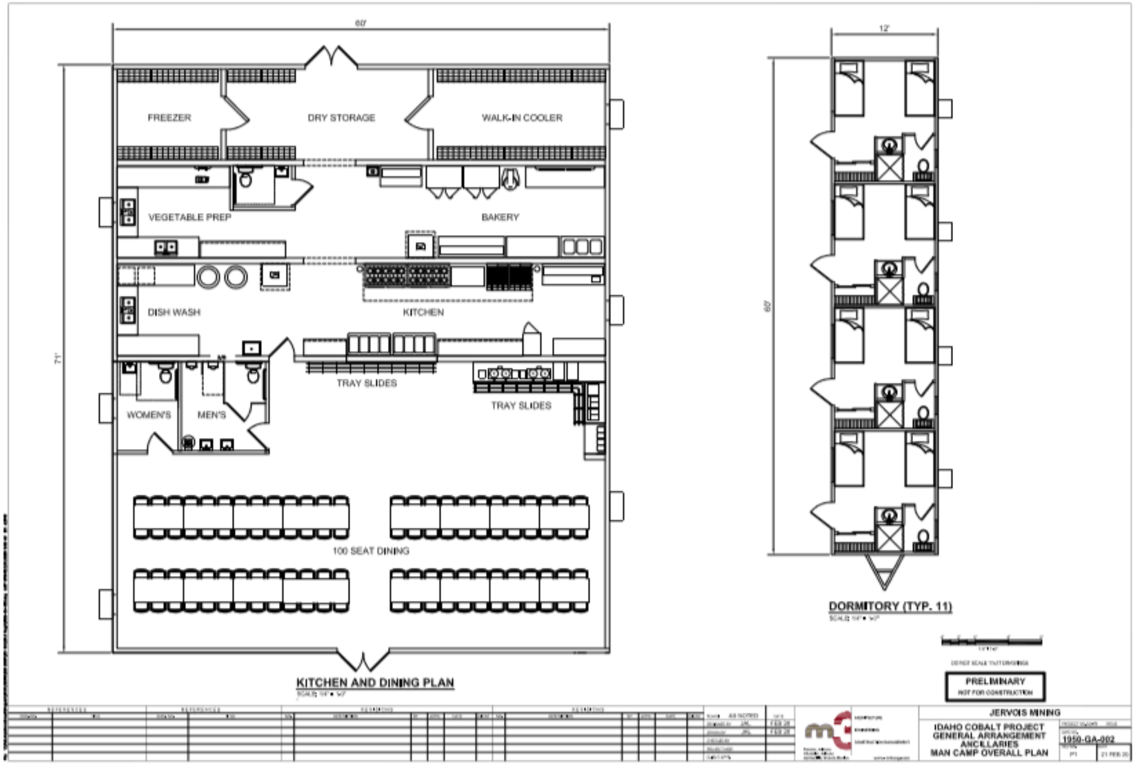


Figure 18-8: ICO Camp Accommodation units and Kitchen/Dining Facility

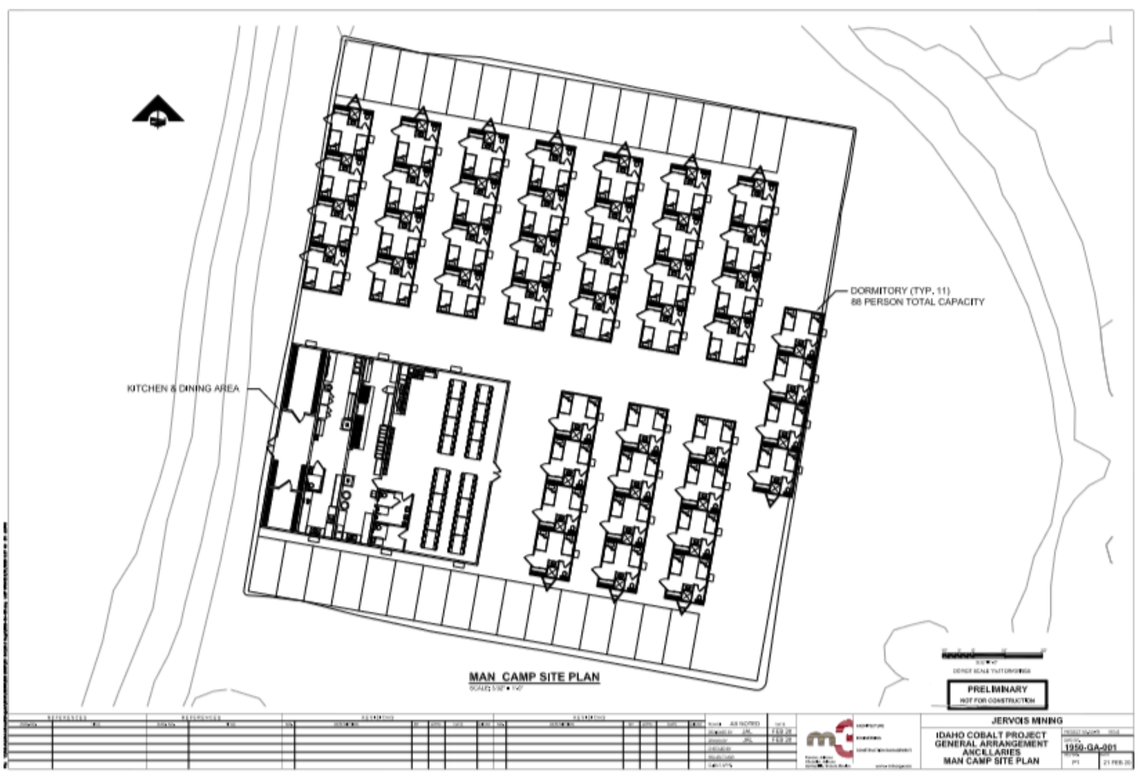


Figure 18-9: ICO Camp General Arrangement

18.6 COMMUNICATIONS

Administrative functions for the mine and concentrator are performed primarily from the existing Jervois office in Salmon Idaho. This includes senior management, human resources, accounting, payroll, accounts payable and procurement. The office is currently connected to all required data and voice communications networks. Mobile phone service is available in Salmon.

Communications to the Mine and Concentrator site is via a radio system and cell phone repeater to allow connection to the cell phone network.

Communications in the mine are via radios and a leaky feeder system. Surface communications is primarily by hand-held, base-station or vehicle-mounted FM radios.

18.7 ELECTRICAL POWER SUPPLY AND DISTRIBUTION

The project mine site is supplied by a 69-kV power line provided by Idaho Power Company. The distribution line and HV power installation, were completed in 2018, has a capacity of 5.5 MW and is expandable to 7.5 MW. The overhead power line to the ICO taps into the Salmon-Blackbird line to the new substation located near the concentrator. The new section of the overhead line is approximately 2.8 miles from the new tap point to the concentrator. This line runs parallel to the mine site access road to the concentrator.

18.7.1 Site Power Distribution

Transformers located at the concentrator substation reduces the voltage to 4.16 kV for further power distribution. The SAG and Ball Mill main drive motors will operate at 4.16 kV. All other loads operate at 480 V, except for lighting, instrumentation and other small loads. A 4.16 kV supply will also feed the concentrator substation and mine portal area. Transformers located at the portal reduces the voltage to 480 V for distribution within the tram loading facility. Power distribution within the mine will be at 4.16 kV.

Power for the water management pond pumping system and the surface-mounted mine vent fans are provided by 4.16 kV overhead power lines that run from the concentrator to the point of consumption. Transformers located at the consumption points reduce the voltage to 480 V to supply the equipment.

18.7.2 Power Requirements

The overall site power demand is based upon completed electrical load analyses for the Process Plant, Mining, and site infrastructure. As part of the feasibility study the following work was also completed:

- Single line diagrams for the entire plant
- Load analyses
- SKM model for load analysis
- Assessment of existing electrical equipment for new mechanical equipment requirements
- Assessed transmission line for power requirements

Table 18-1: ICO Power Requirement

Area	Utilized Load (kW)
Admin. Bldg.	10.2
Clean Water	19.0
Concentrator	1588.5
Man Camp	46.6
Mill	1629.1
Mine	1107.7
Water Mgmt.	59.8
Water Treatment	400.0
Total	**4861.0**

18.8 SURFACE FACILITIES FIRE PROTECTION

Neither local building codes nor the Jervois insurance carrier requires a permanent fire protection system. Fire protection for the surface facilities is provided using a combination of hand-held (20-pound) fire extinguishers and wheel-mounted (120-pound) fire extinguishers. Fire extinguishers will be appropriately located near equipment, particularly conveyor belts, in maintenance areas, motor control centres and other areas that may be prone to fire.

All vehicles and mobile equipment are equipped with fire extinguishers.

Forest Service regulations require that fire suppression equipment be available during the fire season. A water truck equipped with pump, hoses and nozzles is sufficient to meet this requirement. Shovels and other tools for suppressing small fires will also be stocked and made readily available for employees.

In addition, ICO will coordinate with the Forest Service and local fire districts to provide any necessary fire prevention and suppression training for mine and process plant employees and also to devise and maintain a quick alert and response system in the event of any fire that requires outside assistance. ICO will organise and conduct periodic fire drills according to applicable regulations and/or recommendations of local fire officials.

ICO will create an implement a fire prevention plan for any surface blasting events to include appropriate area closures, stationing of fire suppression equipment, and advising local fire authorities.

18.9 WATER SUPPLY, TREATMENT AND DISCHARGE

The use of external make-up water has been minimised as part of the process plant design. Process water is recovered within the circuit using thickeners and filtration unit operations. The two concentrate thickeners (copper/cobalt) allow for the recirculation of process water within the copper and cobalt cleaner flotation circuits. Only a marginal offtake is required from the tails thickener process water for concentrator make-up, allowing most of the water to be recirculated back to the mill circuit. Make-up water from the Water Management Ponds is pumped, when required, up to the tails thickener process water tank when plant make-up is required.

Clean (fresh) water will be withdrawn from local boreholes which will supply the fire water tank, clean water tank and potable water tank. The tanks have sufficient volume to meet the instantaneous process demands of the plant. The clean water tank feed nozzle will be placed mid-way up the fire water tank wall to ensure a reserve volume is always available for fire suppression.

The discharge of paste backfill from the plant allows for a continuous bleed of water. This will prevent a potential buildup of salts within the process which could impact on flotation performance. As further mitigation, if required, the Water Treatment Plant can be used. It is still uncertain as to the long-term effects of salts build up on flotation performance.

18.9.1 Mine Dewatering

Discharge from the mine dewatering system is delivered to a holding sump which is sized to contain the entire backflow from draining the pipeline from the Ram portal to the mill on the Big Flat.

Pumping from the portal to the mill is accomplished via a steel pipe with secondary containment. This pipe is about 2,300 feet long to reach the 8,050-foot high point at the mill site. An air intake with a check valve at the high point allows the line to self-drain in the event of a pump shutdown.

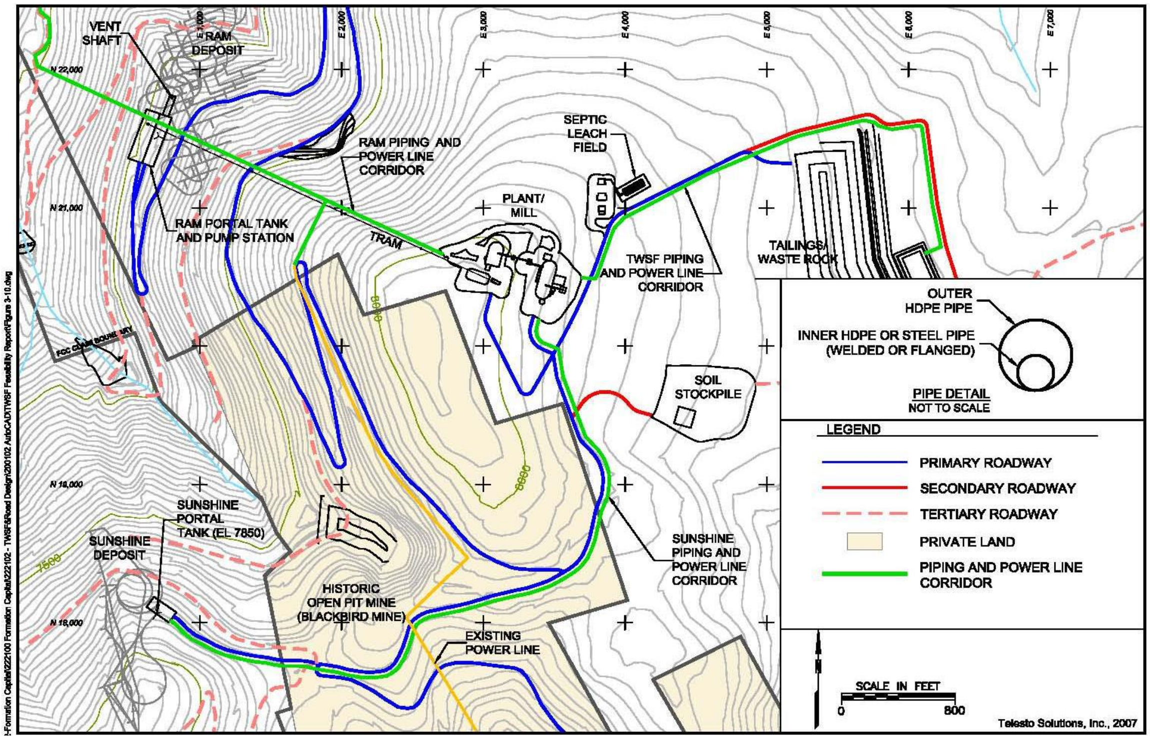


Figure 18-10: Mine Dewatering System Piping and General Arrangement

The pipe is fully winterised to prevent freezing. During an emergency shutdown or production curtailment, the mine pumps continue to operate in order to maintain the entire water balance and control system.

18.9.2 Water Management Pond

A key component of the project and the water management program is the water management pond, which collects drainage from the TWSF and stores mining and process solutions as needed. The water management pond is sized to contain all process and mine drainage waters, and all drainage waters from the TWSF. Additional capacity is required to contain runoff from the 500-year hydrologic condition.

The pond has a total required capacity of 10 million gallons, which is contained within two hydraulically connected cells, each capable of holding 5 million gallons of water. Two feet of freeboard is added to the pond above the level required to maintain 10 million gallons of water, resulting in a total capacity of 12.5 million gallons. The water management pond is designed to fully contain all waters from the operation, other than stormwater and sanitary discharges. The pond cells are hydraulically connected by overtopping the intermediate berm so that water can flow from one cell to the other if the water volume conveyed to either cell exceeds its operating capacity at any time. Each cell is constructed with a sump from which water can be pumped to either cell or the treatment plant. The pipeline from the pond to the mill is double-contained and complete with leak detection at all low points and pipe-to-pipe connections.

The liner design for the water management pond consists of a double synthetic liner system with leak detection and leak collection. The synthetic liner system consists of a primary 80-mil high-density polyethylene ("HDPE") liner over a secondary 40-mil HDPE liner with an HDPE geonet between the two liners. Liner details are shown on the drawings. The 80-mil thickness was selected for the primary HDPE liner due to the potential requirement to withstand damage from ice formation. The liner system is constructed over a subgrade that is graded and smoothed as needed to create a suitable foundation for the construction of the liner system.

The water management pond is designed with a leak detection system. The HDPE geonet placed between the 80-mil primary and 40-mil secondary liners will convey any water entering the system to a gravel collection sump located at the low end of each pond. Riser pipes are used to check the leak detection sump for water and to pump out the leak detection sump as needed.

Additional design details can be found in Section 8 and Section 10 of this Report as well as the Feasibility Design of the Tailings/Waste Rock Facility, and Roads for the Idaho Cobalt Project (Telesto, 2007c) and Conceptual Design of the Tailing/Waste Rock Facility and Water Management Ponds (Telesto, 2006b).

Specific design elements of the water management pond are:

- Seasonal storage for process waters, process influenced stormwater runoff, and drainage collection from the TWSF
- Contingency storage for the 500-year hydrologic condition with an extra two feet of freeboard
- Double synthetic liner systems with leak detection
- Geosynthetic and gravel underdrain system to intercept shallow groundwater
- Influent and effluent management
- Geotechnical stability (criteria for minimum factors of safety under static and pseudo-static conditions were analysed at 1.5 and 1.15, respectively)

18.9.3 Water Treatment

Based on the process flow diagram outlined in Figure 17-1, the water treatment system expects flow inputs associated with mine dewatering, the milling process, and the water management pond that receives drainage from the TWSF.

18.9.3.1 Water Treatment Design and Capacity

Water management is based on operating a water treatment plant and releasing water in accordance with a National Pollutant Discharge Elimination System ("NPDES") permit in conjunction with temporary storage in the water management pond. The water treatment plant has the ability to treat up to 150 gallons of water per minute for discharge through the NPDES outfall. Except during periods of very high inflow, the water treatment plant treats incoming water on an as-received basis, with very little water stored in the water management pond. During periods of high inflow, water will accumulate in the water management pond for treatment during lower inflow periods. Mine water is predicted to contain elevated concentrations of nitrate, sulphate, and metals (aluminum, cobalt, copper, iron, manganese, zinc).

18.9.3.2 Water Treatment Plant Feed Water Quality

Mine water quality is predicted by the Dynamic Systems Model ("DSM"), which includes potential chemical mass loading effects associated with:

- Wall rock weathering
- Waste rock weathering in the stope access slash
- Drainage from the cemented paste backfill tailings

- Ambient groundwater inflow

The projected mine water chemistry results from the relative chemical mass loading from these four potential sources. Table 18-2 summarises the predicted water treatment feed water chemistry based on the maximum projected concentrations as modified by equilibrium geochemical modelling using the PHREEQC code.

18.9.4 Water Treatment Design

Water treatment at the ICO is implemented with the objective of producing the highest quality discharge stream that is reasonably achievable. A secondary objective is to operate the system with as close to a zero liquid-waste discharge condition as possible.

The water treatment plant will incorporate the following treatment methodologies:

- Metals removal by oxidation, precipitation and filtration/settling (co-precipitation),
- Metals polishing removal by cation ion exchange (IX), and;
- Nitrogen removal biological denitrification (biodenitrification) via moving bed bioreactor (MBBR).

Table 18-2: Projected Water Treatment Feed Water Chemistry

Parameter	Mine Water	Locked Cycle (mg/L)	Process Pond
Calcium	38	60	53
Magnesium	59	6-12	77
Potassium	165	10-110	208
Sodium	120	5-125	145
Chloride	1.5	25	0.13
Sulphate	378	200-750	790
Alkalinity	-	10	-
Fluoride	0.20	<0.5	0.10
Nitrate-N	18	10-50	54
Silica	5	10-30	5
Thiosulphate	-	<1-40	-
Xanthate	-	<1-30	-
Aluminum (Al)	100	56-137	150
Silver (Ag)	-	<0.1	-
Arsenic (As)	12	6-7	200
Boron (B)	-	42-54	-
Barium (Ba)	-	2-5	-
Bismuth (Bi)	-	<0.3	-
Cadmium (Cd)	-	<0.1	-
Cobalt (Co)	54	1-3	594
Copper (Cu)	35	1.3-3.9	28
Iron (Fe)	1,750	<20-40	<1
Mercury (Hg)	-	-	-
Lithium (Li)	-	<5	-
Manganese (Mn)	6,000	20-70	5,000
Molybdenum (Mo)	-	18-25	-
Nickel (Ni)	2.2	2-4	5
Lead (Pb)	-	<0.20-0.30	-
Antimony (Sb)	-	0.7-1.0	-
Selenium (Se)	-	8-16	-

Parameter	Mine Water	Locked Cycle (mg/L)	Process Pond
Tin (Sn)	-	<1	-
Strotium (Sr)	-	27-42	-
Titanium (Ti)	-	<3	-
Thallium (Tl)	-	<0.2	-
Vanadium (V)	-	<0.9-1.0	-
Zinc	37	2-4	53

18.9.4.1 Co-precipitation

Influent process water will undergo chemical treatment and equalisation in the raw water storage tank. Sodium hypochlorite (NaClO), Ferric Chloride ($FeCl_2$) and Sodium Hydroxide (NaOH) will be added to promote co-precipitation of heavy metals and dissolved metals that may interfere with the IX system. NaClO is added to assist in the removal of oxidised Arsenic. From the raw water storage tank, a polymer coagulant will be added to the process water entering the lamella clarifier to promote flocculation and settling of suspended solids. The clarifier underflow will be pumped to a cone bottom sludge thickener. An antiscalant will be added to the decant water as it exits the clarifier and the process water will be routed to a cone bottomed clarification tank. Sodium Bisulphite ($NaHSO_3$) will be added to the water just prior to the clarification tank to remove any residual chlorine.

Hydrochloric acid will be added to the water to reduce the pH to 4.0 to promote chelation of metals then pass through a bag filter and granular activated carbon cartridge filter to remove any remaining chlorine residual prior to entering the IX system.

Effluent from the clarifier will be stored in a polished water tank where it will be analysed for reuse, recycle or discharge. Bag and cartridge filters and dewatered solids will be transported to the TWSF.

18.9.4.2 Cation Ion Exchange System

A Cation Ion Exchange (IX) system will be used to remove cobalt, copper, zinc and ammonium which will then feed the MBBR.

Neutralisation water from the IX process will be stored in a backwash water storage tank and slowly fed into the thermal evaporator for reduction. Concentrated neutralisation waste will be delivered annually, off-site to an appropriate disposal facility.

18.9.4.3 Biodenitrification

An above-ground Moving Bed Biofilm Reactor (MBBR) unit will be installed, in which methanol will be used to facilitate growth of heterotrophic microorganisms to consume oxygen from the nitrate ion, releasing nitrogen gas. Reaeration will then be used to restore the dissolved oxygen levels to above the required limit prior to release.

Suspended solids generated from this step will be sent to a cone bottom clarifier for removal. Airlift pumps will be used to remove the suspended solids from the cone which will be dewatered in a reusable dewatering waste container. The sludge generated in the process will be collected and disposed of on in the local landfill.

18.9.5 Treated Water Discharge

The final effluent from the treatment plant will be discharged into Big Deer Creek immediately downstream of WQ-24a in accordance with the NPDES permit which is to be issued by EPA. The discharge pipeline is buried except for small areas where it is brought out of the ground for short distances in order to avoid disrupting the wetlands by burying it there. Vacuum breakers are installed along the pipeline, as needed.

18.10 WASTE ROCK AND TAILINGS STORAGE

A single surface disposal facility is used to store both the tailings from the concentrator and the waste rock material, the TWSF. This facility serves to minimise the area of disturbance by sharing containment and drainage collection facilities while providing storage for these materials.

The TWSF is located east of and downslope from the mill on the Big Flat. This location was chosen as the best site for the facility in the project area because of its relatively flat topography, avoidance of jurisdictional wetlands, soil characteristics, and distance from active drainages and streams.

Specific design elements of the TWSF are:

- Storage of 800,000 tons of waste rock and 960,000 tons of tailing, based on production estimates for the feasibility study, with separation of tailing and waste rock to the extent practicable
- 2.5H:1V interbench side slopes.
- Geomembrane liner system with drainage collection.
- Diversion of runoff around operating areas of the facility.
- Collection of and conveyance of runoff and seepage from tailings and waste rock to synthetically lined water management ponds
- Collection and conveyance of shallow groundwater flow to wetland mitigation ponds
- Toe berm to provide geotechnical stability and stormwater control
- Occupies an area of about 36 acres.

Tailings and waste rock are separated except for a commingled zone at the interface of the two materials. The TWSF is designed with a balanced cut-to-fill grading plan, including a toe berm that will be constructed to provide downstream drainage control and enhance the structural stability of the facility. Perimeter berms will be constructed to provide containment, act as internal stormwater channels, and prevent storm runoff from inundating the facility.

The liner system for the TWSF consists of a 40-mil PVC synthetic liner placed over a geosynthetic clay liner (“GCL”). The colluvium subgrade is graded and compacted in place to create a suitable foundation for liner construction. A PVC liner was selected for use in the TWSF based on chemical resistance, puncture resistance, flexibility and liner/soil interface friction angle. The liner thickness is based on expected subgrade and loading conditions, which in this particular case have greater relevance to liner thickness requirement than the expected differential settlement or tensile strength. The TWSF liner design includes a GCL or compacted clay layer below the PVC; thus, a PVC liner thickness of 40-mil is sufficient. An 80-mil HDPE rubsheet will be placed over the primary PVC liner on the perimeter of the TWSF to protect the primary liner from UV and physical damage. Details of the TWSF liner system are shown in Figure 18-13.

A drainage collection system will be constructed over the PVC liner to collect water infiltrating through the tailing and waste rock. The drainage collection system consists of a series of perforated pipes connected to a header pipe that conveys flow to the water management pond. The drainage collection system will be constructed within a protective sand layer, which also acts to protect the PVC liner from damage during tailing and waste rock placement.

A general layout of the TWSF and water management ponds is shown in Figure 18-11 and Figure 18-12. Detailed design information, including geotechnical stability analysis for the TWSF and water management ponds, can be found in the Feasibility Design of the Tailings/Waste Rock Facility, and Roads for the Idaho Cobalt Project (Telesto, 2007).

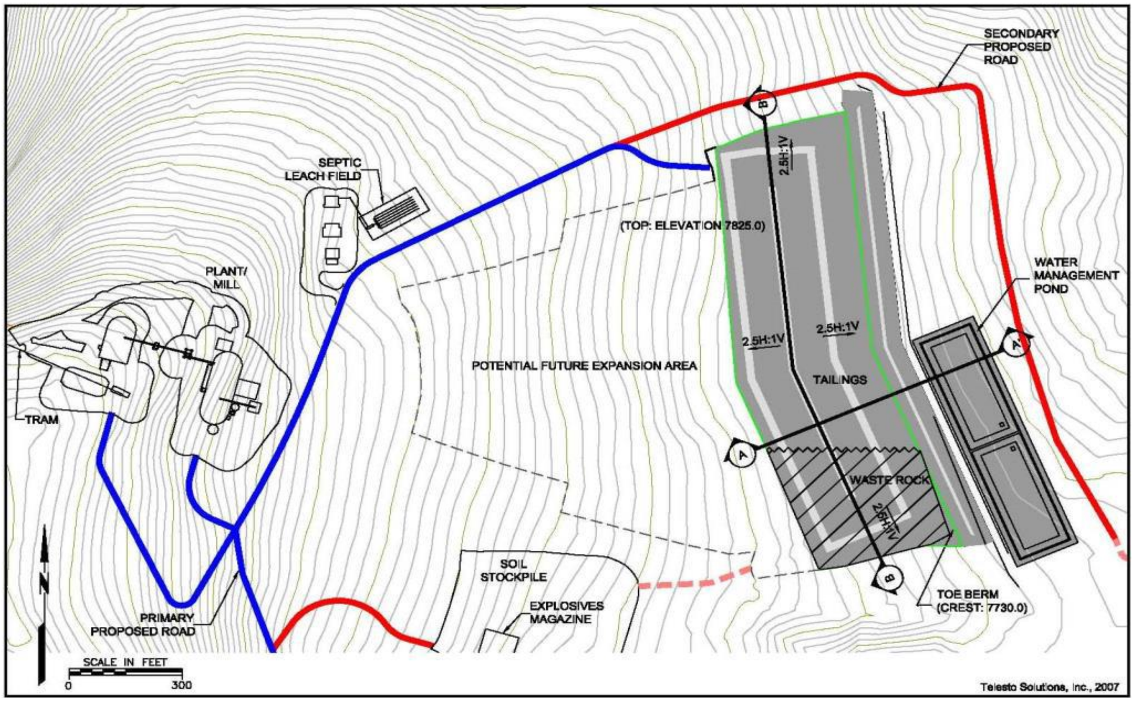


Figure 18-11: TWSF Plan View

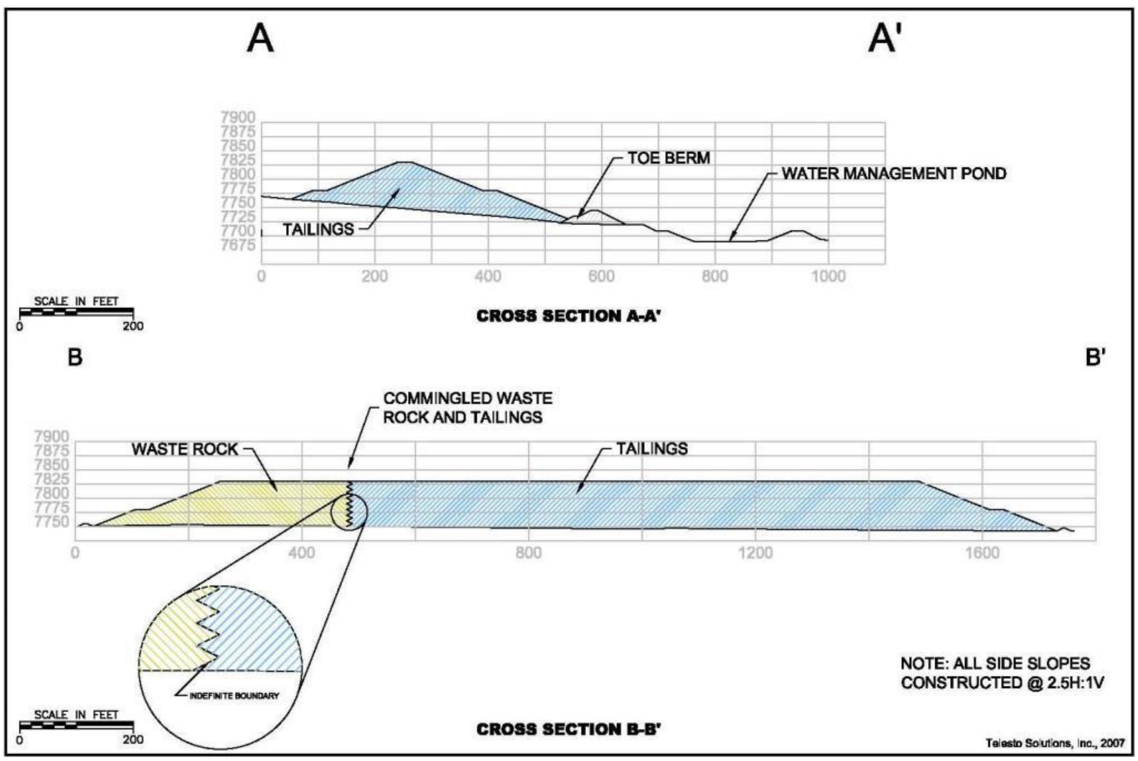


Figure 18-12: TWSF Elevation

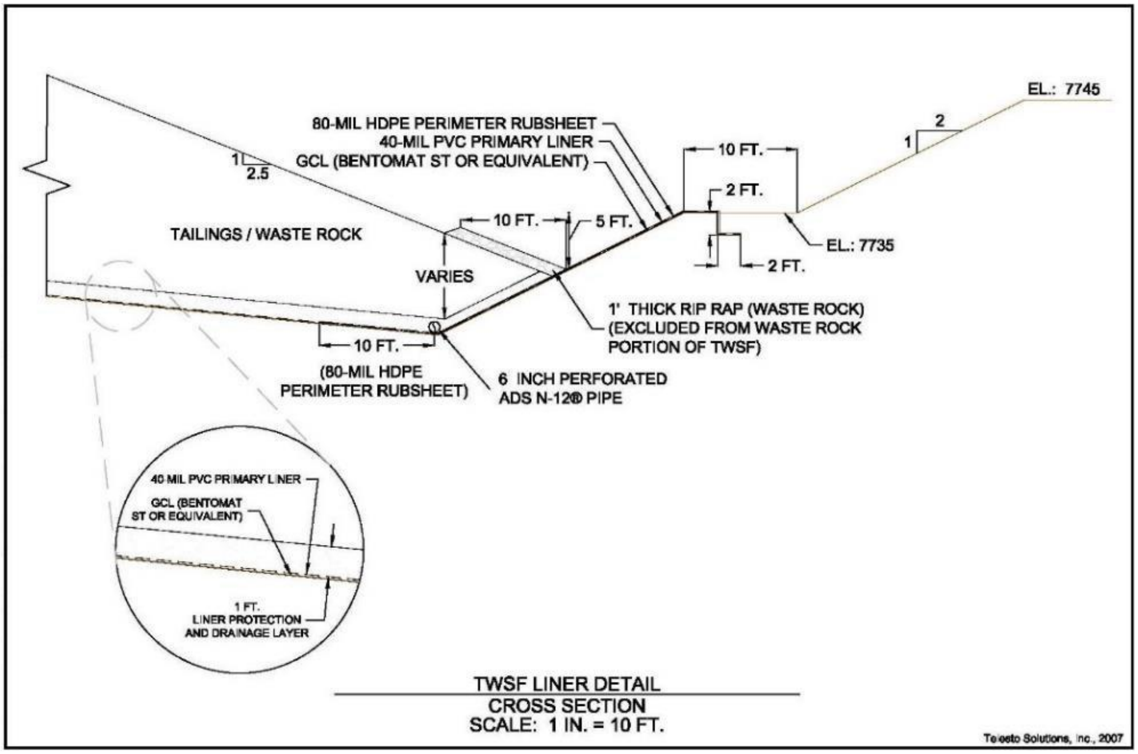


Figure 18-13: TWSF Liner Detail

18.11 TAILINGS HANDLING AND DISPOSAL

Tailings are dewatered with a thickener and vacuum filter. The final filter cake contains approximately 80 per cent solids after moisture conditioning. Filter cake that is not used as backfill is trucked from the dewatering facility and end dumped into the TWSF. For the paste backfill, the tailings are mixed with water and cement and pumped to the mine for use as backfill as outlined in Section 16.

The tailings are levelled and shaped in lifts that have a maximum depth of two feet by a small tracked dozer. Compaction of the tailings to 90 per cent standard Proctor density is achieved by the truck and dozer traffic on the pile. During winter operations, the working areas will be kept small. Snow will be removed from the working area and placed in the snow removal area. The tailings will be spread quickly and compacted following delivery to the TWSF. Freezing of the tailings after compaction is not a concern; however, care must be exercised to prevent incorporating snow and frozen tailings into the pile prior to compaction.

18.12 WASTE MANAGEMENT

18.12.1 Waste Rock Handling and Disposal

Waste rock is delivered to the TWSF by truck from the mill area. It is placed in 5-ft lifts on the prepared surface by mine trucks that use end dumping. The waste rock is spread and levelled with a dozer. Equipment traffic will compact the waste rock during construction.

18.12.2 Topsoil Stockpile

A topsoil stockpile has already been established on the Big Flat for the storage of growth media salvaged during earlier construction phases. The area is located adjacent to the TWSF. Precipitation runoff is diverted around the area by perimeter ditches.

A section of this area will be set aside for the ICO Camp and infrastructure while the remainder will be seeded to stabilise the stockpile.

18.13 EXPLOSIVES STORAGE AND TRANSPORT

Explosive will be delivered by the powder manufacturer/distributor to the designated surface explosive storage facility at the ICO. Explosive requirements for the underground operation will then be transported by designated underground explosive vehicles certified to transport explosives to the underground storage facilities.

From the underground storage facilities, explosive will then be distributed to the working areas with similar designated vehicles.

18.14 SALMON DEPOT

18.14.1 Site Layout

ICO has purchased the Salmon Depot which is located approximately one mile south of Salmon, Idaho along Highway 93. The site is approximately three acres in total area with a single warehouse building at site, with a compacted gravel base across the entire property. The depot will store concentrate from the mine site and provide temporary storage for any partial shipments received prior to transport to the mine site, to minimise traffic along the access road.

18.14.2 Access and Security

The depot is located immediately adjacent to the highway and is surrounded on all sides with a six-foot chain-link fence with lockable gates. All man doors are steel insulated with locking deadbolts, and all overhead doors are steel insulated providing protection from any potential break-ins. The yard is visible from the highway, and any activity in the depot would be readily seen from the highway. The mobile equipment required at the depot will be parked inside the warehouse each night to reduce the opportunity for theft and vandalism. High-value items are not anticipated to be stored at the facility, which further reduces the risk.

18.14.3 Bus Transport

All employees and contractors not accommodated in the ICO Camp will gather at the Salmon Depot every morning for transport to the mine site. No personal vehicles will be allowed to travel to the site. The transport will be operated by a local transportation company which will be selected through a tender process prior to construction and operation.

18.14.4 Power Supply and Distribution

A 240V power distribution system is installed to the building with several receptacles located along with the interior of the building. Interior lighting is installed and is appropriate for a warehouse environment.

18.14.5 Waste Management

General waste disposal and recycling bins will be kept at the depot to disposal of any solid waste generated at site. A contract will be established with a local waste disposal company for the collection of all waste streams.

19 MARKET STUDIES AND CONTRACTS

19.1 COBALT MARKET OVERVIEW

Historically cobalt use has focused on alloys, superalloys, specialty metals, and chemicals. From a technical perspective, cobalt's key properties include ferromagnetism, hardness, and wear-resistance when alloyed with other metals; it has low thermal and electrical conductivity and a very high melting point. These properties make it an essential ingredient in a number of important alloying applications where high temperature service, high mechanical stress and surface stability are critical. Cobalt's magnetic properties also enable the production of permanent and soft magnetic alloys which are utilized in a variety of applications. Furthermore, cobalt's resistance to corrosion and wear and its thermal expansion properties are used in other specialist alloys which are applied in the making of cutting tools and wear-resistant components.

More recently the cobalt market has undergone a remarkable transition from these traditional end use segments of alloys and tooling (which utilize its high temperature and oxidation resistance) across to battery cathode chemistry – particularly in the transportation and personal electronics industries. As such cobalt markets have become increasingly influenced and dominated by the rise of lithium ion batteries, to which it is a key input.

Cobalt's performance, stability, and ability to mitigate thermal runaway (which, if not managed, can cause fire) has made it a critical and difficult-to-replace component in the cathodes of lithium ion batteries. In 2020 over 60% of cobalt is expected to be used in batteries, principally as a stabilizing agent in lithium ion cathodes. Market analysts expect this percentage will continue to rise.

Initially lithium ion battery use was focused on personal electronic devices, such as cell phones and lap-top computers. Increasingly, the electrification of the transportation sector, including automobiles, requires a rapidly rising number of large-scale lithium-ion batteries and, thus, increasingly significant amounts of cobalt.

The supply chain risk for cobalt is profound due to the domination of refined cobalt products by China (around 80% of global cobalt refining capacity), combined by over 75% of the world's cobalt production being mined in a single politically unstable country – the Democratic Republic of Congo ("DRC"). DRC's mining industry is now dominated by Chinese investment. Chinese investors extract cobalt intermediate materials from the DRC primarily in the form of cobalt hydroxide and ship them to mainland China for further processing.

At the onset of COVID-19 the DRC initially imposed mine lockdowns across the Katanga and Lualaba provinces, the key cobalt producing regions of the country, however they were rapidly lifted as Chinese producers in particular resisted closing sites due to health concerns. Producers locked in workforces and stockpiled consumables in anticipation of increased risk around supply chains in and out of the country. Traditional export routes through Zambia and South Africa (Durban port) were closed, with much cobalt subsequently exported via Tanzania (Dar es Salaam) and Kenya (Mombasa). COVID-19 did result in a reduction of supply outside the DRC, with Russia, Canada, Madagascar and Morocco all reducing output due to the virus. Major expansion projects under construction in the DRC (such as Mutoshi operated by Chemaf) were also paused.

Offsetting the interruption to supply was an unprecedented step change in demand across 1H 2020 as the world moved to protect populations from the worse potential impacts of COVID-19 via industry shutdowns and enforced lockdown of populations. Commercial air travel outside cargo ground to an effective halt, with aerospace alloys significantly impacted. Visibility on a recovery in aerospace demand is limited given the continued uncertainty over when travel will resume and at what intensity versus prevailing levels prior to the pandemic. With the collapse in oil prices, cobalt demand for catalysts was also adversely affected.

In the auto industry, whilst China and Europe/North America were adversely and significantly impacted by COVID-19 during Q1 and Q2 2020 respectively, plant utilization rates and consequential demand for raw materials has since recovered strongly. Electric vehicle ("EV") sales in Europe grew faster than anywhere else in the world and replaced much of the decline in alloys demand.

Whilst 2020 growth in electric vehicles expected in January this year will not be reached due to the pandemic, it is too early to determine the medium and long-term impact on electrification of global transportation from COVID-19. Governments in both Europe and China moved aggressively to buttress the transition with increased subsidies and economic incentives.

Despite the COVID-19 pandemic, Jervois estimates that approximately 30,000 metric tonnes of cobalt will be used globally in electric vehicles in 2020, out of a total global market of around 130,000 metric tonnes. Expectations are that the 30,000 metric tonnes in electric vehicles will rise to around 135,000 metric tonnes by 2025 and 270,000 metric tonnes by 2030, dwarfing all other cobalt uses and putting more and more strain on global supply chains.

Chinese government purchasing of physical cobalt and inventory build has supported prices, particularly in China where metal premia and arbitrage arose versus ex China prices. Cobalt hydroxide pricing also strengthened with reports of transactions occurring in the 80's as a percentage of Metal Bulletin Standard Alloy Grade ("SG") pricing, again a high level versus historical averages.

The United States has no cobalt mines or primary cobalt metal production. Once in production ICO will represent the United States' only cobalt mine. There is a limited degree of cobalt recycling in the superalloy and specialty steel sectors, but aside from insignificant levels of mined by-product, all cobalt used in the United States is originally derived from imported materials.

Excess cobalt hydroxide inventories in China, South Africa and the DRC continue to be run down, and market sentiment is improving. Supply side risk from DRC due to COVID-19 and other regional factors remains. Cobalt pricing has begun to recover since the July 2020 lows and was trading at US$15.75/lb (Metal Bulletin SG) as at the date of this filing, 11 November 2020.

Commodity research firms that specialize in cobalt use incentive pricing as their methodology. This methodology will generally result in a higher price than where banks reside as they must use a cobalt price to ensure projects are economically viable for their supply/demand models to balance.

Basis the current future cobalt price predictions and expectations of a demand/supply imbalance as electric vehicle growth expands, a cobalt price of US$25.00/lb was selected for the ICO BFS. This is consistent with or lower than the prices used by Jervois's listed peers in both Australia and Canada, broadly consistent with investment bank long-term consensus price and lower than that of leading independent forecasters.

The medium-term and long-term global outlook for cobalt is one of increasingly large deficits and shortages of physical supply. Jervois considers cobalt supply the key constraint on how rapidly the world will be able to electrify its commercial and personal transportation sectors. As such there is expected to be significant support for cobalt in the future which is forecast to result in a rebasing of cobalt pricing versus historical averages.

19.2 COPPER MARKET OVERVIEW

The largest consumer of global copper production is currently the electrical and electronic industry which consumes approximately 40% of supply. A market which has more than doubled since 2013. The second largest consumer is in building construction materials including roof coverings, plumbing, boilers and wiring which account for approximately 30 percent of world consumption.

Other important uses are transport, mechanical and machinery and consumer goods, each representing approximately 10 percent. Whilst construction will remain an important driver of future copper consumption, the expanding electrical goods manufacture and electrification of the global transportation sector will become increasingly important for future demand trends. In EVs copper is used in wiring and the electric motors themselves forming a significantly greater inclusion than in internal combustion engine ("ICE") vehicles. Glencore has noted that in 2025 EV demand for copper is equivalent to half of new supply from all Probable copper projects globally (assuming 100% are built). By 2030, forecast copper demand in EV's is almost double total new supply from all Probable copper projects globally.

Copper production during the first half of 2020 was less impacted than other base metals by the effects of COVID-19 restrictions. Chile remains the largest copper producer at around 5.6 million tonnes in 2019 with Peru running second at 2.4 million tonnes. China is the largest copper consumer and ranks 3rd for copper production at 1.6 million tonnes in 2019. The DRC and United States are equal 4th producing 1.3 million tonnes each in 2019. How production is affected for the remainder of 2020 largely depends on the effect of COVID-19 in the main producing countries named above.

ICO will produce approximately 2,800 metric tons contained copper in a copper/cobalt concentrate. This concentrate will then be refined at the SMP Refinery, or third parties should Jervois elect to sell concentrate externally once in operation.

Overall, in 2020, despite the pandemic, copper demand remains strong due to economic stimulus package rollouts in China and elsewhere which has kept copper prices buoyant.

An LME copper price of US$3.00/lb was selected, which is slightly below current investment bank long-term consensus and prevailing LME pricing of US$3.15/lb, which has been supported demand optimism in both China and the United States (the latter related to government stimulus after the United States election), a weaker US$ and supply side concerns due to COVID-19. Moving forward copper prices are expected to be well supported over the ICO life of mine, with structural deficits in supply projected.

19.3 GOLD MARKET OVERVIEW

Gold is a readily marketable metal and, in contrast to the ICO concentrate product, pricing is transparent. Supply is made up of mined gold; secondary, recycled gold recovered from previously fabricated products; gold released from the official sector (gold bullion reserves held by central banks, government and supranational bodies), sales or leasing arrangements; and gold released by producer hedging operations.

Gold demand is broadly divided into fabrication demand and investment demand. Fabrication demand includes demand for manufacture of items such as jewellery, coins and medallions, most of which is partly driven by the investment considerations, and gold used in electronics and dental applications, and in minor industrial uses. Investment demand includes ingot and bullion purchased by central banks, governments and other institutions.

Gold is the most malleable metal known and is primarily used in the production of jewellery which accounts for approximately half of physical demand. It is also used as a store of wealth, which has become a more important demand driver over the past couple of years as personal bullion buyers weigh into the market. Other uses of gold are in dentistry, medicine, electronics, infrared radiation shields (spacecraft/suits), manufacturing, 3D printing and gold leaf as coatings on items such as food to buildings.

Global uncertainty related to the COVID-19 pandemic has seen the gold price rise to historically high levels in 2020 on demand for physical bullion as investors seek safe haven for their wealth.

The NI 43-101 FS applies US$1,750/oz which is below the current gold price of US$1,965/oz.

19.4 PRODUCT OVERVIEW

The Idaho Cobalt Operations will produce a bulk cobalt and copper sulphide concentrate product through a conventional concentrate flotation process. Production is scheduled to commence in mid-2022 and will continue through to 2028 based on current ore reserves. Annual available product for sale is detailed in the table below.

Table 19-1: ICO Life of Mine Concentrate Production

Calendar Year		2022	2023	2024	2025	2026	2027	2028
Bulk Concentrate								
Concentrate Prod	(short tons)	7,150	26,745	21,692	20,647	18,307	22,151	20,738
Grade	% Co	10.05	9.87	9.92	9.95	9.87	10.14	10.19
	% Cu	9.48	16.35	17.08	17.72	27.80	8.27	7.22
	g/t Au	9.9	10.97	10.85	14.04	18.29	9.16	7.65
Contained Cobalt	(Mlb's)	1.597	5.836	4.746	4.546	4.039	4.861	4.509
Contained Copper	(Mlb's)	1.368	9.115	7.738	7.690	10.748	3.829	3.114
Contained Gold	(oz)	2,512	9,516	7,022	8,645	12,079	6,053	5,591

The product contains impurity levels in line with typical impurity levels in sulphide concentrates with the exception for arsenic. The arsenic levels in the concentrate are elevated as the arsenic is associated with the cobalt in the orebody and is extracted jointly with cobalt in the flotation process.

19.5 PRODUCT PACKAGING

The bulk sulphide concentrate product will be packed on site in Idaho in one tonne (1,000 kg) lined bulker bags, then stuffed into a standard 20 ft (twenty foot) intermodal container ("TEU") dry cargo containers for shipment to customers. Only sealed containers are employed, and it is not planned to offer Less Container Load ("LCL") shipments. Charter vessels are not expected to uplift bags in bulk due to insufficient cargo per shipment.

One tonne bulk bags are generally acceptable to most potential customers for this material type. This form of packaging is preferred by many customers due to ease of storage, transport, and containment of the material. Bulk bags are easily stored in warehouses, or open areas as they are lined, ultra-violet ("UV") protected and waterproof. They are easily handled by forklift and can be readily transported by truck, on rail, or in containers. The lined and waterproof nature of this packaging ensures low risk of external environmental impacts when compared with other forms of packaging, for example bulk storage, and these bags are also typically approved for transportation of hazardous materials.

19.6 COMMERCIAL STRATEGY

There are existing large metallurgical operations globally which produce cobalt and copper products as well as copper, nickel, and cobalt products. These operations would represent the main target market for the product produced at the Idaho Cobalt Operations. A limiting factor is that any facility that treats the product must also have the ability to extract gold, which is present in significant quantities, and to separate and stabilise the contained arsenic in the product. Currently there are no direct United States based customers for the bulk concentrate product. It must be exported, and Jervois has approached end users located in jurisdictions with strong trade and political ties to the United States.

Jervois's marketing discussions have extended past conventional direct customers for the product outlined above, with extensive discussions being held with non-Chinese precursor and cathode manufacturers, as well as OEM's (automakers) expressing an interest in securing supply of cobalt from a stable jurisdiction such as the United States.

The bulk concentrate from the Idaho Cobalt Operations will be converted to cobalt metal and saleable copper at a Jervois owned refinery in São Paulo, Brazil. Final products will be sold at prevailing market prices.

As Jervois will own and operate its own refining facility in São Paulo Brazil, it is not planned to sell any of the bulk concentrate to market with the concentrate being converted to refined cobalt and copper products at this facility, which shall both be sold to final customers. Gold will be produced as a by-product in the same facility.

While Jervois does not currently plan to sell any of the bulk concentrate, several sales options and customers have been identified as part of Jervois's marketing discussions and sale to market remains an option should Jervois wish to pursue external sales in the future.

20 ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL AND COMMUNITY IMPACTS, ENVIRONMENTAL MITIGATION, AND CLOSURE AND RECLAMATION PLANNING

This chapter provides an overview of the environmental studies performed during Idaho Cobalt Operations ("ICO") scoping. The ICO approval, permit and plan requirements, social and community impacts, major environmental mitigation measures, and closure and reclamation plans and financial assurances are also discussed.

20.1 BASELINE STUDIES AND ENVIRONMENTAL IMPACT STATEMENT

The ICO project site is on U.S. Forest Service ("USFS") Salmon-Challis National Forest ("SCNF") land located west of Salmon, Idaho. The SCNF evaluated the proposed Plan of Operations for ICO under the authority of the U.S. Mining Law and in accordance with USFS regulations governing locatable mineral activities on National Forest System lands (36 CFR 228A) and Council on Environmental Quality regulations implementing the National Environmental Policy Act (NEPA) (40 CFR 1500-1508). The application for a new National Pollutant Discharge Elimination System ("NPDES") discharge permit for ICO was also evaluated according to the NEPA process.

The NEPA process required a thorough series of environmental baseline studies and the development of an environmental impact statement that assessed all environmental effects, including impacts on natural, economic, social and cultural resources. The NEPA process also involved analyzing different alternatives to the proposed project and providing public notice and comment periods. When the NEPA process was complete, the NEPA decision (i.e. the Record of Decision by the SCNF) was subject to an administrative appeals process prior to implementation. Through this process, the public had opportunities to participate in review and approval of the ICO Plan of Operations ("PoO") and NPDES discharge permit.

The lead government agency developing the environmental impact statement was the SCNF with the U.S. Environmental Protection Agency ("USEPA") and the Idaho Department of Environmental Quality ("IDEQ") being major cooperating agencies. Other agencies having permit or review authority included Idaho State Historic Preservation Office, U.S. Fish and Wildlife Service ("USFWS"), National Oceanic and Atmospheric Association National Marine Fisheries Service ("NMFS"), U.S. Army Corps of Engineers ("USACE"), Idaho Department of Water Resources ("IDWR"), Idaho Department of Health and Welfare, and Lemhi County. In response to the USFS Biological Assessment of the proposed ICO PoO and initiation of formal consultation, NMFS issued a Biological Opinion ("BO") for potential impacts to Snake River spring/summer Chinook salmon and Snake River basin steelhead in 2008, and USFWS issued a BO for impacts to bull trout in 2008.

During the preparation of the environmental impact statement, a key component of the consultation process with agencies, organizations, and Tribal governments was the JRP. The Idaho Joint Review Process was developed by the State of Idaho in 1996 to provide for a structured interactive consultation process between the state and federal agencies to address an often-complex interaction of laws and regulations concerning mineral development proposals on public lands in Idaho. The intent of the JRP is to increase communication and cooperation between mining companies, state and federal agencies, the public in Idaho, and Native American tribes.

The SCNF's interdisciplinary team initiated the JRP in May 2001 and continued the JRP throughout the evaluation of the proposed ICO PoO and supporting technical reports, and the development of the ICO environmental impact statement. In addition to routine interagency cooperation and joint reviews of the proposed ICO PoO baseline information and technical reports, several formal JRP meetings were held to discuss the review of pertinent ICO-related information necessary to complete a science-based impact evaluation for the ICO.

Both the ICO and SCNF have conducted numerous consultation meetings with the local community, Native American tribes, stakeholders, special interested groups, and non- governmental organizations. Impacts identified during the

NEPA review have been recorded and addressed during meetings, in the environmental impact statement, and in the Record of Decision ("RoD") (see Section 20.2.1).

The SCNF completed the *Final Environmental Impact Statement* ("FEIS") in 2008 after seven years of analysis conducted with USEPA, Tribal and public participation. The FEIS was prepared in compliance with the NEPA and other relevant Federal and State (Idaho) laws and regulations. The FEIS analysed and disclosed the direct, indirect, and cumulative environmental impacts on resources in and adjacent to the ICO project area that would result from SCNF approval of ICO. Approximately 175 individuals, groups, organizations, Tribal entities, and agencies provided comments regarding project-related concerns and issues. The FEIS addressed the public's concerns and issues expressed in comments.

Baseline studies of existing environmental resources in the ICO project area included the following disciplines:

- Air quality and climate
- Geology and geotechnical characteristics
- Soil resources
- Water resources (hydrology and water quality/quantity of surface and ground water)
- Jurisdictional wetlands and other Waters of the U.S.
- Aquatic biology/fisheries resources and threatened, endangered, and candidate species
- Vegetation resources and threatened, endangered, and candidate species
- Wildlife resources and threatened, endangered, and candidate species (including Forest Service Management Indicator Species)
- Land use and recreation resources
- Cultural and historic resources
- Roadless and wilderness resources
- Visual resources
- Noise resources
- Road and access management
- Social and economic resources
- Heritage resources
- Blackbird Mine Site/CERCLA response and restoration activities

The FEIS compared the direct, indirect, and cumulative impacts of the proposed ICO PoO, three alternative plans, and the No Action alternative on each of these aspects of the ICO project area. A number of design modifications, changes to operational components, and mitigation measures were developed in formulating the agency alternatives to address issues raised in the scoping process, to minimize impacts to resources or to provide an analysis of a wide range of reasonable alternatives.

The FEIS also considered the requirements of the Salmon National Forest Land and Resource Management Plan ("LRMP") during the scoping process. The LRMP contains a management goal related to mineral resources which is to "Encourage the legitimate exploration and extraction of leasable and locatable minerals from National Forest lands while maintaining or improving other resource values."

20.2 ICO APPROVAL AND COMPLIANCE REQUIREMENTS

20.2.1 Record of Decision and ICO Plan of Operations

The Record of Decision ("RoD") describing the decision by SCNF to approve a PoO for ICO was issued in 2009. The RoD documented changes and additions to the proposed ICO PoO that were deemed necessary to meet the requirements of the regulations set forth in 36 CFR 228 Subpart A. The RoD also stated that ICO operations in accordance with the approved ICO PoO would be consistent with the LRMP.

The RoD approved a PoO for ICO that is consistent with Alternative IV (as described in the FEIS) with additional modifications. The RoD states that Alternative IV with additional modifications meets the stated purpose and need for the decision, protects resources to the extent feasible, addresses the public's concerns, and is consistent with applicable State and Federal laws, plans and policies.

The additional modifications to Alternative IV included the following:

- Selection of the Alternative II design (i.e. the originally proposed design) for the TWSF (described in Section 20.4.1.1)
- Construction of mitigation wetlands to replace the non-jurisdictional wetlands covered by the TWSF (described in Section 20.4.2.6)

Based on the selection of Alternative IV with additional modifications, the RoD required changes and additions to the proposed ICO PoO, which was described as Alternative II in the FEIS. These changes, which included design modifications, operational components, and mitigation and monitoring plans, were intended to avoid, reduce, minimize, or mitigate adverse environmental impacts to the extent feasible. The requirements of the RoD were also intended to avoid negative impacts to the Blackbird Mine Site/CERCLA remedy.

Numerous stipulations, mitigations, and monitoring Programs are required by the RoD. The following items were required to be incorporated into the proposed ICO PoO and submitted to SCNF prior to final approval of the ICO PoO:

- Make changes and additions to the proposed ICO PoO in accordance with Alternative IV as described in the RoD
- Submit a reclamation performance surety
- Provide SCNF with a copy of the NPDES discharge permit from the USEPA, and the Clean Water Act ("CWA") Section 401 Certification from IDEQ

The ICO project area is adjacent to the Blackbird Mine Site/CERCLA remedy. Numerous requirements included in the RoD were developed to avoid the release of hazardous substances into the environment that would require a response action or result in the incurrence of response costs under CERCLA. Furthermore, before commencing any surface disturbing activities which required access through private land associated with the Blackbird Mine Site/CERCLA remedy, the following items were required by the SCNF as stipulated in the RoD:

- Documentation of approval by the USEPA CERCLA Remedial Project Manager ("RPM") and the USDA Forest Service CERCLA Project Manager of designs or activity that would modify the existing transportation system and/or which could affect existing wastes, rock piles, water diversions, and other remedial facilities in place to ensure design performance is not adversely impacted and thereby ensure the protection of the Blackbird Mine Site/CERCLA remedy
- Rights-of-way or other access agreements for vehicles and powerlines through patented lands within the Cobalt Townsite and Blackbird Mine Site/CERCLA remedy
- Protocols related to vehicle access and measures to ensure that contaminated materials do not leave the Blackbird Mine Site/CERCLA remedy
- Development of a waste rock disposal plan for disposal of any contaminated mine waste, including but not limited to areas such as the Blackbird Road No. 60115, and overbank materials encountered during pipeline excavation

The RoD incorporated recommendations from the NMFS BO and the USFWS BO that are intended to mitigate impacts of the ICO on fish populations and habitats in the ICO project area and along the ICO transportation route.

Additional mitigation measures, design components, and monitoring programs related to the protection of environmental resources which were required by the RoD prior to SCNF approval of the ICO PoO are addressed in the sections below.

The proposed ICO PoO was modified in accordance with the RoD, and the approved ICO PoO was completed in 2009 and subsequently approved by SCNF.

The SCNF, as the lead federal agency for the ICO, has the primary role in administering the approved ICO PoO and reviewing and approving all final designs and monitoring and mitigation plans. In addition, an Inter-Agency Task Force ("IATF") was established by SCNF in the RoD to oversee development of ICO. The IATF includes SCNF, IDEQ, USEPA, NMFS, USFWS, Nez Perce Tribe and Shoshone-Bannock Tribes. SCNF provides regular updates to the IATF regarding ICO plans, monitoring data, regulatory compliance and quality assurance reviews.

ICO is currently under care and maintenance and remains in compliance with the requirements of the RoD and the provisions of the approved ICO PoO.

Jervois is currently evaluating the following modification to the approved ICO PoO:

- Changing from a temporary man camp during the construction of the mine to a permanent man camp during the life of the mine. The proposed permanent man camp would be located in the soil stockpile area and would not substantially change the disturbance area. In addition, the number of vehicle trips would be significantly reduced, thus minimizing safety concerns associated with daily personnel transport to the mine.

This modification to the approved ICO PoO will require review and approval by SCNF prior to further development.

20.2.2 Permit, Authorization, and Plan Requirements

The RoD identified certain permits and authorizations from SCNF, other Federal agencies, and State agencies that are required for ICO. The permits and authorizations which are currently active are presented in Table 20-1.

Table 20-1: ICO Permits and Authorizations Currently in Effect

Permit / Authorization	Agency	Status
Road Use Permit	SCNF	Current (renewed annually). SCNF is currently reviewing and preparing the 2020 permit.
NPDES (IPDES) Discharge Permit	USEPA, IDEQ	Current (2009; administratively extended). Permit No.: ID-002832-1. Permitting authority transferred to IDEQ in 2019.
CWA Section 401 Certification	IDEQ	Current (2009). Required for NPDES (IPDES) Discharge Permit.
Administrative Settlement Agreement and Order on Consent	USEPA	Current (2011). Provides access/right-of-way for vehicles and powerlines within the Blackbird Mine Site/CERCLA remedy.
Ground water permit	IDWR	Permits to use ground water are in effect; beneficial use of the permitted withdrawals must be proven by 2025 to develop permits into water rights.
Dam Safety Permit	IDWR	Current (2020). Required for Water Management Ponds per ground water permit 75-13977 from IDWR.
Surface water right	IDWR	Water right owned by ICO; currently leased to water supply bank.

Prior to resuming construction, ICO is required to obtain certain additional permits and authorizations listed in the RoD. These permits and authorizations are presented in Table 20-2.

Table 20-2: ICO Permits and Authorizations Required for Construction Phase

Permit / Authorization	Agency	Status
NPDES (IPDES) General Permit for Discharges from Construction Activities (CGP)	USEPA, IDEQ	Permitting authority will be transferred to IDEQ in 2021.
CWA Section 401 Certification	IDEQ	Requirement of NPDES (IPDES) CGP.
CWA Section 404 Permit	USACE	Original permit issued 2009. New permit required for completion of water treatment plant discharge pipeline. Nationwide Permit 12 (Utility Line Activities) includes Sections 10 and 404 permits from USACE, Stream Channel Alteration Permit from IDWR, and Section 401 water quality certification or waiver for impacts to waters within the State of Idaho from IDEQ.
Stream Channel Alteration Permit	IDWR	
CWA Section 401 Certification	IDEQ	
Air Quality Permit to Construct	IDEQ	Original permit issued 2009. Review of updated mill design by IDEQ and modification of existing permit for new design required prior to construction of the mill.
Drilling Permit for Well Construction	IDWR	The need for additional ground water supply wells will be determined when construction resumes.
Building Permits	Lemhi County, Idaho	Additional/ revised permits may be required to develop the proposed operations man camp and the redesigned process buildings.
Septic System Permits	Eastern Idaho Public Health	Existing system permitted in 2018. If another system is required to support the proposed operations man camp, an additional permit will be required.
Approval of paste backfill pipeline/process	SCNF	Design review required.
Approval of updated TWSF design	SCNF	Design review required.
Approval of potable water treatment and distribution system design	IDEQ, SCNF	Original approval received from IDEQ in 2018. Approval of updated potable water treatment and distribution system design required.

Prior to commissioning and operations, ICO is required to obtain certain additional permits and authorizations. These permits and authorizations are presented in Table 20-3.

Table 20-3: ICO Permits and Authorizations Required for Operations Phase

Permit / Authorization	Agency	Additional Information
Injection of Mine Tailings Authorization under Rule OR Injection Well Permit	IDWR	Required prior to mine backfill.
NPDES (IPDES) Multi-Sector General Permit for Stormwater Discharges associated with Industrial Activity (MSGP)	USEPA, IDEQ	Permitting authority will be transferred to IDEQ in 2021.
CWA Section 401 Certification	IDEQ	Requirement of NPDES (IPDES) MSGP.
Air Quality Operating Permit	IDEQ	--
Approval of vegetation reference areas	SCNF	Required to be established in the 1st year of operations.

Permit / Authorization	Agency	Additional Information
Approval of ground water capture and pumpback system	SCNF	Required prior to construction of the mine adits to below the ground water table.

The plans required for the ICO PoO per the RoD and ICO permits and the current status of these plans are presented in Table 20-4.

Table 20-4: ICO Plans

Plan	Status
Waste Rock Amendment Plan	Current version: 2018. Plan will be updated based on updated mine design/process flowsheets. Needs third-party review prior to SCNF approval.
Geochemical Monitoring Plan	Current version: 2011. Plan will be updated based on updated mine design/process flowsheets. Needs third-party review prior to SCNF approval.
Surface Water Management Plan	Current version: 2018.
Water Resources Monitoring Plan	Completed in 2009. Title: Operational Water Monitoring Plan.
Post-Mining Ground Water Capture Plan	Current version: 2018. Plan will be updated to correspond with forthcoming revised Point of Compliance Determination and to include measures for ground water capture and treatment during operations and post-mining.
Weed Control Plan	Current version: 2019.
Wetland Monitoring Plan	Current version: 2014.
Waste Rock Disposal Plan	Current version: 2018. Plan will be updated based on updated mine design/process flowsheets.
Transportation Plan	Current version: 2018.
Spill Control Plan	Current version: 2009. Title: Spill Prevention and Response Plan. Supplemental to the SWPPP. Revised plan currently in development in collaboration with SCNF.
Tailings Placement Plan	Current version: 2018. Plan will be updated based on updated mine design/process flowsheets.
Revegetation Plan	Current version: 2009. Revised plan currently in development in collaboration with SCNF.
Public Access Control plan	Current version: 2009.
Transportation Plan	Completed in 2018. Revised plan currently in development to support updates to the Road Use Permit.
Water Treatment Plan	Current version: 2009.
Health and Safety Plan for Contaminated Materials	Current version: 2009.
Snow Removal Plan	Current version: 2018. Supplemental plan to the SWPPP.
Reclamation Plan	Current version: 2009. Final plan required to be submitted prior to eighth year of operations.
Road Reclamation Plan	Current version: 2009. Final plan required to be submitted prior to eighth year of operations.
Biological Monitoring and Assessment Program (includes Fish Tissue Study Plan and Aquatic Invertebrate Sampling Program)	Current version: 2010. Includes monitoring in accordance with the NMFS BO, USFWS BO, and NPDES (IPDES) discharge permit requirements.

Plan	Status
Copper Loading Demonstration Plan	Current version: 2008. Plan will be updated to correspond with forthcoming revised Point of Compliance Determination. Required for NPDES (IPDES) discharge permit.
Quality Assurance Plan	Current version: 2019. Required for NPDES (IPDES) discharge permit
Best Management Practices Plan (including Mercury Minimization Plan)	Current version: 2020. Required for NPDES (IPDES) discharge permit
Stormwater Pollution Prevention Plan	Updated plan completed in 2020. Required for NPDES (IPDES) CGP.
Fugitive Dust Control Plan	Current version: 2018. Required for Air Quality Permit to Construct.

20.2.3 Safety and Health

The MSHA identification number for ICO is 1002221. Following cessation of construction at the end of 2018, the Mine Safety and Health Administration (MSHA) was notified that the ICO was in care and maintenance period. MSHA has placed the ICO in the Abandoned Mine category to suspend the requirement for reporting. Before construction resumes, MSHA will be notified, the site number will be reactivated and reporting will resume.

As required by the RoD, ICO has incorporated enhanced emergency management capabilities for medical, spill control, and fire situations in the approved ICO PoO.

20.2.4 Permitting and Compliance Audits

Wood Environment & Infrastructure Solutions, Inc. (Wood) was retained to conduct an environmental permitting and compliance audit for ICO. The findings of this work are documented in the *Environmental Permitting Audit, Idaho Cobalt Operations* (the *Audit*; Wood, January 2020). The purpose of the *Audit* was to review the status of environmental permits and approvals to determine if ICO complies with regulatory requirements for the development and operation of the mine. In addition, the *Audit* was conducted to identify risks for ICO regarding permitting and compliance and to identify areas where certain approvals may be in progress, creating a risk that may impact the ICO schedule. The *Audit* focused on the items stipulated in the RoD, current legislation for areas that would require new or renewed permits or authorizations, and project changes being considered as part of the bankable feasibility study.

20.3 SOCIAL AND COMMUNITY IMPACTS

The potential impacts of ICO on heritage and cultural resources, consultations with Native American tribes regarding these resources, the potential impacts of ICO on social and economic resources, community and stakeholder engagement, and the environmental justice review are discussed in this section.

20.3.1 Heritage and Cultural Resources and Impacts and Consultations with Native American Tribes

The FEIS assessed the heritage and cultural resources in the ICO project area. Resources considered included historic properties and prehistoric archaeological properties.

The ICO project area is located within the aboriginal lands of the Shoshone-Bannock Tribes. In addition, the Shoshone-Bannock and Nez Perce Tribes have fishing rights in the ICO project area under the Shoshone-Bannock Tribes Fort Bridger Treaty of 1868 and the Nez Perce Treaty of 1855. In accordance with Federal guidelines, SCNF consulted with the Shoshone-Bannock and Nez Perce Tribes throughout the process of project scoping and evaluation from 2001 through 2007, including informational presentations to the Shoshone-Bannock Tribal Council and staff, periodic project update letters, and other meetings, and on-site tours involving representatives of the Shoshone-Bannock and Nez Perce Tribes. The consultation process included providing all documents requested by the Shoshone-Bannock and

Nez Perce Tribes on Heritage Resources identification on the ICO project area, as well as other documents related to the total scope of environmental analysis of the proposed project. Site inventory work identified no Traditional Cultural Properties in the ICO project area or in consultations with Tribal representatives conducted by SCNF.

The FEIS concluded that ICO is not expected to have any cumulative effects on heritage resources. There are few National Register eligible properties in or near the ICO project area, and sufficient adjustments would be made during project construction to insure impact avoidance. In accordance with the RoD, the original routing of the ICO water treatment plant ("WTP") discharge pipeline was modified to avoid a National Register-eligible prehistoric site, and the Shoshone-Bannock Tribes were notified of the new routing.

If heritage or cultural resources are discovered during any earth disturbing activities, ICO is required per the RoD to immediately cease the activities and notify the SCNF archaeologist. The RoD also requires that survey markers be preserved. Where a survey marker must be disturbed, ICO must submit a plan for referencing the marker to SCNF for approval prior to disturbance.

20.3.2 Social and Economic Resources and Impacts

The FEIS assessed the social and economic resources in Lemhi and Custer Counties and the city of Salmon, the closest major community to the ICO project area. Factors considered included social life, population trends and demographic characteristics, education, law enforcement, fire protection, ambulance service, health care, public assistance, water supply, wastewater treatment, solid waste management, employment and income, and housing.

Jervois anticipates that most ICO employees would be hired from within the local labour market of the ICO project area. The FEIS concluded that the potential positive impacts in the local communities would be direct employment at ICO and secondary employment in the retail and service sectors in the ICO project area, and that due to employment opportunities and income created by the mining operation, workers and their families could enjoy an improved quality-of-life as a result of the ICO. The FEIS concluded that additional positive impacts would occur from ICO employees spending their salaries in local businesses and thereby contributing to sales tax revenue, and additional taxes paid by ICO to local and State jurisdictions.

Potential negative impacts identified in the FEIS would be potential stress on community service providers and housing in the area related to hiring employees from outside the local labour market, primarily during the construction phase. However, since only a small number of construction and mine workers with specialized skills are expected to be hired from outside the local labour market, potential negative impacts were concluded to be minimal.

20.3.3 Community and Stakeholder Engagement

Community and stakeholder engagement are priorities for ICO. Whenever possible, ICO sources supplies, materials and labour from businesses and workers in Lemhi and Custer Counties and the town of Salmon. ICO has hosted multiple public information sessions and open houses, and regularly contributes donations to several organizations in Lemhi and Custer Counties and the town of Salmon, including meals-on-wheels and after school food programs, health care and hospice organizations, and sports and rodeo programs. ICO has also coordinated MSHA training for local contractors.

Members of the Shoshone-Bannock and Nez Perce Tribes are primarily resident outside of Lemhi and Custer Counties; however, these Tribes are members of the IATF, and are thus kept informed by SCNF of ICO project status, plans, and activities. By this mechanism, the Tribes have ongoing opportunities to comment and participate in decision-making for ICO.

20.3.4 Environmental Justice

No environmental justice issues were raised during ICO scoping. Other than members of Native American Tribes within the region, the agencies have not identified any other racial minorities or impoverished populations within the Project area that might be affected by approval of any of the action alternatives. The proposed mine is not located within or adjacent to any Native American reservations. Since the Project is neither adjacent to nor near reservations there would be no risk of direct impacts to the reservation lands. Members of any Tribes living off the reservations and in the ICO project area would be affected to the same extent as other people in the area from an economic standpoint. The FEIS concluded that there are no environmental justice issues related to the ICO that violate the intent of Executive Order 12898.

The RoD states that approval of the ICO PoO will not result in disproportionate adverse human health or environmental effects to minority or low-income communities, and that the social and economic impacts identified with the project are primarily positive.

20.4 ENVIRONMENTAL MITIGATION

As discussed in Section 20.2.1, changes to the proposed ICO PoO required by the RoD were intended to protect resources and mitigate adverse environmental impacts to the extent feasible, address the public's concerns, and be consistent with applicable State and Federal laws, plans and policies.

This section provides additional details regarding the environmental mitigation measures for ICO, including tailings and waste rock storage and management, water management, monitoring, and closure and reclamation plans and financial assurances.

The primary environmental mitigation measures required by the RoD addressed the potential for impacts to surface and groundwater. The tailings and waste rock management, the groundwater pump-back system, and the measures for long-term water treatment included in closure and reclamation planning and financial assurances described in this section are the primary environmental mitigation measures incorporated into the approved ICO PoO for minimizing the potential for these impacts.

20.4.1 Tailings and Waste Rock Management

Tailings and waste rock management for ICO is focused on the protection and conservation of surface and ground water. The design of the TWSF, operations of the TWSF, and the mine backfill plan are discussed in this section.

20.4.1.1 Tailings and Waste Rock Storage Facility Design

Designs of the TWSF and water management ponds (WMP) were prepared by Telesto Solutions Incorporated. The TWSF design is based on three phases of construction in accordance with the RoD, with Phase 1 being the first phase to be completed. The Phase 1 design, site characterization, geotechnical analysis, and project specifications are presented in the following reports:

- Conceptual design of the Tailing/Waste Rock Storage Facility and Water Management Pond, Revision III (September 2006)
- Feasibility Design of the Tailing/Waste Rock Facility, and Roads for the Idaho Cobalt Project (January 2007)
- Technical Specifications Formation Capital Corporation Idaho Cobalt Project Tailings and Waste Rock Facility, Water Management Pond, and Wetland Mitigation Area, Revision 1 (August 31, 2011)
- Idaho Cobalt Project Tailings and Waste Rock Facility, Water Management Pond, and Wetland Mitigation Area for Construction (Drawing Set) 2011 Lidar Correction (August 30, 2011)

The conceptual design and feasibility design reports (Telesto, 2006 and 2007) describe the design elements for the TWSF, WMP, and the wetland mitigation cells (see Section 20.4.2.6). The appendices to the reports include the site geotechnical data, geotechnical stability analysis, geosynthetics specifications, and the construction quality assurance plan for the TWSF and WMP. The reports also address the water balance for the WMP, management of water treatment waste, TWSF operation, and closure and reclamation of these facilities.

The TWSF will be constructed as a hillside fill that is designed to be constructed and filled in three phases. The two-cell WMP is located adjacent to the TWSF. The lined TWSF and WMP will be operated as a zero-discharge facility. Water reporting to the WMP will be managed within the mine water balance or will be treated for discharge as described in Sections 20.4.2.2 and 20.4.2.3. The general layout of the TWSF and WMP are illustrated in Figure 20-1. The design of the TWSF includes details for the following considerations:

- Phased construction, in accordance with the RoD
- Side slopes of 4 horizontal to 1 vertical (4H:1V) slope constructed in three 50-foot raises with two 100-ft wide benches to enhance erosional and structural stability
- Underliner consisting of an impermeable soil (or engineered clay) layer and a synthetic liner, in accordance with the RoD
- Drainage collection system constructed over the liner to collect infiltration water, in accordance with the RoD
- Conveyance system for sending infiltration water to the WMP, in accordance with the RoD
- A toe berm at the base of the TWSF to provide containment for seepage and runoff water
- Co-disposal of tailings and waste rock to reduce the oxidation rate of the higher permeability waste rock component and reduce long-term risk to the environment of metals leaching, in accordance with the RoD (Waste rock will be placed in designated portions of the TWSF in compacted lifts)
- Dewatered tailings placement and compaction in 2-foot lifts
- A plan for placement of tailings into the TWSF during winter to maintain the design density and moisture content of the dry stack tailings, in accordance with the RoD
- Geotechnical stability
- Capacity of 2.6 million cubic yards of dewatered tailings and waste rock, in accordance with the RoD
- Maximum disturbance footprint of 55 acres, in accordance with the RoD

Tailings and waste rock management for the mine will utilize mine backfill (see Section 20.4.1.3) and surface disposal of tailings and waste rock in the TWSF. Dry stacking of tailings not scheduled for use as mine backfill will occur in the TWSF.

The current issued for construction design includes Phase 1 and the WMP with design changes to fulfill geotechnical requirements addressed in the RoD, including a spillway on the WMP and protection of WMP liners from potential ice damage. Final design for Phases 2 and 3 has not been completed and will need to be developed during the initial operation of the TWSF. Design for Phases 2 and 3 will need to incorporate the interim closure design of each phase in order to accomplish concurrent reclamation as defined in the approved ICO PoO. SCNF review and approval of the design for Phases 2 and 3 will need to occur prior to construction of each phase.

The original construction for the TWSF toe berm and WMP occurred during 2011 and 2012. Construction was resumed in 2018, and the WMP liner system was completed. The WMP spillway was completed in 2019. The current plan is that the Phase 1 area will be graded and lined during two separate construction events. The first area, Phase 1A, will be completed in the southern section of Phase 1. Construction of the second area, Phase 1B, will be completed during filling of Phase 1A. Phases 1A and 1B are illustrated Figure 20-2.

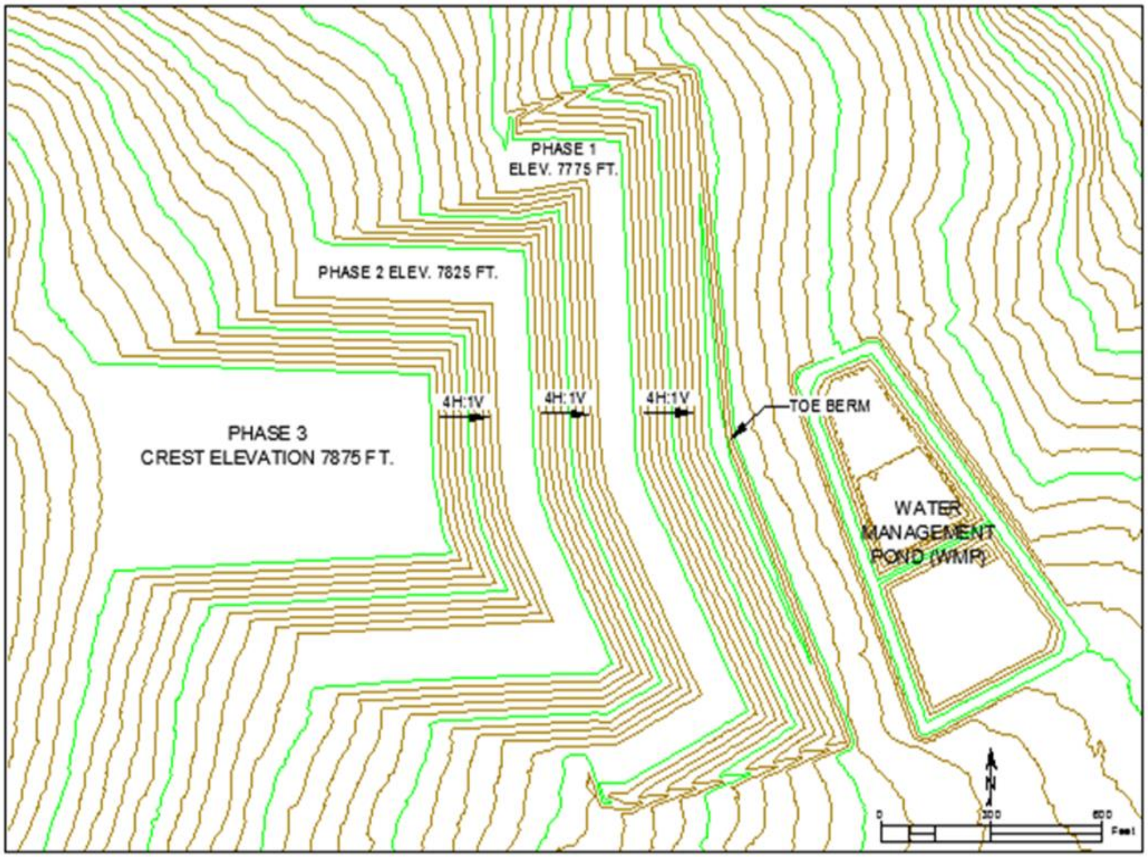


Figure 20-1: TWSF and WMP Conceptual Design

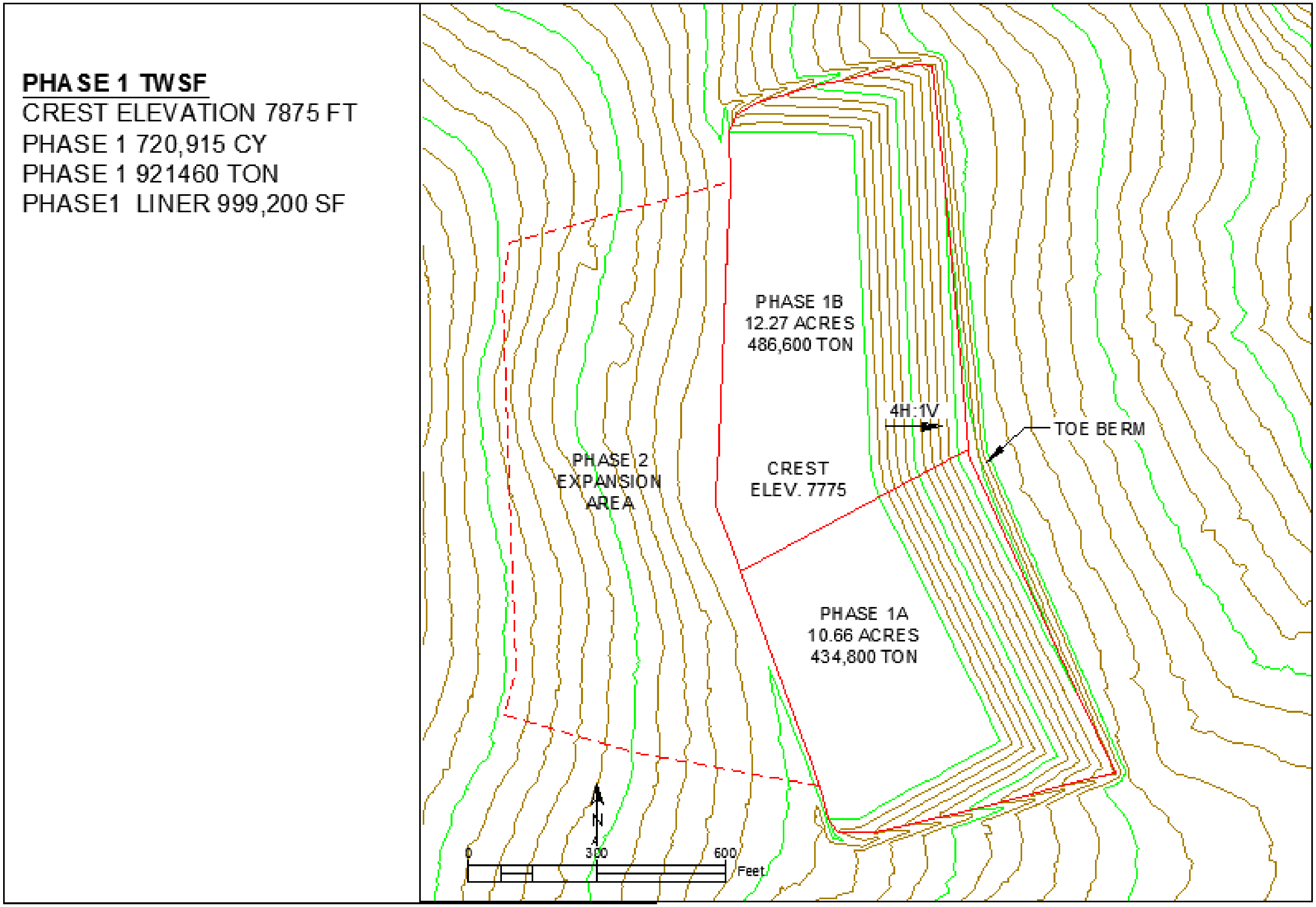


Figure 20-2: Phase 1 TWSF

Phase 1A was previously graded, and partial deployment of the liner system occurred. Quality control testing determined that the seaming of the PVC liner failed to meet the required specifications and mitigation of the liner deficiencies will need to occur when construction resumes. Construction is planned to resume during the summer of 2021. The following construction will be required to prepare Phase 1A and the WMP before operation of these facilities:

- Removal of the defective PVC liner system
- Preparation of the exposed subgrade in the Phase 1A area for liner deployment
- Installation of the new liner system
- Installation of the TWSF liner drainage collection system piping
- Placement of the gravel drainage layer over the Phase 1A area
- Installation of piping between the TWSF, manholes, WMP, and the WTP

The stage-volume relationship for the TWSF has been evaluated in order to schedule the construction of the various TWSF areas. This information was combined with the anticipated mine plan to schedule the time for construction of each area and when the area would be needed for waste disposal. Table 20-5 presents the TWSF crest elevation and capacity for each TWSF phase. The capacity for all three phases is equal to approximately 2,659,509 cubic yards. This volume would accommodate the disposal of approximately 3,400,000 tons of tailings, assuming a dry unit weight of approximately 94.7 pounds per cubic foot.

Table 20-5: Phased TWSF Storage Volumes

TWSF Phase	Crest Elevation (feet)	Volume (cubic Yards)	Cumulative volume (cubic Yards)	Capacity (tons)[1]	Cumulative capacity (tons)
Phase 1A	7775	340,178	340,178	434,809	434,809
Phase 1B	7775	380,694	720,872	486,595	921,404
Phase 2	7825	1,086,025	1,806,897	1,388,135	2,309,540
Phase 3	7875	852,612	2,659,509	1,089,792	3,399,331
	Totals	**2,659,509**		**3,399,331**	

[1]Capacity in tons assumes tailings placed at a dry density of 94.7 pound per cubic foot. This value is based on 90 percent of the standard Proctor density determined by ASTM D698.

Phase 1A is the first area planned for disposal of tailings and waste rock. Phase 1A is anticipated to be filled to capacity by December of 2022 assuming that mine development is initiated during October of 2021. Given this timing, Phase 1B would need to be constructed and lined during the summer and early fall of 2022. Filling of Phase 2 is anticipated to begin during December of 2023; therefore, construction of Phase 2 should be completed during the summer and early fall of 2023. Filling of Phases 1A, 1B, and 2 is illustrated in Figure 20-3. The current mine plan indicates that the amount of tailings and waste rock to be placed in the TWSF is less than the cumulative capacity of Phases 1A, 1B, and 2; therefore, construction of Phase 3 is not anticipated to be required. If future mine plans require additional disposal capacity, Phase 3 will be constructed to accommodate up to an additional 1,089,792 tons.

The tonnage estimates in Table 20-5 assume that tailings will be compacted to a minimum of 90 percent of the standard Proctor density as determined according to ASTM D698. A standard Proctor test performed on a sample of tailings derived from drill core material indicated a maximum dry density of 105.2 pounds per cubic foot and an optimum moisture content of 19.3 percent (SGS; 2020). Design of the tailings filter system assumes tailings will be dewatered to 18 percent moisture.

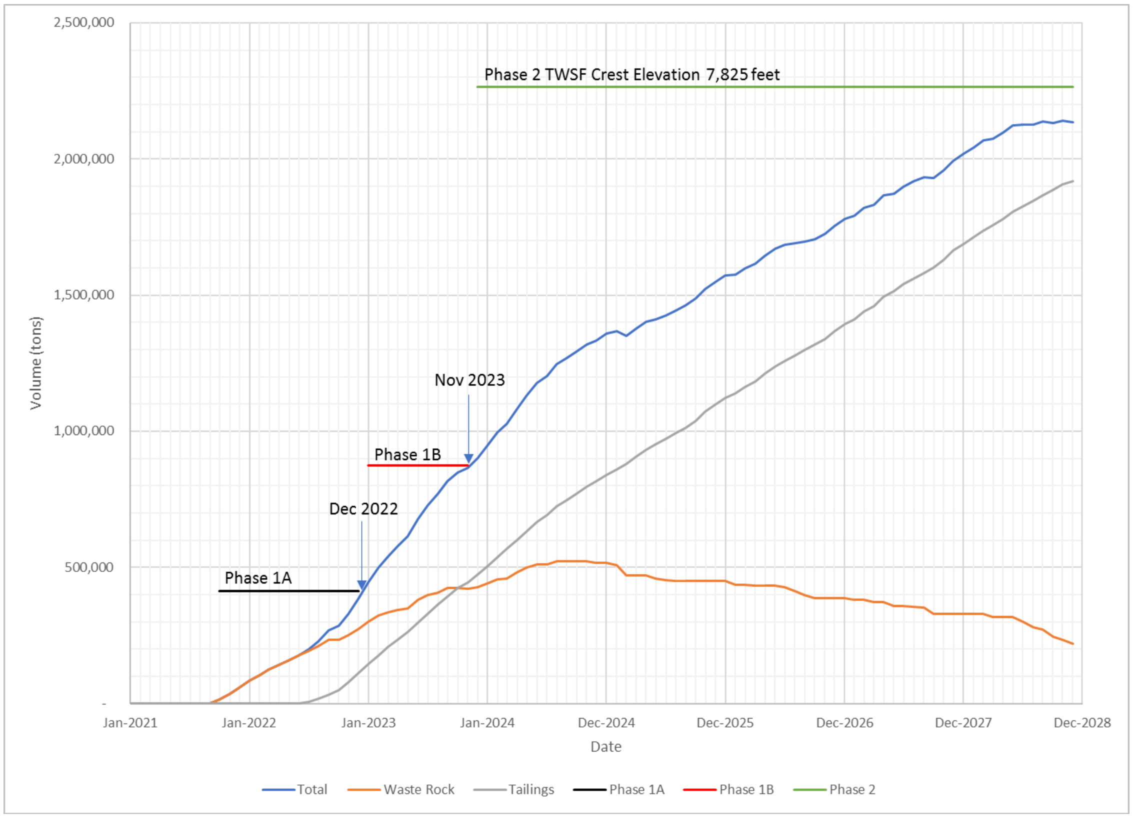


Figure 20-3: TWSF Filling Schedule

The TWSF will also incorporate a portion of the waste rock generated during mine development. To utilize this material for mine backfill, it will need to be strategically placed to accommodate removal, as discussed in Section 20.4.1.2.

20.4.1.2 Tailings and Waste Rock Storage Facility Operations

The RoD requires the ICO Waste Rock Amendment Plan and Geochemical Monitoring Plan to be completed and reviewed by third-party contractors prior to approval by SCNF. In addition, the RoD requires plans for tailings and waste rock placement, including during winter operations, and co-disposal of these materials. Once the revised mine plan is finalized, Jervois will update these plans for submission based on the updated mining methods and production rates.

A draft plan for tailings and waste rock management and tailings placement was prepared by NewFields Mining and Energy Services in 2018. The draft plan was developed based on geochemical characterisation programs reported by Telesto (2004) and Schafer (2006). These characterization data sets include the following:

- Total metals analysis of ore, tailings, and waste rock
- Mineralogy of waste rock and ore samples
- Acid Base Accounting ("ABA") information for tailings and waste rock
- Field Humidity Cell Tests ("HCT") for waste rock
- Laboratory derived amendment protocols for tailings proposed for mine backfill

The draft plan, including the proposed amendment of tailings, is designed to prevent and mitigate potential impacts to the environment, specifically the potential acidity and mobility of metals to surface water and groundwater resources due to potential Acid Rock Drainage ("ARD").

The majority of the waste rock (approximately 80 percent) is not expected to be potentially acid generating since the quartzite has a low pyritic sulphide content. However, approximately 20 percent of the waste rock is predicted to generate slightly acidic solutions containing variable concentrations of soluble arsenic, cobalt, copper, and zinc.

The tailings generated by metallurgical testing were characterized in the baseline geochemical program (Telesto, 2004) by a variety of static and kinetic tests to determine the potential for acid generation and leaching of metal-bearing solutions. The tailings are not considered to be potentially acid generating since the milling process removes the sulphide minerals from the ore. Results of static ABA testing confirm that the tailings materials will be non-Potentially Acid Generating (non-PAG). Tailings paste samples had a neutral pH, and the Net-Neutralization Potential ("NNP") was typically greater than 2.5 kg/t. Tailings should contain a relatively low level of sulphide-sulphur (less than 0.05 percent). However, kinetic tests indicated there is a potential for long-term release of low levels of metals from the tailings.

The TWSF design incorporates liners and a drainage system that will capture any potential ARD in the WMP during operations. The tailings will assist in encapsulating waste rock placed in the TWSF to reduce the potential for ARD generation. The configuration of the TWSF will separate the tailings from waste rock, except for a co-disposal zone where the waste rock will be encapsulated by tailings. In accordance with the RoD, the TWSF closure cover will include a geomembrane liner and a minimum of four feet of soil cover to protect the liner from potential damage from trees growing on the reclaimed surface and prevent meteoric water from infiltrating the dry stack tailings thereby eliminating the potential for seepage. The use of the dry stack tailings disposal method and capping of the TWSF at closure should prevent conditions that would result in seepage from the TWSF during post-closure.

The RoD requires that tailings and waste rock be tested throughout the life of the mine to evaluate potential for acid generation and metals leaching. The ICO Geochemical Monitoring Plan describes the testing program for tailings and waste rock.

Operation of the TWSF is briefly described in the conceptual design report (Telesto 2006). The TWSF operating parameters include the following:

- Dewatered tailings will be trucked to the facility and end dumped
- The tailings will be graded smooth and compacted to 90 percent standard Proctor density in 2-foot thick lifts
- Compaction is expected to be obtained with equipment traffic or a vibratory sheepsfoot roller will be used to achieve compaction
- Tailings and waste rock will be placed in adjacent areas within the TWSF, such that the TWSF can be constructed in a staged manner over the life of the mine
- Waste rock will be hauled to the TWSF, end dumped in 5-foot lifts, and levelled with a dozer
- The out slopes of the waste rock will be flattened to 4H:1V as placement progresses
- Geochemical testing of tailings and waste rock to be placed in the Phase 3 area will be completed prior to construction and operation of this area to verify that the tailings and waste rock are of similar geochemical character to the materials analysed in the FEIS and that the potential leachate from this area will be within the range considered in the FEIS, in accordance with the RoD

The following operating parameters are provided for winter operation, in accordance with the RoD:

- Snow will be removed from active tailings and waste rock disposal areas
- Removed snow will be placed on a liner for collection of meltwater
- Tailings must be prevented from freezing until compacted
- Tailings should be spread and compacted immediately to avoid freezing
- Uniform compaction must be achieved immediately upon tailings placement in freezing weather
- Heated or Teflon-lined truck boxes will be used to prevent tailings from freezing to the inside of the box

- Frozen tailings will be stored without spreading and compaction until thawed
- Wintertime operations would be conducted such that a disposal area is always available for temporary storage
- Ice or compacted snow layers will be removed before placement and compaction of additional lifts of tailings

Tailings will be dewatered through a thickener and vacuum filter and the dewatered tailings would either be trucked to the TWSF or delivered to the paste plant for use as mine backfill.

20.4.1.3 Mine Backfill Plan

The RoD requires that waste rock used as backfill be amended to provide alkalinity to reduce the potential for metals leaching. Tailings produced at the mill will be amended with cement and pumped as a cemented paste backfill into the mine workings during operations and closure. Waste rock will also be used as mine backfill. Backfilling serves the purpose of providing structural support in the mine while reducing the area required for surface tailings and waste rock storage. Nominal design parameters are based on a paste consisting of 65 to 70 percent solids and 30 to 35 percent water. Solids will include between 96 to 98 percent tailings and between 2 to 4 percent Portland cement. Because of the addition of cement, the paste will be alkaline with a stable pH of approximately 9 and will have a high acid-buffering capacity. All remaining tailings will be placed in the TWSF as a dry stack.

The protocols for amendment and placement of paste backfill will be described in the plans discussed in Section 20.4.1.2. The components addressed will include the amendment type, the amendment rate, the method of placement, and performance verification. Protocols will be modified as needed to accommodate conditions and operational knowledge gained during mining.

20.4.2 Water Management

Water management for ICO is focused on the protection and conservation of surface and ground water. Water supply, the modeled water balance, water treatment and the NPDES discharge permit, the Point of Compliance Determination, and water quality monitoring are discussed in this section.

20.4.2.1 Water Supply

Ground water from onsite wells will be used to supply mining (i.e. process) and commercial (i.e. potable) requirements. Surface and ground water withdrawals for mining and commercial use are regulated by IDWR.

ICO has two IDWR permits for ground water withdrawals:

- Mining use (Permit No. 75-13977), 0.67 cubic feet per second (cfs)
- Commercial use (Permit No. 75-14977), 0.5 cfs

ICO must demonstrate beneficial use of the permitted withdrawals by 2025. After ICO has submitted proof of beneficial use, these permits will be developed into water rights by IDWR.

ICO also holds one IDWR water right for surface water withdrawal for mining use:

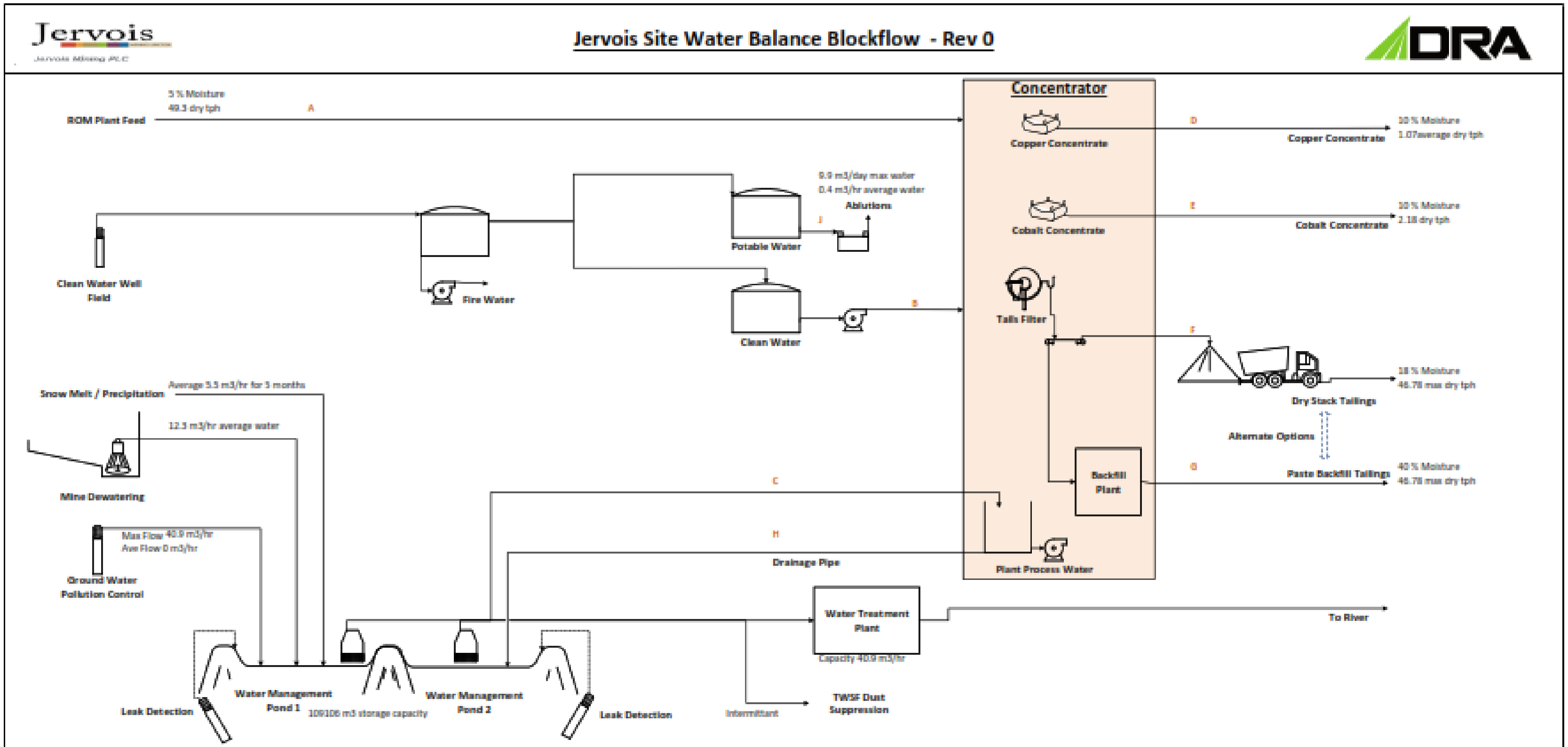
- Mining use (Right No. 75-7073), 1 cfs

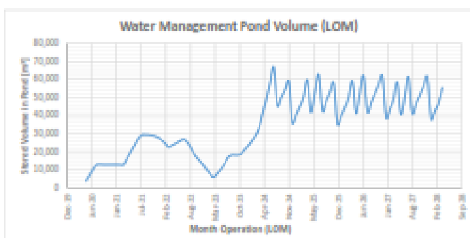
ICO currently leases this water right to the water supply bank administered by IDWR.

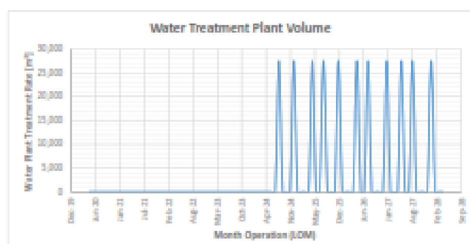
20.4.2.2 Water Balance

The water balance schematic developed for ICO (Figure 20-4) considers the relationships between the ICO components and predicts the mine water balance throughout the life of the mine. The model predicts water requirements based on process uses and water storage requirements in the WMP, based on the water inputs from mine dewatering, the ground water pumpback system, process wastewater, and snowmelt and precipitation; and outflows for process water supply and WTP influent.

Domestic wastewater discharges are not included in the water balance schematic. Domestic wastewater will be conveyed to onsite septic fields for disposal.







	Concentrator Plant Water In	Backfill Tails	Dry Stack Tails	
A	Plant Feed	2.6	2.6	m^3/hr
B	Clean Water Make-Up	17.7	17.7	m^3/hr
C	Process Water MakeUp	10.7	0.0	m^3/hr
J	Potable Water	0.4	0.4	m^3/hr
		31.5	20.7	m^3/hr
	Concentrator Plant Water Out			
D	Copper Concentrate	0.1	0.1	m^3/hr
E	Cobalt Concentrate	0.2	0.2	m^3/hr
F	Tails - Dry Stack	0.0	10.1	m^3/hr
G	Tails - Paste Backfill	30.7	0.0	m^3/hr
H	Excess Process Water	0.0	9.8	m^3/hr
J	Potable Water (Used)	0.4	0.4	m^3/hr
		31.5	20.7	m3/hr

Figure 20-4: ICO Water Balance Schematic

The water balance model includes specific water balance calculations for each year of the mine life based on the anticipated mine plan. The WMP hydrograph (Figure 20-5) indicates that the pond would generally be operated between 10 to 16 million gallons (Mgal). This represents approximately 36 to 59 percent of the WMP capacity to the freeboard level (26.7 Mgal).

Process wastewater, mine dewatering flows, snowmelt and precipitation, and ground water from the pumpback system will be conveyed to the WMP. The water balance model is based on average precipitation for the site and preliminary estimates for mine water production. During periods of high inflow, water will accumulate in the WMP for treatment. This will generally occur in response to system inflows due to snowmelt. As illustrated in Figure 20-6, operation of the WTP to maintain storage capacity in the WMP will not be required until approximately four years after operations begin. The model indicates that after the first water treatment event, four to five water treatment events would be required each year in order to maintain storage capacity in the WMP. For each water treatment event, approximately 6.04 Mgal of water would be treated and discharged to Outfall 001, according to the NPDES discharge permit. Assuming that the WTP is operated at 150 gallons per minute, operation of the WTP for 28 days would be required during each operation event. The water balance simulation indicates an annual discharge of 24 to 30 Mgal of treated water to Outfall 001 after water treatment begins. Discharge of effluent to Outfall 001 will vary annually depending on conditions during each year of operation.

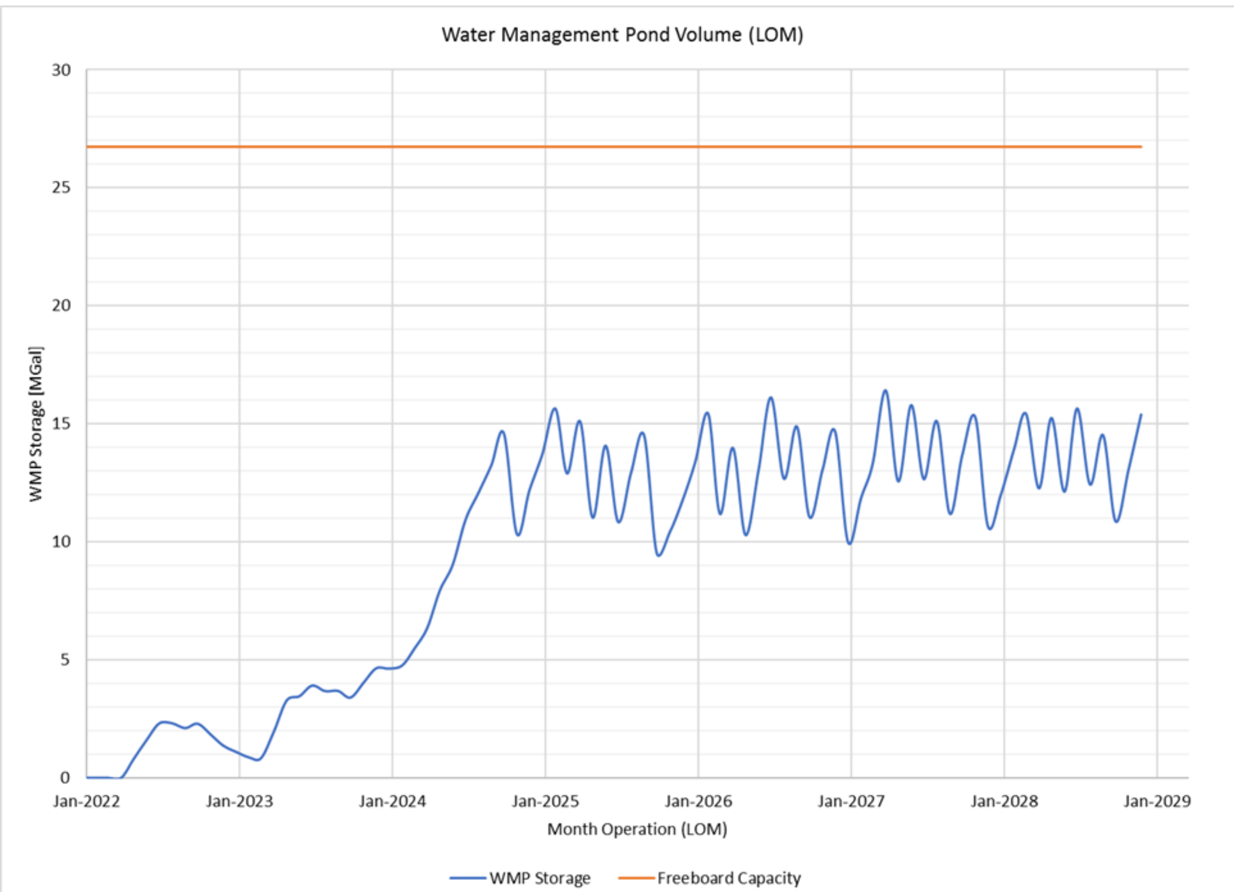


Figure 20-5: Predicted WMP Storage Volume

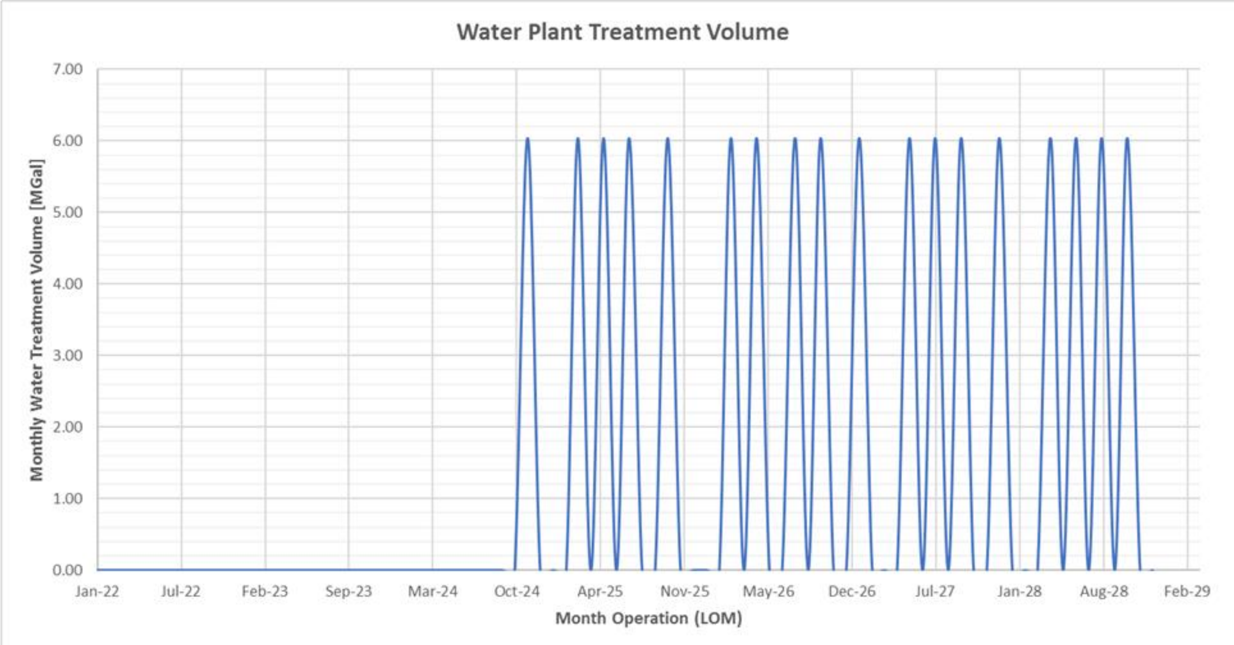


Figure 20-6: Predicted Monthly WTP Treatment Volume

20.4.2.3 Water Treatment and NPDES (IPDES) Discharge Permit

During operations, process wastewater, mine dewatering, and precipitation will be stored in the WMP. Some stored water from the WMP will be used for process water and dust suppression. The WTP will be operated during periods when a positive water balance occurs requiring treatment and discharge of water stored in the WMP. Effluent from the WTP will be discharged to Outfall 001 in accordance with the NPDES (IPDES) discharge permit.

The ICO NPDES (IPDES) discharge permit has been administered under the Idaho Pollutant Discharge Elimination System (IPDES) Program since permitting authority for industrial direct dischargers transferred to IDEQ in 2019. Table 20-4 includes the plans required per the NPDES (IPDES) discharge permit. ICO has developed all required plans and remains in compliance with the permit. The required baseline study of fish tissue and aquatic invertebrates was completed in 2012. The required monitoring is ongoing, as discussed in Section 20.4.2.7.

The WTP is designed to treat up to 150 gallons/min (normal flow rate 100 gallons/min) of water for discharge to Outfall 001. During operation of the mine a negative water balance is initially anticipated so continuous treatment and discharge to Outfall 001 will not typically occur. During periods of high inflow, water will accumulate in the WMP for treatment. This will generally occur in response to system inflows due to snowmelt.

Water accumulated in the WMP is anticipated to require treatment to remove metals, nitrate and ammonia. Veolia Water Technologies, Inc. was contracted for engineering and procurement of the WTP. In accordance with the RoD, the WTP design was based on the effluent limits in the NPDES discharge permit and the requirement to minimize the need for disposal of water treatment waste residues. Third-party engineering review of the WTP design was completed by Tetra Tech in 2018, in accordance with the RoD.

The WTP includes the following processes:

- Metals and total suspended solids removal
  - Chemical precipitation
  - Lamella clarification

- Ultrafiltration (UF)
- Ion exchange (IX)
- Dewatering of metals precipitates using a sludge tank and filter press
- Nitrate and ammonia removal
  - Biological treatment in a moving bed biofilm reactor (MBBR) system for nitrate and ammonia removal
  - Drum filtration for removal of the biological solids

Water treatment residues will be disposed of in the TWSF.

Initial construction of the WTP occurred during 2018. The remaining construction generally consists of final electrical and plumbing installation. Construction of the WTP discharge pipeline has also been partially completed.

20.4.2.4 Point of Compliance Determination

To support monitoring and mitigation of impacts to ground water downgradient of the ICO mine workings, a Point of Compliance "(POC") Determination was developed by IDEQ in 2011 and is currently being updated by IDEQ based on ground water baseline data collected through 2019. The updated POC Determination will establish monitoring locations and upper threshold limits for key constituents at an array of compliance points to allow for early detection of changes in ground water quality during mining. In accordance with the RoD, ICO is required to complete installation of a ground water capture and pumpback system to pump ground water to the WMP for treatment prior to construction of the mine adits to below the ground water table.

The RoD required ICO to develop a post-mining ground water capture plan, but this plan will be expanded based on the POC Determination to include measures for ground water capture and treatment during the operations phase as well as post-mining. The ground water capture plan will include decision criteria and action (trigger) limits for ground water capture.

Prior to stopping the dewatering of the mine, ground water quality must be demonstrated to be of such calibre that use of pumpback system will meet water quality objectives. If post-closure monitoring does not indicate exceedances of the action limits, operation of the ground water pumpback system and ground water treatment will not occur.

As described in Sections 20.4.1.2 and 20.4.1.3, the amendment of tailings used in mine backfill and co-placement of tailings and waste rock in the TWSF are the mitigating measures used to reduce potential ARD and long-term impacts to ground water from the mine backfill and TWSF.

20.4.2.5 Stormwater Management

The ICO Surface Water Management Plan includes the following measures for stormwater and erosion control, in accordance with the ROD:

- Permanent stormwater control structures that will exist beyond the life of the mine, with culverts on streams designed to handle flow from a 100-year storm event
- Road sediment control best management practices ("BMPs") to be designed for a 25 year, 24-hour precipitation event
- BMPs to be utilized for project sediment control
- Sediment control monitoring
- Provisions to demonstrate that construction phase sediments will not be captured by the Blackbird Mine Site/CERCLA remedy facilities unless agreements are in place to utilize those facilities
- A set construction season for soil disturbing construction activities to minimize impacts to soils and sediment production

The stormwater control structures and BMPs described in the ICO Surface Water Management Plan have been constructed and monitoring of BMPs is ongoing in accordance with the ICO Stormwater Pollution Prevention Plan ("SWPPP"), which was prepared in accordance with the NPDES Construction General Permit. The ICO SWPPP is the supporting document to the ICO Surface Water Management Plan that defines the monitoring and maintenance protocols for ICO stormwater and erosion control structures.

20.4.2.6 Wetland Mitigation

ICO activities under the approved ICO PoO are predicted to directly impact about 0.2 acres of jurisdictional wetlands due to mine dewatering and the water treatment plant discharge pipeline crossing streams and riparian zones. As stated in the RoD, this impact is unavoidable and requires CWA Section 404 permit approval from USACE prior to wetland disturbance.

The modification of Alternative IV to include the larger TWSF footprint means that 0.2 acres of non-jurisdictional wetlands will be covered by the TWSF. The RoD requires that replacement wetlands will be constructed to mitigate the impacts to these non-jurisdictional wetlands, as described in the approved ICO PoO.

Two 0.25-acre wetland mitigation cells have been constructed and vegetated, and monitoring is ongoing in accordance with the ICO Wetlands Monitoring Plan. As required by the RoD, the ICO Wetlands Monitoring Plan describes the protocol for monitoring existing wetlands and constructed wetlands to determine impacts to wetlands functions and to modify the constructed wetlands as necessary to assure that they are providing suitable wetland habitat to compensate for project impacts to natural wetlands.

20.4.2.7 Water Quality Monitoring

In accordance with the RoD, the ICO Operational Water Monitoring Plan was developed to provide adequate data to evaluate potential impacts to surface and ground water quality. The plan was developed in accordance with the monitoring requirements set forth in the NPDES discharge permit.

Monitoring is ongoing at surface water locations upstream and downstream of Outfall 001, and ICO reports this ambient water quality data in accordance with the NPDES discharge permit. Outfall 001 sampling will commence when WTP operations begin (see discussion in Section 20.4.2.2). Baseline water quality monitoring is also ongoing at specific seep/spring and ground water locations in the ICO project area to support the development of the POC Determination.

Monitoring requirements of the post-mining and closure phases of the ICO will be addressed in a separate plan that will be submitted prior to the eighth year of operations, in accordance with the RoD.

20.5 CLOSURE AND RECLAMATION PLANS AND FINANCIAL ASSURANCES

The ICO closure and reclamation plans and financial assurances are discussed in this section.

20.5.1 Closure and Reclamation Plans

Section 6.0 of the approved ICO PoO presents the current plan for closure and reclamation of all ICO facilities, roads, and utility routing. The current plan was developed in accordance with the reclamation requirements listed in the RoD. The main objectives of the current plan are as follows:

- Conduct reclamation and revegetation concurrently with the mining program, as much as possible. Concurrent reclamation will be performed on areas no longer required for the mining operation. All exposed soil materials will be stabilized and reclaimed in the same season as the disturbance, unless otherwise authorized by SCNF.
- Minimize all clearing and disturbance, while remaining consistent with ICO needs

- Orient waste rock, tailings, roads, structures, diversions, and the WMP such that they minimize subsequent shaping and recontouring and do not pose a hazard to human health and the environment
- Stockpile soil material from all areas of project disturbance in sufficient quantities to place a minimum of a 1-foot layer on features identified for reclamation
- Recontour surface disturbances and re-establish stable and diverse surface topography and hydraulic features that are compatible with the surrounding landscape
- Confine earth fill construction to the normal operating season (June 1 through November 30) unless specifically authorized by SCNF
- Keep reclaimed slopes to a minimum
- Shape reclaimed slopes to prevent the concentration of water except at points specifically designed to handle flows without erosion
- Establish soil conditions that promote regeneration of stable, diverse, and self-sustaining plant communities through removal, storage, and redistribution of suitable soil materials
- Revegetate areas disturbed by the operation to stable and diverse vegetation communities that provide wildlife habitat and minimize erosion
- Use seed mixes containing native species for reclamation
- Monitor the vegetation reference areas established in the first year of operations to quantify reclamation goals for vegetative cover
- Work with the SCNF to identify opportunities to improve the post-mining land use of the site through reclamation of existing, unnecessary roads
- Provide methods, procedures, and practices for seasonal activities, temporary shutdowns, and final reclamation
- Reclaim all new roads at closure, except those identified by the agencies as needed for administrative purposes

Facility closure plans, conceptual designs, and a conceptual schedule for closure and reclamation of each area of the mine are described in the current plan. Major plans include the following:

- Mine stopes will be backfilled using a combination of slash rock from the stope access ramps and cemented tailings that will be placed as a paste. Adit portals will be sealed to prevent human or animal access.
- The TWSF will be closed by regrading, covering with HDPE liner, capping with a minimum of four feet of soil cover, and revegetating.
- Post-closure surface and ground water quality monitoring in the ICO project area, including discharges from the TWSF, will be conducted and reported for a period of not less than 10 years to verify that post-closure water quality standards defined in the POC Determination have been achieved.

In accordance with the RoD, prior to the eighth year of operations Jervois will summarize the results of all testing for closure purposes and submit a final plan for closure and reclamation to the SCNF and IATF for review and approval. Within one year prior to the closure of ICO, Jervois will review the plan and determine whether it is still valid. If so, ICO will be closed under this plan. If changes have occurred that necessitate modifications to the plan, Jervois will submit a revised final plan to the SCNF within six months of the initiation of closure. In the event that revisions to the final plan are necessary, Jervois will execute the portions of the final plan that will remain unchanged, and refrain from executing portions of the final plan that are undergoing modification until the modification process is complete.

20.5.2 Financial Assurances for Closure and Reclamation Costs

SCNF requires Jervois to furnish financial assurances ("FAs") for surface reclamation and long-term water treatment to ensure that the lands involved with ICO are reclaimed in accordance with the approved ICO PoO and applicable reclamation requirements stated in CFR 228.8 and 228.13. The FAs will be reviewed annually by SCNF to ensure that they are adequate to cover all closure and reclamation obligations.

In accordance with the RoD, the components included in calculating the FAs include:

- Interim operations and maintenance
- Hazardous materials removal and disposal
- Operational water treatment
- Demolition and disposal
- Site re-grading, capping and other earthworks
- Revegetation
- Groundwater capture
- Post-closure operations and maintenance
- Post-closure water treatment, and
- Indirect and overhead costs

The estimated closure and reclamation costs for ICO were determined using a net present value analysis and were based on the preliminary closure and reclamation designs available in the current plan.

The FA amount for surface reclamation was established during 2018 at US$7,206,557 and was placed in the form of a surety bond. The FA for long-term closure and water treatment was negotiated with the SCNF during 2019 to cover the costs of post-closure water treatment for up to 100 years and to complete final closure and reclamation at the end of water treatment.

Because full development of the mine has not occurred, the SCNF initiated the process of updating the surface reclamation FA amount established in 2018. This update required that certain items from the long-term closure and water treatment FA be combined with the current surface reclamation FA, based on the current closure requirements and assuming that development of the mine does not proceed. Jervois anticipates that the update of the surface reclamation FA amount will be finalized in the near future and that the FA amount will be less than that established in 2018.

The FA, for long-term closure and water treatment, is required to be in place prior to mine development. After development of the mine resumes, the annual FA revision will recalculate for the two FAs. The funding instrument for the long-term closure and water treatment FA will be selected by SCNF and Jervois before operations begin.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

This section discusses the Capital Costs Estimate ("CCE") and basis of estimate for ICO. The estimate has been developed from first principles.

21.2 CAPITAL COST SUMMARY

The total initial, pre-production capital costs for ICO as well as the LoM Sustaining Capital are summarized in the tables below.

Table 21-1: Capital Cost Summary by Category (US$ millions)

Category	Initial Capital	Sustaining Capital	LOM Total Capital
Process Plant Direct	25.526	-	25.526
Infrastructure	10.807	1.355	12.162
Mining	18.604	55.861	74.465
Indirect	18.192	0.359	18.552
Contingency	5.274	-	5.274
Total	**78.403**	**57.575**	**135.978**

Table 21-2: Capital Cost Summary by WBS (US$ millions)

Area	WBS Code	Initial Capital	Sustaining Capital	LOM Total Capital
Site Development & Comm. Systems	1100	1.846	-	1.846
Material Handling	1200	10.691	-	10.691
Concentrating	1300	4.776	-	4.776
Tailings & Disposal	1400	5.191	-	5.191
Concentrate Dewatering	1500	1.042	-	1.042
Tailings Storage Facility	1600	-	1.355	1.355
Mining	1700	18.604	55.861	74.465
Reagents	1800	0.273	-	0.273
Utilities	1900	5.208	-	5.208
Temporary Construction Facilities	2210	0.152	-	0.152
Construction Support	2220	0.942	-	0.942
Constr. Equipment, Tools & Supplies	2240	0.815	-	0.815
Pre-commissioning	2250	0.348	-	0.348
Freight	2260	1.309	-	1.309
Spares	2280	0.623	-	0.623
First Fills	2290	0.250	-	0.250
EPCM	3000	3.498	-	3.498
Owner's Cost	4000	10.698	0.359	11.057
Owner's Scope	5000	6.863	-	6.863
Subtotal		**73.129**	**57.575**	**130.704**
Contingency	6000	5.274	-	5.274
Total Capital Cost		**78.403**	**57.575**	**135.978**

21.3 PROJECT CAPITAL – DETAILED

Table 21-3: Idaho Cobalt Operations BFS – Project Capital Breakdown (US$ millions)

PLANT AREA (WBS)	COST US$M
Site Development & Common Infrastructure	**1.846**
Electrical	1.27
Instrumentation	0.576
Material Handling	**10.691**
Milling and Crushing Equipment	4.209
Sitework	0.443
Concrete	1.15
Structural Steel	0.644
Building Erection	0.26
Piping	1.341
Electrical	2.194
Instrumentation	0.451
Concentrating	**4.776**
Flotation and Regrind Plant Equipment	1.247
Sitework	0.001
Concrete	0.076
Structural Steel	0.558
Building Erection	0.26
Piping	0.398
Electrical	2.102
Instrumentation	0.134
Tailings and Disposal	**5.191**
Tailings and Disposal Equipment	1.948
Sitework (Phase 1B)	1.068
Concrete	0.768
Structural Steel	0.322
Piping	0.603
Electrical	0.279
Instrumentation	0.203
Concentrate Dewatering	**1.042**
Concentrate Dewatering Equipment	0.633
Piping	0.208
Electrical	0.131
Instrumentation	0.07
Reagents	**0.273**
Reagent Plant Equipment	0.083
Piping	0.032
Electrical	0.151
Instrumentation	0.007
Utilities	**5.208**
Utilities Plant Equipment	0.845
Sitework	0.08
Man Camp	2.983
Piping	0.338
Electrical	0.882
Instrumentation	0.08
Mining	**18.604**
Mining Pre-production - Capital	18.604

PLANT AREA (WBS)	COST US$M
Freight	**1.309**
Equipment	0.839
Materials	0.47
Owner's Cost	**10.698**
Processing - Owners Cost	5.621
Mining Pre-production - Owners Cost	3.719
Operational Readiness	1.358
Owner's Scope	**6.863**
Pre-Decline Work	4.487
Post-Decline Work	2.376
Project Indirects	**11.902**
EPCM	3.498
Temporary Construction Facilities	0.152
Construction Support	0.942
Constr. Equipment, Tools and Supplies	0.815
Pre-commissioning	0.348
Spares - Capital & Commissioning	0.623
First Fills	0.25
Contingency	5.274
Total Project Capital Costs	**78.403**

21.4 SUSTAINING CAPITAL – DETAILED

Table 21-4: Idaho Cobalt Operations FS – Sustaining Capital Breakdown (US$ millions)

Category	US$M
Sustaining Capital Direct Cost - TWSF Phase 2	1.355
Sustaining Capital Indirect Cost – TWSF Phase 2	.359
Sustaining Capital Costs - Mining	55.861
Sustaining Capital (Total)	**57.575**

BULK CONCENTRATE CASE - Quarterly Project Expenditure													
	2020				2021				2022				
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	
Plant Direct	-	-	-	528,581	2,726,705	6,167,450	7,414,739	5,567,204	1,971,605	984,442	164,889	-	**25,525,616**
Infrastructure	-	-	-	-	-	3,937,074	5,245,142	1,512,519	112,063	-	-	-	**10,806,798**
Mining	-	-	-	-	-	-	135,000	4,515,395	5,564,426	3,996,025	4,392,892	-	**18,603,738**
Indirect	-	-	12,448	453,642	1,409,290	2,730,796	3,770,076	4,678,093	3,658,431	3,790,099	2,963,656	-	**23,466,530**
Total	**-**	**-**	**12,448**	**982,224**	**4,135,995**	**12,835,320**	**16,564,957**	**16,273,210**	**11,306,526**	**8,770,566**	**7,521,438**	**-**	**78,402,682**

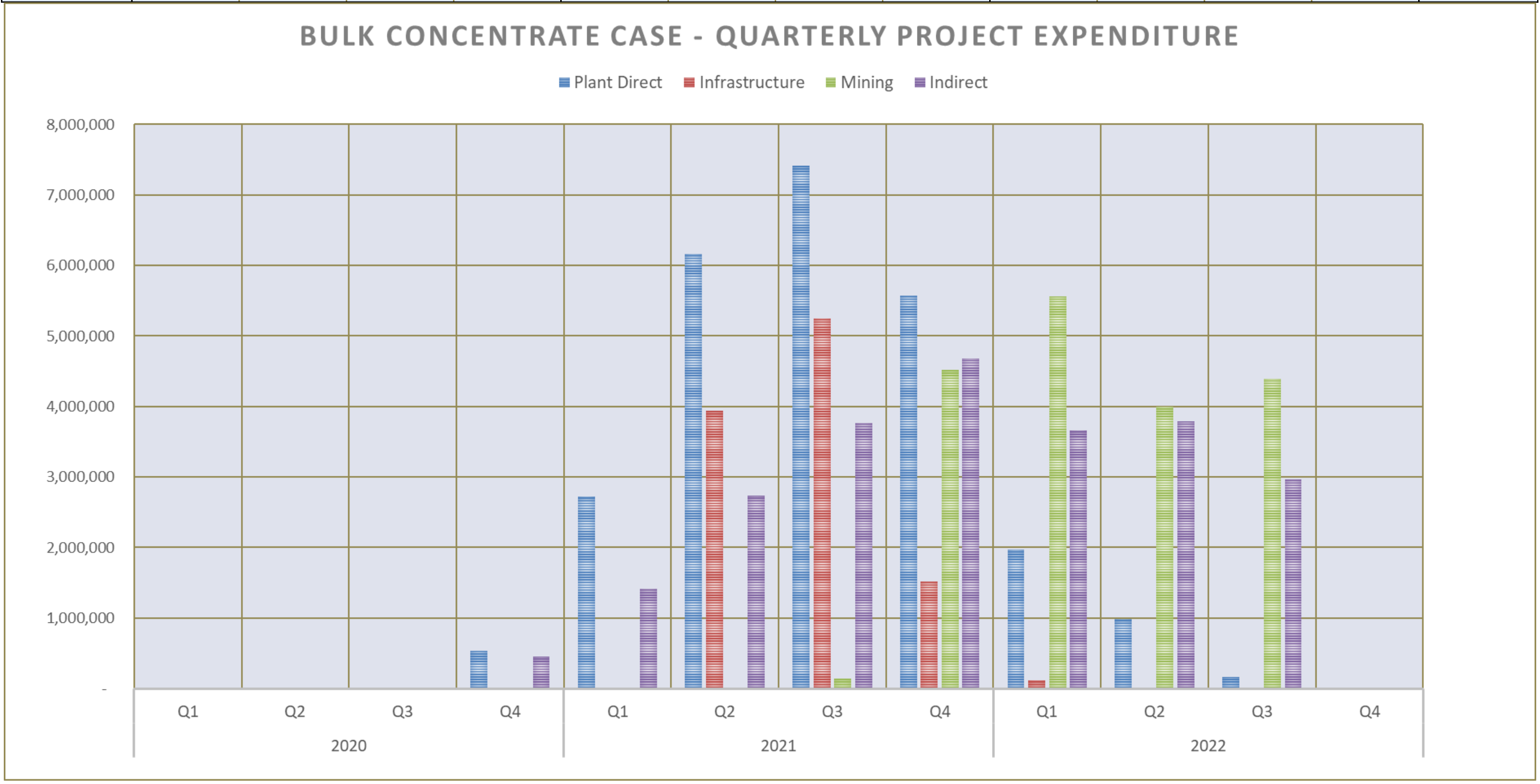


Figure 21-1: Idaho Cobalt Operations Project – Initial Capital Cashflow

21.5 BASIS OF ESTIMATE

The Basis of Estimate covers the site development, material handling, concentrating, tailings, dewatering, reagents, utilities, sustaining capital costs, and associated indirect costs for the ICO, for which DRA and M3 Engineering ("M3") provided the estimating input.

The estimate consolidates the estimate produced under M3/DRA's scope of work with contributions from third party consultants and client (Jervois). Third-party and Jervois contributions are subject to their own defined accuracy levels.

21.5.1 Accuracy

The estimate meets the required accuracy criteria of + 15% - 15%.

21.5.2 Base Date

The base date for the capital cost estimate is August 2020.

21.5.3 Base Currency and Foreign Currency

The estimate has been presented in US Dollar (USD) currency terms dated September 2020. Prices obtained in other currencies have been converted to USD using the applicable exchange rates stated in the quotations. Fluctuations in the exchange rates from the date of quotation to the date of project implementation have not been catered for and should be included in the client's financial model. A schedule of all foreign content has been provided in the body of the CCE.

Table 21-5 Exchange Rates

Exchange Rates	
USD	1.0000
AUD	0.7064
EUR	1.1679
GBP	1.2771
CAD	0.7497

21.5.4 Scope

The scope of the estimate covers the total capital cost associated with the materials handling plant, process plant, related infrastructure, sustaining capital and indirect costs as more fully described in the body of the main documents.

21.5.5 Escalation

Escalation has not been considered past the quoted base date and is therefore not included in the capital cost estimate and should be included for in the client's financial model.

21.5.6 Labour Rates and Costs

Typical contractor distributable costs which were considered based on local requirements and in accordance with the anticipated or agreed contracting strategy are listed below:

- Home office support.
- Mobilisation and demobilisation.

- Office equipment, supplies, furniture and reimbursable expenses
- Safety equipment, personal protective equipment (“PPE”), and clothing
- Utilities Cost
- Contractor’s warehousing costs.
- Non-manual labour including supervision and office staff
- Indirect manual labour
- Construction equipment
- Contractor’s corporate Overhead and Profit (“OH&P”)

The all-in labour rates are based on an average hourly rate of a blended crew for the appropriate trade, comprising of supervisors, journeymen, apprentices and general labour. These rates include the following costs:

- Base labour rate for each crew member
- Benefits and burdens
- Scheduled overtime premiums
- Casual overtime
- Small tools and consumables
- Scaffolding
- Safety awards
- Contractors construction equipment (owned and leased)
- Contractor distributable costs which refer to the indirect components required to complete an individual task

21.5.7 Productivity Factors

The labour units are based on standard industry installation unit hours, which are adjusted for the anticipated productivity performance expected on-site as influenced by, but not limited to the following items:

- Type of plant and size
- Craft availability and skills
- Weather and climate
- Local economy
- Plant location
- Travel time
- Supervision
- Types of contracts
- Shift work
- Overtime

- Engineering support and rework

21.5.8 Estimating System and Format

The estimate has been prepared within the M3 estimating format system in MS Excel.

21.5.9 Estimate Report Requirements

The CCE has been presented as a fully detailed estimate, together with a summary sheet. The estimate takes into account the required work breakdown structure, and the summary sheet consists of a WBS matrix with areas against disciplines.

21.5.10 Work Breakdown Structure (WBS)

A project-level 4 WBS was prepared clearly defining the individual work area for ease of reference and cost management. The cost estimate has been compiled in line with the WBS.

21.5.11 Estimating Methodology

The identification codes as utilized in the estimate are in accordance with the Mechanical Equipment List ("MEL") and PFD's, cover the various WBS areas associated with the materials handling plant, mining and the associated infrastructure. The following documents provided the basis for the CCE's:

- Block Plan & Layouts
- Process Flow Diagrams ("PFD's")
- Process Design Criteria ("PDC")
- Engineering Design Criteria ("EDC")
- General Arrangement drawings
- Supplementary sketches as required
- Equipment quotations from vendors
- Fabrication and erection rates from vendors and suppliers
- Mechanical Equipment List ("MEL")
- Electrical motor lists
- Preliminary electrical cable schedules
- MV Single Line Diagrams ("SLD")
- Project Execution Plan
- Mining development production schedule

The estimate has been produced using solicited bids from vendor quotations for all major equipment supply packages based on the MEL and datasheets provided. Any changes to the equipment would require a modification to the cost estimate. The balance of the CCE is comprised of a combination of detailed, semi-detailed and limited factored costs.

The estimate for the plant has been based on an assumption of a continuous engineering, procurement and construction effort with no interruption of the implementation plan after project approval has been obtained.

21.5.12 Mining

21.5.12.1 Initial Capital Cost

The mine development initial capital during pre-production is based on the proposal by Small Mine Development ("SMD"), an underground mining contractor. Mining costs reflect all capital costs directly associated with mining pre-production over 13 months including completion of the portal bench during the first three months of construction. Initial capital costs for mining are comprised mainly of the contractor's fees for underground development, but also include surface installations for the stabilisation and expansion of the portal bench area, underground power supply, ventilation and pumping equipment, and light vehicles for the transport of personnel and supplies.

The initial mining capital costs include:

- Fuel costs
- Power cost
- Contract mine management support, survey and planning support based on quoted rates
- Mining labour costs were all included on a rates basis of mining, backfill and development requirements of the Life of Mine plan
- Haulage costs based on quoted rates for all ore transportation and loading. Included and separate to these costs were haulage to the treatment plant and Tailings Storage Waste facility

The unit costs were applied to the development production schedule detailed in Section 16 of this report, and the resulting cost estimates were allocated to the relevant cash flow periods. Table 21-6 and Table 21-7 shows the budgetary unit prices developed by SMD and the initial capital using the production plan and applied unit rates.

21.5.12.2 Sustaining Capital Cost

Ongoing development of the underground mine is treated as a sustaining capital expense and has been based on the unit rates obtained from SMD in Table 21-6 and the LoM production schedule, detailed in Section 16, from September 2022 to December 2028.

Table 21-8 provides a breakdown of the annual sustaining mining capital expenditure for the Life of Mine (LoM).

Table 21-6: SMD Proposal - Unit Rates (US$)

SMD Pricing for ICO 2020 Request for Proposal			
Item	Description		Unit
1	Mobilization	$135,000.00	Lump sum
2	15' w x 15' h, Decline, Ramp, Multiple Heading, Fair to Good Ground	$1,550.00	Linear foot
3	15' w x 15' h, Decline, Ramp, Multiple Heading, Poor to Fair Ground	$1,900.00	Linear foot
4	14' w x 14' h, Level Access, Fair to Good Ground	$1,300.00	Linear foot
5	Sill Mining Ore, > 14' width, Fair to Good Ground	$59.50	Ton
6	Sill Mining Ore, > 14' width, Poor to Fair Ground	$70.26	Ton
7	Retreat stoping (L2,L3), (6' or wider x 24'), drill, blast, remote, haul to surface	$44.75	Ton
8	Backstoping (L5,L6), (6' or wider x 24'), drill, blast, remote, haul to surface	$46.75	Ton
9	Shotcrete from Thiessen Bags delivered to Salmon, cu.yd.	$550.00	cu.yd.
10	Placing U/G Gob into drift and fills	$0.00	Ton
11	Backhauling Gob or CRF from surface back into the mine, Average 5,000' haul	$12.50	Ton
12	Placing CRF into drift and fills	$6.00	Ton
13	Placing and Jamming CRF into drift and fills (tight jam)	$9.00	Ton
14	Paste Filling Support Crew (1 per shift)	$1,600.00	Day
15	Drop raising, 9x9, < 100' breakthrough	$800.00	Linear foot
16	Supply and install 45" Escapeway raise cans, with ladders & landings	$800.00	Linear foot
17	Supply and install 9' diameter raise cans, with ladders & landings, grout annulus	$1,825.00	Linear foot
18	Ventilation Bulkhead supply & installation	$20,000.00	Each
19	Supply and install drop raise deck	$12,500.00	Each
20	Optional - Surveying support	$900.00	Day
21	Optional - Mine Planning support	$1,000.00	Day
22	Demobilization	$95,000.00	Lump sum

Table 21-7: Mining Initial Capital Cost Estimate (US$ millions)

Description	Initial Capital (US$M)
Mobilization	0.135
15' w x 15' h, Decline, Ramp, Multiple Heading, Fair to Good Ground	7.897
15' w x 15' h, Decline, Ramp, Multiple Heading, Poor to Fair Ground	1.076
14' w x 14' h, Level Access, Fair to Good Ground	6.058
Sill Mining Ore, > 14' width, Fair to Good Ground	1.188
Sill Mining Ore, > 14' width, Poor to Fair Ground	0.156
Backstoping (L5,L6), (6' or wider x 24'), drill, blast, remote, haul to surface	0.101
Paste Filling Support Crew (1 per shift)	0.147
Drop raising, 9x9, < 100' breakthrough	0.313
Supply and install 9' diameter raise cans, with ladders & landings, grout annulus	0.715
Ventilation Bulkhead supply & installation	0.112
Supply and install drop raise deck	0.07
Optional - Surveying support	0.302
Optional - Mine Planning support	0.335
Total	**18.604**

Table 21-8: Mining Sustaining Capital Cost Estimate (US$ millions)

Description	LoM	2022 Sep-Dec	2023 Jan – Dec	2024 Jan – Dec	2025 Jan – Dec	2026 Jan – Dec	2027 Jan – Dec	2028 Jan – Dec
Decline - Fair to Good Ground	**21.220**	2.698	9.784	6.428	2.310	-	-	-
Decline - Poor to Fair Ground	**2.890**	0.368	1.333	0.876	0.315	-	-	-
Level Access	**22.667**	2.294	10.974	6.332	3.067	-	-	-
Drop raising	**1.581**	0.202	0.612	0.503	0.265	-	-	-
9' diameter raise	**3.607**	0.460	1.396	1.146	0.604	-	-	-
Vent Bulkhead	**0.565**	0.072	0.219	0.179	0.095	-	-	-
Drop raise deck	**0.353**	0.045	0.137	0.112	0.059	-	-	-
Infill Drilling	**1.431**	0.112	0.466	0.465	0.388	-	-	-
Demobilization	**0.095**	-	-	-	-	-	-	0.095
Sustaining Capital	**54.409**	**6.251**	**24.920**	**16.042**	**7.102**	**-**	**-**	**0.095**

21.5.13 Process Plant Direct Costs

M3/DRA developed Material Take-Offs ("MTOs") for the Process Plant and the subsequent estimate without using any previously developed MTOs or estimates other than for existing equipment and material that will be used under the current scope of work.

Engineering specifications were based on M3 standards. Specifications were detailed enough to acquire reliable budget quotations for equipment representing significant costs within the discipline's scope of work. The estimated quantities were based on the buildup from the following disciplines.

21.5.13.1 Bulk Earthworks

The bulk earthworks scope of work consisted of the following key activities related to the Mill terrace:

- In-situ rip and compact
- Cut to Fill / Spoil
- Import of final capping layer
- Structural backfill to the ROM Tip retaining wall

21.5.13.2 Civil Works

The civil scope of work comprised of the following:

- Restricted earthworks
- Concrete work including formwork, rebar and concrete placement
- Grouting
- Cast-in plates/rails and holding down bolts excluding chute work
- Joints and sealant

Civil construction includes foundations, slabs, rafts, QC decking floor, retaining walls and where required mass concrete MTO's for all the related civil scope.

21.5.13.3 Structural Steelwork Supply and Erection

Steelwork quantities for major and minor structures were estimated by M3 from flow sheets provided by DRA and General Arrangement drawings produced by M3 as well as previous "As-Built" MTO's produced for similar structures in similar projects.

An MTO was produced by M3 to cover all steelwork and ancillary bought outs including sheeting, grating, stairtreads, handrailing, etc. The steelwork material quantities in this MTO was used to calculate cost estimates for the steel, based off of historical project data and current steel cost rates.

21.5.13.4 Platework and Lining Supply and Erection

Platework and lining items were quantified by M3 from the equipment list, PFD's, and existing "As-Built" BOQ's produced for similar structures.

An MTO was produced by M3 to cover all platework and lining requirements. The platework material quantities in this MTO was used to calculate cost estimates for based off of historical project data.

21.5.13.5 Corrosion Protection

In accordance with the Engineering Design Criteria ("EDC"), all steelwork and platework will be painted. The rates for painting were quoted by the fabrication contractor, and these have been included in the estimate as part of the fabrication rate.

The cost of galvanising or painting for piping is included in the piping price.

An allowance has been provided for the touch-up of steelwork and platework on site after installation.

21.5.13.6 Mechanical Equipment

From the MEL and PFD's, an enquiry register was compiled and issued to selected vendors for costing. Quotations from selected vendors were obtained for equipment supply. The erection cost for most of the mechanical equipment was based on rates from historical M3 similar project data.

The cost for the refurbishment of existing mechanical and process plant equipment was calculated by applying a factor of 25% to the original purchase price of the equipment.

21.5.13.7 Piping and Valves

The cost for the process plant piping and valves was derived as a factor of 35% of the mechanical equipment supply cost in line with plants of a similar size and nature from the M3 historical database.

Utility piping was quantified and priced and included in the Utilities section of the estimate.

21.5.13.8 Electrical, Control and Instrumentation

Various documents were compiled to determine the electrical equipment quantities, including an Electrical Load List and Single Line Diagram. These documents were based on the Mechanical Equipment List ("MEL"), Process Control Diagrams ("PCDs"), General Arrangement Drawings, and Site Block Plans.

The Electrical MTO includes major equipment, as well as bulk electrical and associated materials such and cables, racking, light fittings etc. Estimated bulk electrical quantities for conveyor installations, as well as equipment installed along conveyor sections, were derived based on individual conveyor lengths. Bulk electrical materials to be installed inside main plant buildings are calculated as an average based on overall plant building dimensions, as per the available general arrangement drawings and block plan.

The cost of Control and Instrumentation was derived as a factor of 12% of the mechanical equipment supply cost in line with plants of a similar size and nature from the M3 historical database.

Major equipment supply costs were obtained from solicited bids received from various vendors. Datasheets, together with MTO, were compiled for the following electrical equipment and issued to Phase 1 vendors for pricing:

- Medium voltage switchgear
- Distribution transformers
- Motor control centres
- Variable speed drives
- Overhead Line
- Power factor correction equipment
- Earthing and Lightning Protection System
- Main power transformer
- Medium voltage switchgear
- Distribution transformers
- Stand-by diesel generators
- Power factor correction equipment
- Mill isolators and liquid resistance starters
- Earthing and Lightning Protection System

Reference rates, for the supply and installation of bulk electrical and instrumentation materials (cables, cable terminations, cable racking, steelwork etc.) as well as for the installation of vendor-supplied equipment, were used to populate the estimate.

21.5.14 Infrastructure Costs

21.5.14.1 Initial Capital Cost

T The initial capital cost related to the infrastructure scope includes the following key areas;

- Construction of the Milling and Concentrator Buildings
- Process Plant utilities including HVAC systems, potable water reticulation, and sewer treatment and reticulation
- Supply and construction of a 90-bed camp including utilities and services
- Construction of the Phase 1B Tailings Storage Facility

Environmental systems included under Owner's Scope which consists of the pre and post decline development work required to enable mining activities.

Table 21-9: Infrastructure Initial Capital Cost Estimate

Environmental Systems Scope of Work	Capital Cost Estimate	Duration	Description
Pre-Decline Work	***US$***		
Pumpback System	1,035,311	4 months	Completion of dirt work, mechanical, and electrical scopes, as well as commissioning
TWSF – Phase 1A	804,475	3 months	Includes new liner cost, dirt work, install, gravel overlay, and QA/QC
Process Water Pipe System	249,000	2 months	Includes completion of pipe system and setting of reclaim pumps in the pond
WTP Completion	840,544	3 months	Includes commissioning cost and Veolia supervision
Portal Bench	1,160,000	3 months	Includes completion of Phase 1, Ground Support, and Completion of Phase 2.
Miners Dry	398,000	1 month	Includes set up cost and delivery to site from Salmon
Water Management Ponds		1 month	Final as-built report to be completed and submitted to USFS
Post-Decline Work			
Mobile Equipment	525,000	NA	Lease of mobile equipment to support site services, mill operations, and TWSF.
Cafeteria	100,000	1 month	Single wide trailer
Maintenance Facility	280,000	One month	2-3 bay truck shop.
Potable Water System	350,000	3 months	Includes piping, pumps, tanks, and treatment system
Paste Backfill Line Drillhole	158,043	1 month	Includes drill site, drilling, sleaving/grouting, for 2x 500' holes
Paste Backfill Pipeline	962,592	4 months	Includes supply and installation of paste line piping
Sub-Total	**6,862,965**		

21.5.14.2 Sustaining Capital

Construction of the Phase 2 Tailings Storage Facility detailed in Section 18 of this report has been included as sustaining capital in the CCE. The estimate is based on the designed quantities, MTO's and unit rates produced for the supply and installation of the liner material as well as bulk earthworks required under the scope of work.

Table 21-10: Infrastructure Sustaining Capital Cost Estimate

TWSF - Phase 2	Sustaining Capital (US$)
Rough Grading 12.3 Ac-ft	60,790
Subgrade Compaction 12.3 Acres	273,340
40 mil PVC liner	603,829
GCL	274,916
80 mil Rub Sheet	38,632
Berm Fill	57,028
Freight	46,177
Sub-Total	**1,354,712**

Provision for indirect costs related to the Phase 2 TWSF and contingency have also been included in the sustaining capital estimate.

21.5.15 Indirect Costs

21.5.15.1 EPCM

The EPCM costs cover the project management, detailed engineering, procurement and construction management costs directly associated with the implementation of the Project.

Project Management and EPCM costs relating to the Process Plant and Associated Infrastructure have been based on EPCM hours calculated by M3. The rates used are M3 rates and are valid to the end of 2020, after which they must be reviewed. The EPCM hours were calculated using typical industry-standard percentages.

21.5.15.2 Spares, First Fill, and Consumables

Allowances have been made for first fill and consumables based on M3's estimate of the requirements and from information provided by the vendors. The first fill allowance apply only to lubricant requirements.

Spares costs have been allowed for as an overall percentage of the required discipline:

- Capital Spares - 5% of Plant Equipment Cost
- Commissioning Spares – 0.5% of Plant Equipment Cost
- First Fills – Estimated by M3 to be US$250,000

21.5.15.3 Pre-Production Costs

Allowances have been made for the following as per requirements on similar projects and as per client's requirements:

- EPCM Site Establishment
- Ramp-up & Commissioning

21.5.15.4 Project Services

Provision has been made for consultants to carry out specialized work. Allowance has been made for the following:

- Logistics / Shipping Management Fee
- Geotechnical Consultant
- Surveys During Construction
- Earthworks & Civil Laboratory
- Quantity Surveyors
- Catering & Management

21.5.15.5 Packages by Others

The following costs have been allowed for:

- Fire protection
- PLC Programming
- Laboratory

21.5.15.6 Owner's Cost

The following costs have been allowed for:

- Overheads (Owners Team)
- Travel
- Insurance – Works, Construction and Transport
- Operational Readiness

21.6 EXCLUSIONS

The scope of the estimate is restricted to the battery limits as shown on the block plan, PFD's, equipment list and more specifically as described in the scope of work. The following are excluded in the capital cost estimate:

- Scope changes, Labour disputes
- Cost of financing
- Acquisition costs
- Any owner's team or pre-production costs not specified in the estimate
- Sunk costs
- Additional studies prior to project implementation
- All VAT, import duties, surcharges and any other statutory taxation, levies or government duties
- All royalties, commissions, lease payments, rentals and other payments to landowners, titleholders, mineral rights holders, surface right holders, or any other third parties not mentioned in this documentation
- Forward cover for any foreign content
- All operating costs
- Any work outside the defined battery limits
- Any provision for project risks outside of those related to design and estimating confidence levels
- Any costs to be expended following completion of the feasibility study and prior to Board approval for project implementation
- Mineral rights and the purchase or use of land

21.7 PROJECT / OWNERS CONTINGENCY

A contingency allowance of 11.3% on the Process Plant is allowed to address the unforeseen risks applicable to this Project. Similarly, a contingency allowance of 6% has been included for mining development as part of the initial capital.

Table 21-11: Contingency

Description	Budget (US$)
Process Plant Contingency	4,158,014.02
Mining Pre-Production Contingency	1,116,138.78
Contingency	**5,274,152.80**

21.8 OPERATING COST ESTIMATE

This section presents the Operating Cost Estimate for ICO. The estimate has been developed from first principles.

The estimate reflects all on-site costs directly or indirectly associated with mining of the Ram deposit, treatment of ore in a flotation concentrator at 1,200 short ton/day (438,000 s ton/annum), to produce and sell a bulk concentrate.

21.9 SUMMARY OF OPERATING COST ESTIMATE

The total estimated cash operating costs compiled in the feasibility study are summarized in Table 21-12 below.

Table 21-12 Operating Costs for Life of Mine (US$ millions)

Major Project Area	LOM (US $m)	2022 Sept - Dec	2023 Jan - Dec	2024 Jan - Dec	2025 Jan - Dec	2026 Jan - Dec	2027 Jan - Dec	2028 Jan - Dec
Mining Cost	**$203.46**	$8.88	$31.074	$32.291	$32.672	$33.031	$33.194	$32.315
Processing Cost	**$51.98**	$2.42	$8.164	$8.212	$8.321	$8.320	$8.362	$8.175
Concentrate Logistics	**$16.00**	$0.83	$3.114	$2.525	$2.405	$2.130	$2.580	$2.415
G&A Cost	**$33.21**	$1.75	$5.244	$5.244	$5.244	$5.244	$5.244	$5.244
Total Cost	**$304.65**	$13.88	$47.596	$48.272	$48.642	$48.726	$49.380	$48.149
US$ / s Ton	**$111.86**	$124.8	$108.7	$109.9	$111.1	$111.2	$112.7	$114.3
US$ / m Tonne	**$123.30**	$137.6	$119.8	$121.2	$122.4	$122.6	$124.3	$126.0

21.10 BASIS OF ESTIMATE

The estimate is based on underground mining and the treatment of ore in a flotation concentrator at 1,200 short tpd (438,000 s ton/annum) to produce and sell a bulk concentrate. The estimate is based upon the LoM plan developed by the project team.

21.10.1 Operating Cost Definition

Operating costs in this feasibility study include all costs incurred by ICO in mining and processing ore to produce and sell a bulk concentrate. The operating costs begin to be incurred from the date of production status being declared in September 2022 with the concentrator capacity treating at approximately 65% of design throughput. Operating costs include general expenses and on-site administration costs.

The operating cost estimate excludes:

- Sustaining capital costs;
- Exploration costs;
- Depreciation and amortization;
- Corporate head office costs;

- Royalties and taxes;
- Bond Costs; and
- Financing costs or interest repayments on loans.

The operating cost estimate also excludes pre-production costs incurred by ICO prior to production status being declared, such as:

- Mining costs including development, mobilization and management;
- Processing costs including labour and supervision, power, consumables and laboratory;
- General and Administrative ("G&A") costs including environmental, safety, road use fees and insurances, as well as camp costs and owners team costs; and
- Spares procurement.

Treatment charges and refining charges, if any, are included in the revenue amounts paid to ICO per ton of concentrate.

21.10.2 Base Date

The base date of the estimate is August 1st, 2020.

21.10.3 Estimate Accuracy

The accuracy of the operating cost estimate has been prepared according to AACE Class level 3 guidelines.

21.10.4 Estimate Currency

The estimate is compiled in US dollars. All cost values in the report are in US dollars unless nominated otherwise. All the quotations received were in US dollars.

21.11 ESTIMATING METHODOLOGY

21.11.1 General

The estimate has generally been developed from first principles. Exceptions are the plant maintenance materials cost which is a factored estimate based on the plant direct capital cost, and parts of the G&A cost which are allowances based on ICO's previous project experience in the area.

All development and processing costs obtained prior to production status being declared have been omitted from the operating cost estimate (pre-production costs), along with mine development and TWSF expansion costs (sustaining capital).

21.11.2 Mining

Mining costs reflect all costs directly associated with mining of the Ram deposit including drilling, blasting, excavation, waste handling, paste backfill ore handling, mining vehicles, roads, fuel, power and labour. Costs associated with the pre-production period (year -1) and ongoing development (sustaining capital) were capitalized and are not included.

The mining cost estimate was developed in-house and is supported by:

- Budget estimates from three contractors; and
- Estimation of haulage profiles for each source and destination over the life of the project.

The mining costs include:

- Fuel costs;
- Power cost;
- Backfill cement cost. An additional cost of paste backfill support has also been allowed for;
- Contract mine management, survey and planning based on quoted rates;
- Mining labour costs were all included on a rates basis of mining, backfill and development requirements of the Life of Mine plan; and
- Haulage costs based on quoted rates for all ore transportation and loading. Included and separate to these costs were haulage to the Treatment Plant and Tailings Storage Waste facility.

The mining costs exclude:

- Sample assaying which is provided for within the plant laboratory estimate;
- Mobilization costs (included in pre-production costs) and demobilization (included in sustaining capital costs); and
- Mine development is included as pre-production or sustaining capital costs.

The mining contractor's cost rates are tabulated in Table 21-13.

Table 21-13: Mining Contractor Cost

Unit Costs	Unit	Cost
Sill - good	US$/s ton	59.50
Sill - poor	US$/s ton	70.26
Retreat L3/4	US$/s ton	44.75
Back Stope L5/6	US$/s ton	46.75
Waste Fill haul from surface	US$/s ton	12.50
Paste fill support (per ton)	US$/day	1,600
Survey support	US$/day	900
Mine planning	US$/day	1,000

Tabulated below are the owner's cost rates utilized in Table 21-14.

Table 21-14: Owners Cost Rates

Description	Consumption	Unit	Cost	Unit
Power	15	KWH/s t	0.055	$/KWH
Fuel	0.85	Gal/s t	3.00	$/gal
Haul Portal to-from ROM/TWSF	1	Unit	2.5	$/unit
Tails to TWSF	1	Unit	0.75	$/unit
Cement	5%	Unit	200	$/unit
Mine Management & Geo	1	Unit	5776.8	$/unit

Depicted below is the mining operating cost breakdown in Figure 21-2.

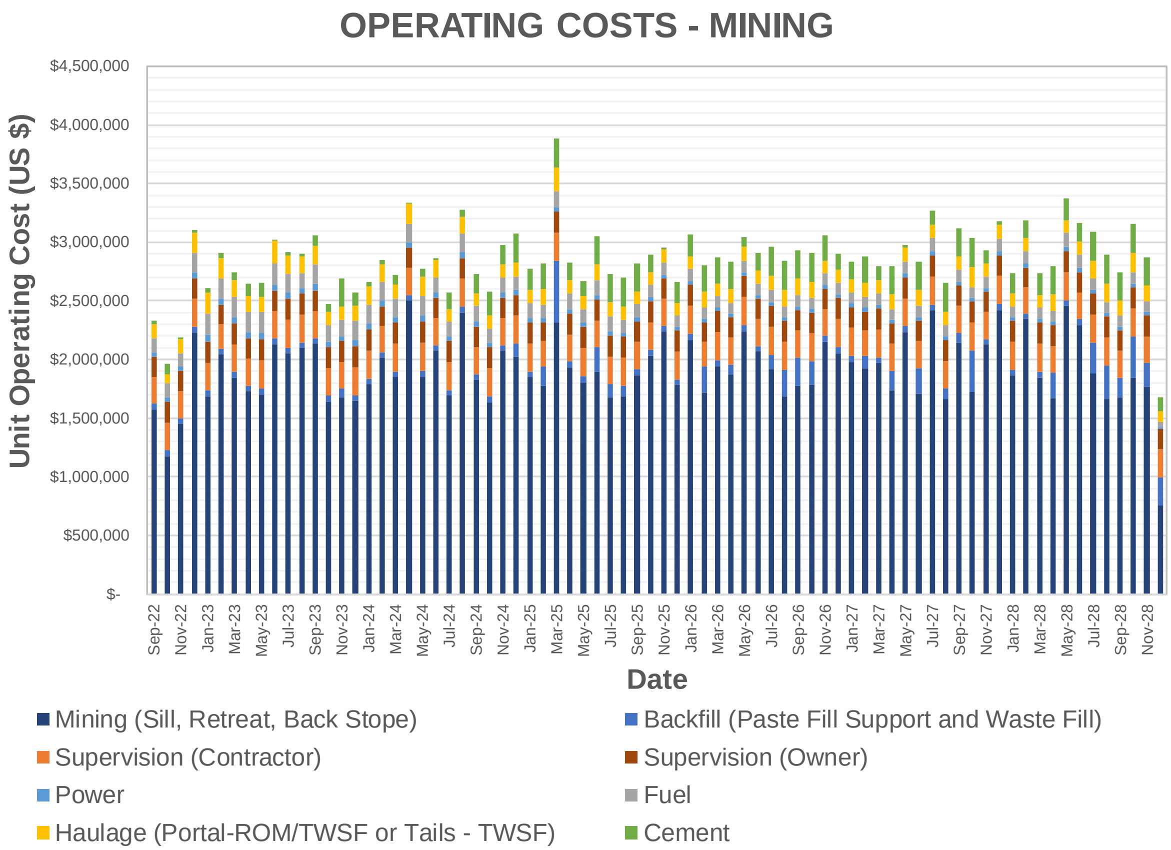


Figure 21-2: Mining Operating Cost Breakdown

Tabulated below are the annualized operating cost breakdowns in Table 21-15.

Table 21-15: Mining Operating Cost Breakdown (US $ Million)

Description	LOM (US$M)	2022 Sept-Dec	2023 Jan - Dec	2024 Jan - Dec	2025 Jan - Dec	2026 Jan - Dec	2027 Jan - Dec	2028 Jan - Dec
Contractor	156.864	6.855	23.685	25.077	24.996	25.466	26.053	24.730
Owner	46.595	2.025	7.389	7.214	7.676	7.565	7.141	7.584
Mining Total	203.458	8.880	31.074	32.291	32.672	33.031	33.194	32.315

21.11.3 Processing

Processing costs reflect all costs directly associated with the treatment of Ram deposit in the flotation concentrator at 1,200 short tons per day. Processing costs include:

- Labour;
- Power;
- Reagents and consumables;
- Maintenance materials;

- Laboratory;
- Water Treatment Plant; and
- Mobile equipment leasing costs.

21.11.3.1 Labour

Plant supervision and labour costs were derived from a workforce plan estimate developed by the project team together with market-related rates obtained by the ICO for 2019 with an additional 3% escalation. The labour costs have been developed based on a four shift (8 hours) plant operation for shift workers and a dayshift only for non-shift workers. Employees housed in the camp will generally work a two week on, one week off roster. Salary bands and pay rates have been developed from information advised by ICO. The pay rates include a 40% burden overhead. Tabulated below is the annual supervision cost for processing in Table 21-16.

Table 21-16: Processing Supervision Cost

Position	Salary Scale	Quantity	Total Annual Cost	
			$ / annum	
Mill Manager	Prevailing	1	$	173,040
Mill maintenance foreman	15% over Tech 5	1	$	118,386
Metallurgist	Prevailing	1	$	144,200
Operations Foreman	15% over Tech 5	4	$	473,544
Annual		**7**	**$**	**909,170**

Tabulated below is the annual supervision cost for processing in Table 21-17.

Table 21-17: Processing Labour Cost

Position	Salary Scale	Quantity	Total Annual Cost	
			US$ / annum	
Mechanic	Tech 5	5	$	514,722
Electrician	Tech 5	4	$	411,778
Flotation/Control	Tech 5	4	$	411,778
Grind	Tech 4	4	$	390,205
Dewatering	Tech 3	4	$	370,663
Ore Pad/Crushing	Tech 3	4	$	370,663
Operations labour	Tech 1	6	$	466,343
Annual		**31**	**$**	**2,936,151**

21.11.3.2 Power

The unit power cost is based on the overhead line cost of supply at US$0.055/KWh as derived from a commercial contract established with Idaho Power, inclusive of any required capital expenditure (i.e. fully loaded). The power consumption calculation has considered:

- Installed power;
- Absorbed power;

- Duty or standby operation;
- Continuous or intermittent operation; and
- Operating time per year.

The annual power cost is the product of annual power consumption and unit power cost tabulated below in Table 21-18.

Table 21-18: Annual Processing Power Cost

Area	Power Absorbed	Consumption	Power Cost
	KW	MWh/annum	US$ / annum
Crushing and Milling	1170	10219	$562,027
Flotation	357	3121	$171,651
Paste Backfill	289	2523	$138,753
Water Reticulation and Air	300	2619	$144,018
Services	427	3729	$205,085
Total	2542	22210	$ 1,221,534
Total tons treated / annum		s tons	438000
Power Cost		US$ / s ton	US$ 2.79

21.11.3.3 Consumables

Consumables were calculated based on the following:

- Reagent dosages were derived from locked cycle test work conditions, while the reagent costs were selected from three budget quotations obtained. Transportation and delivery the reagent to site were included in the quotations. An allowance for the addition of 10 g/t mill feed of sulfuric acid was included in order to mitigate any alkaline conditions that could potentially occur with backfilling in the mining process;
- The wear media costs (liners, grates, screen panels, etc.) was determined by the project team, estimating the number of annual changeouts and the costs selected based on vendor budget unit prices;
- Grinding media consumption rates were calculated based on test work inputs (ore abrasivity) and the costs selected based on three vendor budget quotations. Transportation and delivery the grinding media to site was included in the quotations; and
- Costs of concentrate bagging were calculated based on calculated concentrate production profiles expected and the capacity of lined bags and US$0.35/ bag.

Table 21-19: Annual Reagent Processing Cost

Consumables	Consumption			Selected Price	Cost	
	g / t	kg/hr.	t / month	US $ / kg	US $ / annum	US $ / s ton
PAX (Potassium Amyl Xanthate)	210	10.35	6.95	$ 2.43	$ 202,766	$ 0.46
Frother (Aeroflot AF65)	2.50	0.12	0.08	$ 2.98	$ 2,957	$ 0.01
Acid (H_2SO_4)	10	0.49	0.33	$ 1.10	4,380	$ 0.01

Consumables	Consumption			Selected Price	Cost	
	g / t	kg/hr.	t / month	US $ / kg	US $ / annum	US $ / s ton
Flocculant (Hychem AF 304)	25.00	1.23	0.83	$ 3.31	$ 32,880	$ 0.08
Total					$ 242,983	$ 0.55

Table 21-20: Annual Wear Media Processing Cost

Wear Media	Cost / Set (USD)	Sets / year	US $ / year	US $ / s ton
Jaw Crusher Liners	$ 7,541	1	$ 7,541	$ 0.02
SAG Mill Trommel Panels	$ 5,500	1.5	$ 8,250	$ 0.02
SAG Liners	$ 261,600	1	$ 261,600	$ 0.60
SAG Discharge Grate	$ 35,000	1	$ 35,000	$ 0.08
Ball Mill Liners	$ 30,000	1	$ 30,000	$ 0.07
Ball Mill Discharge Grate	$ 25,455	1	$ 25,455	$ 0.06
Ball Mill Trommel Panels	$ 4,000	1.5	$ 6,000	$ 0.01
Trash Screen Panels	$ 5,100	2	$ 10,200	$ 0.02
Total			$ 384,046	$ 0.88

Table 21-21: Annualized Grinding Media Costs

Grinding Media	Consumption				Selected Vendor Pricing	Cost
	Range	Selected	t / month	US $ / metric t	US $ / annum	US $ / s ton
SAG Mill Balls (100 mm)	122 - 249 g / t	200 g / t	6.62	1080.00	$ 85,827	$ 0.20
Ball Mill Balls (50 mm)	200 - 490 g / t	392 g / t	12.98	1080.00	$168,221	$ 0.38
Total					$ 254,058	$ 0.58

21.11.3.4 Maintenance

The plant maintenance materials cost was calculated using factors applied to the installed capital cost of new equipment purchased by the project. The purchasing price of existing equipment was escalated and added to the new equipment capital costs. The cost of refurbishing the existing equipment previously purchased has been subtracted from the CAPEX principal amount.

Table 21-22: Processing Maintenance Cost Estimate

Description	US$ Cost
Capex Estimate (2020) Feasibility Study (total mechanicals, incl existing refurbishment costs)	$ 10,483,624
Less Refurbishment Costs (2020) Feasibility mechanicals refurbishment costs	$ (882,253)
Additional Pre-Purchased Mechanical Capex Estimate	$ 3,515,755
Total Mechanical Capex	$ 13,117,126
Mechanical Equipment cost factor	4.00%
Maintenance Cost (US $ / annum)	$ 524,685
Maintenance Cost (US $ / s ton)	$ 1.20

21.11.3.5 Laboratory

Laboratory analysis costs for mining development, processing and concentrate sampling, included the following based on a sampling plan derived by the project team:

- Contract laboratory services supervision and labour;
- ICO will provide the capital for the laboratory to be built and therefore this is not included in the $ / ton rate;
- Costs of samples per month of production based upon a $ / ton rate; and
- Two product composite samples per day to be fire assayed off-site based on market rates provided by ICO.

Table 21-23: Laboratory Services Cost

Description	Total Cost (US $ / a)	Total Cost (US $ / s ton)
Onsite Lab Variable Cost	$ 103,442	$ 0.24
Onsite Lab Fixed Cost	$ 988,953	$ 2.26
Offsite Gold Fire Assay	$ 65,520	$ 0.15
Total	$ 1,156,915	$ 2.64

21.11.3.6 Water Treatment Plant

The Water Treatment Plant costs obtained by ICO were estimated based upon:

- Running absorbed power estimates of 781.8 MWh/annum at the cost of US$42,994/annum or US$1.74/1,000 gal treated. This excluded the power for the pump-back system which would only be utilized under leak detection conditions;
- Reagent quantities and pricing based upon vendor estimates. Total reagents cost US$56,666/annum or US$2.29/1,000 gal treated;
- The vendor advised IX Resin Exchange media recharge/disposal costs inclusive of haulage of US$38,556 (or US$1.56/1,000 gal treated). It is assumed that all non-hazardous filter press waste is placed within the TWSF;
- No labour is allowed for as it was included within the treatment plant operations duties; and
- An advised allowance for maintenance spare parts of US$21,209 (or US$0.86/1,000 gal treated).

A site-wide water balance allowed for the level of the Water Management Ponds to be estimated over the life of mine. Any water in excess of 50% of the Water Management Pond capacity was then treated by the WTP in month-long campaigns. The monthly site water balance was based upon the following assumptions:

- Seasonal fluctuation of 38 – 52 gpm from mine dewatering from the second year of operation;
- A seasonal quantity of snowmelt/precipitation 0 – 20 gpm;
- A seasonal TWSF dust control usage of 0 – 5 gpm;
- Processing treatment plant concentrator make-up and run-off as per its water balance; and
- A WTP treatment capacity of 150 gpm.

The estimated number of WTP operating months per annum is tabulated below in Table 21-24.

Table 21-24: Months of Water Treatment Plant Operation

Description	2022 Sept - Dec	2023 Jan - Dec	2024 Jan - Dec	2025 Jan - Dec	2026 Jan - Dec	2027 Jan - Dec	2028 Jan - Dec
Months of Operation	0	0	1	4	4	5	4

21.11.4 Concentrate Logistics

Product concentrate logistics was calculated by using budgetary quotations for concentrate bags to potential third-party customers of the bulk product. These costs included for all road and rail activities required to transport concentrate from the site.

All concentrate will be transported in lined bags placed inside 20 ft shipping containers prior to leaving the site. Each container will hold 18 tonnes of product. It is assumed that no concentrate is shipped prior to production status being declared.

21.11.5 Tailings Waste Storage Facility (TWSF)

Haulage and compaction of material to the TWSF is included in the haulage costs within the mining estimate. No additional costs have been included for the TWSF.

21.11.6 General and Administration

21.11.6.1 General

G&A costs excluding labour and camp cost were provided by ICO, calculated as the sum of all non-mining and non-processing general and administration costs as tabulated in Table 21-25.

Table 21-25: G&A Cost Summary Excluding Labour and Camp

Cost Item	US$/Annum	Description
Accounting (excluding labour)	25,000	Software fees, audits
Safety (excluding labour)	100,000	PPE, IH Sampling, training
Human Resources (excluding labour)	50,000	Recruiting, fees
Environmental Dept. (excluding Labour)	250,000	Compliance sampling, seeding, runoff control
Security (excluding labour)	10,000	Fuel, consumables
Janitorial Services (contract)	25,000	trash fees, consumables
Community Relations (excluding labour)	50,000	Outreach meetings, forums, community events
Office Operating Supplies and Postage	5,000	
Maintenance Supplies	10,000	Maintenance for non-mining or process assets (warehouse, admin)
Power	15,000	Power for non-operations (admin and warehouse)
Propane	10,000	Heat for warehouse
Phone/Communications	60,000	Includes contract IT service & supplies
Licenses, Fees, and Vehicle Taxes	100,000	
Legal	50,000	
Subs, Dues, PR, and Donations	25,000	
Travel, Lodging, and Meals	50,000	Assumes corporate travel to site not included
Training	25,000	Trades, technical
Road Use Fees - USFS and County Maintenance	200,000	Use cost of access road for resurfacing by USFS ($50k/yr.) and County Maintenance Allowance for snow removal, grading, and dust control ($150k/yr.)

Cost Item	US$/Annum	Description
Total per year	1,060,000	
Cost per short ton	2.42	

21.11.6.2 Camp

The required camp size was calculated based on the required manning estimates for the various mining, concentrator and owner labor requirements, in consideration of the various leave rotations. The camp running and catering costs are based on quotations received from contractors with experience in the area.

- Estimated number of rooms required 88 rooms
- Contingency 7 rooms
- Estimated camp capacity 95 rooms

Table 21-26: Camp Operating Cost Calculation

Description	Quantity
Grand Total Operations	164
Grand Total Camp	156
Dayshift Camp Total	20
Rotation Camp Total	136
Annual Person-Day	28991.4
Cost per Person Day	US$75.00
Total Annual Cost	US$2,174,357
Cost per short ton	US$4.96

G&A supervision and labor costs were derived from a workforce plan estimate tabulated below in Table 21-27. The ICO obtained Market-related rates.

The labour costs have been developed based on a four shift (8 hours) plant operation for shift workers and a dayshift only for non-shift workers. Employees housed in the camp will generally work a two week on, one week off roster. Salary bands and pay rates have been developed from information advised by ICO. The pay rates include a 40% burden overhead.

Table 21-27: G&A Labour Cost (US$)

Position	*Quantity*	Total Cost / Annum
G&A		
General Manager	1	$ 260,000
Environmental Manager	1	$ 130,000
Environmental Technician (a)	1	$ 70,200
Environmental Technician (b)	1	$ 70,200
Manager of Human Resources	1	$ 130,000
Admin/HR Assist	1	$ 54,600
Controller	1	$ 195,000
Purchasing Superintendent	1	$ 117,000
Accounting Clerk	1	$ 54,600
Warehouseman	1	$ 65,000

Position	*Quantity*	Total Cost / Annum
G&A		
Camp Staff	8	$ -
Totals	**10**	**$ 1,146,600**
Health & Safety		
Safety Manager – Site	1	$ 130,000
Safety Tech	1	$ 70,200
Paramedic	3	$ 160,618
Site Security & Piloting	5	$ 233,200
Totals	**10**	**$ 594,018**
Surface		
Surface Support Manager	1	$ 104,000
Surface Equip Operator	3	$ 165,000
Totals	**4**	**$ 269,000**
Summary	**Number**	**Totals [US $/annum]**
G&A	10	$ 1,146,600
Mine (refer to Mine OPEX)	0	
H&S and Security	10	$ 594,018
Mill Processing (Refer to Processing OPEX)	0	
Surface	4	$ 269,000
Total	24	$ 2,009,617.60
Total G&A Labor / s ton		$ 4.59

21.12 ESTIMATING CONTRIBUTORS

Table 21-28 summarizes the primary contributors to the operating cost estimate.

Table 21-28: Operating Cost Estimate Contributors

Item Description	Contributor
Mining	ICO / 9140697 Canada Inc
Processing	
Labor	ICO / DRAA
Power	ICO / DRAA
Consumables	DRAA
Maintenance	M3 Engineering / DRAA
Laboratory	ICO / DRAA
Water Treatment Plant	ICO
Concentrate Logistics	ICO
G&A Costs	
General	ICO
Camp	M3 Engineering
Labor	ICO

21.13 CONTINGENCY

The estimate does not include any contingency allowance.

21.14 ESCALATION AND INFLATION

The estimate does not include any allowance for escalation or inflation.

21.15 TAXES AND DUTIES

All costs have been calculated excluding customs excise, duties and taxes.

22 ECONOMIC ANALYSIS

The economic assessment of the Idaho Cobalt Operation ("ICO") is based on Q3-2020 price projections and cost estimates in U.S. currency all on a real basis (excluding inflation). Discount rates and Internal Rate of Return ("IRR") are also in real terms. The evaluation was carried out on a 100% equity basis. Current U.S. federal and Idaho state tax regulations were applied to assess the project's tax liabilities. US dollar price forecasts of US$25.00/lb for cobalt, US$3.00/lb for copper and US$1,750/oz for gold are assumed.

The financial indicators under base case conditions are listed in Table 22-1.

Table 22-1: Base Case Financial Indicators

Base Case Financial Results	Unit	Value
Pre-tax NPV @ 8%	M USD	113.4
Post-tax NPV @ 8%	M USD	95.7
Pre-tax IRR	%	41.8
Post-tax IRR	%	37.6
Pre-tax Payback Period	years	2.6
Post-tax Payback Period	years	2.8

A sensitivity analysis reveals that the Project's viability will not be significantly vulnerable to variations in capital and operating costs, within the margins of error associated with Feasibility Study ("FS") estimates. However, the Project's viability remains more vulnerable to the larger uncertainty in future market prices.

22.1 ASSUMPTIONS

22.1.1 Macro-Economic Assumptions

The main macro-economic assumptions used in the base case are given in Table 22-2. Details on the derivation of the price forecasts are given in Section 19 of this Report. The sensitivity analysis examines a range of prices 30% above and below this base case forecast.

Table 22-2 Macro-Economic Assumptions

Item	Unit	Base Case Value
Cobalt Price Forecast (Metal Bulletin SG)	US$/lb	25.00
Copper Price Forecast (LME Cash)	US$/lb	3.00
Gold Price Forecast	US$/oz	1,750
Real Discount Rate	% per year	8
Real Discount Rate Variants	% per year	6 and 10

Current U.S. Federal and Idaho state tax regulations were used to assess the Project's annual tax liabilities. These consist of federal and state corporate taxes as well as the Idaho Mining License tax. The federal and state corporate tax rates currently applicable over the Project's operating life are 21.0% and 5.47% of taxable income, respectively. The Mining License tax is assessed at the rate of 1.0% of [Sales less Total Cash Costs]. The tax model was reviewed by Ernst & Young.

The assessment was carried out on a 100% equity basis. Apart from the base case discount rate of 8% (real), two (2) variants of 6 and 10% (also both real) were used to determine the Net Present Value ("NPV") of the Project. These discount rates represent possible weighted-average costs of capital.

22.1.2 Technical Assumptions

The main technical assumptions used in the base case are given in Table 22-3. The commercial (payable) terms that underpin the FS are approximately 75% of the contained metal value in concentrate weighted across cobalt, copper and gold.

Table 22-3: Technical Assumptions

Item		Unit	Base Case Value
Reserve Milled		k metric tonnes	2,486.1
Average Mill Head Grades[1]	Cobalt	%	0.55
	Copper	%	0.80
	Gold	g/mt	0.64
Design Milling Rate		mt/day	1,090
Mine Life		years	7
Average Process Recoveries	Cobalt	%	91.0
	Copper	%	95.4
	Gold	%	84.9
Bulk Concentrate Grade		%	10.0
Bulk Concentrate Production (dry)		k metric tonnes	124.4
Average Mining Costs		(US$/metric tonne milled)	81.81
Average Processing Costs		(US$/metric tonne milled)	20.14
Average Mine & Mill G&A Costs		(US$/metric tonne milled)	14.24
Average Concentrate Transport Cost		(US$/metric tonne milled)	6.98
Average Total Costs[2]		(US$/metric tonne milled)	123.17
[1] All averages based on financial model [2] Excludes Mining License Tax, Real & Property Taxes and Corporate G&A			

22.2 FINANCIAL MODEL AND RESULTS

Figure 22-1 illustrates the post-tax cash flow (after all capital expenditure and working capital movements) and cumulative cash flow profiles of the Project for base case conditions. The intersection of the post-tax cumulative cash flow curve with the horizontal dashed line represents the payback period measured from the start of concentrate sales (Periods are quarterly, Period 1, i.e., Q3 2022).

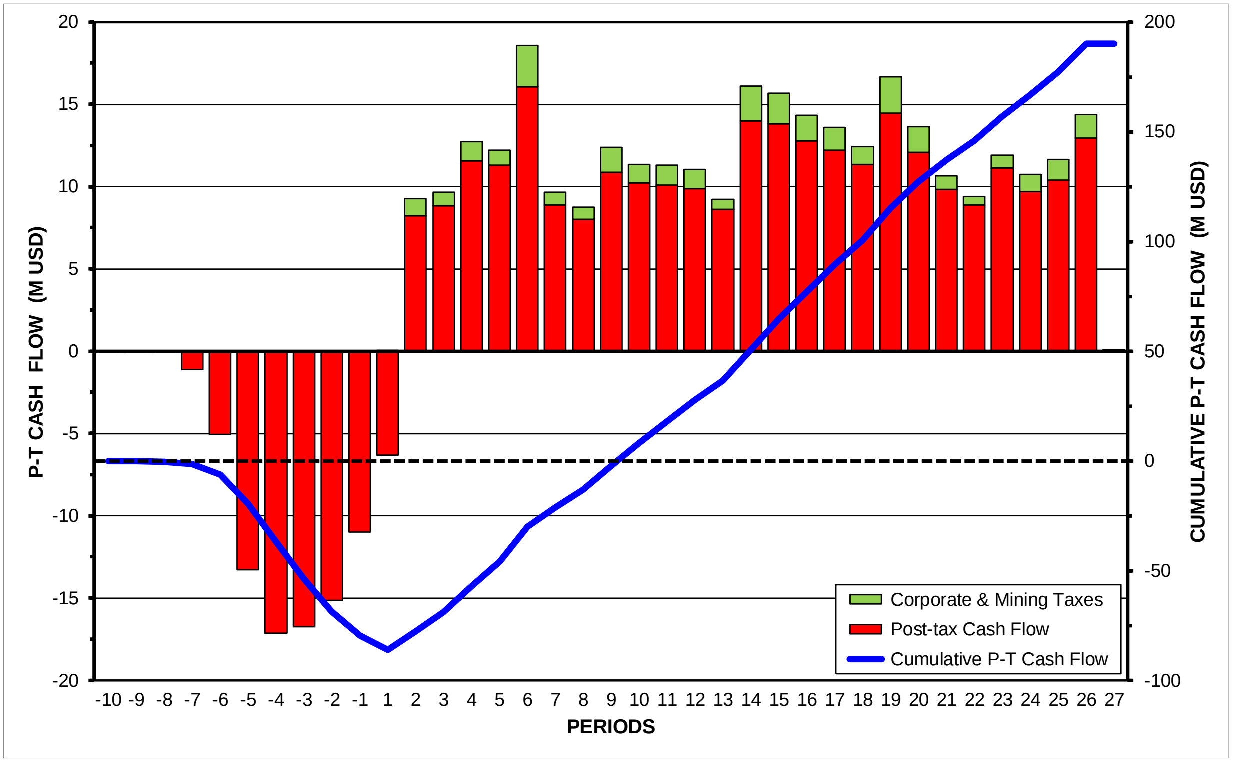


Figure 22-1: Quarterly Post-tax Cash Flow and Cumulative Cash Flow Profiles

A summary of the evaluation results based on the financial model is given in Table 22-4 and Table 22-5 gives the cash flow statement, both for base case conditions.

The summary table and cash flow statement indicate that the total pre-production (initial) capital costs were evaluated at US$78.4M. The sustaining capital requirement was evaluated at US$56.1M. Mine closure and rehabilitation costs were estimated at an additional US$21.2M.

The cash flow statement shows the estimated capital spending schedule (initial and sustaining) over the life of the Project. Working capital requirements were estimated using 15 days of inventory, 30 days of receivables and 45 days of payables. Since operating costs vary annually over the mine life, additional amounts of working capital are injected or withdrawn as required. Mine closure and rehabilitation costs occur from the time production ends and continues for 20 years.

The total revenue derived from the sale of the concentrate products was estimated at US$667.4M (US$541.2M for Co, US$99.7M for Cu, and US$53.4M for Au), or on average, US$268.46/metric tonne milled. The total operating costs were estimated at US$315.7M, or on average, US$127.01/metric tonne milled.

The financial results indicate a pre-tax NPV of US$113.4M at a discount rate of 8%. The pre-tax IRR is 41.8% and the payback period is 2.6 years. The NPV is assessed at the start of Q1, 2020. The payback period is measured from the end of Q4, 2021.

The post-tax NPV is US$95.7M at a discount rate of 8%. The post-tax IRR is 37.6% and the payback period is 2.8 years.

Table 22-4: Project Evaluation Summary – Base Case

Item	Unit	Value
Total Revenue	M USD	667.4
Total Operating Costs	M USD	315.7
Initial Capital Costs (excludes Working Capital)	M USD	78.4
Sustaining Capital Costs	M USD	56.1
Mine Closure & Rehabilitation Costs	M USD	21.2
Average Operating (EBITDA) Margin	%	53.4
Total Pre-tax Cash Flow	M USD	198.5
Pre-tax NPV @ 6%[1]	M USD	130.8
Pre-tax NPV @ 8%	M USD	113.4
Pre-tax NPV @ 10%	M USD	98.1
Pre-tax IRR	%	41.8
Pre-tax Payback Period[2]	Years	2.6
Total Post-tax Cash Flow	M USD	170.9
Post-tax NPV @ 6%	M USD	111.1
Post-tax NPV @ 8%	M USD	95.7
Post-tax NPV @ 10%	M USD	82.1
Post-tax IRR	%	37.6
Post-tax Payback Period[2]	Years	2.8
[1] NPVs based on mid-period discounting convention [2] Measured from the end of Q4, 2021		

Table 22-5: Cash Flow Statement – Base Case

		Periods	-10	-9	-8	-7	-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8
JERVOIS ICO PROJECT		Quarters	Mar-20	Jun-20	Sep-20	Dec-20	Mar-21	Jun-21	Sep-21	Dec-21	Mar-22	Jun-22	Sep-22	Dec-22	Mar-23	Jun-23	Sep-23	Dec-23	Mar-24	Jun-24
Mill Feed		kt											32.659	84.550	97.976	99.065	100.153	100.153	99.065	99.065
Co Grade		%											0.58%	0.64%	0.64%	0.68%	0.63%	0.71%	0.53%	0.54%
Cu Grade		%											0.46%	0.57%	0.96%	1.03%	0.89%	1.28%	0.91%	0.87%
Au Grade		g/t											0.70	0.65	0.71	0.67	0.71	0.88	0.55	0.52
Processing Recoveries																				
Cobalt		%											81.6%	91.9%	90.7%	90.7%	90.9%	90.3%	90.4%	90.5%
Copper		%											84.2%	96.0%	96.0%	96.2%	96.0%	96.4%	95.4%	95.5%
Gold		%											84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%
Bulk Concentrate Production		k dmt											1.542	4.971	5.657	6.109	5.746	6.415	4.702	4.805
Payable Content	Payable																			
Cobalt	75%	mt											115.650	372.840	424.270	458.208	430.928	481.162	352.639	360.392
Copper	80%	mt											101.802	367.307	719.279	785.247	683.653	985.488	684.693	654.765
Gold	70%	Oz											436.28	1053.19	1320.41	1273.39	1364.58	1683.59	1035.50	978.33
Revenue																				
Cobalt		M USD											6.374	20.549	23.384	25.254	23.751	26.519	19.436	19.863
Copper		M USD											0.673	2.429	4.757	5.194	4.522	6.518	4.528	4.331
Gold		M USD											0.763	1.843	2.311	2.228	2.388	2.946	1.812	1.712
Total		M USD											7.811	24.822	30.452	32.676	30.660	35.984	25.776	25.906
Operating Expenses																				
Mining Costs		M USD										0.000	0.175	0.562	0.691	0.701	0.637	0.620	0.640	0.629
Processing Costs		M USD										0.638	1.056	1.208	1.247	1.250	1.253	1.253	1.250	1.250
Mine & Mill G&A Costs		M USD										1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311
Concentrate Transport Costs		M USD											0.215	0.693	0.789	0.852	0.801	0.895	0.656	0.670
Real & Property Taxes		M USD	0.096	0.092	0.088	0.085	0.084	0.086	0.110	0.144	0.177	0.193	0.202	0.212	0.212	0.209	0.208	0.203	0.197	0.191
Corporate G&A		M USD				0.032	0.164	0.370	0.445	0.334	0.118	0.059	0.010							
Total Operating Costs			0.096	0.092	0.088	0.117	0.247	0.456	0.555	0.478	0.295	2.201	5.199	10.849	12.133	12.277	12.768	11.768	11.914	12.665
EBITDA Margin (%)													33.4%	56.3%	60.2%	62.4%	58.4%	67.3%	53.8%	51.1%
Capital Expenses																				
Initial Capital Costs		M USD			0.012	0.982	4.136	12.835	16.565	16.273	11.307	8.771	7.521							
Sustaining Capital Costs		M USD											1.387	4.864	7.029	7.874	5.859	5.871	5.479	4.638
Working Capital		M USD					0.663	0.000	0.000	0.000	3.527	0.000	0.000	0.000	1.823	0.000	0.000	0.000	-1.149	0.000
Mine Closure & Rehabilitation Costs		M USD																		
Idaho Mining License Tax		M USD											0.029	0.143	0.187	0.208	0.183	0.247	0.142	0.136
Federal & State Corporate Taxes		M USD												0.897	0.613	0.956	0.715	2.302	0.665	0.607
Cash Flows																				
Pre-tax Free Cash Flows		M USD	-0.096	-0.092	-0.101	-1.099	-5.046	-13.291	-17.120	-16.751	-15.129	-10.971	-6.297	9.109	9.467	12.525	12.033	18.344	9.533	8.603
Post-tax Free Cash Flows		M USD	-0.096	-0.092	-0.101	-1.099	-5.046	-13.291	-17.120	-16.751	-15.129	-10.971	-6.297	8.212	8.853	11.569	11.318	16.042	8.868	7.996

Periods	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	To	
JERVOIS ICO PROJECT	Sep-24	Dec-24	Mar-25	Jun-25	Sep-25	Dec-25	Mar-26	Jun-26	Sep-26	Dec-26	Mar-27	Jun-27	Sep-27	Dec-27	Mar-28	Jun-28	Sep-28	Dec-28	Mar-29	Dec-47	TOTAL
Mill Feed	100.153	100.153	97.976	99.065	100.153	100.153	97.976	99.065	100.153	100.153	97.976	99.065	100.153	100.153	99.065	99.065	100.153	82.749			2486.063
Co Grade	0.58%	0.52%	0.55%	0.55%	0.48%	0.48%	0.47%	0.44%	0.44%	0.50%	0.64%	0.58%	0.51%	0.48%	0.49%	0.53%	0.54%	0.61%			
Cu Grade	0.92%	0.84%	0.95%	0.63%	0.58%	1.35%	1.35%	1.44%	1.30%	0.82%	0.50%	0.45%	0.42%	0.38%	0.40%	0.38%	0.30%	0.41%			
Au Grade	0.55	0.58	0.51	0.53	0.54	1.12	1.25	1.07	0.93	0.53	0.52	0.47	0.45	0.44	0.35	0.49	0.42	0.60			
Processing Recoveries																					
Cobalt	90.5%	90.5%	90.5%	91.5%	91.6%	89.0%	88.9%	88.6%	88.8%	90.9%	92.6%	92.6%	92.4%	92.5%	92.4%	92.8%	93.5%	93.0%			
Copper	95.7%	95.4%	95.6%	95.6%	95.1%	95.1%	95.0%	94.8%	94.8%	95.2%	96.1%	95.7%	95.3%	95.1%	95.2%	95.4%	95.5%	95.9%			
Gold	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%	84.9%			
Bulk Concentrate Production	5.248	4.755	4.921	4.986	4.424	4.302	4.092	3.827	3.878	4.565	5.834	5.309	4.728	4.489	4.481	4.840	5.060	4.718			124.405
Payable Content																					
Cobalt	393.614	356.616	369.089	373.972	331.769	322.637	306.933	287.034	290.864	342.366	437.568	398.139	354.594	336.664	336.042	363.021	379.517	353.825			9330.357
Copper	703.582	641.827	708.485	477.914	438.725	1027.022	1003.247	1079.250	988.163	621.876	372.971	342.316	322.869	287.300	305.257	287.061	230.399	260.964			15081.462
Gold	1056.19	1110.01	955.25	1004.29	1035.38	2138.14	2334.42	2022.04	1787.02	1005.83	982.45	898.93	861.93	848.36	669.83	925.35	797.89	948.78			30527.369
Revenue																					
Cobalt	21.694	19.655	20.343	20.612	18.286	17.782	16.917	15.820	16.031	18.870	24.117	21.944	19.544	18.555	18.521	20.008	20.917	19.501			514.247
Copper	4.653	4.245	4.686	3.161	2.902	6.793	6.635	7.138	6.536	4.113	2.467	2.264	2.135	1.900	2.019	1.899	1.524	1.726			99.747
Gold	1.848	1.943	1.672	1.758	1.812	3.742	4.085	3.539	3.127	1.760	1.719	1.573	1.508	1.485	1.172	1.619	1.396	1.660			53.423
Total	28.196	25.843	26.700	25.530	22.999	28.317	27.637	26.497	25.694	24.743	28.303	25.781	23.187	21.940	21.712	23.526	23.837	22.888			667.417
Operating Expenses																					
Mining Costs	0.589	0.547	0.545	0.546	0.521	0.477	0.464	0.464	0.465	0.468	0.461	0.463	0.484	0.475	0.472	0.495	0.461	0.411			13.461
Processing Costs	1.253	1.253	1.247	1.250	1.253	1.253	1.247	1.250	1.253	1.253	1.247	1.250	1.253	1.253	1.250	1.250	1.253	1.205			32.874
Mine & Mill G&A Costs	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311	1.311			35.397
Concentrate Transport Costs	0.732	0.663	0.686	0.696	0.617	0.600	0.571	0.534	0.541	0.637	0.814	0.741	0.660	0.626	0.625	0.675	0.706	0.658			17.354
Real & Property Taxes	0.184	0.174	0.161	0.151	0.140	0.129	0.113	0.101	0.088	0.076	0.064	0.054	0.045	0.036	0.026	0.020	0.013	0.006			4.371
Corporate G&A																					1.532
Total Operating Costs	12.349	12.335	13.189	12.238	11.858	12.138	12.303	12.296	12.224	12.432	12.275	12.270	12.628	12.637	12.094	12.909	12.298	11.069			315.743
EBITDA Margin (%)	56.2%	52.3%	50.6%	52.1%	48.4%	57.1%	55.5%	53.6%	52.4%	49.8%	56.6%	52.4%	45.5%	42.4%	44.3%	45.1%	48.4%	51.6%			53.4%
Capital Expenses																					
Initial Capital Costs																		-2.553			75.850
Sustaining Capital Costs	3.609	2.316	2.448	2.401	2.019	0.233	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.095			56.123
Working Capital	0.000	0.000	-0.081	0.000	0.000	0.000	-0.202	0.000	0.000	0.000	-0.484	0.000	0.000	0.000	-2.176	0.000	0.000	0.000	-1.922		0.000
Mine Closure & Rehabilitation Costs																			1.824	19.346	21.170
Idaho Mining License Tax	0.162	0.138	0.138	0.136	0.114	0.165	0.156	0.144	0.137	0.125	0.163	0.137	0.107	0.094	0.097	0.107	0.117	0.119			3.632
Federal & State Corporate Taxes	1.387	0.975	1.065	0.999	0.505	1.964	1.708	1.430	1.251	0.964	2.034	1.427	0.711	0.413	0.681	0.918	1.136	1.333			27.655
Cash Flows																					
Pre-tax Free Cash Flows	12.237	11.191	11.143	10.891	9.122	15.946	15.536	14.201	13.470	12.311	16.511	13.510	10.559	9.303	11.794	10.617	11.539	14.276	0.098		217.876
Post-tax Free Cash Flows	10.850	10.216	10.078	9.892	8.617	13.982	13.828	12.771	12.219	11.347	14.478	12.084	9.848	8.891	11.113	9.699	10.403	12.943	0.098		190.221

22.3 SENSITIVITY ANALYSIS

A sensitivity analysis has been carried out, with the base case described above as a starting point, to assess the impact of changes in pre-production (initial) capital expenses (Capex), sustaining capital expenses (Sus Capex), operating costs (Opex), the cobalt price and all prices (varied together) on the Project's NPV @ 8% discount rate and IRR. Each variable was examined one-at-a-time. An interval of ±30% with increments of 10% was used for all variables.

The pre-tax results of the sensitivity analysis, as shown in Figure 22-2 and Figure 22-3, indicate that, within the limits of accuracy of the cost estimates in this FS (±15%, as shown by the vertical dashed lines), the Project's pre-tax viability does not seem significantly vulnerable to the under-estimation of capital and operating costs, taken one at-a-time. As seen in Figure 22-2, the NPV is more sensitive to variations in Opex than Capex and Sus Capex, as shown by the steeper slope of the Opex curve. As expected, the NPV is most sensitive to variations in price. The NPV remains positive at the lower limit of the cobalt price interval examined (US$17.50/lb) but becomes negative when all prices are varied together by about -26.5% (US$18.40/lb for cobalt, US$2.20/lb for copper and US$1,286/oz for gold).

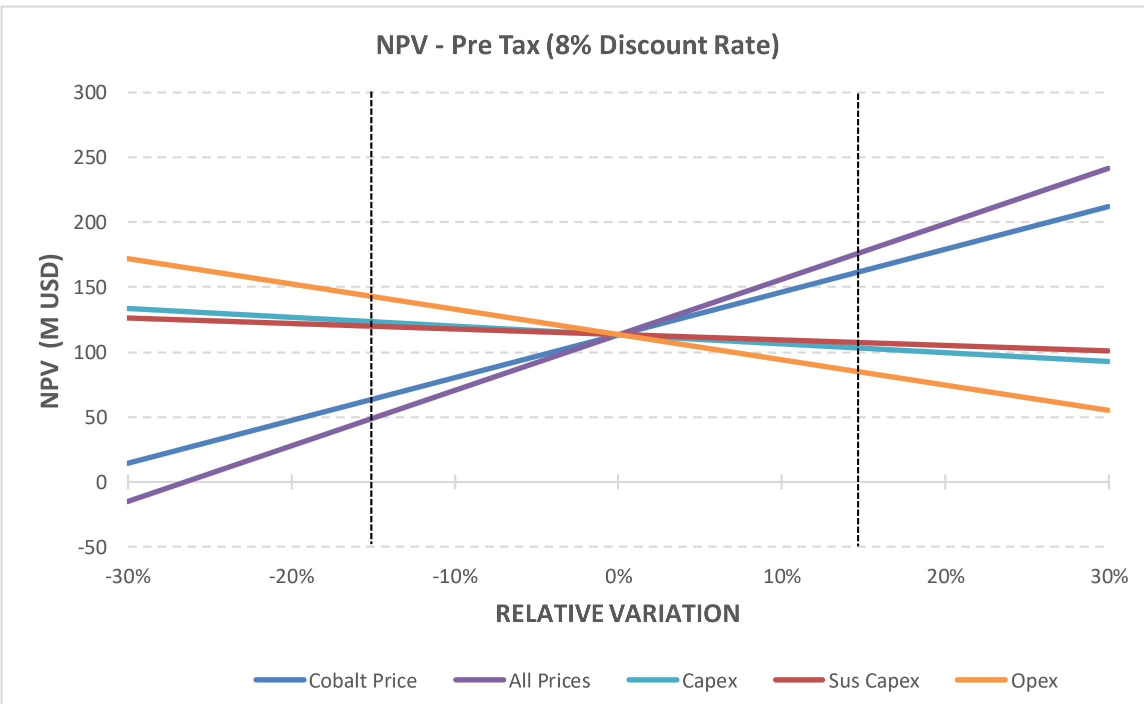


Figure 22-2: Pre-tax NPV$_{8\ \%}$: Sensitivity to Capital Expenses, Operating Costs and Prices

Figure 22-3, showing variations in internal rate of return, provides the same conclusions. The horizontal dashed line represents the base case discount rate of 8% (real). Because of the different timing associated with Capex versus Opex, the IRR is more sensitive to changes in Capex than Opex for negative variations.

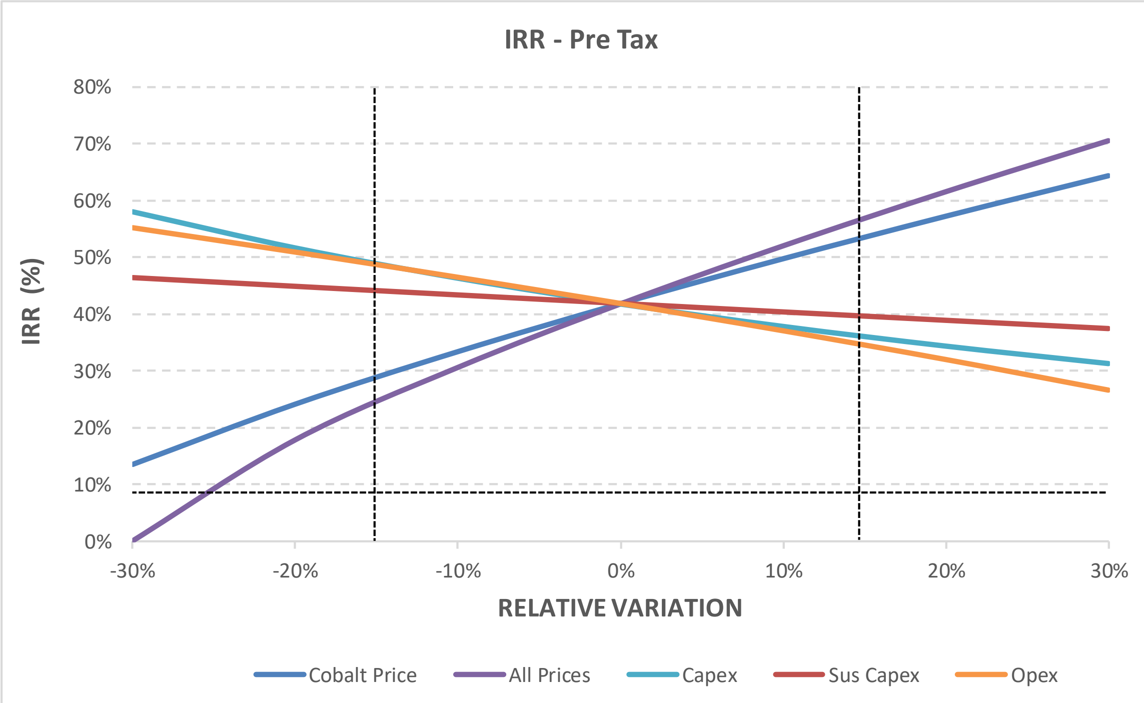


Figure 22-3: Pre-tax IRR: Sensitivity to Capital Expenses, Operating Costs and Prices

The same conclusions can be made from the post-tax results of the sensitivity analysis as shown in Figure 22-4 and Figure 22-5. Figure 22-4 indicates that the Project's post-tax viability is mostly vulnerable to a price forecast reduction, while being less affected by the under-estimation of capital and operating costs. The NPV remains positive at the lower limit of the cobalt price interval examined (US$17.50/lb) but becomes negative when all prices are varied together by about -25.4% (US$18.60/lb for cobalt, US$2.25/lb for copper and US$1,305/oz for gold).

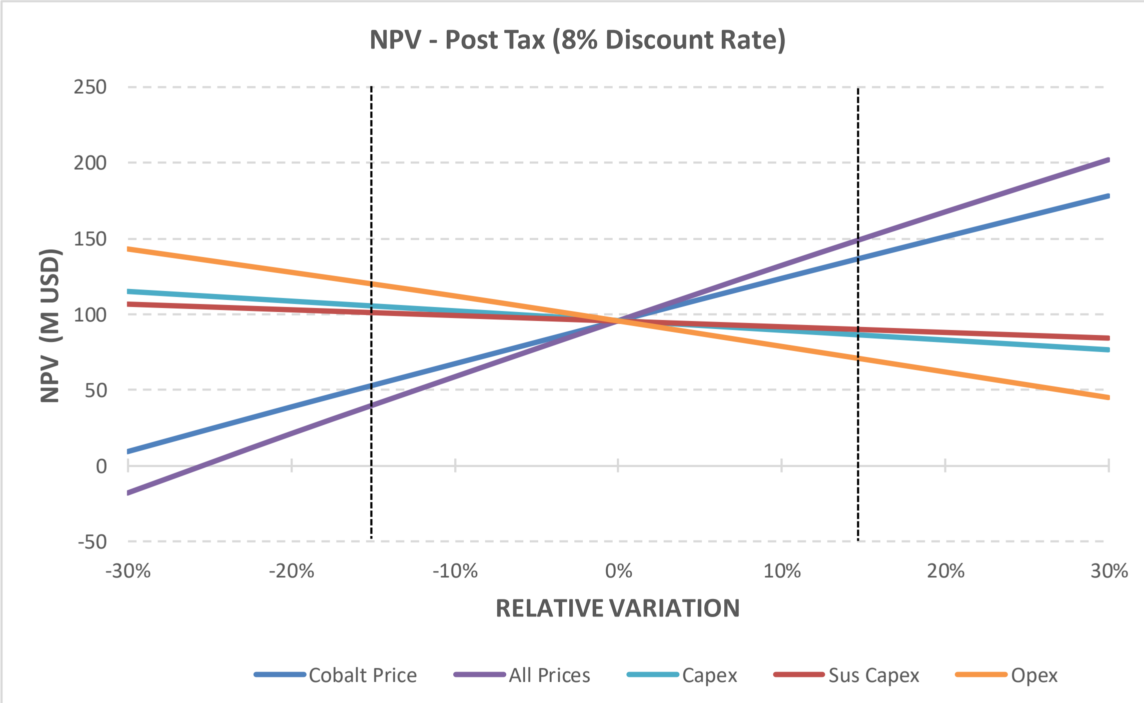


Figure 22-4: Post-tax NPV$_{8\ \%}$: Sensitivity to Capital Expenses, Operating Costs and Prices

Figure 22-5, showing variations in the internal rate of return, provides the same conclusions.

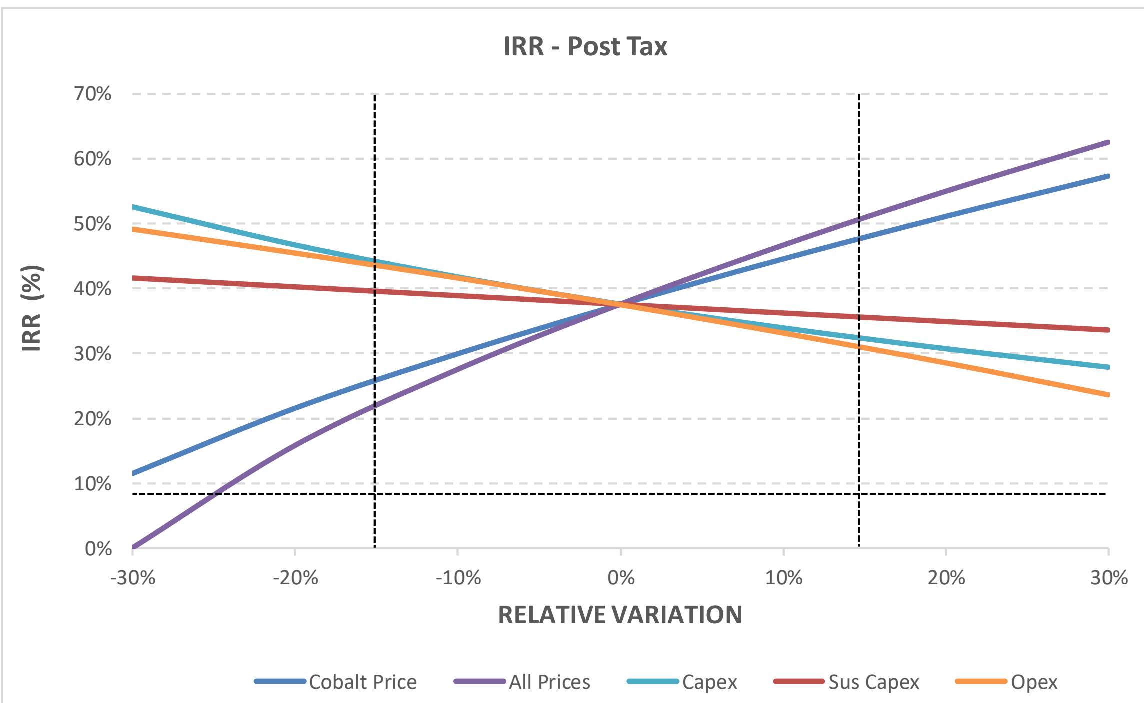


Figure 22-5: Post-tax IRR: Sensitivity to Capital Expenses, Operating Costs and Prices

23 ADJACENT PROPERTIES

The historical Blackbird mine is enclosed by the ICO property mineral claims, see figure 4.4. The Blackbird mine is no longer in production, has undergone remediation and continues with water treatment for the mine and tailings runoff waters.

In the course of producing the Resource Model for the Ram orebody, Orix Geoscience developed a geological model incorporating historical data for the ICO property and data from the 2019 drilling campaign. It is generally agreed that the Ram is an extension, northwards of the Blackbird mine sequence, with fault offsets. Technical papers authored by the Geological Society of America discuss in detail the mineralization of the Blackbird mine and its associations with the Ram deposit. In particular the Geological Society of America Special Paper 522, Bookstrom et al.

To the north of the Ram mineral claims is the Tinkers Pride and Bonanza cobalt prospects which are currently owned by Battery Mineral Resources. These 2 prospects were historically small scale mined for copper and cobalt and more recently surface sampling has been undertaken by Battery Mineral Resources.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT IMPLEMENTATION PLAN

The ICO endeavours to leverage as much of the engineering, procurement, fabrication, construction, and infrastructure available from the previously completed feasibility studies and work completed to date, as it pertains to permits, service agreements, human resources, sub-contractors, management systems, and other synergies.

The implementation plan outlines the contracting strategy selected for the feasibility study, which forms the basis of the project schedule summarized below and the capital cost estimate as detailed in Section 21.

Described herein is the implementation plan considered most appropriate for the design, engineering, transportation, construction and commissioning of the facilities, infrastructure and services for the Project. The contracting strategy also describes the number of contracts proposed to execute the Project and the type of contract commercial instrument proposed for each.

24.2 OBJECTIVES

The key objectives are as follows:

Business Objective

- To maximize value* of the ICO resource by optimizing its processing capacity and leveraging existing infrastructure, while minimizing risks to the business.

  ** Value is defined as maximum Net Present Value (NPV) while considering Earnings Before Interest, Taxes, Depreciation and Amortization (EBITDA) and impact to Internal Rates of Return (IRR)*

Project Objectives

- Deliver a technically robust process plant that meets the Process and Engineering Design Criteria and that is simple to operate and maintain
- Identify and appropriately manage project risk
- Pursue opportunities to improve the project schedule and cost
- Capture value from existing infrastructure
- Proactively manage the execution/implementation phase to minimize business risk.
- Proactively manage stakeholder expectations
- Deliver the Project in a safe and socially responsible manner, while maintaining ICO's external relations and environmental standards

24.3 ORGANIZATION

An integrated team from Jervois's owners team at ICO and contractors will be mobilized to execute final ICO construction and commissioning.

The ICO build will be led by Jervois executives in key positions: the ICO project director, ICO construction manager and health and safety manager, with support from a core owner's team that will ultimately form the basis of the operations team.

The EPCM contractor will provide engineering, procurement, fabrication, quality assurance, and logistic support services. The EPCM contractor will have limited involvement in material control, construction management, pre- and cold commissioning which will be led by Jervois.

24.4 PROJECT DELIVERABLES

- Implement ICO health, safety and environmental policies and standards, and follow throughout project execution
- Comply with the approved Plan of Operations ("PoO")
- Achieve an operational throughput of 1,200 stpd by 31 August 2022
- Deliver the Project on schedule
- Deliver the Project within the Capital Cost Estimate ("CCE") budget
- Integrate effectively with ICO operations and utilize existing infrastructure, and resources, to ensure fit-for-purpose execution
- Ensure execution of the operational readiness plan, to enable seamless transition from Construction to operation.

24.5 SCHEDULE

The schedule includes an eight-month period for procurement of long-lead items and detailed engineering, followed by an eleven-month construction and a five-month commissioning period, including ramp-up to commercial production.

Due to the constraints of seasonal construction, the environmental systems which are required to commence mine development will commence in June 2021. These environmental systems include completion of the Phase 1A TWSF, the Pump back system, the water treatment plant, water management ponds and completion of the portal bench.

Mining pre-production development based on the production schedule detailed in Section 6 of this report will be completed in 10 months to coincide with commissioning of the concentrator.

The start date for implementation/execution is September 2020, with commissioning completed and first ore delivered in June 2022. The planned milestone dates are summarized in Table 24-1.

Table 24-1 Key Date Schedule

Milestone Description	Milestone Date
Jervois ICO 1200 stpd Project Implementation	1-Oct-20
Start Detail Design & Engineering	1-Nov-20
Mobilization to Site	1-May-21
Start Construction	1-Jun-21
Completion of Portal Bench	31-Aug-21
Start Mining Development	1-Sep-21
Complete Concentrator Construction	30-Mar-22
Complete Mine Pre-Production Development	30-Jun-22
Start Hot Commissioning	1-Jun-22
Handover	31-Aug-22

24.6 CAPITAL COST

The Project will move forward using the CCE for cost control, monitoring, change management and reporting purposes. Table 24-2 illustrates the May 2020 go forward full funds request. Details of the CCE are provided in Section 21 of this report.

Table 24-2 Capital Cost Summary by Category

Category	Initial Capital
	US$ millions
Process Plant Direct	25.526
Infrastructure	10.807
Mining	18.604
Indirect	18.192
Contingency	5.274
Total	**78.403**

24.7 RISKS AND OPPORTUNITIES

The Feasibility Study ("FS") included a review of the Project Risk/Opportunity Register as well as a HAZOP study conducted in February 2020. The Risk Register will be reviewed and updated before implementation and will be regularly updated during implementation.

Activity	Start	Finish	Duration (Months)
Jervois ICO 1200 stpd Project	**1-Oct-20**	**15-Aug-22**	**23.5**
Detail Design & Engineering	**1-Nov-20**	**15-Aug-21**	**9.5**
Structural	1-Nov-20	31-Mar-21	5.0
Civil & Infrastructure	1-Dec-20	30-Apr-20	5.0
Mechanical	1-Dec-20	31-Mar-21	4.0
Piping	1-Jan-21	31-May-20	5.0
Electrical	1-Feb-21	30-Jun-21	6.0
Control & Instrumentation	15-Mar-21	15-Aug-21	6.0
Procurement	**1-Nov-20**	**31-Aug-21**	**11.0**
Mechanical & Process Equipment	1-Nov-20	15-May-21	7.5
SAG Mill	1-Nov-20	31-Dec-20	2.0
Environ Systems & Infrastructure	**1-Mar-21**	**31-Jan-22**	**11.0**
Fabrication & Delivery	**1-Nov-20**	**30-Sep-21**	**11.0**
Mechanical & Process Equipment	1-Jan-21	31-Aug-21	8.0
SAG Mill	1-Nov-20	30-Sep-21	10.0
Construction	**1-May-21**	**31-Mar-22**	**11.0**
Civil & Infrastructure	**1-May-21**	**30-Nov-21**	**7.0**
Site Establishment	1-May-21	31-May-21	1.0
Mill Building Civils	1-Jun-21	30-Aug-21	3.0
SMP Erection	**15-Jul-21**	**31-Dec-21**	**5.5**
Mill Building	15-Oct-21	30-Nov-20	1.5
ROM Bin	15-Jul-21	31-Aug-21	1.5
Milling Structure / Mech / Platework	15-Jul-21	15-Oct-21	3.0
Ball Mill Installation	1-Sep-21	31-Oct-21	2.0
Sag Mill Installation	1-Oct-21	30-Nov-21	2.0
Piping Installation	**15-Sep-21**	**15-Feb-22**	**5.0**
Milling Area Piping	15-Nov-21	31-Jan-22	2.5
EC&I Installation	**15-Oct-21**	**30-Mar-22**	**5.5**
Milling EC&I Installation	15-Nov-21	30-Mar-22	4.5
Mining	**1-Jun-21**	**30-Jun-22**	**13.0**
Portal Bench	1-Jun-21	31-Aug-21	3.0
Portal to first Ore Access	1-Sep-21	30-Nov-21	3.0
Mine Pre-Production Development	1-Dec-21	30-Jun-22	7.0
Commissioning	**1-Apr-22**	**15-Aug-22**	**4.5**
C1 Commissioning	1-Apr-22	7-Apr-22	0.5
C2 Commissioning	8-Apr-22	15-May-22	1.0
C3 Commissioning	1-May-22	31-May-22	1.0
C4 Commissioning	1-Jun-22	30-Jun-22	1.0
C5 Commissioning	1-Jul-22	15-Aug-22	1.5

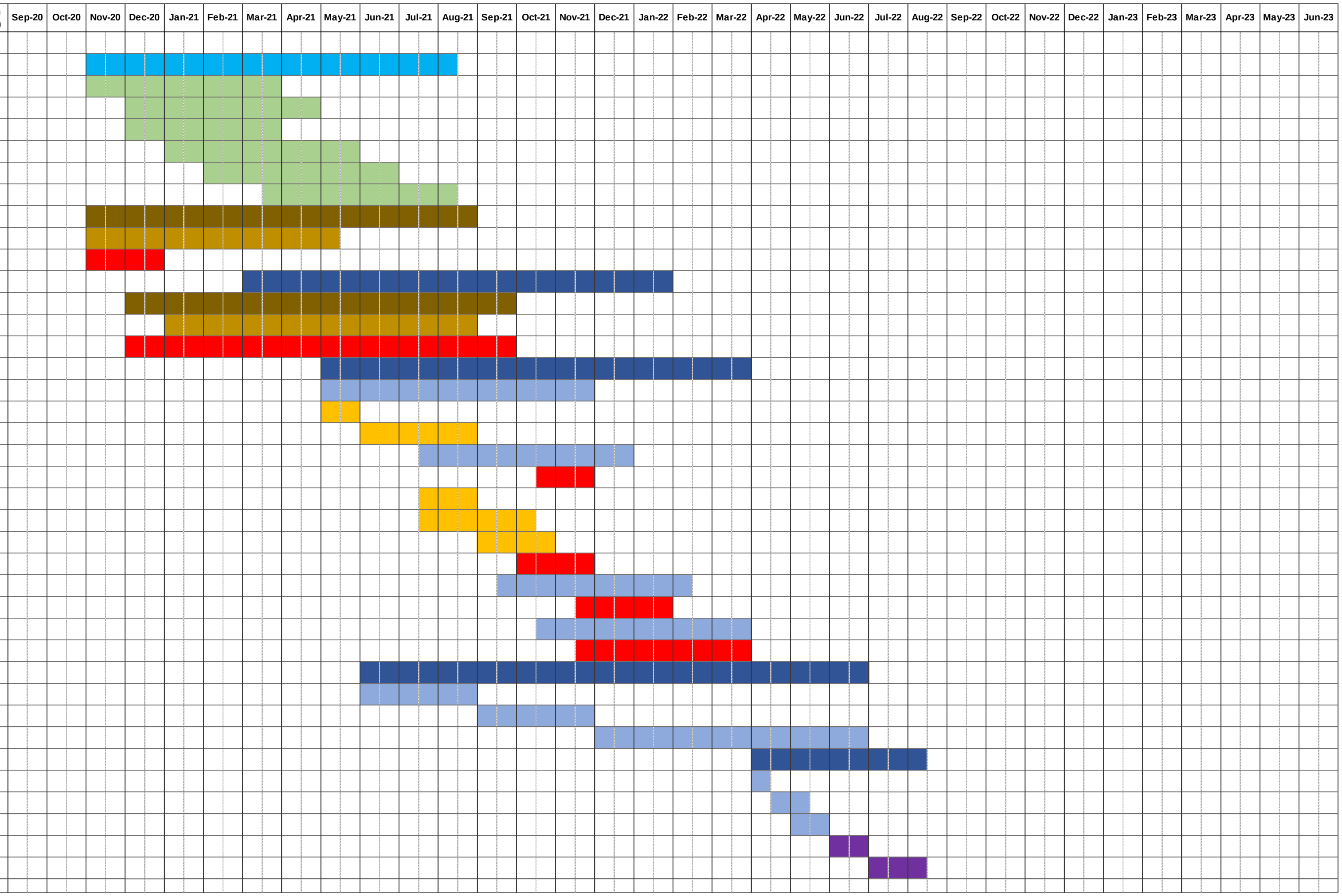


Figure 24-1: Summarized ICO Implementation Schedule

24.8 SCOPE OF FACILITIES

The ICO Project consists of a 1,200 stpd concentrator process plant, mining development and associated infrastructure;

Process Plant

- Ore Receiving
- Milling
- Copper Rougher Flotation
- Copper Cleaner Flotation
- Cobalt Rougher Flotation
- Cobalt Cleaner Flotation
- Copper and Cobalt Concentrate
- Tails Thickening
- Tails Filtration
- Water Reticulation
- Reagents
- Utilities

Mining

- Portal Bench
- Mining development
- Miners Dry
- Mining Infrastructure supplied by the Mining Contractor

Infrastructure

- Tailings Waste Storage Facility
- Water Management Ponds
- Water Treatment Plant
- Pumpback System
- Paste fill system
- Maintenance Facility and Cafeteria
- Construction/Operations Camp
- Electrical Substations

24.9 PROJECT EXECUTION METHODOLOGY

24.9.1 Organization and Responsibilities

The strategy is to integrate the project Owner's team and the EPCM team, into a single cohesive ICO Project team.

The project Owner's team, including Jervois team members, will have varying types and levels of responsibilities and subsequent interaction with the Project team. The appropriate levels of involvement are as follows:

- Part-time or full-time secondment: Secondment of resources directly into the Project team to perform certain deliverable-based tasks.

- External assistance to the Project team: Support will be required for project deliverables that require on-going review and approval, including but not limited to, designs, drawings, specifications, procedures, plans, commercial documentation, fabrication QA/QC and construction QA/QC.

24.9.2 Project Controls Plan

The EPCM controls team will perform all estimating, scheduling and cost management activities, under the guidance of the ICO Project Director.

Project controls focus on scope management, cost management and time management, beginning early with planning and end with project review and closeout. Project controls play a central role in periodic performance reporting.

- Scope management processes include initiation, planning, definition, verification and change control.
- Cost management requires resource planning, cost estimation and cost control.
- Time management includes activity definition, sequencing, duration estimation, schedule development, and schedule control.

Note: All changes to the scope, cost and schedule, require approval by the ICO Project Director.

24.9.2.1 Work Breakdown Structure

A Work Breakdown Structure ("WBS") was developed in accordance with the existing ICO WBS. The agreed WBS details the geographical areas of the project and facilities (i.e. specific plant areas).

24.9.2.2 Schedule

The execution schedule was completed in Primavera P6 in accordance with the capital cost estimating and project schedule standards.

24.9.2.3 Resourcing

The schedule was resource-loaded with man-hours from the EPCM budget hours and a forecast for Construction and commissioning. The hours were split by phase for design, procurement, manufacture, delivery, Construction and commissioning.

In order to establish confidence with the construction and commissioning man-hours, an exercise comparing the ICO construction hours with those of other projects was carried out. Several similar previous DRA and M3 projects were equalized with the ICO project based on size.

24.9.2.4 Critical Path

The critical path includes all activities with no primary float days. The critical path of the Project is highly dependent on the following activities:

- Full funds approval.
- Procurement of the SAG Mill
- Installation of the SAG Mill
- Construction of the milling building

The critical path runs through the milling area, while the near-critical path runs through the civil Construction of the concentrator area.

24.9.2.5 Progress and Performance Measurement

Earned value is a method for measuring project performance. The Project will track progress using earned value. The Project will use the project EPCM-hour budget as the baseline, while updating current and forecasted progress.

The Project will track fabrication progress (steel, platework, piping and EC&I) by major commodities.

The Project will track construction discipline progress (earthworks, concrete, steel, platework, piping and EC&I) by major commodities installed (concrete cubic meters, etc.).

Progress will be reported monthly and will be in line with the progress measurement and Earned Value Analysis standard.

24.9.2.6 Cost Control

The Project will move forward utilizing the CCE for control, monitoring and reporting purposes as outlined in the Cost Control Procedure. The CCE was coded in accordance with the WBS structure and will be imported into the EPCM cost management system.

24.9.2.7 Change Management

All team members, contractors and a third-party quantity surveying services will monitor scope, unit rate, and quantity changes. The Project controls progress, and performance tracking provides metrics to identify deviations.

Project controls will maintain a formal change management system using the EPCM cost management system in accordance with the Change Management Procedure. No manual changes to budget or "estimate-at-completion" will be allowed.

Project controls will require an approved "Project Change Notice" before implementing the scope change.

24.9.2.8 Contingency Management

The Project contingency take into consideration current risks associated with the execution of the Project.

Project controls will manage the contingency forecast. The ICO Project Director will approve all forecast and budget changes for contingency as documented in a "Project Change Notice".

The Project team will develop an authorization matrix for approval of non-contingency variations.

24.9.2.9 Escalation Management

Escalation has not been considered past the quoted base date and is therefore not included in the capital cost estimate and should be included for in the client's financial model.

During execution, the Project team will distribute escalation into forecast accounts periodically, based on documentation showing actual price escalation. A "Project Change Notice" will document the changes.

24.10 PROJECT ADMINISTRATION PLAN

24.10.1 Project Procedures

The Project Execution Plan ("PEP") is supported by project-specific execution procedures and plans, developed by the ICO team. These project-specific documents define in detail the processes to be followed in the various execution activities. These are working documents and will be revised as and when required during execution/implementation.

This list of procedures may be expanded upon, depending on the complexity and requirements of the Project, and will be supported by additional documentation when required.

24.10.2 Document Management Plan

The Project will use Onehub, a web-based data-repository system for document management. The EPCM may use different internal systems for tracking document development, but Owner approval and transmission to the Owner will be through Onehub.

The on-site construction team will use document controllers to ensure the latest approved drawing and specifications are issued through a formal document control and transmission system to the respective contractors.

The on-site document controller will conduct periodic audits to ensure the use of current documents. At the end of the project, the Project team will transfer all relevant documents to ICO through Onehub.

24.10.3 Project Communication Plan

As part of project management, the following meetings will be scheduled during execution:

- Weekly Project progress
- Monthly Project progress
- Weekly construction meetings

The following reports will be distributed to the relevant stakeholders:

- Weekly EPCM progress
- Monthly project progress
- Monthly dashboard, Minutes of all meetings

24.10.4 Project Audit and Review Plan

During the project implementation phase, audits will be conducted as follows:

- ICO Corporate will perform audits, when required.
- The ICO project team will audit the project periodically. This will include, but will not be limited to, cost control, planning, logistic, document control, procurement, fabrication QA/QC and contracting systems and their implementation.
- The ICO Corporate team will also conduct site visits periodically to perform external audits on the ICO construction team and contractors. This will include, but will not be limited to, construction QA/QC, HR, IR, materials control and safety systems and their implementation.
- Members from the ICO construction team consisting of ICO representatives will perform audits on the appointed construction contractors, ensuring that on-site business systems comply with ICO standards. This

will include, but will not be limited to, document control, construction QA/QC, HR, IR, materials control and safety systems and their implementation.

24.11 ENGINEERING PLAN

Engineering will be completed in accordance with the following reference documents:

- Project Execution Plan
- Engineering Design Criteria

The three-dimensional (3D) model will be completed during execution/implementation. All earthworks, civil, steel, platework, piping and cable rack fabrication and installation drawings will be generated from the 3D model and made available to the EPCM construction team and all construction contractors, to assist with the visualization and confirmation of construction activities.

The standard EPCM operating procedure and process adopted by ICO gives responsibility to discipline lead engineers for the design, specification, procurement, technical/commercial evaluations and supply recommendation. The lead engineer will also follow their designs into the field for construction, where they will act as the responsible technical lead.

The discipline lead engineers will assist in providing technical continuity during pre- and cold commissioning of the Project, in full technical collaboration with the ICO engineering & maintenance team. The lead engineers will assess the necessity to schedule vendors and other special Contractor's site activities.

24.12 PROJECT COMMERCIAL/SUPPLY CHAIN MANAGEMENT PLAN

ICO commercial support will be managed by the ICO project team, overseen by the ICO project director. The EPCM will be responsible for technical datasheets and preparation of a scope of work document during the procurement phase.

ICO's scope of contracts management for this Project includes:

- Contract Administration of Prime Contract
- Management of Owner procurement activities in support of the Project

24.12.1 EPCM Contract

The EPCM contractor has the responsibility to manage all engineering, procurement and project management for the scope of work and provide strict cost-control methods to ensure the work is completed on time and subcontractor control is effective.

The EPCM contractor will have limited involvement during fabrication, construction and commissioning, primarily consisting of technical assistance.

The EPCM contract will be a fixed unit rate with a dual fee (fixed and discretionary). The EPCM labour will be on a cost-reimbursable basis with a not-to-exceed amount.

24.12.2 Procurement and Services

The objective of procurement and services contracting during execution is to obtain fair market pricing for equipment and services from a minimum of three companies, selecting the best price at the required quality, thereafter, placing the orders and contracts with the preferred suppliers or service provider.

Procurement will be managed by the ICO project team. Approvals will be done in accordance with the project RACI matrix. Equipment and services will be tendered to parties ICO deem able to supply and deliver such equipment or services. The approved vendor list developed during the FS will be utilized during execution.

Where applicable, sole sourcing will be allowed following the established ICO procedure. Horizontal contracting strategies will be followed for all construction services including earthworks, civil, structural, and mechanical installation, piping and EC&I installation.

24.12.3 Transport and Logistics

A specialist logistic services provider (“LSP”) be appointed by ICO to manage and coordinate delivery of out of gauge and oversized equipment. The LSP will provide all related logistical and freight forwarding services for the project under the guidance of the ICO project director.

The LSP will receive information of each order and expedite with the ICO project director and logistics coordinator all materials once released from the supplier.

Each supplier will be responsible for making goods available, export packaged according to international shipping specification and the project packing requirements at the suppliers and sub-contractors premises or delivered to the Contractor’s premises.

24.13 FABRICATION PLAN

Suppliers will be responsible for fabricating all materials and equipment in accordance with the quality and schedule, as stipulated in the purchase order, with the management assistance of the EPCM Contractor.

24.14 QUALITY ASSURANCE PLAN

The objective of the quality assurance plan is to ensure that contractors, suppliers, installers and construction contractors are implementing a well-documented and auditable quality management system and that all fabrication, installation and construction are performed to ICO’s requirements.

The following are fundamental quality principles that will be applied to all activities associated with this Project:

- Individuals are accountable for the quality of the work assigned to them.
- The degree of application of quality criteria is dependent on the importance of the item, i.e. criticality rating.
- The quality department of each contractor and supplier will oversee the work under their supervision, with the guidance of the ICO project director.

On a day-to-day basis, the following quality assurance plan will be implemented:

- During fabrication of equipment, ICO will have inspectors or third-party inspection authorities visiting the supplier to monitor quality and progress.

- All equipment and materials will be checked and released by the engineer (the engineer can second an inspector or appoint a third-party inspection authority) in accordance with the contract, specifications and drawings.
- Every supplier, installer or construction contractor will be required to submit a quality control plan ("QCP") to the engineer and the inspector or construction supervisor responsible for the specific equipment, materials or construction work.
- The QCP will be signed and agreed to by the engineer and inspector or construction supervisor and the supplier, installer or construction contractor. The QCP will define the witness and hold points during fabrication, installation and construction. The relevant parties will sign the points during the fabrication or construction process as defined by the QCP.
- Equipment and material fabrication and supply quality assurance will be done in accordance with Quality Assurance and Quality Control Requirement.
- Construction and installation quality assurance will be done in accordance with the Construction Procedure and the Site Quality Control Procedure.

24.15 EXPEDITING PLAN

The ICO project director will monitor all equipment and material purchases will have expediting responsibilities that include:

- Developing an equipment/materials specific expediting plan
- Scheduling expediting visits
- Oversight of expediting plans
- Review of expediting reports
- Track fabrication progress and report
- Release equipment for packing
- Capture planned, forecast and actual release dates, for progress and logistics management purposes

The primary function of the expediting role is to anticipate problems, rather than simply report problems, and to be pro-active in the implementation of resolutions.

24.16 HEALTH AND SAFETY PLAN

The ICO Project Health and Safety Plan ("HASP") addresses the construction of ICO and any other authorized work on the project. The HASP covers all owners personnel, contractors, and vendor personnel as well as any temporary visitors working on the site.

The HASP specifies regulatory compliance requirements, training, certifications and medical requirements necessary to complete the Project.

Each Contractor shall provide their own Health and Safety Plan for their respective work activities which is equivalent to this HASP (i.e., which contains requirements at least as stringent as those outlined in the HASP and PoO and Specifications).

In all cases, each Contractor shall be responsible for site safety related to or affected by their own field operations (i.e., heavy equipment or Construction related operations).

24.16.1 HASP Purpose

This HASP identifies potential site hazards (both physical and chemical) along with the safety requirements and preventative actions for reducing the risk of accident or injury. Preventative actions include:

- Identifying proper personnel protective equipment
- Defining proper work procedures
- Providing personnel health and safety training
- Implementing emergency response procedures
- Ensuring appropriate first aid equipment and training are available
- Maintaining accurate accident/injury records
- Scheduling safety and maintenance inspections

24.16.2 Accident Reporting

All independent contractors working on the ICO Project are required to report all injuries requiring medical treatment to the company representative and the safety department as soon as possible. MSHA requires contractors to submit a specific accident reporting form (7000-1) within ten (10) days of the injury-producing accident that required medical treatment, as well as immediate notification for certain serious accidents. Any contractor is required to furnish the ICO project director with a copy of the worker's compensation first report of injury and illness form and a copy of the MSHA 7000-1 form on all reportable injuries requiring medical attention; however, submission of more detailed accident investigation reports is encouraged, and these reports must be made available upon request in the event of a serious accident.

24.16.3 Emergency Procedures

All Contractors and others engaged in work activities on site will be expected to make arrangements to provide emergency medical care for their employees who may be injured while on the job. This may include notification of community or plant paramedics or rescue teams and providing on-site trained employees.

24.16.4 Accident Costs

All costs associated with accidents that occur under the control of a Contractor and its subcontractors are to be borne by the Contractor and not passed on to the EPCM or. These costs may include, but are not limited to; compensation costs; medical treatment costs; state or federal government penalties for safety and health violations; lawsuits against the Contractors, the EPCM or ICO for injury or damages as a result of the Contractor accident; and liquidated damage agreements against the Contractor for failure to complete a project on schedule as a result of an accident. Contractors agreed to hold ICO harmless from all liability resulting from injuries and property damage arising out of Contractors' activities.

24.16.5 Insurance

Each Contractor and Subcontractor is responsible for submitting proof of Worker's Compensation Insurance prior to the start of work. This form must be submitted and received by ICO prior to contractor or subcontractor commencing onsite work. Such insurance shall meet the minimums required of the contract and the state. It shall be on a standard ACORD form.

24.16.6 Documentation Requirements

The ICO project team and contractors are committed to operating and working under a quality management process defined by ANSI/ASQC Q9001-1994 Quality Standard.

This quality assurance system provides positive management control, a record of conformance to specified requirements, and together with approved work plans and instructions establishes the operational and administrative requirements for the project. Within the quality assurance system, individual and organizational responsibilities are assigned for the project. Control measures necessary to achieve, verify and document conformance to specified requirements are established and maintained.

Where applicable, all contractors shall obtain an MSHA identification number through the local MSHA office.

All MSDS, hazard communication plans, accident reports, inspection reports and audits are also to be submitted to ICO for filing.

24.16.7 Standard Operating Procedures Overview

All work performed at the site will be done in compliance with all federal, state and local requirements. All personnel performing work will meet the required training and certifications necessary to meet the regulatory requirements for job-related tasks.

24.16.8 Federal, State And Local Laws, Regulations And Codes

The ICO Project is subject to regulation by the MSHA Act which provides for specific standards and regulations affecting employee safety and health. The MSHA Act requires all contractors working on mine property to comply with the standards and provisions of the legislation including specific safety and health standards, accident reporting, and employee training requirements, as well as its general intent.

All contractors will be required to secure MSHA identification numbers and provide the safety department with a copy. Because the MSHA Act is a mine safety and health act, some of the work contractors perform may not be addressed in the MSHA standards. When the MSHA Act is silent, contractors are expected to comply with other applicable federal, state and local standards. Regulations such as the OSHA standards, the National Electric Code, the Associated General Contractors Accident Prevention Manual, and state and local safety and health rules and regulations must be reviewed and followed where appropriate. If a contractor is uncertain about a working condition or practice that may affect employees' safety and health, the ICO project director and Safety Department will be consulted.

All contractors shall obtain a legal identification number through the local MSHA office as described in 30 CFR Part 41, Notification of Legal Identity.

24.16.9 Responsibilities

The ICO construction manager has overall responsibility for health and safety at the site. The contractor site manager has responsibility for the health and safety of his or her employees. A site safety manager will be responsible for carrying out relevant site safety training when required.

Each contractor is responsible for having a safety program equal to the ICO Project Health & Safety program. The contractor may adopt the ICO Project Health & Safety program in addition to its own program in order to cover any potential differences.

Each contractor is responsible for ensuring that each of their superintendents and supervisors is aware of his or her individual responsibilities and enforce safety. Each contractor shall be responsible for the compliance of its subcontractors and shall provide written acceptance by the subcontractor of its acceptance of the contractor and ICO Project's Health and Safety Plan.

Each contractor shall designate a responsible member of his or her organization as the contractor Health and Safety Representative ("CHSR") whose duties shall include the following:

- Be responsible for the daily enforcement and monitoring of their HASP for their employees and subcontractors.
- Document that proper medical monitoring and training is provided to their employees and subcontractors.
- Advise the ICO Safety Supervisor, and the Construction Manager, of any accident, injury or potential site-related illness.
- Ensuring their employees arrive on-site with proper personal protective equipment.
- Contractors and subcontractors are responsible for accident investigation, and accident follow up reports for their employees.
- Ensure all employees and contractor's employees have received the proper MSHA training, and copies of the form (5000-23) have been submitted.

Each Contractor is responsible for filing all necessary documents with the governing safety authorities, including MSHA and OSHA.

24.16.10 Visitors

All visitors entering the site must be briefed on the applicable provisions of this HASP. In addition, visitors will be expected to comply with relevant OSHA requirements such as medical monitoring, training, and respiratory protection (if applicable). Visitors will be required to wear prescribed proper personal protective equipment ("PPE").

Visitors not adhering to the provisions of the HASP will be escorted from the site.

24.16.11 Training and Medical Prerequisites

Contractors that are scheduled for work on-site must meet owners required training standard. In addition to these prerequisites, site-specific training is also required for all project personnel before actually performing the work on site. Contractors will be responsible for certifying that their employees meet the training requirements specified in MSHA 30 CFR 48.

Contractors and all personnel working on site will be required to provide a document certifying that each general site worker has received 24 hours of MSHA training or has had an 8-hour refresher course within the last year. Underground workers shall have had 40 hours of training and 8 hours of refresher every year.

Contractors must be certain that all mobile equipment operators have received adequate training (and have submitted their Operators Certification, if required) prior to their operating the equipment.

Medical monitoring programs are designed to track the physical condition of employees regularly as well as survey pre-employment or baseline conditions. The medical surveillance program is a part of each employer's Health and Safety Program.

Contractors will maintain the medical records for their employees. Still, they shall provide ICO and the construction manager with written documentation certifying that each employee at the site has met the requirements of the Medical Surveillance Program.

Employees will be informed of their right to access their medical records.

24.16.12 Alcohol and Drug Policies

No alcohol is permitted on the site. The contractors will be responsible for implementing a program to ensure there are no drug users. Random drug testing of critical craft, e.g. crane, and heavy equipment operators shall be used to ensure compliance.

24.17 CONSTRUCTION PLAN

24.17.1 Construction Strategy

Construction management of facilities will be executed by an integrated ICO team to manage construction work.

Under the guidance of the project team, the construction manager will drive the construction management philosophy on health, safety, quality and productivity, across all construction areas in line with the project schedule.

24.17.2 Construction Roles and Responsibility

The construction manager will report to the ICO project director and be responsible for providing overall construction and site management services to ICO. In addition, he will work closely with the operational team and contractors.

A direct relationship between the construction manager and each contractor's representative will ensure work is executed safely, on time, and to the required specifications.

The site-based quantity surveying and contract administration personnel will support the construction supervisors in site administration of the contractors. Supervisors will report to their discipline lead engineers on contractual issues and the construction manager on schedule, H&S and construction issues.

24.17.3 Construction Infrastructure

ICO Site Office

The existing admin office will be utilized as the project team office. Vendor representation during commissioning will be accommodated within the office.

Contractor Site Offices and Laydown areas

In accordance with the site establishment plan, fenced off contractor site offices and laydown areas are allocated to each contractor. Each construction laydown area shall accommodate the contractor and 'subcontractor's site offices, stores, materials area, fabrication shop, ablutions, eating areas and construction vehicles parking areas.

A site plan and detailed layout for the contractor site offices and laydown areas will be submitted by the contractor for approval to the construction manager, safety manager and discipline supervisor. The plan shall indicate a clear separation of amenities, structures and services.

Contractors must ensure that management of hazardous waste (i.e. hydrocarbons, corrosives, etc.) and non-hazardous waste (i.e. putrescible materials, building materials, etc.), is compliant with the ICO waste management standard.

Contractors shall supply fire extinguishers in accordance with ICO specifications within all work areas and laydowns.

Contractors shall establish tool sheds/workshops. The size and position of these tool sheds/workshops must be approved by the construction manager prior to construction.

Contractors will be required to have an emergency site laydown specific response procedure, including emergency assembly points, and evacuation route or routes identified. Information signage must be visible and in accordance with the emergency response procedure.

24.17.4 Traffic and Access Plan

The primary objective of the Traffic Management Plan is to ensure optimal safety of all personnel associated with the Project, when operating within the ICO project area, as well as complying to the PoO and approved transportation plan.

The transportation plan describes the type and numbers of vehicles which will be required to travel over Forest Service roads to access the property. WEP models have been developed as well as road logs describing the lines of sight at curves etc. The Spill Prevention and Response Plan, Snow Removal Plan and Conceptual Road Design have been incorporated into this document. A separate document has been prepared for and submitted to Glencore outlining methods which will be utilized to ensure the protection of their facilities as access will be through the Blackbird Mine site.

24.17.5 Security Plan

During construction, the ICO security contractor will manage all gatehouses and be responsible for managing access control in line with the existing ICO security policies, procedures and systems.

24.17.6 Material Control Plan

The material controllers will receive, inspect, store, issue, report and maintain records of all incoming materials and equipment purchased by the project for the subsequent issue to construction contractors.

The construction supervisor or site engineer will be responsible for performing an inspection of all critical equipment received by the material controllers, to ensure that goods are technically sound for installation and commissioning and stored as per the vendor instructions.

The projects commercial inventory control activities will be integrated with the existing site inventory control function.

Strategic spares, surplus commissioning spares and surplus construction materials will be handed over from the project to ICO operations after completion of cold commissioning.

Contractors will be responsible for the handling of their construction consumables, plant and equipment from origin to their laydown areas and work sites.

24.17.7 Plant and Equipment

Equipment

Contractors shall be responsible for their lifting and rigging equipment. The tender adjudication process will ensure adequate craneage compliance. Contractors will provide all other required lifting equipment (e.g. cranes, scissor lifts, elevating working platforms, boom lifts, etc.) for their respective construction activities.

A specialist lifting and rigging contractor will be appointed to supply the required equipment and perform the mill lifts with supervision support from the ICO construction manager.

Rigging studies shall be conducted by the responsible contractor and approved in writing by the construction manager for all heavy lifts.

The main construction contractor will be responsible for supplying all equipment and personnel to assist the with offloading and issuing of free issue materials to the contractors for construction.

ICO will lease, as part of the project, a combination of mobile equipment to assist contractors during construction.

Fuel

Fuel for all contractors will be managed in the following manner:

- Fuel facilities are available on site, and relevant permission to access such facilities must be obtained from the Construction Manager and the relevant ICO departments.
- Contractors will be charged for what they draw. The cost will be deducted from the invoices submitted for work completed, as part of the ICO Project.
- Contractors shall not be permitted to establish permanent fuel facilities at the construction site.

24.17.8 Site Communication

Telecommunication

Desktop telephones, mobile phones, internet/intranet data networks are available to the team at the admin/project office. Internet and telecommunication systems will be jointly managed by the ICO IT Department (from a technical perspective), and the Construction Manager (from a management perspective).

Contractors shall be responsible for the establishment of their telecommunication systems. The primary mode of communication will be mobile phones.

Radio Communication

Radios will be issued to supervisors, commissioning engineers and the site managers of contractors during the construction and commissioning phases. ICO will supply all radios. There will be various construction and commissioning channels on each device. At the end of construction, radios will be handed over to the ICO operation team.

Contractors will be authorized and responsible for the implementation and use of their radio communication systems internally if required.

Site Meetings

A schedule for site meetings will be established during construction.

The Contractor shall be furnished an advanced copy of the meeting agenda, with a request to provide feedback on agenda items.

24.17.9 Site Planning

The objectives of site planning and monitoring are as follows:

- To implement a monitoring system, to accurately follow the progress of contractors.
- To review and approve Contractors' schedules in conformance with the master schedule, and contractual milestones.

- To collect and monitor progress data from field engineering, discipline supervisors and contractors, against project objectives and contractors' performance reports.
- To measure contractors reported completion percentages against actual concrete, steel, plate work, piping and cables installed.
- To provide inputs for the co-ordination of Contractor's activities and eliminate interference issues among the various contractors.
- To assist the Construction Manager and supervisors in detailed construction planning, when rescheduling is required to mitigate delays.

The Construction Manager will update his site schedule and advise the Project Manager of delays or slippage and a proposed mitigation plan.

Construction activities on the critical path are monitored as described above. Reports produced will be integrated into the project monthly report, weekly dashboards, and distributed to the Project team/applicable stakeholders.

The control of work progress and contractors performance is based on identifying early negative or positive trends to allow management to react and take appropriate action. A two-week look-ahead and rolling horizon will be implemented and distributed to the construction team.

24.17.10 Construction Completion

During construction, the plant operations team, together with the ICO project director and project team, shall perform regular walkabouts and surveillance inspections. The aim is to identify any potential deviations or shortcomings in terms of the project requirements, before making a punch list.

There are four-punch list levels categorized as A, B, C, and D to the degree of seriousness of a said defect, or punch list item.

Following completion of all category A and B punch list items by the Contractor, the area will be handed over to the Commissioning Manager for pre- or cold commissioning. The Construction Manager will issue a C1 construction certificate to the Commissioning Manager, before commissioning can commence.

24.17.11 Vendor Assistance

As part of construction completion and pre-commissioning, vendors will inspect and approve their equipment installation as required. Vendors will certify that the equipment is ready for commissioning by signing the equipment pre-commissioning checklist, and acceptance of the installation checklist.

In addition to typical vendor assistance, the SAG and Ball mill, and flotation cell vendors will have representation on site during installation. These vendor representatives are required to guide the installation contractor in technical aspects during installation.

24.18 COMMISSIONING

This section describes the commissioning and start-up strategy for the various facilities and operable systems for the ICO Project.

The ICO project director will form a team with a designated commissioning manager to plan, coordinate and execute all commissioning activities to achieve completion. The pre-commissioning, cold commissioning, hot commissioning, performance testing, and handover methodology adopted, will be a systems-based approach.

The total scope of facilities for the Project will be divided into subsystems and assets (e.g., crusher, mill, etc.) based on operational functions. A systems and asset index will be developed and maintained as a key control document. The subsystems and assets will be grouped into operable systems that will be identified on the systems index and project schedule.

Handover packages, including construction completion and commissioning documentation, will be managed by related operable systems, with detailed tasks being performed at the subsystem level. Change management, risk review and a review of safety procedures, will be conducted prior to each level of commissioning.

The commissioning plan will be reviewed and signed off by the commissioning manager, who will coordinate with the commissioning team and the ICO process manager, and will assign an ICO designated owners team to take part in all commissioning activities (pre-planning and execution) for each operable system being commissioned.

This will ensure proper QA/QC and owner acceptance, before final sign off and handover occurs.

The ICO Commissioning team will consist of trained and experienced process and maintenance personnel, together with process supervisors assigned to the Project for the duration of commissioning and start-up.

On completion of all mechanical construction operable systems, pre-commissioning and cold commissioning activities shall be performed to confirm all parts of the facilities are ready for hot commissioning, and performance testing.

Each stage of facilities handover and testing shall be planned in detail and coordinated between ICO and the contractors. Check sheets shall be prepared before and for each commissioning phase and completed copies retained as a record of commissioning activity and verification, that facilities or parts thereof were tested.

The ICO project director will prepare a commissioning plan for the facilities based on operable systems or areas, in conjunction with and as directed or approved by ICO commissioning manager.

24.19 POST COMMISSIONING ASSISTANCE

As part of post-commissioning assistance, the following provisions were made:

The capital estimate made provision for an additional man-hours over two months, after wet commission completion. The following will be covered:

- Project closure.
- Post wet commissioning assistance, to optimize programming, control and instrumentation systems.
- Field engineering and construction supervision to assist the Owners team during hot commissioning and ramp-up to commercial production.

24.20 PROJECT CLOSURE

After final handover, the following activities may remain to be completed by the ICO project director:

- Compilation of a post-commissioning report
- Updates to design documentation to reflect modifications, made during commissioning
- Complete all as-build drawings CAD format, and hand over to ICO
- Hand-over all commissioning spares not used during commissioning and all strategic spares to ICO, in accordance with the Spares Procedure
- Ensure all contractors demobilize from site as per their contractual agreement with ICO
- Compile and handover all equipment manuals and data books through Onehub

- Closeout and handover of all service contracts and purchase orders, in accordance with the Procurement Closeout Procedure.
- The Project team will capture information throughout the Project and will complete a project closeout report.

25 INTERPRETATION AND CONCLUSIONS

The authors have reviewed the project data, including the available drill-hole and metallurgical information, and have visited the project site. The authors believe that the data provided by Idaho Cobalt Operations, as well as the interpretations Jervois has derived from the data, are generally an accurate and reasonable representation of the Idaho Cobalt property. Based on the positive results of this Feasibility Study, the project should continue on a path to a production decision.

25.1 GEOLOGY AND MINERAL RESOURCES

The Ram Deposit is hosted by fault-bounded, meta-sedimentary sequence believed to be marine in origin. The rock types are dominantly quartzite and argillite. The sequence strikes north-northwest, and dips 50° to 60° to the west. The mineralization is hosted in a transitional sedimentary package with variable biotite/chlorite alteration which is associated with the cobaltite mineralization. The deposit was modeled using a combination of nine (9) wireframes representing mineralized units as well as offsetting fault wireframes. The overall mineral resource is summarized in Table 25-1.

Table 25-1: Summary of the Ram Deposit Mineral Resources at 0.2% Co Cut-off

Category	Co% Cut-off	Resource (S Tons "st")	Co (%)	Co (lbs)	Au (oz/st)	Au (ounces)	Cu (%)	Cu (lbs)
M + I	0.2	3,436,000	0.59	40,577,700	0.016	54,200	0.73	50,435,500
Inferred	0.2	1,543,000	0.51	15,593,800	0.012	18,700	0.68	21,032,200

M + I = Measured & Indicated

The main ("mmh") zone contains the bulk of the mineralization, including all of the Measured and Indicated resources. Jervois successfully targeted and updated the resource confidence in its recent drilling campaign, resulting in an increase in Measured and Indicated resources over previous estimations, and future drilling could be planned to accomplish this again. The mineralization of the Ram deposit remains open at depth (down-dip) and along strike.

The geological corridor/structure controlling the mineralization is persistent for the entire strike length of Jervois's ICO area and beyond. The already known Sunshine deposit is within easy reach (i.e., only one mile south) from the infrastructure at the Ram. Deep hole drilling successfully intersected what is interpreted to be the Blackbird horizon, in an effort to prove the continuity of nearby zones. It is also worth noting that previous drill-testing by earlier operators in the greater region identified additional areas of mineralization near the ICO deposits. These mineralized zones represent promising targets for future drilling. Hence, the outlook in terms of increasing the resource is favourable.

25.2 MINING AND MINERAL RESERVES

Table 25-2 summarizes the mineral reserve estimate for the Idaho Cobalt Operations.

Table 25-2: Mineral Reserve for the ICO at 0.24% Recovered and Payable Equivalent Co Cut-off

Category	Short Tons	Co Grade %	Co lbs	Cu Grade %	Cu lbs	Au gpt	Au Oz
Proven	1,586,961	0.563	17,864,643	0.670	21,258,983	0.481	24,633
Probable	1,160,322	0.528	12,260,846	0.962	22,322,281	0.715	26,758
Total	2,747,283	0.548	30,125,489	0.793	43,581,264	0.580	51,391

25.3 ECONOMIC EVALUATION

An economic analysis based on the production and cost parameters of the ICO Project was prepared and selected results are summarised in Table 25-3. In the analysis, price forecasts of US$25.00/lb for cobalt, US$3.00/lb for copper and US$1,750/oz for gold were assumed.

Table 25-3: Summary of Life of Project Production, Revenues, and Costs

Description	Units	Value
Resource Milled	k tonnes	2,486.1
Bulk Concentrate @ 10 % Co	k tonnes	124.4
Revenue	M USD	667.4
Operating Costs	M USD	315.7
Initial Capital Costs (excludes Working Capital)	M USD	78.4
Sustaining Capital Costs	M USD	56.1
Mine Closure & Rehabilitation Costs	M USD	21.2
Total Pre-Tax Cash Flow	M USD	198.5
Total After-Tax Cash Flow	M USD	170.9

The financial indicators associated with the economic analysis are summarised in Table 25-4.

Table 25-4: Base Case Financial Indicators

Financial Results	Unit	Value
Pre-tax NPV @ 8%	M USD	113.4
Post-tax NPV @ 8%	M USD	95.7
Pre-tax IRR	%	41.8
Post-tax IRR	%	37.6
Pre-tax Payback Period	years	2.6
Post-tax Payback Period	years	2.8

26 RECOMMENDATIONS

The authors believe that the Idaho Cobalt Operations property is a project of merit and warrants the proposed program and level of expenditures outlined throughout this report. Based on the positive results of this Feasibility Study, the project should continue on the path of project execution through to production.

26.1 GEOLOGY AND MINERAL RESOURCE

The following two-phase program is recommended as a result of the geological modeling and resource estimation.

26.1.1 Phase 1

1600 metres of drilling is recommended to test up-dip positions on the Blacktail North zone. As development of the underground mine is anticipated, further testing of the hanging wall zones, or Main zone is not recommended at this time.

Cost of this program is anticipated at approximately US$340/meter x 1600 meters for a total of US$544,000.

26.1.2 Phase 2

Phase 2 includes follow-up drilling based on the results of Phase 1 and definition drilling of the hanging wall zones and Main zone from underground positions (in conjunction with grade control drilling).

For Blacktail North, an additional 1600 meters of drilling is recommended if Phase 1 produces promising results.

For the hanging wall and Main zones, 1000 meters of definition drilling from underground (meaning shorter holes).

Cost of this program is anticipated at approximately US$340/meter x 1600 meters for a total of US$544,000 for the Blacktail North, and US$250/meter x 1000 meters for a total of US$250,000 for the hanging wall and Main zones, for a total of US$794,000.

26.1.3 Additional Recommendations

Orix, Scott Zelligan, and CSA Global recommend the following be implemented for all future programs (including the above two phases):

- The use of four-acid digestion for the over limit arsenic (As) values may be problematic because As, an element of interest, can volatize with this method leading to a potential underreporting. CSA recommends some of the OG62 arsenic over limit pulps to be analyzed with aqua regia digestion OG46 method to determine if there may have been any volatilization of As.
- The use of a regression formula to define densities for the resource calculation proved adequate. However, Standardization of SG measurements potentially using whole core intervals, should be a considered for any oncoming drill program.
- Although the spread use of standard material produced in the late 1990s has yielded decent results, Jervois could re-examine and likely produce new CRMs for ongoing programs.
- QA/QC could be improved by the inclusion of field duplicates more frequently in the sequence, as well as the inclusion of true certified blanks as opposed to the red brick material used during the latest programs.
- Currently all historic and 2019 data is hosted in an excel format database. Ideally a commercial relational database should be used which has built-in error checking, audit documentation and QA/QC.
- Given the gradational nature of the sedimentary package, Orix recommends a detailed analysis of the existing geochemical data, to look for geochemical signatures that could highlight marker horizons. These marker

horizons, particularly if found in the hanging wall zones, could prove very useful to determine true lateral continuity of some of those units, which could possibly give a positive impact in the re-definition of the resource categories.
- The current resource model presents the opportunity to test areas where plunges/ore shoots may be present. If said ore shoots do exist and continue at depth, they could represent prospective economic areas at depth.
- The Ram deposit lays less than 2 km north of the famous Blackbird group of deposits, mined by several operators in the 1900s. It is recommended that Jervois use the opportunity to explore the footwall of the Ram, in search of equivalent stratigraphic horizons that exist in the Blackbird mine.

26.2 MINING

The following summarizes the recommendations observed during the preparation of the current feasibility:

- Geotechnical - Classification and characterization of the rock mass in relation to it spatial location during initial development and access to ore zones will assist in stope dimension and overall mine design validation in coordination with the geotechnical borehole and study conducted in 2017 at the recommendation of the prior NI43-101 technical report.

- Ventilation - A mine ventilation study utilizing best practice modeling software was conducted on the proposed mine design. The study identified the potential for a relatively short, bored ventilation raise to surface above the south ramp system would reduce ventilation costs. It is recommended the cost and permitting requirements for a ventilation raise be examined prior to development of the South Ramp system (Y3).

- Electrification – Additional optimization of the mine design, plan and especially the use of battery/electric haulage equipment to enable automation of mine systems should be examined early in the mine life to reduce costs and the operational carbon footprint. Preliminary trade-off studies indicate additional power supply to the site will be required to support electrification. Studies should be advanced prior to ore production. Unused re-mucks along the ramp system may function as battery bays to reduce excavation requirements in this scenario.

- Contract Mining – The contract for underground mining and portal development should be formalized and executed as the project is approved to move forward.

26.3 PERMITTING

It is recommended that Idaho Cobalt Operations evaluate and audit all existing permits to ensure they remain valid. Any exceptions and updates to permits should be initiated upon the start of detailed engineering and carried through to completion.

In general, the permits in place appear valid and viable and will likely not require any major modification moving forward.

26.4 EARLY WORKS

It is recommended that Idaho Cobalt Operations finalize previously initiated early works construction and move to completion of these activities as soon as possible. The cost for this work is carried within the capital cost estimate presented in the report.

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APPENDIX A - FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

Certificate of Author

To accompany the technical report titled "NI 43-101 Bankable Feasibility Study Technical Report for the Jervois Mining Idaho Cobalt Operations (ICO) Project", dated November 13, 2020 (the "Technical Report") prepared for Jervois Mining Limited. (the "Company").

I, Matthew Sletten, P.E., do hereby certify that:

1) I am employed by M3 Engineering & Technology Corp., 2175 W. Pecos Rd. Suite 3, Chandler, AZ 85224;

2) I graduated with a BS in Civil Engineering and an MS 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 a as an engineer and project manager in the base metals and precious metals industry for a total of 15 years;

5) My work experience includes detailed engineering, engineering management, project management, corporate management, capital and operating cost development and report development for major mining projects throughout the world;

6) I have read the definition of "qualified person" set out in the NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;

7) I am independent of the Company as described in Section 1.5 of NI 43-101;

8) I have no prior involvement with the property that is the subject of the Technical Report;

9) I am responsible for Sections 2, 3, 4, 5, 6, 18, 21, and 23 of the Technical Report and I am co-responsible for Sections 1, 24, 25, 26 and 27 of the Technical Report;

10) I visited the Idaho Cobalt Operations site on August 13, 2019 and reviewed the plant location site;

11) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Technical Report;

12) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Jervois Mining Limited, or any associated or affiliated entities;

13) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Jervois Mining Limited, or any associated or affiliated companies;

14) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Jervois Mining Limited, or any associated or affiliated companies;

15) I have read NI 43-101 and the relevant Sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101; and

16) As of the effective date of the Technical Report, the best of my knowledge, information and belief, the relevant Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of November 2020.

Matthew Sletten, P.E.

Vice President

M3 Engineering & Technology Corp.

Certificate of Author

To accompany the technical report titled "NI 43-101 Bankable Feasibility Study Technical Report for the Jervois Mining Idaho Cobalt Operations (ICO) Project", dated November 13, 2020 (the "Technical Report") prepared for Jervois Mining Limited. (the "Company").

I, Scott Zelligan, B.Sc. (Honours), P.Geo. (ON), do hereby certify that:

1) I am currently an independent Consulting Geologist residing at 3357 Beechwood Drive, Coldwater, Ontario, L0K 1E0;

2) I graduated with a degree in Bachelor of Science Honours, Earth Sciences, from Carleton University (Ottawa, Ontario) in 2008;

3) I am a Professional Geoscientist (P.Geo.) registered with the Professional Geoscientists Ontario (No. 2078);

4) I have worked as a Geologist in various capacities since my graduation from university in 2008;

5) My work experience has included more than 12 years as a geologist including: as an employee of major and junior mining companies, as an employee of engineering consulting firms, and as an independent consultant, including: five months working underground in a producing gold mine; three years working in exploration for numerous commodities (including base, precious, and other minerals); and nine years of resource estimation work including modelling, estimating, and evaluating mineral properties of all types (including base, precious, and other minerals) throughout North America and occasionally globally. I have previously been the primary author on seven NI 43-101 technical reports as well as secondary author or contributor on numerous others. I have worked on several properties with similar or comparative mineralization styles to the Project;

6) I have read the definition of "qualified person" set out in the NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;

7) I am independent of the Company as described in Section 1.5 of NI 43-101;

8) I have no prior involvement with the property that is the subject of the Technical Report;

9) I am responsible for Sections 7-12 (prepared in conjunction with Orix Geoscience and reviewed by myself) and 14; and I am co-responsible for Sections 1 and 28 of the Technical Report;

10) I visited the Idaho Cobalt Operations site from October 4-6, 2019 and reviewed the core shack, the active drilling operation, the historical core storage, and the general site and infrastructure. This included a review of current sampling practices and historical assay

results. Details are discussed in Section 12.6;

11) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Technical Report;

12) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Jervois Mining Limited, or any associated or affiliated entities;

13) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Jervois Mining Limited, or any associated or affiliated companies;

14) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Jervois Mining Limited, or any associated or affiliated companies;

15) I have read NI 43-101 and the relevant Sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101; and

16) As of the effective date of the Technical Report, the best of my knowledge, information and belief, the relevant Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of November 2020.

Scott Zelligan, B.Sc., P. Geo.

Independent Resource Geologist and Associate to Orix Geoscience

Certificate of Author

To accompany the Report titled "NI 43-101 Bankable Feasibility Study Technical Report for the Jervois Mining Idaho Cobalt Operations (ICO) Project", dated November 13, 2020 (the "Technical Report") prepared for Jervois Mining Limited. (the "Company").

I, Nick Yugo, M.Eng, P.Eng., do hereby certify that:

1) I am self-employed as the Director and Principal Engineer of 9140697 CANADA Inc with office at 2848 Apple Drive, Campbell River, British Columbia Canada;

2) I graduated from the University of Toronto with a Bachelor of Applied Science in Mineral Engineering in 2011 and a Master of Engineering in Geomechanics in 2012;

3) I am a registered Professional Engineer in British Columbia (192206), Ontario (100229341), Yukon (2400) and Northwest Territories/Nunavut (L3246).

4) I have worked as a Mining Engineer since 2012 including as a researcher in mining since 2011.

5) My work experience has included numerous underground narrow-vein deposits ranging from base metals to precious metals in both operations and consulting functions over the past 8 years. I have previously signed National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") technical reports involving narrow vein mining;

6) I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101;

7) I am independent of the Company as described in Section 1.5 of NI 43-101;

8) I have had prior involvement with the property that included independent consultation for eCobalt Solutions Inc. prior to the purchase by Jervois Mining Limited.

9) I responsible for Sections 15 and 16 of the Technical Report;

10) I visited the Idaho Cobalt Operations site December 11 to 13, 2018 and reviewed the planned mine portal location;

11) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report;

12) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Jervois Mining Limited, or any associated or affiliated entities;

13) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Jervois Mining Limited, or any

associated or affiliated companies;

14) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Jervois Mining Limited, or any associated or affiliated companies;

15) I have read NI 43-101 and the relevant Sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101; and

16) As of the effective date of the Technical Report, the best of my knowledge, information and belief, the relevant Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 10th day of November 2020.

PROFESSIONAL PROVINCE OF N. YUGO # 43023 BRITISH COLUMBIA ENGINEER

10 Nov 2020

Nick Yugo, P.Eng.

Director and Principal Engineer

9140697 CANADA INC

Certificate of Author

To accompany the technical report titled "NI 43-101 Bankable Feasibility Study Technical Report for the Jervois Mining Idaho Cobalt Operations (ICO) Project", dated November 13, 2020 (the "Technical Report") prepared for Jervois Mining Limited. (the "Company").

I, David P. Cameron, P.E., do hereby certify that:

1) I am employed by KC Harvey Environmental, LLC in the role of Principal Engineer, with my office located at 376 Gallatin Park Drive, Bozeman, Montana, USA;

2) I graduated from the University of Colorado, Denver, Colorado with a Bachelor of Civil Engineering in 1993;

3) I am a professional engineer in good standing in Montana, in the area of civil engineering, Registration Number PE-13019;

4) I have worked as an Engineer in various capacities since my graduation in 1993;

5) My work experience includes engineering design, environmental permitting, reclamation and closure planning including projects requiring analysis and design for disposal of mine waste including tailings and mine overburden;

6) I have read the definition of "qualified person" set out in the NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;

7) I am independent of the Company as described in Section 1.5 of NI 43-101;

8) I have prior involvement with the property that is the subject of the Technical Report. KC Harvey Environmental, LLC is currently providing ongoing consulting services to the project related to reclamation planning and determination of closure cost for the project.

9) I am responsible for Section 20 of the Technical Report and I am co-responsible for Sections 1 and 27 of the Technical Report;

10) I last visited the project site on September 23 and 24, 2020;

11) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Technical Report;

12) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Jervois Mining Limited, or any associated or affiliated entities;

13) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Jervois Mining Limited, or any associated or affiliated companies;

14) Neither I, nor any affiliated entity of mine, have earned the majority of our income during

the preceding three (3) years from Jervois Mining Limited, or any associated or affiliated companies;

15) I have read NI 43-101 and the relevant Sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101; and

16) As of the effective date of the Technical Report, the best of my knowledge, information and belief, the relevant Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of November 2020.

MONTANA
DAVID
CAMERON
No. 13019PE
REGISTERED
PROFESSIONAL ENGINEER

David P. Cameron P.E.

Principal Engineer

KC Harvey Environmental, LLC

29th Floor, 20 Queen Street West, Toronto, M5H 3R3 / Canada
T +1 416 800 8797 / E info@draglobal.com / W draglobal.com

13 November 2020

Certificate of Author

To accompany the technical report titled "NI 43-101 Bankable Feasibility Study Technical Report for the Jervois Mining Idaho Cobalt Operations (ICO) Project", dated November 13, 2020 (the "Technical Report") prepared for Jervois Mining Limited. (the "Company").

I, David Frost, FAusIMM, B. Met Eng, do hereby certify that:

1) I am employed by DRA Americas in the role of Vice President Process Engineering, with my office located at 29th Floor, 20 Queen Street West, Toronto, Canada;
2) I graduated from the Royal Melbourne Institute of Technology (RMIT), Melbourne, Australia with a Bachelor of Metallurgical Engineering in Metallurgy in 1993;
3) I am a registered Fellow Member of the Australian Institute of Mining and Metallurgy (FAusIMM) membership # 110899;
4) I have worked as a Metallurgist and Process Engineer in various capacities since my graduation from university in 1993;
5) My work experience has included more than 26 years of process plant operations and engineering design experience including the oversite of flotation circuit processing in operations and the engineering design of several sequential polymetallic flotation flowsheets including copper gold flowsheets. I have also been involved in the supervision and interpretation of numerous metallurgical testwork programs used for the derivation of process plant flowsheets involving flotation. I have previously signed National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") technical reports involving conventional flotation flowsheets;
6) I have read the definition of "qualified person" set out in the NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101;
7) I am independent of the Company as described in Section 1.5 of NI 43-101;
8) I have no prior involvement with the property that is the subject of the Technical Report;
9) I am responsible for Sections 13, 17 and 21.2.3 of the Technical Report and I am co-responsible for Sections 1, 2, 3, 21, 24, 25, 26 and 27 of the Technical Report;

10) I visited the Idaho Cobalt Operations site from August 27 to 28, 2019 and reviewed the plant location site. I have also personally viewed a selection of cores on site;

11) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Technical Report;

12) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Jervois Mining Limited, or any associated or affiliated entities;

13) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Jervois Mining Limited, or any associated or affiliated companies;

14) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Jervois Mining Limited, or any associated or affiliated companies;

15) I have read NI 43-101 and Form 43-101F1 and have reviewed the relevant sections involved in the Technical Report in compliance with NI 43-101 and Form 43-101F1; these Sections were prepared in conformity with generally accepted - mining industry best practice, and;

16) As of the date of the certificate, the best of my knowledge, information and belief, the relevant Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of November 2020.

David Frost Signed and Sealed

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David Frost, FAusIMM, B. Met Eng

Vice President Process Engineering

DRA Americas Inc. Inc.

555 Boulevard René-Lévesque Ouest / 6ième étage / Montréal / Québec / Canada / H2Z 1B1
T +1 514 288-5211 / E info@draglobal.com

13 November 2020

Certificate of Author

To accompany the technical report titled "NI 43-101 Bankable Feasibility Study Technical Report for the Jervois Mining Idaho Cobalt Operations (ICO) Project", dated November 13, 2020 (the "Technical Report") prepared for Jervois Mining Limited (the "Company").

I, Céline M. Charbonneau, P. Eng., do hereby certify that:

1) I am Senior Project Manager with Met-Chem, a division of DRA Americas Inc, with an office at 555 René-Lévesque Blvd. West, 6th Floor, Montréal, Canada;
2) I am a graduate from "*École Polytechnique de Montréal*" with B.Eng. in Geological Engineering in 1985;
3) I am a registered member of "*Ordre des Ingénieurs du Québec*" (#41764);
4) I have worked as a Geological Engineer or Project Manager continuously since my graduation from university;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent 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 have no prior involvement with the property that is the subject of the Technical Report;
8) I am responsible for Sections 1.14 and 19.0 and contributed part of Section 3.0;
9) I have not visited the project site;
10) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Technical Report;
11) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Jervois Mining Limited, or any associated or affiliated entities;
12) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Jervois Mining Limited, or any associated or affiliated companies;
13) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Jervois Mining Limited, or any associated or affiliated companies;

14) I have read NI 43-101 and Form 43-101F1 and have prepared the relevant Sections involved in the Technical Report in compliance with NI 43-101 and Form 43-101F1; and these Sections were prepared in conformity with generally accepted mining industry best practice, and;

15) As of the date of the certificate, to the best of my knowledge, information and belief, the relevant Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of November 2020.

Céline M. Charbonneau (sign and seal)

Céline M. Charbonneau, PEng., M. Sc.
Senior Project Manager
Met-Chem, a division of DRA Americas Inc