Independent Technical Report for the Condor Gold Project, Zamora Chinchipe Province, Ecuador

Report prepared for
Silvercorp Metals Inc.

SILVERCORP METALS INC.

Report prepared by
SRK Consulting (Canada) Inc.

srk consulting

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Cover Image(s):
A hill side viewed from the exploration camp at Condor showing erosion likely caused by the activities of artisanal miners.

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Useful Definitions

This list contains definitions of symbols, units, abbreviations, and terminology that may be unfamiliar to the reader.

Abbreviation / Symbol / Term / Unit	Definition
Ag	Chemical symbol for silver
Ag Eq	Silver equivalent is a term used express the value or amount of a metal in terms of its silver content, based on their relative market values
Au	Chemical symbol for gold
CCD	Counter current decantation
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
COG	Cut-off grade
CRM	Certified reference material
CSA	Canadian Securities Administrators
DGPS	Differential global position units
doré	A semi-pure alloy of gold and silver
ft^3	Cubic foot
g/t	Grams per tonne
kg	Kilogram
kg/t	Kilogram per tonne
km	Kilometre
km^2	Square kilometre
kt	Kilotonnes
lb	Pound
m	Metre
Ma	Millions of years
masl	Metres above sea level
Mineral Resources	Defined under the CIM Standards
m^3	Cubic metre
μ	Micron
MCF	Mechanical cut and fill is a mining technique where ore is excavated and the resulting void is backfilled with waste rock or other materials to maintain ground stability

Abbreviation / Symbol / Term / Unit	Definition
MRMR	Mineral Resources and Mineral Reserves
MSO	Minable stope optimization
NI 43-101	National Instrument 43-101 (Standards of Disclosure for Mineral Projects), developed by the Canadian Securities Administrators
NSR	Net Smelter Return
oz	Troy ounce (31.1035 g)
Pb	Chemical symbol for lead
ppb	Parts per billion
ppm	Parts per million
P_{80}	Particle size at which 80% of the material in a sample is finer than a specified size
QA/QC	The combination of Quality Assurance (QA), the process or set of processes used to measure and assure the quality of a product, and Quality Control (QC), the process of ensuring products and services meet consumer expectations
QP	Qualified Person
RPEEE	Reasonable prospects for eventual economic extraction, as defined under the CIM Standards
SG	Specific gravity
SRM	Standard reference material
t	Tonnes
tpd	Tonne per day
US$	United States dollar
Zn	Chemical symbol for zinc

1 Executive Summary

1.1 Introduction

This report has been prepared by SRK Consulting (Canada) Inc. on behalf of Silvercorp Metals Inc. (Silvercorp). The purpose of this report is to provide a technical report that documents all supporting work, methods used and results relevant to a Preliminary Economic Assessment (PEA) that fulfills the reporting requirements in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.2 Property Description and Ownership

The Condor Project is located in the Province of Zamora-Chinchipe, near the Ecuador-Peru border and the southern end of the Cordillera del Condor. The Project is approximately 400 kilometres (km) south-southeast of Quito, 149 km east of the city of Loja, and 76 km east of the town of Zamora.

The ownership history of the Condor Project began with artisanal and small-scale miners operating in the area pre-1988. In 1988, modern exploration commenced through a joint venture between ISSFA and Prominex UK. This partnership lasted until 1991 when Prominex UK withdrew, and in 1993, TVX Gold, Inc. (TVX) and Chalupas Mining joined the venture. They remained involved until 2000, after which Goldmarca (formerly Hydromet Technologies Ltd.) formed a new joint venture with ISSFA in 2002. Goldmarca was rebranded to Ecometals Ltd. in 2007 and continued operations until the Ecuadorian government imposed a moratorium on mineral exploration from April 2008 to November 2009. In 2010, Ecometals sold its interest to Ecuador Capital, which was later renamed Ecuador Gold and Copper Corp. (EGX). Lumina Gold Corp (Lumina) acquired EGX in 2016, and in 2018, Lumina spun out Luminex Resources Corp. (Luminex), leading to the Condor Project being 90% owned by Condormining, a Luminex subsidiary, with ISSFA retaining a 10% stake. However, ISSFA has made no funding contribution to the continuing operation of the project; consequently, its share has been diluted to 1.3% to date. In January 2024, Adventus Mining Corporation (Adventus) merged with Luminex. In July 2024, Silvercorp acquired Adventus and assumed the ownership of the Condor Project.

1.3 Geology and Mineralization

The Condor Project is located in the Cordillera del Condor in the Zamora copper-gold metallogenic belt. The Project area comprises epithermal gold-silver, porphyry copper-gold ±molybdenum, and numerous alluvial gold deposits.

The Condor Project's geology is both diverse and complex, particularly in the Condor North area. This region is characterized by distinctive low- to intermediate-sulphidation epithermal vein swarms located in the northern part. These vein swarms form a series of north-northwest-striking, narrow, high-grade gold and electrum-bearing manganoan carbonate veins, often accompanied by base metals and hosted in dacite porphyry. The Condor breccia, dyke, and dome complex is further divided into four main zones: Camp, Los Cuyes, Soledad and Enma. Gold-silver mineralization in these zones is linked with sphalerite-pyrite/marcasite veins, which typically occur within breccias, along the contacts of rhyolite dykes, and as replacements and disseminations. These veins are often disrupted by post-mineral extensional faults.

Camp: The Camp deposit features gold and silver mineralization linked to a swarm of northwest-striking rhyolite-dacite dykes, likely originating from a larger buried rhyolite intrusion. These dykes are concentrated at the contact between a volcanic/intrusive complex and a major granodiorite intrusion. The mineralized zone, dipping steeply at 85° to the northeast, extends over 700 m along strike and is 200 m wide. Gold occurs within veins containing pyrite, marcasite, iron-rich sphalerite (marmatite), galena, ± chalcopyrite, pyrrhotite, quartz, and rhodochrosite gangue. Host rocks include altered granodiorites, breccias, flow-banded rhyolite, and phreatomagmatic breccia. The area is capped by 30 to 80 m of trachyte to rhyolitic welded tuff, with the Camp ridge bounded by the Camp Fault and Piedras Blancas Fault.

Los Cuyes: Los Cuyes is hosted within an oval-shaped diatreme measuring 450 m northeast-southwest, 300 m northwest-southeast, and extending to at least 350 m in depth. This diatreme, resembling an inverted cone plunging approximately 50° to the southeast, consists of an outer shell of polymictic phreatomagmatic breccia and an internal fill of well-sorted rhyolitic lapilli tuffs, breccias, and volcanic sandstones. Amphibolite and quartz arenite fragments occur around its periphery, with dacite and rhyolite ring dykes intruding the steep margins. Lithological contacts, such as dykes cutting through the diatreme and its outer breccia shell, favoured vein development. The mineralization and alteration at Los Cuyes post-date all local rock types, including blocks of the Hollín Formation, indicating that the mineralization is post-Early Cretaceous.

Soledad: The Soledad Zone features a 700-m diameter oval-shaped rhyolite intrusion within the Zamora Batholith, surrounded by discontinuous pyritic breccias. The overall mineralization at Soledad is described as a north-south elongated wine glass-shaped body, tapering between 200 to 300 m below the surface and extending approximately 110 m northwest by 50 m northeast. Sphalerite transitions to pyrite as the dominant sulphide at around 100 m below the surface, leading to diminished gold and silver grades similar to Los Cuyes.

Enma: Gold and silver mineralization at Enma is hosted in a west-northwest-trending rhyolitic breccia that occurs at the contact between andesite lapilli tuffs and the Zamora batholith. The deposit has dimensions of 280 m east-northeast, is approximately 20 to 75 m wide, and has a vertical extent of 350 m. Alteration mineralogy is primarily chlorite with minor quartz-sericite ± alunite-kaolinite. Gold is associated with pyrite-sphalerite-quartz and locally rhodochrosite veins. At depths greater than 200 m, gold-poor, pyrite-pyrrhotite ± chalcopyrite veins are more dominant.

Camp, Los Cuyes, Soledad and Enma Deposits are consistent with low- to intermediate sulphidation epithermal mineralization. Characteristics of such deposits are:

- Occur at convergent plate settings, typically in calc-alkaline volcanic arcs.
- Form at shallow depths (<2 km) from near-neutral pH, sulfur-poor hydrothermal fluids, often of meteoric origin, with metals derived from underlying porphyry intrusions.
- Structural permeability created by hydrothermal fluid over-pressuring allows for mineralized fluids to permeate, with gold precipitated by boiling.
- Sub-types include sulphide-poor deposits with rhyolites, sulphide-rich deposits with andesites/rhyodacites, and sulphide-poor deposits with alkali rocks.
- Hydrothermal alteration is zoned and subtle, characterized by sericite, illite, smectite, and carbonate.

- Features quartz, quartz-carbonate, and carbonate veins with various textures.
- Sulphide content varies (1-20%), typically <5%, with pyrite, sphalerite, galena, and low copper (chalcopyrite).
- High gold, silver, arsenic, antimony, mercury, zinc, lead, selenium, and low copper, tellurium.

1.4 Exploration Status

Since 1994, the Condor Project has undergone extensive drilling by various operators. The drilling campaigns of Condor Project from 1994 to 2023, totaling 538 holes with 157,312 m, focused primarily on the Condor North Area and Condor Central Area.

No QAQC data are available for the TVX drilling programme.

From 2004 to August 2007, the Certified Reference Materials (CRMs or standards Standards), blanks and quarter core duplicate samples were used on the Project. The QAQC procedure from July 2007 to 2011 involved inserting a blank every 6 samples, a standard after 7 samples, a duplicate after 6 samples, followed by another blank. Checks by SRK indicate that this methodology was not strictly adhered to in terms of the number of blanks and standards. From July 2007, OREAS standards and blanks were used, mine waste material was no longer used.

During the Goldmarca / Ecuador Gold and Copper Corp. (EGX) drill programs from 2012 to 2014, CRMS, blanks and quarter core duplicate sample were inserted after every 20 samples as part of the QAQC procedure.

Quality control failures for programs from 2012–2015 were addressed with programs of remedial assay analysis.

During the 2017–2018 drill program, QAQC samples are inserted after every six core samples. These include three certified standards (high, medium and low gold grades), a blank, and a coarse duplicate.

During the 2019–2021 drill program, QAQC samples are inserted with the insert rate about 2% - 4% for each type, including the certified standards, blank, coarse duplicate and fine duplicates.

The author considers that quality control measures adopted for assaying of the Condor Mineral Resource drilling have established that the assaying is representative and free of any biases or other factors that may materially impact the reliability of the analytical results. The author considers that the sample preparation, security and analytical procedures adopted for Condor drilling provide an adequate basis for the current Mineral Resource estimates.

SRK conducted a site inspection of the Condor Project from June 19 to 20, 2024. The inspection was led by Principal Geologist Mark Wanless (QP) from SRK Canada, Falong, Hu (Principal Mining Consultant) and Yanfang Zhao (Principal Geologist) from SRK China, who carried out a series of verification steps. These included a thorough examination of the Project area, meetings with company representatives, and discussions with geologists regarding sample collection, preparation, storage, and QAQC procedures. The team also reviewed geological interpretations, inspected outcrops, mineralization, and fault structures, and verified drillhole sealing marks. Additionally, they visually checked stratigraphy against interpreted drilling sections and visited the drill core storage facility and core catalog room to assess the company's core storage protocols and procedures.

The QP was provided the database named CN_DH_Export_Database_8Sept2023.xlsx which covers the QAQC data for several deposits from 1994 to 2023. A review of historical QAQC data was conducted by SRK.

Based on SRK's site visit, review of the previous and ongoing exploration datasets, communication with the Condor's technical personnel and consideration of the mineralization characteristics of the deposit, SRK is satisfied with the quality and result of the sample preparation and assay conducted by related analytical laboratories. The analytical procedures are consistent with generally accepted industry practices and the primary sample results are therefore suitably reliable for use in Mineral Resource estimation.

1.5 Mineral Resource Estimates

Condor Project comprises several deposits, however this section only focuses on the Camp, Los Cuyes, Soledad and Enma deposits in the Condor North area, which are included in the Mineral Resources estimation.

The Mineral Resource estimation work of Condor Project was completed by SRK in 2025. The estimates are based on drilling samples information available up to 2023. The QP believes the drilling information is sufficiently reliable to interpret with confidence the boundaries for the deposits and that the assay data are sufficiently reliable to support Mineral Resource estimation. Mr. Mark Wanless (Pr.Sci.Nat, FGSSA), and Ms. Yanfang Zhao (MAusIMM), who are Principal Geologists from SRK have reviewed the drillhole database, geological model and the mineralisation domains generated by Silvercorp, made some adjustment, performed the grade estimation, classified the Mineral Resources and prepared the Mineral Resource estimate using Datamine, Isatis.Neo and Leapfrog Geo and Edge.

The Qualified Person responsible for the Mineral Resources is Mr. Mark Wanless, who is a full time employee of SRK Consulting (Canada) Inc. (SRK Canada) and registered with the South African Council for Natural Scientific Professionals as Pr.Sci.Nat, 400178/05, Fellow of the Geological Society of South Africa, Member of the Geostatistical Association of South Africa and a Member of the South African Institute for Mining and Metallurgy (SAIMM). Mr. Mark Wanless visited the Condor Project between the 19th and 20th of June 2024.

The Mineral Resources have been estimated in accordance with generally accepted CIM Definition Standards and are reported in compliance with NI 43-101.

The Company considered future operation on Soledad and Enma using surface mining. However at Camp and Los Cuyes the Company plans underground mining due to the steep terrain conditions, relative complexity, high grade tabular mineralization, and that the surface infrastructure might best be located in Camp and/or Los Cuyes area.

The optimization parameters reflect a conventional open pit operation although the cost and revenue assumptions on Soledad and Enma used are not related to any mine plan or financial analysis, they were used only to define the reasonable prospects for eventual economic extraction (RPEEE) envelope, and the figures were derived from the current information.

For the higher-grade and thicker tabular domains at Camp and Los Cuyes, there is the opportunity of using a bulk mining method such as long hole open stoping for underground extraction. For the open pit deposits, the Mineral Resource is constrained by a conceptual pit, designed using Whittle software. For the underground Mineral Resources, SRK used a Mineable Shapes Optimiser (MSO) to outline areas of the mineralization domain that have suitable continuity and grade to sustain underground mining operations. The summary of the estimated Mineral Resources is shown in Table 1-1 for Mineral Resources with underground mining potential, and in in Table 1-2 for Mineral Resources with open pit mining potential.

The commodity prices are sourced from an independent analyst, Consensus Market Forecast (CMF) for gold, silver, lead, and zinc. The projected outlook (in real USD) was issued by CMF in November 2025. The long-term prices were used for the consideration of the Reasonable Prospect for Eventual Economic Extraction (RPEEE), with a 15% premium used for the Mineral Resource evaluation.

Within the current mining license area, as of 30 November 2025, Mineral Resources are reported for the Condor Project, above a COG of 0.5 g/t and 0.4 g/t gold equivalent for Enma and Soledad respectively which are amenable for open pit extraction.

Table 1-1: Underground Extraction Mineral Resource Statement for Condor Project, November 30, 2025

		Average Grade					Contained Metal				
Deposit	**Tonnes (Mt)**	**AuEq (g/t)**	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**AuEq (koz)**	**Au (koz)**	**Ag (koz)**	**Pb (lb'000)**	**Zn (lb'000)**
Indicated											
Camp	5.93	2.46	1.94	15.51	0.06	0.61	468	370	2,956	7,914	79,864
Los Cuyes	4.22	2.07	1.84	11.06	0.05	0.36	280	249	1,500	4,301	33,067
Total	**10.15**	**2.30**	**1.90**	**13.66**	**0.05**	**0.50**	**748.9**	**620**	**4,456**	**12,215**	**112,931**
Inferred											
Camp	20.04	2.42	1.87	14.83	0.05	0.68	1,557	1,202	9,558	23,042	298,873
Los Cuyes	10.06	2.63	2.37	13.26	0.07	0.36	849	767	4,287	14,936	80,696
Total	**30.10**	**2.49**	**2.03**	**14.31**	**0.06**	**0.57**	**2,406**	**1,969**	**13,846**	**37,978**	**379,569**

Notes: Mineral Resources are reported within a MSO shape for Camp and Los Cuyes with no additional cut-off value applied. Including must take material. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. The Mineral Resources are reported on a 100% basis, and not the portion attributable to Silvercorp.

The resource statement does not include mineralization in the Halo domain of the Los Cuyes, and its economic potential remains to be further investigated in future studies. Optimisations are undertaken using a gold price of USD/oz 3,000, silver price of USD/oz 40, zinc price of USD/lb 1.47 and lead price of USD/lb 1.05.

1 troy ounce = 31.1034768 metric grams

1 metric tonne = 2204.62 lb

Table 1-2: Open Pit Mineral Resource Statement for Condor Project, November 30, 2025

Deposit	Tonnes (Mt)	Average Grade AuEq (g/t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Contained Metal AuEq (koz)	Au (koz)	Ag (koz)	Pb (lb'000)	Zn (lb'000)
Indicated											
Soledad	4.63	1.06	0.98	6.86	0.05	0.54	158.0	146	1020	4,651	55,499
Enma	0.02	1.20	1.12	6.73	0.04	0.34	0.9	1	5	21	180
Total	**4.65**	**1.06**	**0.98**	**6.86**	**0.05**	**0.54**	**158.9**	**147**	**1025**	**4,672**	**55,679**
Inferred											
Soledad	19.99	0.73	0.66	5.97	0.04	0.46	467.8	422	3839	16,588	202,758
Enma	0.01	0.95	0.86	7.82	0.04	0.28	0.2	0	1	4	34
Total	**20.00**	**0.73**	**0.66**	**5.97**	**0.04**	**0.46**	**468.0**	**422**	**3841**	**16,592**	**202,792**

Notes: Mineral Resources are reported in relation to a conceptual pit shell for Soledad and Enma. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. The Mineral Resources are reported on a 100% basis, and not the portion attributable to Silvercorp.

Open pit Mineral Resources are reported at a cut-off grade of 0.5 g/t AuEq for Enma and 0.4 g/t AuEq for Soledad. Open pit optimizations have been determined using a gold price of USD/oz 3,000, silver price of USD/oz 40, zinc price of USD/lb 1.47 and lead price of USD/lb 1.05.

1 troy ounce = 31.1034768 metric grams

1 metric tonne = 2204.62 lb

1.6 Mineral Processing and Metallurgical Testing

A large amount of metallurgical testwork was carried out by Plenge laboratory in Peru between 2020 and 2023 using the mineralized samples from the domains of Camp, and Los Cuyes. The metallurgical testwork included the gravity concentration, whole-ore cyanide leach, bulk flotation, cyanide leach of the bulk flotation concentrate, and sequential selective flotation of gold/silver/lead/zinc from the cyanide leached residue. Some early testwork was completed by Goldmarca Mining Peru, Independent Metallurgical Laboratories and Lehne & Associates Applied Mineralogy.

Most of gold is free milling. The whole-ore cyanide leach resulted in gold recovery on the order of 96% for the Camp domain and 89% for the Los Cuyes domain. A significant amount of gold is recoverable by gravity concentration. The preliminary gravity concentration testwork achieved 34% gold recovery for the Camp domain, 23% gold recovery for the Los Cuyes domain. These results indicate that the flowsheet of gravity concentration followed by cyanide leach is preferred so that the final product will be gold/silver dore. Subject to the metal prices and operating cost, the remaining gold, silver, lead and zinc in the cyanide leached residue may be recovered by selective flotation to generate the marketable lead/silver concentrate and zinc/silver concentrate.

1.7 Mine Geotechnical

A reasonable drillhole based geotechnical data set has been established for both the Camp and Los Cuyes areas, including an extensive point load test data set for each area. The majority but not all veins have sufficient geotechnical data covering the current mining areas. 3D geological models have been built for both areas including a limited structural model.

Camp geotechnical conditions are generally good with little variability, and only occasional weaker or altered zones concurrent with the NW veins. The stope design at Camp includes options for both longitudinal and transverse orientations with a maximum long-wall hydraulic radius of 6.3 m (20mH x 35mL).

Geotechnical conditions at Los Cuyes are generally fair to good with evidence of adverse matrix alteration or matrix weakening associated with some geological contacts which results in the presence of poor ground conditions. Longitudinal and transverse stope designs have maximum long-wall hydraulic radii of 4.2 m (20mH x 15mL) and 4.7m (20mH x 18mL) respectively.

Ground support has been designed for all permanent excavations using resin grouted rebar, and temporary areas using inflatable (Swellex or Omega type) anchors; walls and back require welded wire mesh and an allocation of shotcrete has been included for areas of broken or lower quality ground. Ground support for stoping assumes cable bolting is required for all transverse stope backs, and for longitudinal stopes over 6.0mW (hangingwall to footwall distance).

Future geotechnical work should include: geotechnical re-logging of historic core in the Los Cuyes area; dedicated geotechnical drill holes in particularly the footwall and critical infrastructure areas; an intact rock properties laboratory testing program; update to the structural model and extents of weakening alteration types; review of the temporary pillar stability strategy; and numerical modeling of stope and pillar geometries, and global mine extraction sequence.

1.8 Mining Method

The underground mine design is based on updated geological block models that incorporate the latest resource estimation and NSR values derived from metal prices, metallurgical recoveries, and operating cost assumptions. The NSR was used to define stope envelopes and establish the economic limits for underground extraction. Both the Camp and Los Cuyes zones contain multiple steeply dipping, gold-bearing vein systems with variable widths and grades that are amenable to longhole open stoping methods.

The proposed mine will be accessed via a main portal at approximately 1,100 m elevation, providing access to both Camp and Los Cuyes through ramps and production levels. The life-of-mine plan spans approximately 13 years of production, excluding the pre-production and construction period. The design supports a nominal production rate of 1.8 million tonnes per annum, or 5,000 tonnes per day (tpd).

Each deposit is subdivided into five mining fronts to facilitate concurrent development and maintain consistent mill feed production. Stope sequencing within each mining front will follow a bottom-up extraction sequence. The overall mining sequence between fronts will proceed top-down, allowing for progressive access and ventilation development. At the mining level scale, both transverse primary–secondary stoping and longitudinal retreat stoping methods will be employed, depending on vein geometry and mineralization thickness.

Run-of-mine (ROM) material will be hauled to surface using 50-tonne capacity trucks and directly dumped onto the mill ROM pad for processing. Waste rock generated underground will be hauled by 30-tonne capacity ejector-type trucks either to mined-out stopes for backfilling or, when backfill voids are unavailable, to a surface temporary waste stockpile.

Backfill will be primarily sourced from development waste, with any shortfall supplemented by truck-hauled waste material or gravel aggregate obtained from nearby riverbeds.

The mine ventilation system is designed to deliver approximately 675 cubic metres per second (m^3/s) of airflow to support a maximum production rate of 5,000 tpd. This capacity includes a contingency allowance comprising a 20% leakage factor and a 10% transition factor. Fresh air intake will be provided through the main adit and a network of fresh-air raises (FARs), while exhaust air will exit through the return-air raise (RAR) system. Both the FAR and RAR will be developed using raise boring methods. Each of the Camp and Los Cuyes zones will be equipped with dedicated FAR and RAR systems, ensuring sufficient airflow capacity for underground operations.

Underground main sumps and pump stations will be installed at approximately 100 m vertical intervals, starting from the bottom level of the mining front below the main adit level. Water collected at each level will be pumped in stages to the sump at the next higher level and ultimately discharged to the surface via the main adit. No main sumps or pump stations are planned above the main adit level, as water in these areas will be drained by gravity to the adit level before being directed to the surface.

1.8.1 ROM Material in Mine Plan

There are no mineral reserves declared for the Condor project. In the PEA, Indicated, and Inferred Mineral Resource categories, were considered for inclusion into the mine plans. The resource block models were used for the designs of mining shapes targeting all mineral resources within a mining shape above in situ net smelter return (NSR) cut-off values (COV) of $105/t for the Camp zone and $100/t for the Los Cuyes zone respectively, which were based on gold price of $2,450/oz, silver price of $27.25/oz, lead price of $0.88/lb, zinc price of 1.20/lb, initial estimated total site costs of $95, and varied initial metallurgical recoveries with rock types provided by processing QP. Mining recovery and dilution parameters were applied based on the selected mining methods and geotechnical considerations. External dilution ranges from 7% to 68% depending on the mining method. Mining recoveries vary from 85% to 95% depending on the mining method.

For reporting purposes, a separate set of commodity prices of $2,600/oz for gold, $31.00/oz for silver, $0.91/lb for lead, and $1.27/lb for zinc was used to derive updated NSR formulae. Table 1-3 presents the ROM material estimate by resource class.

The PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

Table 1-3: Run of Mine Material by Resource Class

Mine	Category	Run-of-Mine Plant Feed					
		Tonnes (Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	NSR ($/t)
Camp	Measured	-	-	-	-	-	-
	Indicated	3.07	2.14	15.09	0.06	0.60	181
	Measured + Indicated	3.07	2.14	15.09	0.06	0.60	181
	Inferred	10.75	1.99	14.78	0.05	0.65	169
Los Cuyes	Measured	-	-	-	-	-	-
	Indicated	1.79	2.09	12.16	0.05	0.38	169
	Measured + Indicated	1.79	2.09	12.16	0.05	0.38	169
	Inferred	5.73	2.48	13.27	0.07	0.35	199
Total	Measured	-	-	-	-	-	-
	Indicated	4.86	2.12	14.01	0.06	0.52	176
	Measured + Indicated	4.86	2.12	14.01	0.06	0.52	176
	Inferred	16.48	2.16	14.26	0.06	0.54	179
Total ROM	**Measured + Indicated + Inferred**	**21.33**	**2.15**	**14.20**	**0.06**	**0.54**	**179**

Notes:

1. Totals may not sum due to rounding.
2. The estimated run-of-mine is partly based on Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment based on these mineral resources will be realized.
3. The reader is cautioned that the mineralized material should not be misconstrued as a mineral resource or a mineral reserve. The quantities and grade estimates are derived from the block model and include mining dilution and losses.

1.8.2 Mine Production Schedule

The Condor Project is planned to be contractor-operated for the full duration of the mine life, from start-up through final production. Collaring of the main adit is planned to begin in September of Year-2, initiating underground development activities. Total ROM material mined from Year-1 through Year 1 is estimated at approximately 1.37 million tonnes (Mt), consistent with the planned mill construction and commissioning timeline.

Year 1 marks the start of commercial production, during which mine output continues to ramp up until steady-state ROM throughput of 5,000 tpd or 1.8 million tonnes per annum (Mtpa) is reached in Year 2. Steady production is maintained from Year 2 through Year 12, with Year 13 serving as the final year of mining. The mine schedule is based on 360 operating days per year to allow for routine maintenance and operational downtime. Total LOM ROM material is estimated at 21.34 Mt, grading an average NSR value of $179/t.

LOM lateral development is estimated at approximately 136.7 (km), comprising 42.6 km of capitalized development and 94.1 km of operating development. LOM vertical development totals 4.8 km, all of which is capitalized.

Total LOM waste rock broken is estimated at 3.70 Mt. The underground backfill requirement is 12.34 Mt, to be supplied by approximately 3.70 Mt of development waste and 8.65 Mt of riverbed gravels. As a result, no waste rock is expected to remain on surface at the end of the mine life.

Definition drilling costs have been included in the operating cost estimates; however, a detailed definition drilling schedule has not yet been developed.

1.9 Water Management

A scoping level design for a water management system was prepared. The objectives of the water management system were to provide sufficient water for processing, segregate non-contact and contact water and treat discharge contact water. The sizing of water management infrastructure (e.g. diversions, channels, ponds and the water treatment plant) were not based on climate and hydrologic data collected from site. Climate and hydrologic data from a nearby site (<50km distant) were used.

Sources of contact water are groundwater inflows to the underground mine, runoff from the waste rock pad and process plant pad and excess process water from the tailings storage facility. Contact water from the waste rock and process plant is assumed to be slightly acidic with elevated metal concentrations and the tailings water has a circum-neutral pH and elevated metal concentrations. Contact water is conveyed to a pond from which influent to the water treatment plant is drawn. The water treatment process is high density sludge lime treatment to neutralize acidity and precipitate metals. The sludge from the treatment process will discharge into the tailings line and be deposited in the tailings storage facility.

Costs for the water management and treatment system were estimated.

1.10 Recovery Methods and Processing

There are several distinct mineralization zones within the Condor property. The study focuses on the Camp Zone and the Los Cuyes Zone. The Condor process flowsheet is developed based on the metallurgical test work results described in Section 13 and the mine plan presented in Section 16. The primary metal values are gold and silver, with minor associated values from lead and zinc. The metallurgical test results indicate that the Condor mineralization is amenable to gold and silver recovery through a combination of gravity concentration and cyanidation. Although the lead and zinc grades of the mineralization are relatively low, the lead and zinc minerals also respond well to a conventional flotation process.

The proposed process plant will treat the mineralized material at a milling rate of 5,000 t/d, or 1.8 Mt/a, with an average LOM head grade of 2.15 g/t gold and 14.2 g/t silver. The overall gold and silver recoveries to doré in cyanidation circuit are estimated to be approximately 93% and 46%, respectively. A two-stage grinding circuit, integrated with a gravity concentration, is proposed to grind the cyanide leach feed to 80% passing (P80) approximately 74 µm. The ground mill feed is processed in a carbon-in-pulp (CIP) cyanidation circuit. The loaded carbon is washed and stripped, and resulting pregnant solution is treated by an electrowinning unit to recover gold and silver, producing gold-silver doré. The leach residue is treated to destroy residual weak acid dissociable (WAD) cyanide. Subsequently, the leach residue is further processed by conventional differential flotation to produce marketable silver-lead and zinc concentrates separately. The flotation tailings is thickened and pumped to Tailings Storage Facility (TSF) for storage.

1.11 Tailings Management

A cross-valley embankment TSF was designed to retain conventional slurry. The TSF location is approximately 3 km southwest of the proposed processing plant and was selected after a comparison of several options based on the embankment volume to storage capacity ratio and the TSF catchment area. The TSF was designed to accommodate 21.3 Mt of tailings over the life of the mine. The zoned rockfill embankment will be constructed in stages using downstream construction methods over the mine life to suit tailings and water storage requirements. A 40-m high starter dam will be constructed to accommodate the first two years of production followed by five raises throughout the 13 year mine life.

1.12 Environmental

The Project currently holds all necessary environmental permits for the advanced exploration phase and complies with all applicable legal and regulatory obligations.

In 2015, Condor project obtained EIA certification, granting them an environmental license for advanced exploration.

A new EIS report was submitted to the Ministry of Environment in March 2025, to obtain a new environmental permit for exploitation (the current one is for exploration only), under the regime of Small Mining. With this new environmental permit, underground development can occur to provide access for underground resource definition drill programs. Currently, the report has been reviewed and approved by various functional departments of the ministry, with a final statement of approval to be announced by the regularization directory of the ministry. Once the approval of the EIS is announced, the PPC process can be initiated to get the approval or consent of the local communities. Silvercorp project team has been working together with the local communities to get their social consent. Once the PPC is completed and assuming it is in favour of the proposed project, the ministry can issue the new environmental permit. The EIS announcement is expected to be delivered prior to the end of 2025.

Illegal and informal mining represent a high risk, as they operate within and around the project's concession areas. The Shuar Indigenous communities are vulnerable to accepting the presence of informal and illegal miners due to limited State support, poverty, and weakened social organizations. Silvercorp recognize this risk and are developing a strategy to address this risk. Additionally, employment demands must comply with Ecuadorian law, which requires that 80% of workers be local residents.

1.13 Economic Analysis

The Condor Gold Project has been evaluated on a discounted cash flow basis in constant 2025 US dollars assuming all equity project financing. Economic results are presented for the entire project. SRK understands that as of the date of this report Silvercorp owns 98.73% if the Condor Gold Project through its ownership of Condormining S.A.

The economic analysis includes capital costs that are forecast to be incurred after the start of a two-year construction period. Condor Gold Project expenditures that will be incurred prior to this point, such as costs for further exploration drilling, field investigations and analysis, more detailed technical and environmental studies, and surface rights land acquisition, are excluded from the PEA economic analysis.

Base case economic results are summarized in Table 1-4. The results of the analysis show the Condor mineral resources to be potentially viable and relatively strong project economics. At a base case gold price of $2,600/oz, the potential pre-tax present value of the net cash flow at the start of the projected two-year construction period using a 5% discount rate (NPV5%) is estimated at $720M, and potential project post-tax NPV5% is estimated at $522M. Potential internal rates of return (IRR) are respectively 36% pre-tax and 29% post-tax.

At the base case gold price and project cost estimates payback of the initial capital is forecast to occur in the third year of the 13-year operating mine life. The payback period is defined as the time after process plant start-up that is required to recover the initial expenditures incurred developing the Condor Gold Project. At this point in time the project's cumulative undiscounted net cash flow is zero.

Table 1-4: Base Case Economic Results Summary

	Unit	Total
Plant Feed	**Mt**	**21.34**
Payable Gold	Oz (000)	1,375
Payable Silver	Oz (000)	5,266
Payable Lead	lbs (000)	8,448
Payable Zinc	lbs (000)	95,656
Equiv. Payable Gold	**Oz (000)**	**1,487**
Net Smelter Return	$/t	179
Operating Costs		
Mining	$M	875
Processing	$M	392
Water Management	$M	15
Mining Supervision Fees	$M	17
Conservation Fees	$M	2
Refining and Freight	$M	54
Royalties	$M	191
Profit Sharing State	$M	164
Profit Sharing Employee	$M	41
All Other G&A	$M	288
Total Operating Cost	**$M**	**2,038**
Total Operating Cost	**$/t-milled**	**95.51**
Capital Costs		
Initial Capital	$M	292
Sustaining Capital	$M	382
LOM Total Capital	**$M**	**674**
Project All-In Cost	$M	3,002
Cash Cost	$/EqOz-Payable	1,118
All-in Sustaining Cost (AISC)*	**$/EqOz-Payable**	**1,359**
Project All-in Cost	$/EqOz-Payable	2,018
Economic Indicators		
Project Pre-tax Cash Flow	$M	1,156
Pretax NPV 5%	$M	720
Pre-tax IRR		36%
Payback from Mill Start	Yr	3.0
Post-tax Cash Flow	$M	865
Post-tax NPV 5%	$M	**522**
Post-tax IRR		**29%**

* Based on World Gold Council June 27,2013 Press Release: "Guidance Note on Non-GAAP Metrics - All-In Sustaining Costs and All-in Costs"

The sensitivity analysis was performed on the base case considering variations in gold price, operating cost, and capital cost.

Like most greenfield mining projects, the key economic indicators of NPV5% and IRR are most sensitive to change in gold price (i.e. revenue), as it affects directly the revenue stream. A 10% reduction from the US$2,600/oz base case gold price reduces Condor’s post-tax NPV5% and IRR by 28.5% and 6 percentage, respectively. A 10% increase from the US$2,600/oz base case gold price increases Condor’s post-tax NPV5% and IRR by 28.5% and 5 percentage, respectively. The sensitivity analysis shows that the project is less sensitive to operating cost and capital expenditure.

The PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

1.14 Conclusions and Recommendations

Silvercorp has reviewed, re-logged, and remodelled the mineralization at the Condor Project. At the Los Cuyes and Camp deposits the updated model of mineralization has included identification of several high-grade tabular domains which are potentially amenable to extraction using underground mining methods. At Soledad, Enma and outside of the high-grade domains at Los Cuyes Silvercorp have modelled a lower grade disseminated mineralization which has the potential for extraction using an open pit mining method.

This mineralization interpretation at Los Cuyes is a change from the previous model which only considered a disseminated mineralization style, and did not isolate the high-grade zones separately. For some domains at Los Cuyes (such as the LCW domain) the data strongly support the revised interpretation, with good continuity in the mineralization observed over the project area. While for other domains, the continuity is less clear, and the quantity of data supporting these is less, resulting in lower confidence in these interpretations. The lateral extents of some of the domains are based on wider spaced drilling which naturally carries some additional risk to the confidence in the interpretation of the domain continuity.

At Camp, the previous models relied on interpolated domain definition using indicators, and the current interpretation is supported by a more geologically rigorous interpretation using a combination of the grade and geological logs to link up intersections between drill holes into more coherent and continuous domains.

The geological interpretation at Soledad and Enma is not as well developed as that of Los Cuyes and Camp, relying on grade shells to constrain the mineralization. At Soledad, there is sufficient dense sampling in several locations to confirm the continuity of the mineralization despite the lower understanding of the mineralization controls, and SRK considers this sufficient to support an Indicated Mineral Resource classification.

For all the deposits, the metallurgical test work indicated that there are reasonable prospects for achieving the recoveries applied to the economic assessment. However, further work is required to be able to confirm the optimal processing configuration for each style of mineralization. As such, there is a risk that these recovery factors may change with additional test work and depending on the ultimate processing flow sheet that is selected if the project is developed.

To confirm the interpretation of the high-grade domains at Camp and Los Cuyes, SRK recommends that an exploration program should be undertaken. Silvercorp has planned an initial exploration program of underground drilling program from an exploration drive (pending the approval of an environmental permit which is currently in progress to intersect the mineralization, and to provide platforms for drilling which will allow for better targeted drilling of shorter holes from the underground development. The cost of the development and a 30,000 m drilling program is estimated at US$10.5M.

Geotechnical confidence at Los Cuyes should be improved by expanding core-based data coverage through re-logging or machine-learning analysis of historical drill core, supplemented by dedicated oriented geotechnical drilling with focus on hangingwall, vein, footwall and critical infrastructure areas. Laboratory strength testing to confirm intact rock strength for dominant lithologies and correlations for point-load testing. The structural model and geotechnical domains should be updated to reflect structure, alteration, and rock mass variability, and used to reassess pillar stability and refine stope designs. These updates should be validated through numerical modeling of stopes, pillars, and the overall mine extraction sequence to confirm ground stability and design assumptions.

The updated block models for the Camp and Los Cuyes zones provide a robust foundation for mine planning, incorporating NSR values derived from metallurgical recoveries, processing costs, and metal prices to support realistic economic evaluation, stope design, and cut-off grade determination. The selected longhole stoping method is technically appropriate for the steeply dipping vein systems, with preliminary designs indicating that orebody geometry and geotechnical conditions can sustain stable stopes with acceptable dilution and recovery. The main portal at approximately 1,100 m elevation provides efficient access for haulage, ventilation, and services, and the mine plan defines an operating life of approximately 13 years (excluding pre-production development) at a steady-state throughput of 1.8 Mtpa (5,000 tpd) from the Camp and Los Cuyes zones.

To advance the project to PFS level, metal prices, treatment charges, and recoveries should be re-benchmarked and sensitivity analyses completed to validate NSR cut-offs, while stope geometries should be refined using updated geotechnical and structural data and validated through numerical modeling or trial stope simulations. Further work is also required to evaluate backfill options, confirm pillar stability, and optimize ramp and level spacing to balance capital efficiency and production flexibility. A detailed mine scheduling study should be completed to confirm production targets and ore delivery consistency, supported by a review of mobile equipment fleet sizing, utilization, and ventilation compatibility, including potential adoption of low-emission equipment. In parallel, the design of key underground infrastructure systems, including power, pumping, dewatering, and materials handling, should be advanced to a PFS level of definition.

No fatal flaws related to water management have been identified; however, the current level of site data is insufficient to support detailed system design. Advancement of the project requires additional studies to characterize climate, hydrology, hydrogeology, geochemistry, and water quality, including investigation of waste rock, tailings, quarry materials, and baseline water conditions, with particular attention to potential mercury contamination from artisanal mining. A comprehensive program should include installation of meteorological and hydrologic gauging stations, characterization of river and creek flows and floodplains near planned infrastructure, assessment of groundwater conditions and geotechnical properties within water management footprints, and development of an integrated site-wide water balance. Hydrogeological field data should be collected through exploration or geotechnical drilling, including hydraulic testing and monitoring well installation, with targeted testing of geological structures that may represent dominant groundwater inflow pathways.

A proposed TSF located 3 km southwest of the processing plant is designed to accommodate 21.3 Mt of tailings over the life of the mine. A 40-m high starter dam will be constructed to accommodate the first two years of production, followed by five raises throughout the 13-year mine life. The cost of this facility will be US$91.8M, with initial capital cost estimated at $18.1M.

A required upgrade to the current Cumbaratza substation includes a new bay with 138/69 kV and transformer of 25 MVA to supply power to the mine in 69 kV. An overhead power transmission line with single-circuit configuration of Aluminum Conductor Alloy Reinforced (ACAR) with incorporated Optical Ground Wire (OPGW) Project Infrastructure is required to transfer power to the project. An independent 13.8 kV feeder from the main switchgear is required for each mine with a 13.88/4.16 kV substation, switchgear, and 4.16 kV feeder cables to the portable underground substations. A single 13.8/4.16 kV transformer will power the mill motors at medium voltage, all others are 13.8/0.48 kV for primarily low-voltage motors. Dry-type transformers are considered for environmental, safety and ease of installation criteria.

The social context surrounding the project is marked by the expansion of informal and illegal mining, the vulnerability of Shuar communities, and the growing local demand for employment and services. Silvercorp is aware of this risk and are developing a strategy to address the social risks associated with development in this region in order to secure a social license to operate the Condor Project. Compliance with national labor regulations and proactive management of population dynamics will be essential to ensure the project's long-term social viability and positive community relations.

The economic analysis indicates that the Condor Project is potentially favorable, with a sensitivity analysis demonstrating that project value is most strongly influenced by gold price, followed by capital costs and, to a lesser extent, operating costs. A key project risk is the potential for gold prices to underperform the long-term assumptions applied in this study, which would materially impact economic outcomes. Additional fiscal risk exists related to potential unforeseen taxation; however, the assumption that the project, as a gold mining operation and exporter, would not be subject to Ecuadorian VAT or customs duties, or would be eligible for full VAT recovery during pre-production and operations, is supported by current mining regulations and publicly available information, but should be formally confirmed with the relevant Ecuadorian tax authorities.

2 Introduction and Terms of Reference

The Condor Project is an advanced stage gold exploration project, located in the Province of Zamora-Chinchipe, near the Ecuador-Peru border and the southern end of the Cordillera del Condor. The Project is approximately 400 km south-southeast of Quito, 149 km east of the city of Loja, and 76 km east of the town of Zamora. Silvercorp Metals Inc. (Silvercorp) assumed ownership of the Condor Project in July 2024 through the acquisition of Adventus Mining Corporation.

In May 2024, Silvercorp commissioned SRK Consulting (Canada) Inc. (SRK) to generate a Mineral Resource model and provide a Mineral Resource statement. In June 2024, the Mineral Resource QP, Mr. Mark Wanless and Mining Engineer Mr. Falong Hu conducted a site visit the property. During this mandate, Mr. Wanless verified the project and validated geological information required to produce a Mineral Resource estimate. The Mineral Resource statement reported herein was disclosed publicly by Silvercorp in a news release on May 12, 2025.

In May 2025, Silvercorp commissioned SRK to assume the role of mining lead and QP. During a site visit in July 2025, Mr. Benny Zhang reviewed Silvercorp's current procedures and provided guidance for the underground geotechnical assessment and input to the surface and ground infrastructure. A review of the current environmental and social aspects related to the local communities and artisanal miners was also conducted. The services were rendered between May to December 2025 leading to the preparation of the Preliminary Economic Assessment (PEA).

This technical report documents a PEA based on the Mineral Resources disclosed in May, 2025, for the Camp, Los Cuyes, Soledad and Enma deposits of the Condor Project prepared by SRK and updated in this report based on the change in mining method planned for Camp and Los Cuyes and on updated metal price assumptions for all for deposits. This PEA report is focussed on the development of the Camp and Los Cuyes underground deposits. This report was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. The Mineral Resource statement reported herein was prepared in conformity with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines*.

This technical report summarizes the technical information available on the Condor Project. In the opinion of SRK, this property has merit warranting additional exploration expenditures.

2.1 Scope of Work

The scope of work, as defined in a letter of engagement executed on June 09, 2025 between Silvercorp and SRK includes an update to the PEA based on the re-interpretation of the geology and controls on mineralization, completed by Silvercorp, wherein it was determined that an underground mining approach is more suitable to exploit the ore bodies at the Camp and Los Cuyes deposits.

A strategic assessment to evaluate key project parameters was undertaken to identify areas to maximize the NPV and bring forward value. SRK evaluated three strategic options at a high level (production rate, mining rate, cut-off grade and stockpiling), and determined the capital and mining costs for the selected production rate and mining method. This process determined:

- The capacity of the Camp and Los Cuyes orebodies and their ability to deliver to a processing facility. A suitable production rate that matched with the design of the processing facility, and no trade-off analysis of alternative options.
- A suitable underground mining method optimized for the orebodies at a minimized operating cost, with no trade-off.
- A cut-off grade and stockpiling strategy using the identified optimal cut-off grade(s) for underground areas and evaluate ways to feed higher grade/value material.
- A mine plan includes stope optimization, design and access, development, production and schedule. This is inclusive of equipment selection and materials handling, ventilation analysis, air heating and fan section as well as labour requirements.

SRK also considered ways to optimize the Environmental and Social (E&S) performance of the project during the strategic assessment. This examined approach considered alternatives or design options that:

- Avoided or minimized potential negative E&S impacts or enhanced positive impacts
- Considered and protected the rights of surrounding land and water users
- Optimized waste and energy usage and greenhouse gas emissions
- Are resilient to future climate change risks and scenario
- Reduced future closure liabilities

SRK prepared an economic analysis and discounted cashflow financial model developed on an annual basis that included:

- A post-tax, pre-finance basis US$ dollar real money terms at a determined date
- A production model from mine to point of sale, for the life of the mine
- Operating costs, initial capital and sustaining capital expenditures
- Commodity price forecasts
- Depreciation, taxes and working capital movements
- Project unit operating costs, all-in costs and all-in sustaining costs
- Project NPV (with appropriate discount rate applied), IRR and payback period
- A sensitivity analysis of the project economics to changes in product prices, capital expenditure, operating costs, recoveries, royalties and discount factor

SRK has previously prepared an independent technical report in compliance with National Instrument 43-101 and Form 43-101F1 guidelines (May 2025).

2.1.1 Work Program

The Mineral Resource statement reported herein is a collaborative effort between Silvercorp and SRK personnel. The exploration database was compiled and maintained by Silvercorp and was audited by SRK. The geological model and outlines for the gold mineralization were constructed by SRK from a two-dimensional geological interpretation provided by Silvercorp. In the opinion of SRK, the geological model is a reasonable representation of the distribution of the targeted mineralization at the current level of sampling.

The geostatistical analysis, variography and grade models were completed by SRK during the months January and March 2025. The Mineral Resource statement reported herein represents an update to that disclosed publicly in a news release dated June 12, 2025. The update reflects the changes in the economic assumptions, and considers the changes to the underground mining methods selected during the development of this PEA.

The Mineral Resource Statement reported herein was prepared in conformity with the generally accepted CIM *Exploration Best Practices Guidelines* and CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines*. This technical report was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

Following the release of the Mineral Resource technical report dated May 12, 2025, SRK has led the development of a PEA on the Los Cuyes and Camp deposits at Condor. SRK is responsible for the Mineral Reosurces, mining and underground infrastructure, ventilation design, environmental assessment, surface and underground water management, and mining related surface infrastructure. The process facility and tailings storage design has been undertaken by consultants from Tetra Tech, and the metallugical testwork assessment is undertaken by JJ Metallurgical Services Inc.

Through testing mineral processes and metallurgical processes, which include preliminary gravity concentration testwork, a marketable concentrate was determined. Geotechnical data was utilized to determine ground support required for all excavations, permanent and temporary, stoping and longitudinal stoping.

NSR values based on current metal prices, metallurgical recoveries, and operating cost assumptions was used to define stope envelopes and establish economic limits for underground extration. A detailed mine plan was developed to access Camp and Los Cuyes areas include a life-of-mine schedule, pre-production and construction timelines, and mill feed requirements. Stoping methods, backfill strategies, and waste management practices, along with the design of ventilation systems, pumps, and sumps were determined to support safe and efficient underground operations.

A scoping level design for water management was designed to determine the source and procedures around incoming water, contact water, and water treatment. A tailings storage facility was designed to accommodate the tailings of the life of mine and costs associated are included. The environmental stoping methods, backfill strategies, and waste managemnet practices, along with the design of ventiallation systems, sumps and pump station to support a safe and efficient underground operation.

Utilizing the results from all components mentioned, an economic analysis including capital costs during construction and a sensitivity analysis were determined.

The technical report was assembled in Toronto during the months of June to December, 2025.

2.2 Basis of Technical Report

The basis of this report will be the SRK Mineral Resource estimate, mine plan and PEA, with additions and sectional updates on the SRK (2025) technical report. The mine plan and underground aspects of the PEA were reviewed during a site visit performed between July 08 to 10, 2025 and additional information provided by Silvercorp throughout the course of SRK's investigations. SRK has no reason to doubt the reliability of the information provided by Silvercorp. Other information was obtained from the public domain. This technical report is based on the following sources of information:

- May 12, 2025, SRK Technical Report “Independent Technical Report for the Condor Project, Ecuador” (SRK, 2025)
- Inspection of the Condor project area during two separate site visits, including outcrop and drill core
- Review of exploration data collected by Silvercorp
- Technical and cost information provided by Silvercorp
- Technical information provided by Tetra Tech, including the capital cost and layout of the processing facility in Sections 13 and 17
- Discussions with Silvercorp personnel
- Review of exploration data collected by Silvercorp
- Additional information from public domain sources
- Siting and design of the tailings storage facility by Tetra Tech and waste rock dumps by SRK
- Review of available geotechnical data and logging to support mining method selection, stope design and ground support recommendations
- Review of selected core photos, geotechnical data, deposit context and widely accepted empirical assessments to define optimum geometries and estimates of dilution
- Review of the design, layout, costing and reporting of the tailings provided by Tetra Tech

2.3 Qualifications of SRK and SRK Team

The SRK Group comprises more than 1,700 professionals, offering expertise in a wide range of resource engineering disciplines. The independence of the SRK Group is ensured by the fact that it holds no equity in any project it investigates and that its ownership rests solely with its staff. These facts permit SRK to provide its clients with conflict-free and objective recommendations. SRK has a proven track record in undertaking independent assessments of Mineral Resources and mineral reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies, and financial institutions worldwide. Through its work with a large number of major international mining companies, the SRK Group has established a reputation for providing valuable consultancy services to the global mining industry.

The resource evaluation work and the compilation of this technical report was completed by SRK, Mr. Yanfang Zhao (MAusIMM), Ms. Jessica Elliott, GIT (EGBC#195174) and Ms. Joycelyn Smith, PGeo (PGO#4963), under the supervision of Mr. Mark Wanless, FGSSA, Pr.Sci.Nat, (400178/05). The underground mine planning work was prepared by Mr. Benny Zhang, MEng, PEng (PEO#100115459) of SRK, the Qualified Person taking professional responsibility, and Mr. Eric Wu, PEng (PEO#100604418). Mr. Sean Kautzman, PEng (PEO# 100159892) is responsible for the mining surface infrastructure and underground infrastructure.

The metallurgical testwork was reviewed and compiled by Dr. Jinxing Ji, PhD, PEng (EGBC#59305) with JJ Metallurgical Services Inc., and a Qualified Person as defined by Nation Instrument 43-101. Dr. Ji is responsible for the metallurgical test work and processing recoveries.

Mineral Processing was conducted by Dr. Jianhui Huang, PhD, PEng (EGBC# 30898), a Principal Process Engineer with Tetra Tech Canada Inc. (Tetra Tech), and a Qualified Person as defined by National Instrument 43-101. Dr. Huang is responsible for process design and process related capital and operating cost estimates.

Mr. Chris Johns, PEng (EGBC# 39423) a Principal Geotechnical Engineer with Tetra Tech, and Qualified Person is responsible for tailings facility design and related capital cost estimate.

The environmental studies, permitting and social or community impact was prepared by Mr. Raul Pastor, Mr. Mijail Camborda, and Mr. Rasul Camborda of SRK Peru, under the supervision of Mr. Mark Liskowich, PGeo (#10005) a Professional Geologist and associate Principal Consultant with SRK Canada, and a Qualified Person as defined by National Instrument 43-101 responsible for Environmental, Social and Governance issues.

By virtue of their education, membership to a recognized professional association and relevant work experience, Mr. Mark Wanless, Mr. Benny Zhang, Mr. Mark Liskowich, Mr. Sean Kautzman, Dr. Jinxing Ji, Dr. Jianhui Huang, and Mr. Chris Johns are independent Qualified Persons as this term is defined by National Instrument 43-101. Complete list of persons responsible are listed in Table 2-1.

Mr. Glen Cole, PGeo (PGO#1416), a Principal Consultant and Practice Leader with SRK, reviewed drafts of this technical report prior to their delivery to Silvercorp as per SRK internal quality management procedures. Mr. Cole did not visit the project.

Table 2-1: List of Qualified Persons Responsibilities

Qualified Person	Company	QP Responsibility/Role	Report Section(s)
Mark Wanless	SRK	Resource Estimation	Sections 3, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23. Shared responsibility for Sections 1, 2, 24, 25, 26, and 27
Benny Zhang	SRK	Mining Engineering	Sections 15, 16, 19, 21.3.2-21.3.8, 21.3.10, 22. Shared responsibility for Sections 1, 2, 24, 25, 26, and 27
Mark Liskowich	SRK	Environment and Permitting	Sections 4 and 20. Shared responsibility for Sections 1, 25, and 26
Sean Kautzman	SRK	Infrastructure	Shared responsibility for Sections 1, 18, 21, 25, and 26
Jinxing Ji	JJ Metallurgical Services Inc.	Metallurgical Test Work and Recoveries	Section 13. Shared responsibility for Sections 1, 25, and 26
John (Jianhui) Huang	Tetra Tech	Mineral Processing	Section 17. Shared responsibility for Sections 1, 21, 25, 26 and 27
Chris Johns	Tetra Tech	TSF Geotechnical	Section 18.5. Shared responsibility for Sections 1, 21, 25, 26, 27

2.4 Site Visit

In accordance with National Instrument 43-101 guidelines, Mr. Wanless, Ms. Zhao, and Mr. Hu travelled to the Condor Project in June 2024 to undertake an inspection of the project, the drill core available at the Camp site for these projects, and to review the exploration procedures, data capture and geological interpretation with the Silvercorp exploration team.

The purpose of the site visit was to review the digitalization of the exploration database and validation procedures, review exploration procedures, define geological modelling procedures, examine drill core, interview project personnel, and collect all relevant information for the preparation of a revised Mineral Resource model and the compilation of a technical report. During the visit, a particular attention was given to the treatment and validation of historical drilling data.

The site visit also aimed at investigating the geological and structural controls on the distribution of the gold mineralization in order to aid the construction of three dimensional gold mineralization domains.

A three day site visit from July 8 to 10, 2025 was conducted by two SRK employees, Mr. Benny Zhang from the Toronto office and Mr. Mikhail Camborda from the Lima office. Mr. Zhang undertook the role of mining lead and QP, overseeing the mining capital and working cost estimation and building the financial model. During the site visit Mr. Zhang provided guidance for the underground geotechnical assessment and input to the surface and underground infrastructure. Mr Camborda led the engagement on the environmental and social aspects of the project, and reviewed the current state of these investigations and assessments.

The siting and design of the storage tailings facility and the design and costing of the processing facilities were undertaken by a team of consultants from Tetra Tech in Vancouver lead by Mr. Chris Johns, PEng (Tailings) and Dr. Jianhui Huang, Ph.D., P.Eng (Processing). Mr. Johns undertook a project site visit from July 8 to10, 2025 with other members of the study team. Mr. Johns was responsible for the project tailings storage facility design and during the visit visually assessed potential tailings storage facility locations and reviewed design criteria and tailings management options with the study team.

The teams were given full access to relevant data and conducted interviews with exploration staff to obtain the information required for technical reporting.

SRK was given full access to relevant data and conducted interviews with Silvercorp personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store and analyze historical and current exploration data.

2.5 Acknowledgement

SRK would like to acknowledge the support and collaboration provided by Silvercorp personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

2.6 Declaration

SRK's opinion contained herein and effective **November 30, 2025** is based on information collected by SRK throughout the course of SRK's investigations. The information in turn reflects various technical and economic conditions at the time of writing this report. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Silvercorp, and neither SRK nor any affiliate has acted as advisor to Silvercorp, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

3 Reliance on Other Experts

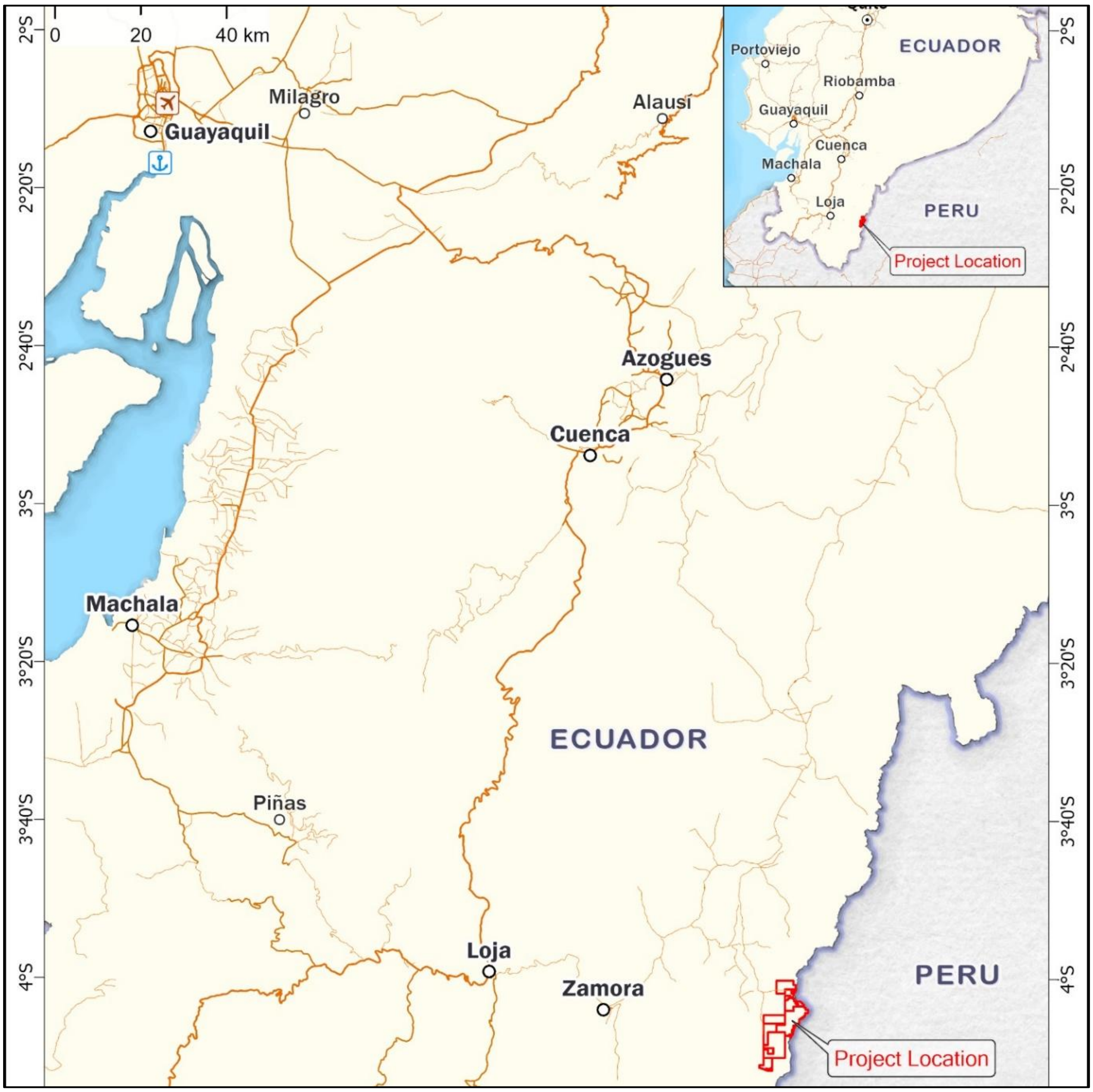
SRK has not performed an independent verification of land title and tenure information as summarized in Section 3 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but has relied on Flor, Bustamante, Pizarro, Hurtado Abogados as expressed in a legal opinion provided to Silvercorp on August 26, 2024. The reliance applies solely to the legal status of the rights disclosed in Sections 3.1 and 3.2 below.

SRK was informed by Silvercorp that there are no known litigations potentially affecting the Condor project.

4 Property Description and Location

The Condor Project is located in the Province of Zamora-Chinchipe, near the Ecuador-Peru border and the southern end of the Cordillera del Condor (Figure 4-1). The Project is approximately 400 km south-southeast of Quito, 149 km east of the city of Loja, and 76 km east of the town of Zamora. The approximate centre of the Project properties is located at 95523500 m North and 768000 m East (geographic projection: Provisional South American Datum 1956, UTM Zone 17M).

Figure 4-1: Condor Project Location

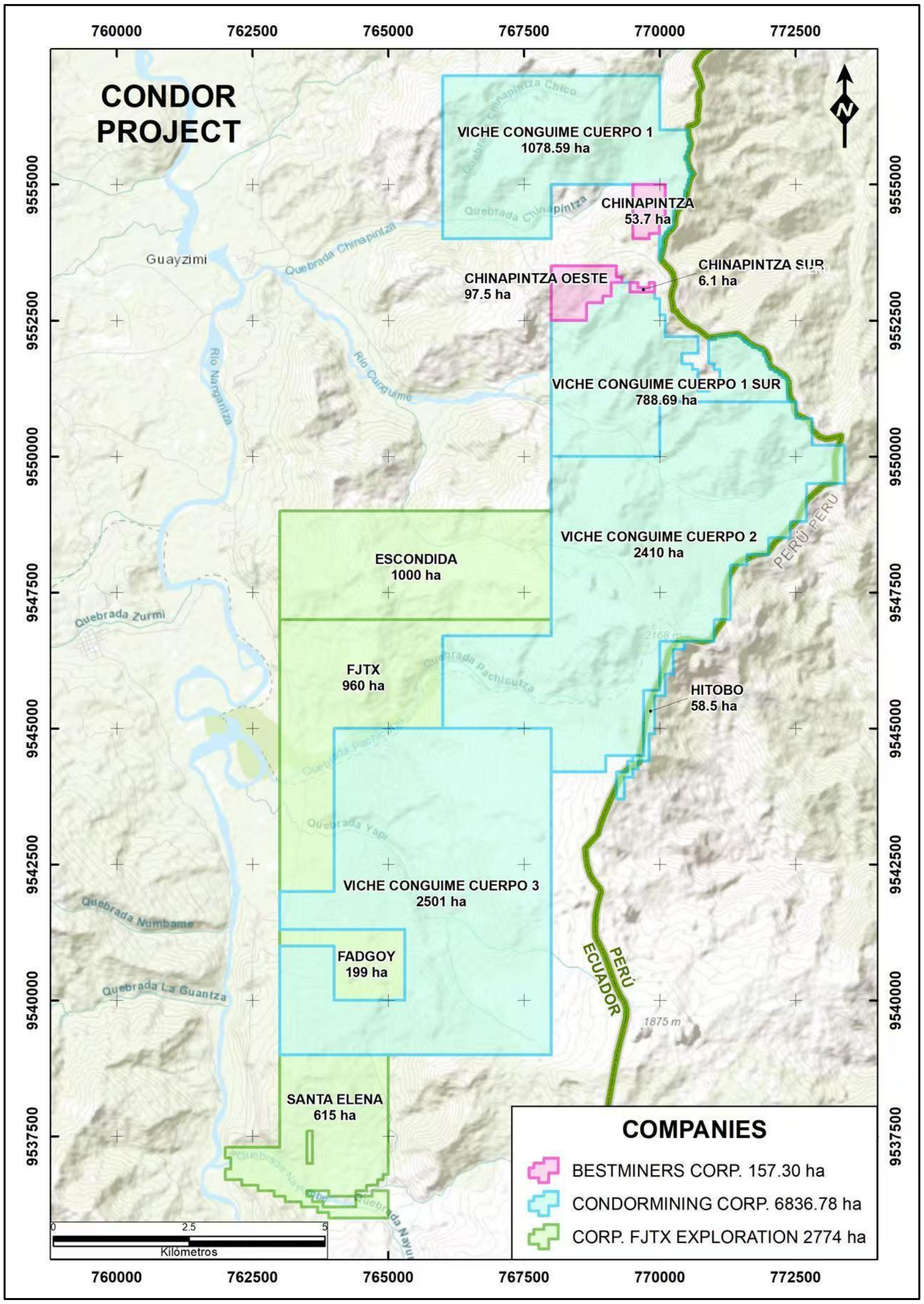


Source : Independent Technical Report for the Condor Project, Ecuador (SRK, 2025).

4.1 Mineral Tenure

The Condor Project consists of a number of concessions i: Chinapintza (including Chinapintza Oeste and Chinapintza Sur), Escondida, FADGOY, Hitobo, Santa Elena, Viche Congüime I (including Viche Congüime Cuerpo I and Viche Congüime Cuerpo I Sur), Viche Congüime Cuerpo II (including Viche Congüime Cuerpo III and FJTX,) (Figure 4-2).

Figure 4-2: Mineral Tenure Information



The mining concessions are held under Condormining S.A, Corporación FJTX S.A, and Bestminers Ecuador S.A., subsidiaries of Adventus Mining Corporation which was acquired by Silvercorp in 2024. Adventus Mining Corporation owns 100% of Ecuador Gold Holdings Ltd., which owns 98.73% of Condormining S.A through its 100% owned subsidiary EMH S.A.. Corporación FJTX is owned by Adventus Mining Corporation through EMH S.A., which holds 100% of the common shares of Corporación FJTX. Bestminers S.A. is 100% owned by Adventus Mining Corporation through Condormining Corporation S.A.

Condormining S.A. holds five mining concessions that are part of the Condor Project, namely:

- Viche Congüime Cuerpo 1 (registered May 20, 2010, valid for ~21 years)
- Viche Congüime Cuerpo 1 Sur (registered January 21, 2025, valid for 6 years)
- Viche Congüime Cuerpo 2 (registered May 21, 2010, renewed in 2021 for 25 years)
- Viche Congüime Cuerpo 3 (registered May 20, 2010, valid for ~22 years)
- Hitobo (registered May 25, 2010, valid for ~21 years)

Corporación FJTX S.A. holds four mining concessions also included in the Condor Project, namely:

- Escondida (registered February 17, 2017, valid for 25 years)
- Santa Elena (registered February 17, 2017, valid for 25 years)
- FJTX (registered May 25, 2010, valid for ~21 years)
- Fadgoy (registered May 20, 2010, valid for ~21 years)

Bestminers S.A. holds three mining concessions as part of the Condor Project, namely:

- Chinapintza (registered January 29, 2014, valid for ~17 years)
- Chinapintza Sur (registered January 21, 2025, valid for ~6 years)
- Chinapintza Oeste (registered January 21, 2025, valid for ~6 years)

4.2 Underlying Agreements

Condormining previously held a joint venture agreement with Minera Guangsho Ecuador and JV Chinapintza Mining S.A.(JVC) (signed November 2, 2012). As of June 2025, seven years had elapsed since the initiated liquidation process of Codormining and JVC. According to the Superintendence of Companies (Supercias) the JVC is considered cancelled and its legal life ended. At the time of this report, the JVC is still listed as in the process of liquidation on the Ecuadorian government websites.

A memorandum dated August 2024 outlines the details of the claims against JVC and the liquidation process. SRK has not performed an independent verification of land title and tenure information.

4.3 Permits and Authorization

The Mineral Resources in the Condor North area are located within the three northernmost contiguous concessions shown in Figure 4-2. According to MEM and MAATE, advanced exploration works have been conducted in these concessions since 2013 in compliance with an approved EIS (Ambienconsul, 2006), biennial environmental audits, and regularly updated PMAs.

In March 2025, a new Environmental Impact Statement (EIS) was submitted for review to the Ministry of Environment and Energy through the Single Environmental System (SUIA). The new environmental permit would allow for exploitation beyond exploration, primarily for underground resource definition drill programs. This categorization provides full legal support for initiating processes related to regularization, monitoring, mining control, and environmental management based on the established mining rights.

The Mining Law allows concessionaires to enter pre-negotiation agreements with the Government of Ecuador related to the development of exploitation contracts. Such discussions may commence following a formal request during the Economic Evaluation Period.

Before the construction of the mine and the commencement of mineral production, the Condor Project will be subject to the guidelines and directives required by the current Ecuadorian laws and regulations on mining and environment. Considering previous experience with projects of a similar scale in Ecuador, it is estimated that the main permitting actions will take up to 24 months to complete. These actions are summarized in the following: Change of Mining Phase, Environmental Licensing Process, Water Permits, Safety and Health Planning Actions, Electricity-Related Permits, Fuel and Explosives Permits, among others.

Further information around the current permit and authorization process for the Condor project is found in Section 20:Environmental Studies, Permitting, and Social or Community Impact.

4.4 Environmental Considerations

Concession areas are dominated by naturally mineralized soils with high background metals concentrations that are considered unsuitable for agriculture. The physiography of the area is steep terrain, with abundant rainfall during rain seasons. The streams drain into the larger Nangaritza River, and is surrounded by secondary tropical forest, as part of the highland tropical climate.

Environmental studies for current exploration activities were completed as part of the exploration licensing process. These studies include:

- Meteorological studies
- Biodiversity studies
- Vegetation studies
- Hydrological studies
- Biological studies

Further environmental information is discussed in Section 20.

4.5 Mining Rights in Ecuador

In Ecuador, mining concessions are granted by the Ministry of Energy and Mines (MEM) through a Mining Title. Condormining is the lawful title holder of five mining concessions. The Viche Congüime Cuerpo 1, Viche Congüime Cuerpo 2, Viche Congüime Cuerpo 3, Hitobo concessions under Condormining and FJTX and Fadgoy concessions under Corporación FJTX S.A. were originally granted in 2001. In 2009, the Mining Law was reformed and it provided that existing mining titles shall be substituted with new mining titles in accordance with the new provisions of the Mining Law. Therefore, in 2010, new/substituted mining titles were granted to Condormining and Corporación FJTX S.A. for these six concessions. The concession information is summarized in Table 4-1 to Table 4-3.

Table 4-1: Condor Mining Concessions

Concession Name	Cadastral Code	Surface Area (hectares)	Registration Date[1]	Term of the Concession[2]
Viche Congüime Cuerpo 1	2024	1,078.59	May 20, 2010	21 years, 3 months, 11 days
Viche Congüime Cuerpo 1 Sur	50001609	788.69	January 21, 2025	6 years, 3 months, 29 days
Viche Congüime Cuerpo 2	2024A	2,410	May 21, 2010	25 years counted since February 4, 2021, because it was renewed for an additional 25 years
Viche Congüime Cuerpo 3	500802	2,501	May 20, 2010	22 years, 11 months, 5 days
Hitobo	500115	58.5	May 25, 2010	21 years, 4 months, 17 days

Source: Silvercorp provided Independent Legal Opinion – Flor, Bustamante, Pizarro, Hurtado.
[1] Date in which the Mining Title was registered in the Mining Register.
[2] Term of the concession (counted since the date of registration in the Mining Registry).

Table 4-2: Corporación FJTX S.A. Concessions

Concession Name	Cadastral Code	Surface Area (hectares)	Registration Date[1]	Term of the Concession[2]
Escondida	50000497	1000	17/02/2017	25 years
Santa Elena	50000655	615	17/02/2017	25 years
FJTX	500135	960	25/05/2010	21 years, 4 months, 17 days
Fadgoy	500245	199	20/05/2010	21 years, 3 months, 25 days

Source: Silvercorp provided Independent Legal Opinion – Flor, Bustamante, Pizarro, Hurtado.
[1] Date in which the Mining Title was registered in the Mining Register.
[2] Term of the concession (counted since the date of registration in the Mining Registry).

Table 4-3: Bestminers S.A. Concessions

Concession Name	Cadastral Code	Surface Area (hectares)	Registration Date[1]	Term of the Concession[2]
Chinapintza	2024.1	53.7	29/01/2014	17 years, 7 months, 2 days
Chinapintza Oeste	2024.1	97.5	21/01/2025	6 years, 4 months, 13 days
Chinapintza Sur	2024.1	6.1	21/01/2025	6 years, 4 months, 13 days

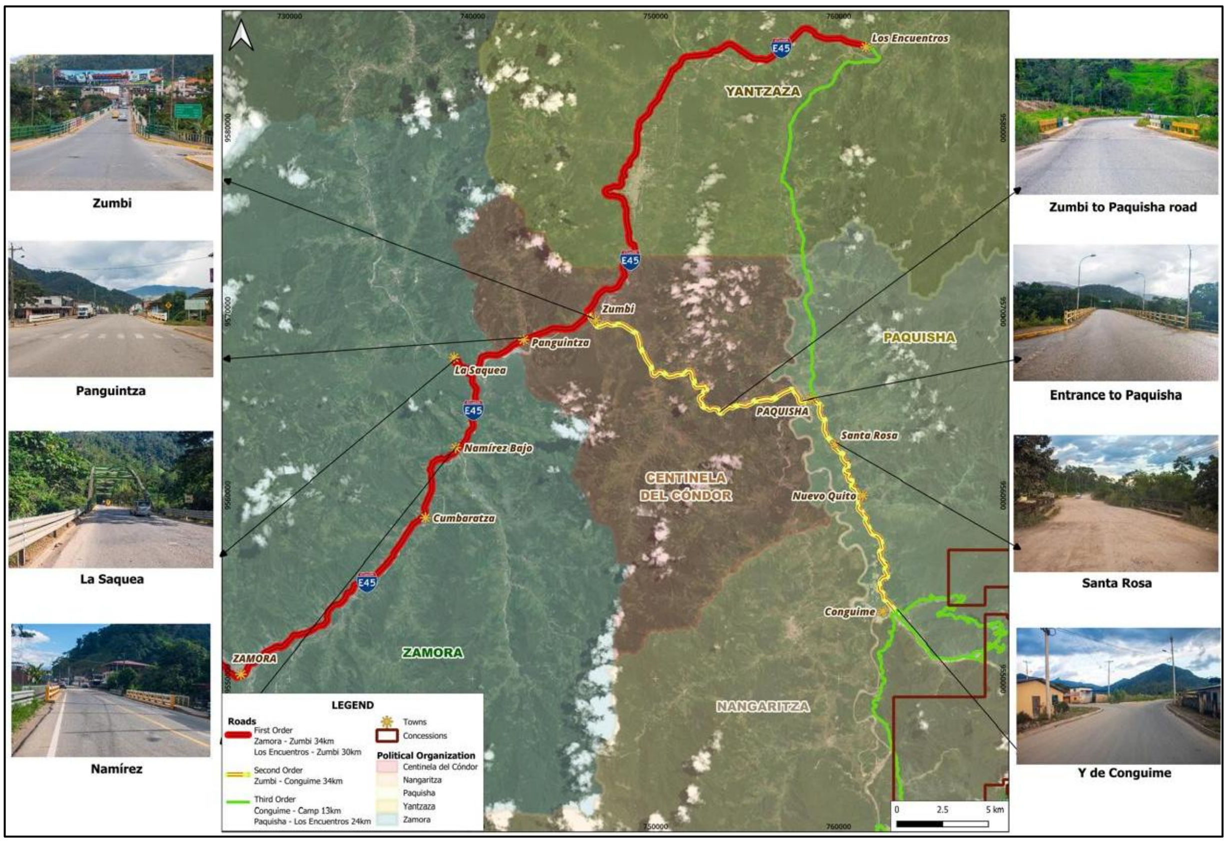
Source: Silvercorp provided Independent Legal Opinion – Chinapintza.
[1] Date in which the Mining Title was registered in the Mining Register.
[2] Term of the concession (counted since the date of registration in the Mining Registry).

5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility

The Condor Project is located along the Ecuador-Peru border in southeast Ecuador, approximately 149 km southeast of the City of Loja and 76 km east of the town of Zamora in the province of Zamora-Chinchipe (Figure 5-1). Access is provided by paved and gravel roads.

Figure 5-1: Access to Condor Project



Source: SRK Site Visit 2025, on behalf of Silvercorp

5.2 Local Resources and Infrastructure

The city of Loja (population ~181,000) is the largest regional centre in the area of the Project and will be a major source of basic goods and services for advanced phases of exploration as well as mine construction and operation. Loja is served by regular daily flights with Quito via Ciudad de Catamayo Airport, located 20 km to the west. Skilled labour can be retained in Loja and Zamora and towns closer to the Project; unskilled labour is typically sourced in the smaller villages nearest to the Project.

The Project is connected to Loja, Zamora (population ~14,000) and other regional centres via the national highway network).

Initial estimates indicate that the national electric grid is capable of providing all necessary power to the Project.

Current infrastructure at the Condor Project consists of a fully equipped 70-man exploration camp, located at 1,456 masl directly above the Camp deposit. The camp consists of dormitories, canteen, medical clinic, administrative offices, warehouse, emergency generator, water treatment plant, septic system, diesel storage tanks and fueling station, a meteorological station, various security installations, and a large core logging and storage facility. Ancillary core storage, warehousing, and waste segregation/accumulation facilities are also located near the camp. The camp is connected to the national grid and has full internet and cellular telephone access.

The Congüime River and numerous smaller streams and springs within the Project concessions can serve as sources of water for all anticipated mining, mineral processing, potable usage, and other Project requirements.

5.3 Climate

The climate in the Project area is highland tropical, with an average daily temperature ranging from 21−24°C, and an average annual rainfall of approximately 2,000 to 3,000 mm. There is a distinct annual rainy season that typically occurs between January and June. A meteorological station has been fully operational at Condor Camp (at 1,456 masl) since January 2021. Relevant historical rainfall data are also available from the National Institute of Meteorology and Hydrology (Instituto Nacional de Meteorología en Hidrología (INAMHI)) stations in Yantzaza and El Pangui; however, neither station is currently operational.

5.4 Physiography

The Condor Project is located in steep, high-relief terrain, near the southern end of the Cordillera del Condor. Elevations range between 960 m and 1,830 m above sea level. The Project drains into the Congüime River, which flows to the Nangaritza River, a main tributary of the Zamora River.

The Condor Project area is surrounded by secondary tropical forest (Figure 5-2), which has been heavily impacted by illegal mining and other intrusive anthropic activities for at least the last 30-40 years. The Condor Project area is subject to frequent landslides and mudflows, due to the steepness of terrain, underlying geology, periodically extreme precipitation events, and the accumulated exacerbating impacts of illegal mining clearances.

Figure 5-2: Typical Landscape in the Project Area

6 History

6.1 Ownership History

The ownership history of the Condor Project commenced with artisanal and small-scale miners operating in the area since pre-1988. In 1988, modern exploration commenced through a joint venture between ISSFA and Prominex UK. This partnership lasted until 1991 when Prominex UK withdrew, and in 1993, TVX Gold, Inc. (TVX) and Chalupas Mining joined the venture. They remained involved until 2000, after which Goldmarca (formerly Hydromet Technologies Ltd.) formed a new joint venture with ISSFA in 2002.

Goldmarca rebranded to Ecometals Ltd. in 2007 and continued operations until the Ecuadorian government imposed a moratorium on mineral exploration from April 2008 to November 2009. In 2010, Ecometals sold its interest to Ecuador Capital, which was later renamed Ecuador Gold and Copper Corp. (EGX). Lumina Gold Corp (Lumina) acquired EGX in 2016, and in 2018, Lumina spun out Luminex Resources Corp. (Luminex), making the Condor Project 90% owned by Condormining, a Luminex subsidiary, with ISSFA retaining a 10% stake. ISSFA has, however, not contributed any funding to the continuing operation of the project, and consequently its share has been diluted to 1.3% as of May 2025. In January 2024, Adventus Mining Corporation (Adventus) merged with Luminex.

In July 2024, Silvercorp acquired Adventus and assumed the ownership of the Condor Project.

6.2 Exploration History

The exploration of the Condor Project area has been extensive and spans several decades.

From 1988 to 1991, ISSFA and Prominex UK conducted regional stream sediment sampling and geological mapping. When TVX Gold, Inc. and Chalupas Mining joined in 1993, they expanded the exploration program to include soil, rock, and stream sampling, trenching, geophysical surveys, and drilling 195 holes totalling 42,101.5 m. They also completed 1,081 m of underground development at the Chinapintza veins.

After TVX and Chalupas withdrew in 2000, Goldmarca / Ecometals took over and continued with reconnaissance mapping, IP and magnetic surveys, and drilling 154 holes totalling 33,322.9 ms from 2002 to 2008.

Exploration was stopped due to a moratorium imposed from April 2008 to November 2009. Resuming in 2012, EGX focused on geological mapping, rock sampling, and diamond drilling 37 holes totalling 22,051.7 m until 2016.

Under Lumina Gold Corp from 2016 to 2018, the project saw additional mapping, sampling, and geophysical surveys, leading to the drilling of nine holes totalling 1,907.4 m.

Since 2018, Luminex Resources Corp. has continued these efforts, conducting a property-wide airborne ZTEM geophysical survey and drilling 28 holes totalling 14,801 m at the Camp deposit.

A compiled list of exploration and ownership history is found in Table 6-1.

Table 6-1: History of Exploration and Ownership

Time Period	Company	Details
pre-1988	Misc.	Artisanal and small-scale miners operated in the area.
1988	ISSFA, Prominex UK	Joint Venture
1988-1991	ISSFA, Prominex UK	Joint venture (1988). Regional stream sediment sampling and geological mapping. Prominex UK withdrew (1991).
1993-2000	ISSFA, TVX Gold Inc, Chalupas Mining	TVX and Chalupas Mining joined the venture. Expanded exploration to include soil, rock, stream sampling, trenching, geophysical surveys. Drilled 195 holes (42,101.5 m). Underground development of 1,081 m at Chinapintza veins. TVX and Chalupas withdrew venture in 2000.
2000-2008	ISSFA, Goldmarca [Ecometals (2007)]	Formed new joint venture (2000). Conducted reconnaissance mapping, IP and magnetic surveys, and drilled 154 holes (33,322.9 m) from 2002-2008. Goldmarca rebranded to Ecometals Ltd. (2007).
April 2008 - November 2009	Ecuador Government	Ecuador government imposed a moratorium on mineral exploration.
2010-2016	ISSFA, Ecuador Capital [Ecuador Gold and Copper Corp. (EGX)]	Ecometals sold interest to Ecuador Capital. Later renamed Ecuador Gold and Copper Corp. (EGX). EGX completed geological mapping, rock samples, and drilled 27 holes (22,051.7 m) until 2016.
2016-2018	ISSFA, Lumina Gold Corp. (Lumina)	Lumina acquired EGX. Luminex spun from Lumina. Condormining - a Luminex subsidiary held 90% of the Condor Project, and ISSFA held 10%. Lumina conducted geological maps, collected samples and geophysical surveys and drilled nine holes (1,907.4 m)
2018-2024	Luminex	Conducted property-wide airborne ZTEM geophysical survey and drilled 28 holes (14,801 m) at the Camp deposit.
2024	Adventus Mining Corporation (Adventus), Luminex	Merged. January 2024. ISSFA did not contribute funding and share was diluted.

Source: SRK, Summary from 2021 Condor Project PEA, published in SRK 2025 Independent Technical Report for the Condor Project, Ecuador

6.3 Geophysics History

Since the early 1980s, extensive geochemical work (Table 6-2) has been conducted at the Condor Project, Stream, soil, and rock surveys have been carried out, identifying well-defined gold-copper soil anomalies at Santa Barbara and a copper-molybdenum soil anomaly at El Hito. Other areas also show anomalous gold and copper values.

As of 2018, previous operators completed 703 trenches totalling 14,650 m, mainly around the Condor breccia pipes.

From 2017 to 2018, Soil surveys were conducted at Santa Barbara, Prometedor, Camp, Wanwintza Bajo, and Wanwintza Alto by Luminex. Detailed results are available in the 2018 Technical Report.

In 2019, Luminex conducted two soil sampling grids in the Camp area, collecting 110 samples.

Since 2018, Luminex has continued property-wide sampling activities, advancing Prometedor and Nayumbi to drill-ready stages.

Table 6-2: Geochemical Surveys of Condor Project

Time Period	Activity	Details
1980s–Present	Geochemical Surveys	Stream, soil, and rock surveys; gold-copper soil anomalies at Santa Barbara; copper-molybdenum soil anomaly at El Hito
Pre-2017	Trenching and Channel Sampling	703 trenches totaling 14,650 m, mainly around the Condor breccia pipes
2017–2018	Soil Surveys by Luminex	Conducted at Santa Barbara, Prometedor, Camp, Wanwintza Bajo, and Wanwintza Alto
2019	Soil Sampling by Luminex	Two soil sampling grids in the Camp area, totaling 110 samples
Since 2018	Ongoing Sampling by Luminex	Property-wide sampling activities; Prometedor and Nayumbi brought to drill-ready stage

Source: SRK, Summary from 2021 Condor Project PEA, published in SRK 2025 Independent Technical Report for the Condor Project, Ecuador

Geophysical surveys (Table 6-3) have played a crucial role in identifying targets within the Condor Project, Magnetic Surveys did not yield significant useful data before 2006.

CSAMT Surveys were Conducted by previous owners, these surveys identified areas of low resistivity correlating with the sulphide-rich Chinapintza veins before 2006.

In 2006, A Pole-Dipole IP Survey with 100 m spacing on northwest-trending lines covered the Condor breccias. High-chargeability values reflecting sulphide mineralization were found only at the Enma breccia deposit. High-chargeability zones near other breccia zones remain untested.

In 2019, A helicopter-supported ZTEM survey by Geotech Ltd. covered 780-line kilometres in the Condor North area. This survey revealed several conductive zones correlating with precious-metal showings, including Prometedor and the Soledad Baja target, aligning with the Camp discovery.

Table 6-3: Geophysical Surveys of Condor Project

Time Period	Survey Type	Details
pre-2006	Magnetic Surveys	Did not yield significant useful data
pre-2006	CSAMT Surveys	Identified areas of low resistivity correlating with sulphide-rich Chinapintza veins
2006	Pole-Dipole IP Survey	Covered the Condor breccias; high-chargeability values at Enma breccia deposit
2019	ZTEM Survey	Helicopter-supported; covered 780-line km in the Condor North area; revealed several conductive zones correlating with precious-metal showings

Source: SRK, Summary from 2021 Condor Project PEA

6.4 Previous Mineral Resource Estimates

The Condor Project has seen several updates to its Mineral Resource estimates over the years, reflecting the evolving understanding of the area's geology and mineral potential:

From 1993 to 2000, TVX Gold, Inc. and Chalupas Mining conducted extensive exploration, including drilling 195 holes totaling 42,101.5 m. This period's exploration provided initial insights into the mineralization but did not culminate in a formal resource estimate.

Between 2002 and 2008, Goldmarca/Ecometals continued drilling, completing 154 holes totaling 33,322.9 m across various gold deposits. Their work helped delineate significant mineralized zones, although specific resource estimates from this period are not detailed in the provided history.

Ecuador Gold and Copper Corp. (EGX) conducted diamond drilling from 2012 to 2016, completing 37 holes totaling 22,051.7 m at several deposits. Their efforts contributed to a better understanding of the mineralization, leading to more refined resource estimates.

In 2015, A Preliminary Economic Assessment (PEA) was completed for the Santa Barbara Project (Short et al., 2015). This PEA included updated Mineral Resource estimates, providing a more comprehensive understanding of the project's economic potential.

Lumina Gold Corp released an updated Mineral Resource estimate in May 2018, covering four deposits: Santa Barbara, Los Cuyes, Soledad, and Enma. This estimate was further detailed in a technical report released on July 10, 2018. This update significantly advanced the project's resource understanding and laid the groundwork for further exploration and development by Lumina and later Luminex.

The Mineral Resources for Santa Barbara, Los Cuyes, Soledad, Enma deposits were restated in a subsequent PEA Technical Report released on July 28, 2021 by Luminex Resources Corp., using updated metal prices and other parameters. Additionally, an underground Mineral Resources estimate for Camp was also released in the PEA. The results of the estimation are tabulated in Table 6-4.

Table 6-4: Previous Condor Project Mineral Resources for Selected Projects Effective 28 July 2021

Deposit	Tonnes (Mt)	Average Grade			Contained Metal		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (Moz)
Indicated							
Los Cuyes	50.8	0.71	0.65	5.2	1,161	1,059	8.5
Soledad	19.4	0.68	0.63	4.8	426	390	3
Enma	0.66	0.78	0.64	11.6	17	14	0.25
All	**70.9**	**0.70**	**0.64**	**5.2**	**1,604**	**1,463**	**11.8**
Inferred							
Los Cuyes	36.4	0.65	0.59	5.3	761	687	6.2
Soledad	15.1	0.5	0.46	3.4	245	225	1.7
Enma	0.07	0.93	0.81	9.7	2	2	0.02
Camp	6	3.45	3.28	27.8	663	631	5.3
All	**57.6**	**0.90**	**0.83**	**7.1**	**1,671**	**1,545**	**13.2**

Sources: MBT 2021

Notes: Mineral resources exhibit reasonable prospects of eventual economic extraction using open pit extraction methods at Los Cuyes, Soledad and Enma and using underground mining methods at the Camp deposit. At Los Cuyes and Soledad, the base case cut-off grade is 0.30 g/t AuEq and at Enma, the base case cut-off grade is 0.37 g/t AuEq. At Los Cuyes, Soledad, and Enma, AuEq = Au g/t + (Ag g/t × 0.012). The base case cutoff grade for the Camp resource is 1.33 g/t AuEq, where AuEq = Au g/t + (Ag g/t x 0.0062). Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The qualified persons responsible for the mineral resource estimates in this technical report have not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves and Silvercorp is not treating these historical estimates as current Mineral Resources. The previous Mineral Resources are being provided herein for reference and to provide comparative information as against the current Mineral Tesource estimates.

The major changes to the Mineral Resource between 2021 and 2025 include:

- The additional data from Camp and Los Cuyes from the 2022 and 2023 exploration.
- The wireframes of Camp and Los Cuyes were updated based on the new data and the interpretation of the mineralization.
- The grade shells of Soledad and Enma were updated.
- The Mineral Resource of Los Cuyes was planned as an open pit operation in 2021 PEA but this has switched to Underground mining in this estimate.

RPEEE assumptions (different commodity prices and recoveries) as well as changes in the reported cut-offs.

6.5 Production

Despite extensive exploration efforts, the Condor Project has not yet achieved commercial mineral production. However, artisanal mining has been a significant activity in the area since the 1980s. Legal and illegal artisanal miners have been extracting gold from the Chinapintza veins, and this activity continues to the present day. Unfortunately, there are no official production records available for this artisanal mining, highlighting the need for more formal and regulated mining operations to fully realize the project's potential.

7 Geological Setting and Mineralization

The contents of this section are mainly sourced from the Condor Project, Ecuador NI 43-101 Technical Report, Condor Project NI 43-101 Technical Report on Preliminary Economic Assessment Report in 2021 (Elfin et al, (2021)) and adapted from the Independent Technical Report for the Condor Project (SRK, (2025)).

The Condor Project is located in the Cordillera del Condor in the Zamora copper-gold metallogenic belt. The Project area comprises epithermal gold-silver, porphyry copper-gold ±molybdenum, skarn gold-copper, and numerous alluvial gold deposits (Morrison, 2007; Williams, 2008).

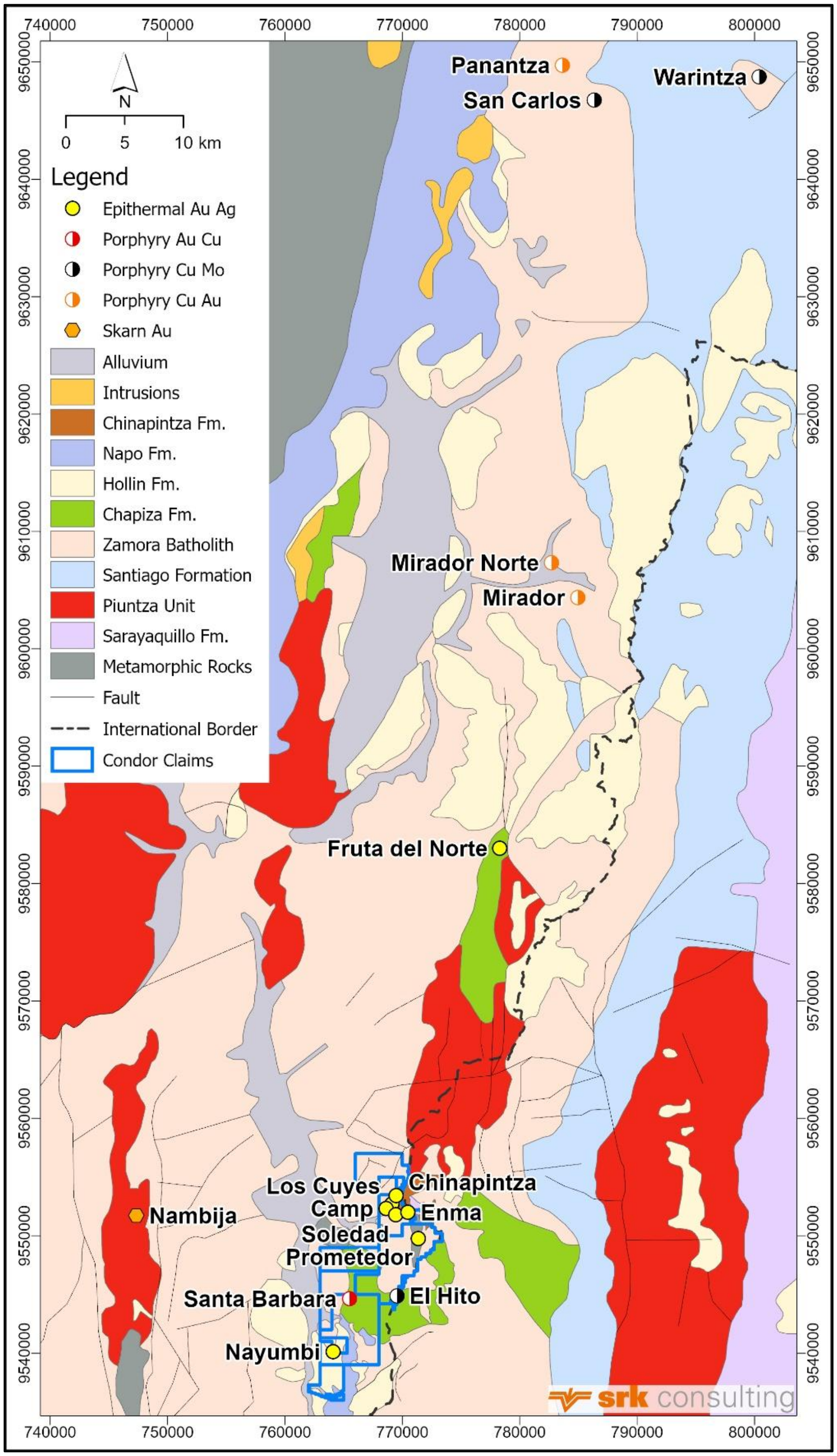
7.1 Regional Geology

The Condor Project is located in the Cordillera del Condor in the Zamora copper-gold metallogenic belt. The Project area comprises epithermal gold-silver, porphyry copper-gold ±molybdenum, and numerous alluvial gold deposits (Morrison, 2007; Williams, 2008). The Fruta del Norte and Mirador Mines, and the San Carlos-Panantza and Warintza deposits are also located within the Zamora copper-gold metallogenic belt (Drobe et al., 2013).

The geologic make-up of the Cordillera del Condor is dominated by the Middle to Late Jurassic Zamora batholith, dated between 153–169 Ma (Litherland et al., 1992; Drobe et al., 2013). Calc-alkaline, I-type batholith lithologies form components of a continent-scale remnant magmatic arc emplaced along an Andean-type continental margin. Batholith magmas intrude supra-crustal sequences of Palaeozoic to Mesozoic sedimentary and arc-related igneous and volcanic rocks. The Zamora batholith is exposed along a 200 km north-northeast trend, is over 100 km wide, and is dissected by predominantly north-south faults forming part of a laterally extensive fold and thrust belt.

The regional geology and key mineral deposits are shown in Figure 7-1.

Figure 7-1: Regional Geology Setting



Sources: SRK, Modified from 2021 Condor Project PEA

Batholith magmas are typically composed of equigranular, medium-grained monzonites and granodiorites along with younger sub-volcanic porphyritic (plagioclase-hornblende ±quartz) intrusions, the latter spanning rare gabbroic to more commonplace andesitic to rhyolitic compositions. Porphyritic intrusions form every 15 km to 20 km along the north-northeast axis of the Zamora batholith and are commonly associated with copper and gold mineralization.

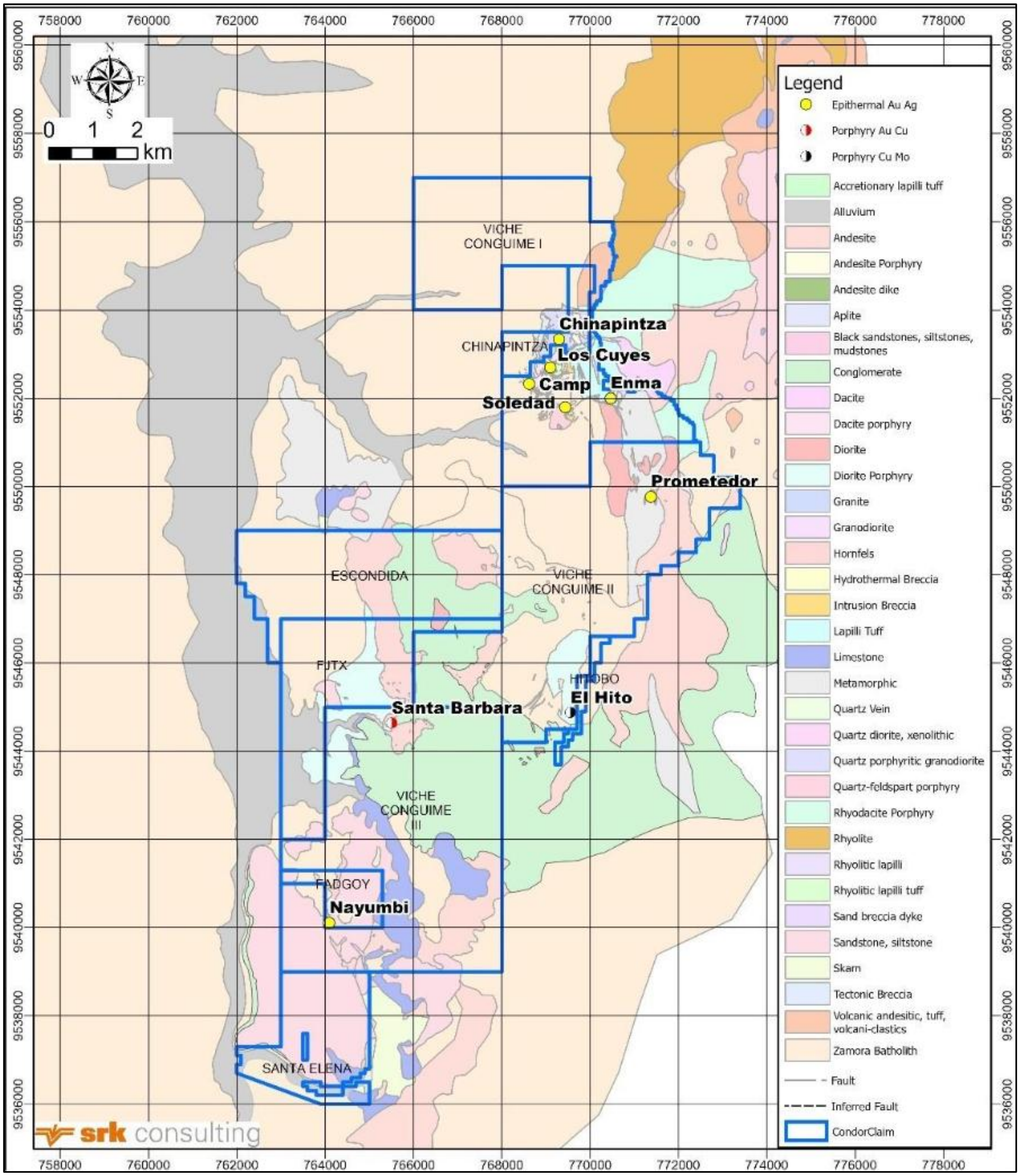
The Zamora batholith intrudes Late Triassic to Early Jurassic Santiago Formation sedimentary and volcanic rocks, locally incorporating them as faulted blocks or roof pendants. Late Jurassic Chapiza Formation sedimentary rocks and Misahuallí volcanic rocks unconformably overlie the batholith. Early Cretaceous quartz arenites of the Hollín Formation as well as sandstones, mudstones and limestones of the Napo Formation further cover portions of the eroded Jurassic volcano-sedimentary sequence and the batholith (Hedenquist, 2007; Drobe et al., 2013). This sequence is locally overlain by rhyolitic to dacitic volcanoclastic rocks of the Early Cretaceous Chinapintza Formation. Late Cretaceous felsic to intermediate stocks and dykes are aligned with regional fault structures.

North-south-trending detachment faults form the principal structural grain, precursors of which controlled the emplacement of the batholith and its subsequent uplift. A series of younger northeast-, northwest- and east-northeast-striking cross structures control the emplacement of younger intrusions.

7.2 Property Geology

The Condor Project encompasses a diverse and geologically complex area with at least three distinctive mineral sub-districts, each characterized by unique mineralization styles and deposits. Only the Condor North area is discussed in this report. The sub-districts highlight the geological diversity and significant exploration potential within the Condor Project, underscoring the presence of various mineral deposits and targets across the concession. A concession-scale geology map of the Condor Project is shown in Figure 7-2.

Figure 7-2: Local Geology Setting



Sources: SRK, Modified from 2021 Condor Project PEA.

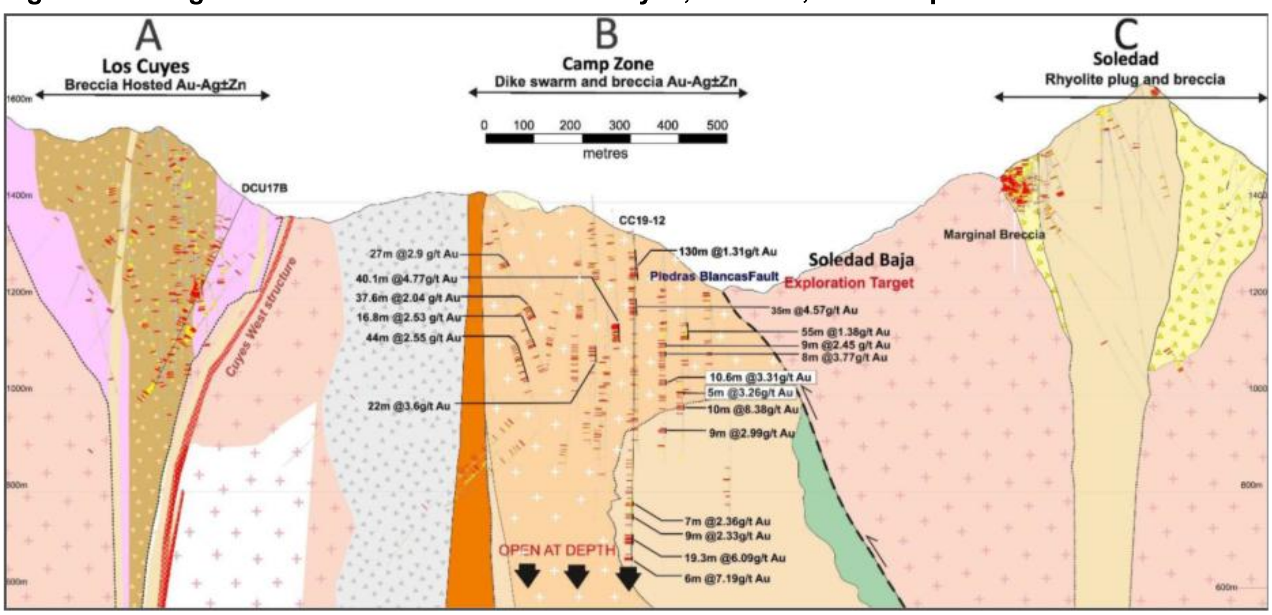
7.2.1 Condor North Area

The Condor Project's geology is both diverse and complex, particularly in the Condor North area. This region is characterized by distinctive low- to intermediate-sulphidation epithermal vein swarms located in the northern part. These vein swarms form a series of north-northwest-striking, narrow, high-grade gold and electrum-bearing manganoan carbonate veins, often accompanied by base metals and hosted in dacite porphyry.

Notably, the Chinapintza vein district extends along strike for 1.5 km over a zone 0.6 km wide, traversing the former Jerusalem concession and continuing into Peru. In the 1990s, TVX conducted more than 45,000 m of drilling followed by underground trial mine development to explore these veins. Although sufficient data for an accurate Mineral Resource evaluation is lacking, artisanal mining continues to exploit these veins.

Immediately south of the Chinapintza vein district lies the Condor breccia, dyke, and dome complex. This complex is hosted by Early Cretaceous rhyodacite to dacite intrusions and volcaniclastics of the Chinapintza Formation, encircled by the Zamora Batholith. Within this area, several diatreme breccias, dykes, plugs, and sub-volcanic domes are associated with these intrusions. Rhyolite dykes, in particular, play a crucial role in localizing vein mineralization. The Condor breccia, dyke, and dome complex is further divided into four main zones: Los Cuyes, Soledad, Enma, and Camp (Figure 7-3). Gold-silver mineralization in these zones is linked with sphalerite-pyrite/marcasite veins, which typically occur within breccias, along the contacts of rhyolite dykes, and as replacements and disseminations. These veins are often disrupted by post-mineral extensional faults.

Figure 7-3: Diagrammatic Cross-section of Los Cuyes, Soledad, and Camp



Source: Hathaway (undated)

7.3 Mineralization

The Condor breccia, dyke and dome complex hosts the Camp, Los Cuyes, Soledad, Enma and the Chinapintza vein deposits and the un-drilled Prometedor prospect (Prometedor lies to the southeast of the area shown in Figure 7-4).

7.3.1 Camp

The Camp deposit features gold and silver mineralization linked to a swarm of northwest-striking rhyolite-dacite dykes, likely originating from a larger buried rhyolite intrusion. These dykes are concentrated at the contact between a volcanic/intrusive complex and a major granodiorite intrusion. The mineralized zone, dipping steeply at 85° to the northeast, extends over 500 m along strike and is 80 to 130 m wide.

Gold occurs within veins containing pyrite, marcasite, iron-rich sphalerite (marmatite), galena, ± chalcopyrite, pyrrhotite, quartz, and rhodochrosite gangue. Host rocks include altered granodiorites, breccias, flow-banded rhyolite, and phreatomagmatic breccia. The area is capped by 30 to 80 m of trachyte to rhyolitic welded tuff, with the Camp ridge bounded by the Camp Fault and Piedras Blancas Fault.

Anomalous surface copper mineralization and stockwork porphyry clasts with molybdenite in the nearby Los Cuyes diatreme suggest a deeper common mineralized porphyry underlying the Condor breccia, dyke, and dome complex.

7.3.2 Los Cuyes

Los Cuyes is hosted within an oval-shaped diatreme measuring 450 m northeast-southwest, 300 m northwest-southeast, and extending to at least 350 m in depth. This diatreme, resembling an inverted cone plunging approximately 50° to the southeast, consists of an outer shell of polymictic phreatomagmatic breccia and an internal fill of well-sorted rhyolitic lapilli tuffs, breccias, and volcanic sandstones. Amphibolite and quartz arenite fragments occur around its periphery, with dacite and rhyolite ring dykes intruding the steep margins.

Alteration within the diatreme is primarily sericite-illite, with localized carbonate and intense phyllic alteration at the margins, indicating focused hydrothermal fluid flow. Gold and silver mineralization occurs in veins containing pyrite, sphalerite, galena, chalcopyrite, and pyrrhotite. The entire diatreme exhibits a low background level of gold, primarily in disseminated pyrite and sphalerite. The highest gold values are found in veins of massive sphalerite, pyrite, and marcasite, with minor quartz, galena, and rhodochrosite, similar to the nearby Chinapintza veins.

Lithological contacts, such as dykes cutting through the diatreme and its outer breccia shell, favoured vein development. The mineralization and alteration at Los Cuyes post-date all local rock types, including blocks of the Hollín Formation, indicating that the mineralization is post-Early Cretaceous.

7.3.3 Soledad

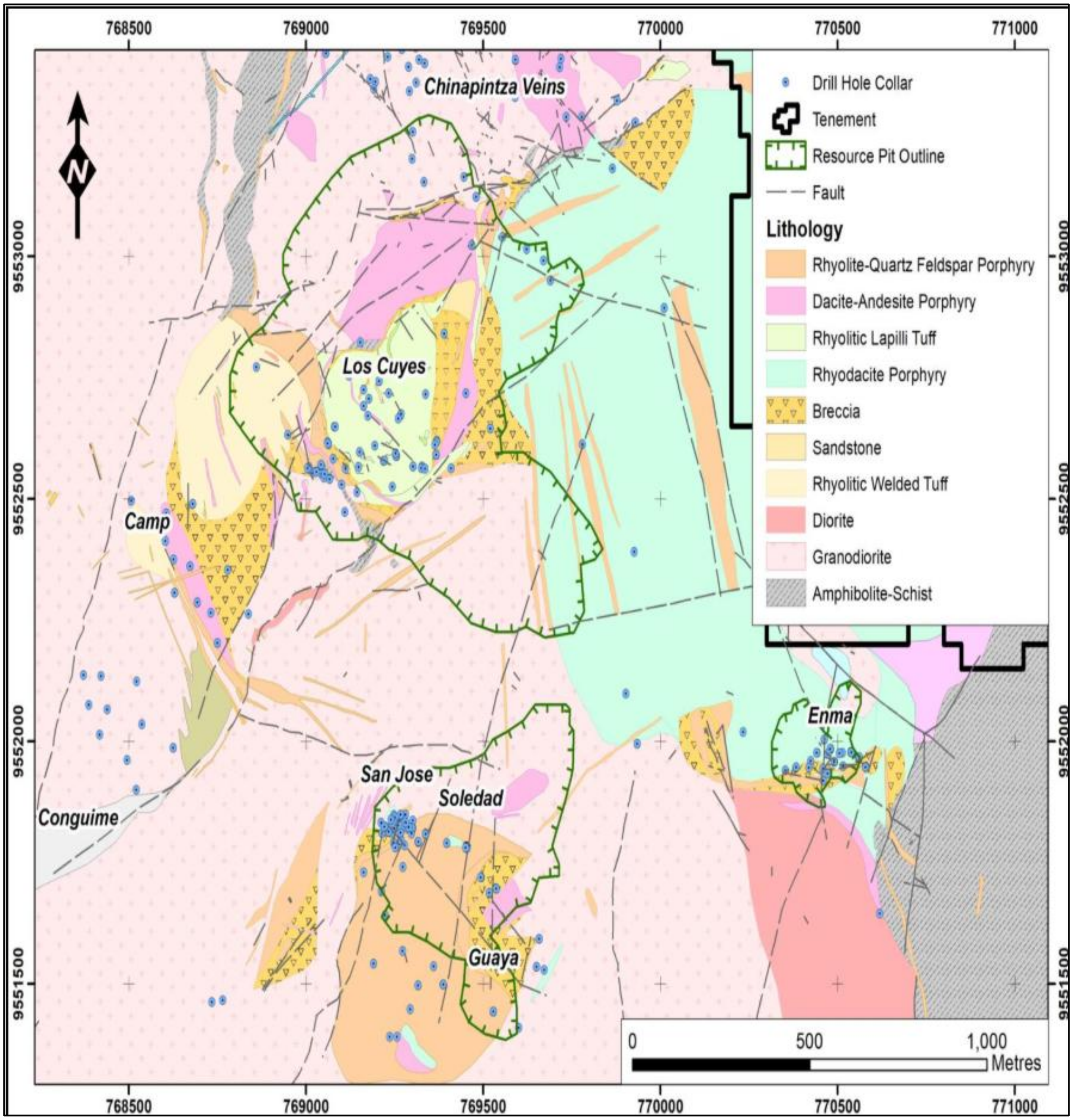
The Soledad Zone features a 700-meter diameter oval-shaped rhyolite intrusion within the Zamora Batholith, surrounded by discontinuous pyritic breccias. It includes individual mineralized zones named Soledad, San Jose, Bonanza, and Guayas. Epithermal gold-silver mineralization at Soledad resembles that of the Camp deposit, with patchy matrix replacement by sulphides, grain-scale replacement of rhyolite feldspars by sphalerite and pyrite, and irregular sphalerite veinlets. Unique to Soledad are the pyritic hydrothermal matrix breccias at the upper margins of the intrusion at San Jose and Guayas.

The overall mineralization at Soledad is described as a north-south elongated wine glass-shaped body, tapering between 200 to 300 m below the surface and extending approximately 110 m northwest by 50 m northeast. Sphalerite transitions to pyrite as the dominant sulfide at around 100 m below the surface, leading to diminished gold and silver grades similar to Los Cuyes.

7.3.4 Enma

Gold and silver mineralization at Enma is hosted in a west-northwest-trending rhyolitic breccia that occurs at the contact between andesite lapilli tuffs and the Zamora batholith. The deposit has dimensions of 280 m east-northeast, is approximately 20-75 m wide, and has a vertical extent of 350 m. Alteration mineralogy is primarily chlorite with minor quartz-sericite ± alunite-kaolinite. Gold is associated with pyrite-sphalerite-quartz and locally rhodochrosite veins. At depths greater than 200 m, gold-poor, pyrite-pyrrhotite ± chalcopyrite veins are more dominant.

Figure 7-4: Condor Volcanogenic Breccia and Dome Complex



Source: 2021 Condor Project PEA

8 Deposit Types

In the Condor North area, gold and silver mineralization is located within the Condor breccia, dyke and dome complex as well as in the adjacent Chinapintza veins. The recently identified Nayumbi prospect located in the Condor South area, is consistent with low to intermediate sulphidation epithermal mineralization (Hedenquist et al., 1996). Notable examples of epithermal gold deposits include Fruta del Norte (Ecuador), McLaughlin (California), Hishikari (Japan), Waihi (New Zealand) and parts of Porgera (Papua New Guinea). The Condor Project is reported to display the characteristics of low to intermediate sulphidation epithermal deposits (as described by Sillitoe, 1993; White and Hedenquist, 1995; Leary et al., 2016).

The Camp, Los Cuyes, Soledad, and Enma prospects are consistent with low to intermediate sulphidation epithermal mineralization. Characteristics of such deposits are:

- Occur at convergent plate settings, typically in calc-alkaline volcanic arcs.
- Form at shallow depths (<2 km) from near-neutral pH, sulphur-poor hydrothermal fluids, often of meteoric origin, with metals derived from underlying porphyry intrusions.
- Structural permeability created by hydrothermal fluid over-pressuring allows for mineralized fluids to permeate, with gold precipitated by boiling.
- Sub-types include sulphide-poor deposits with rhyolites, sulphide-rich deposits with andesites/rhyodacites, and sulphide-poor deposits with alkali rocks.
- Hydrothermal alteration is zoned and subtle, characterized by sericite, illite, smectite, and carbonate.
- Features quartz, quartz-carbonate, and carbonate veins with various textures.
- Sulphide content varies (1-20%), typically <5%, with pyrite, sphalerite, galena, and low copper (chalcopyrite).
- High gold, silver, arsenic, antimony, mercury, zinc, lead, selenium, and low copper, tellurium.

9 Exploration

In 2024, Silvercorp took ownership of the Condor Project through the acquisition of Adventus Mining Corporation. As part of the 2024 Silvercorp relogging program (Figure 9-1), the geology team completed the evaluation of 100 DDH, totalling 46,942 m, including 38 DDH from Camp Zone and 62 DDH from Los Cuyes. The program focused on understanding and confirming the project characteristics including lithology types, structural setup, and mineralization style.

Figure 9-1: Silvercorp Relogging Program (2024)

SILVERCORP METALS INC. srk consulting

Source: Silvercorp, 2024

10 Drilling and Trenching

10.1 Drilling

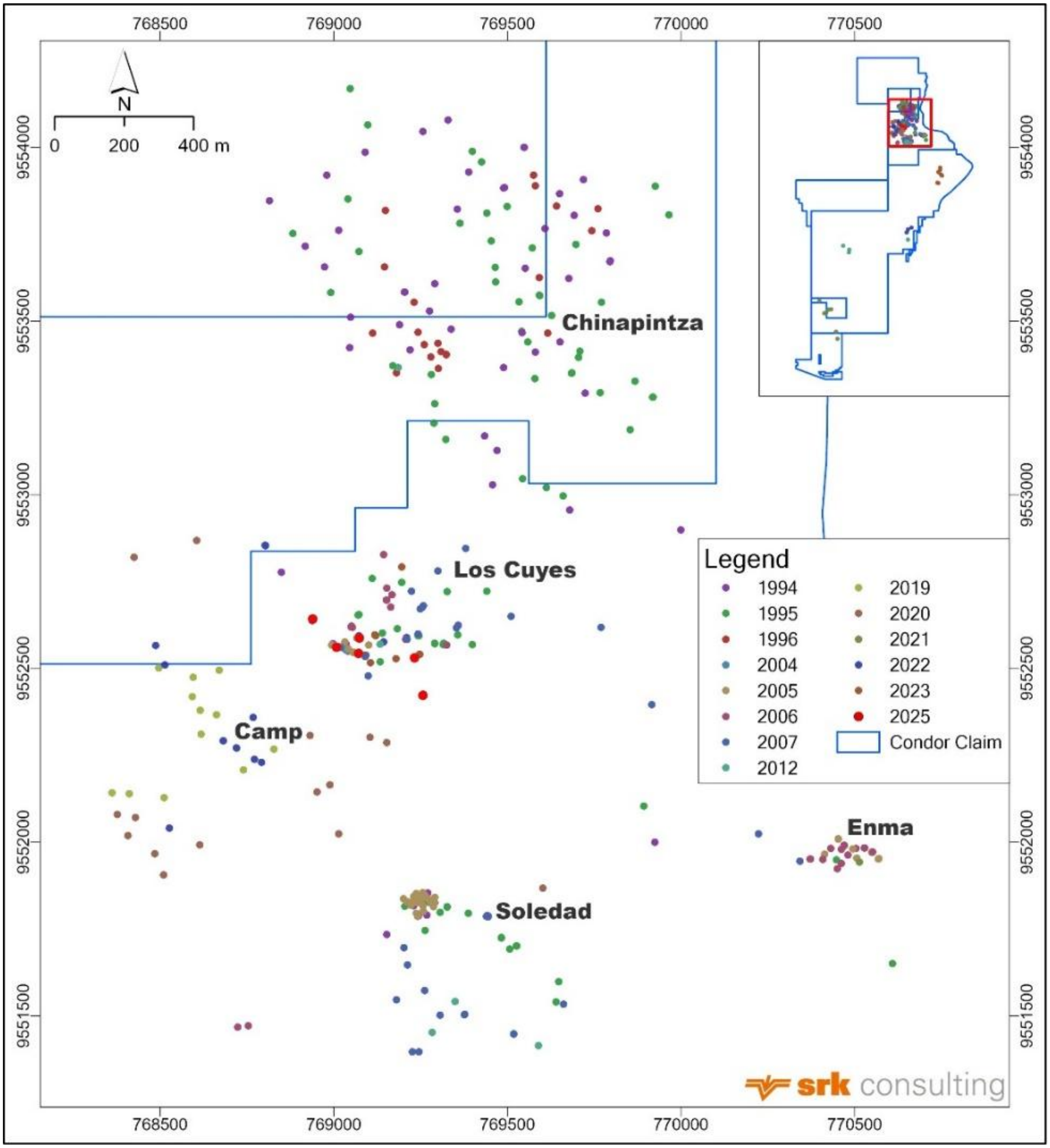
Since 1994, the Condor Project has undergone extensive drilling by various operators. The drilling campaigns of Condor Project from 1994 to 2021, totalling 538 holes with 157,312 m, focused primarily on the Condor North Area and Condor Central Area.

Drilling campaigns from 2022 to September 2023 totalled 21,838 m, mainly distributed in Camp Condor, Los Cuyes, 4 holes in El Hito, and 7 holes in Prometedor.

The Condor property was acquired by Silvercorp in 2024, and a small drilling campaign was completed in July 2025. All six drill holes targetted Los Cuyes and totaled 2,250.37m. These were completed after the estimation of the current MRE and were therefore not reviewed or included in the current MRE.

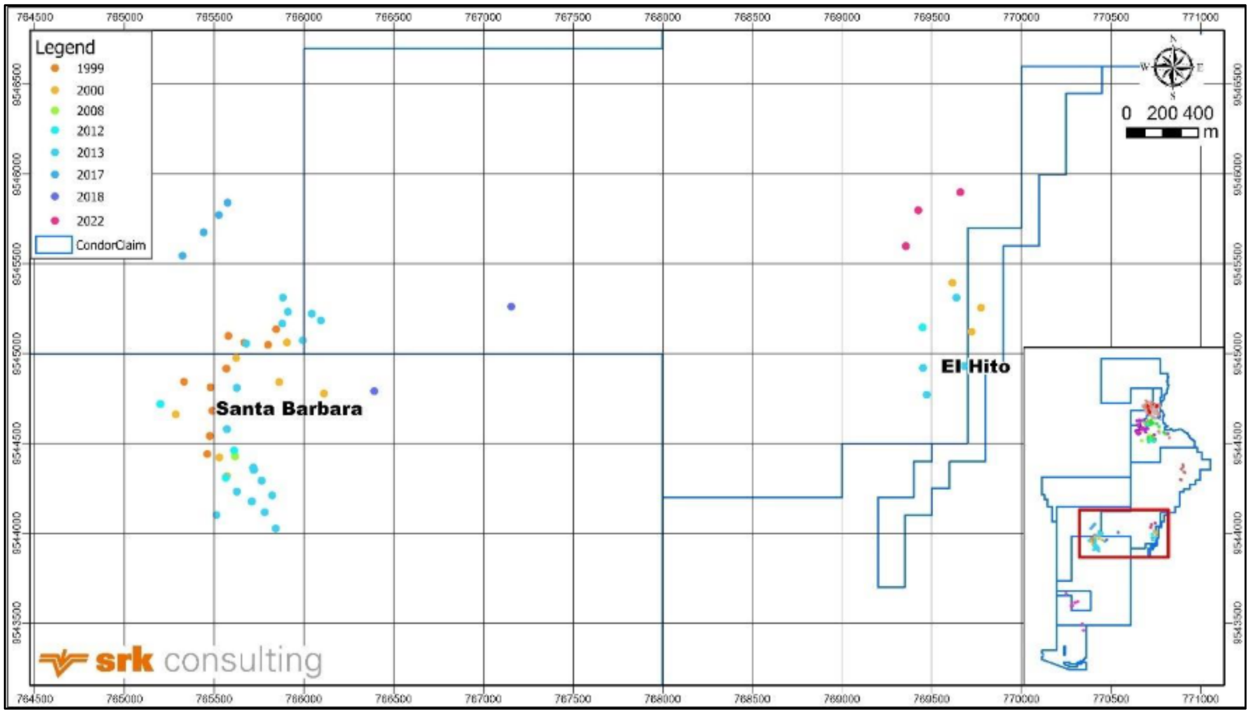
Figure 10-1 and Figure 10-2 display the locations of the drillholes in North and Central Area of Condor Project respectively.

Figure 10-1: Map Showing the Distribution of Condor North Area Drilling (1994 – 2025)



Source: SRK

Figure 10-2: Map Showing the Distribution of Condor Central Area Drilling (1994 – 2025)



Source: SRK

10.1.1 Historical Drilling (Pre-2019)

Condor Project has experienced extensive drilling by various operators during 1994 to 2018. The drilling programs summary is presented in Table 10-1.

TVX Gold, Inc. initiated drilling between 1994 and 2000, testing the Chinapintza veins (75 holes; 20,489 m), Condor breccias (97 holes; 16,128 m), Santa Barbara (19 holes; 4,296 m), and El Hito (4 holes; 1,188 m). It used worker-portable drills that produced HQ- or NQ-size core. Downhole surveys were completed, but the specific method is unknown, except at Santa Barbara where a Pajari instrument was used. Most of the collars are marked with a concrete pad.

From 2004 to 2007, Goldmarca drilled the Condor breccia pipes (124 holes; 21,612 m), followed by Ecometals in 2008, focusing on the Condor breccias (29 holes; 11,111 m) and Santa Barbara (1 hole; 600 m). All holes were drilled using HQ-size core, reducing to NQ as needed. Holes were located using a handheld Garmin GPS instrument. Downhole surveys were completed for 33 of the drill holes using a FLEXIT instrument which takes readings at 3 m or 6 m intervals. Core recoveries for holes drilled by Goldmarca and Ecometals were generally >90% (Hughes, 2008).

Between 2012 and 2014, Ecuador Gold and Copper Corp. (EGX) conducted further drilling on the Chinapintza veins (1 hole; 757 m), Los Cuyes and Soledad breccias (4 holes; 2,574 m), Santa Barbara (27 holes; 15,223 m), and El Hito (5 holes; 3,498 m). Two contractors were used for this drilling: Roman Drilling Corp. S.A. and Hubbard Perforaciones Cia., Ltda. (Hubbard); both are based in Cuenca, Ecuador.

All holes were drilled using HTW-size (HQ) core, reducing to NTW (NQ) as needed. The Hubbard drills were worker-portable and similar to Hydracore 4,000 rigs. Holes were located using a handheld Garmin GPS. When a hole was completed, the hole location was marked with a cement monument displaying the hole number, azimuth and dip. A Reflex EZ-SHOT™ was used to provide downhole orientation data at 50 m intervals. Core recoveries during this period of drilling average approximately 93%.

From 2017 to 2018, Lumina used Hubbard Perforación Cia. Ltda.to complete nine HTW (HQ) drill holes (1,907 m) in the Santa Barbara area. Three targets peripheral to the main Santa Barbara mineralization were tested: Santa Barbara northwest, northeast, and southeast. A Hydracore 2000 drill was used, and the drill was moved using a small tractor. Drill holes were located using a handheld Garmin GPS. A Reflex EZ-SHOT™ was used to provide downhole orientation data at 50 m intervals. Core recoveries in holes drilled by Lumina average just over 91%.

During Lumina's 2017 to 2018 drill program, drillers initially placed the HQ drill core in plastic boxes (four rows; total of approximately 2.5 m per box). Wooden tags, marked with the downhole depth, were placed in the box. Lids were placed on the box and taped shut. The core was then transported to the nearest road and trucked to Lumina's core facility at the Luminex exploration camp. Once unloaded on core inspection racks, Lumina field assistants checked the depth and core recovery and recorded the "FROM and TO" intervals on the outside of the boxes. The core was washed, and wet and dry photos were taken of the whole core. Lumina geologists examined the whole core first and prepared geotechnical and geological logs. The geotechnical log recorded RQD, core recovery, fracture and vein quantity, and vein angles.

Table 10-1: Drilling Programs of Condor Project (Pre-2019)

Year	Company/Entity	Core Boreholes	Total Metres Drilled	Focus Area
1994-2000	TVX Gold, Inc.	75	20,489	Chinapintza veins
		97	16,128	Condor breccias
		19	4,296	Santa Barbara
		4	1,188	El Hito
2004-2007	Goldmarca	124	21,612	Condor breccia pipes
2008	Ecometals	29	11,111	Condor breccias
		1	600	Santa Barbara
2012-2013	Ecuador Gold and Copper Corp.	1	757	Chinapintza veins
		4	2,574	Los Cuyes and Soledad breccias
		27	15,223	Santa Barbara
		5	3,498	El Hito
2017-2018	Lumina Gold Corp	9	1,907	Geochemical and IP anomalies around Santa Barbara

10.1.2 Luminex Resources (2019-2023)

From 2019 to 2020, Luminex Resources Corp. has completed 46 holes (23,683 m) focusing on geochemical anomalies and delineation drilling at Camp and Soledad deposits, and additional holes to recover metallurgical material from Cuyes and Enma. Drilling was completed by two contractors, Kluane Drilling Ecuador S.A. and Rumi Drilling Services Ecuador (RDSEC) S.A. Each used a Hydra core 2000.

All holes were collared with HQ-size (or HTW) core and reduced to NQ (or NTW) when needed. Access trails to drill pads were constructed by hand as well as using a small excavator. Rig movements were facilitated by a Bobcat and, where possible, a larger Morooka all-terrain vehicle was used.

All holes were drilled as oriented core via Reflex ACT II or III equipment with downhole surveys completed by either DeviShot TM or Reflex EZ-TRACTM XTF tools. Data from downhole surveys were collected at 30 m to 50 m intervals. Collars were initially spotted via handheld Garmin GPS and later surveyed using a total-station theodolite (Sokkia model 105) to a 5-mm accuracy.

Core recoveries average 98% for drilling conducted by Luminex.

In 2021, Luminex completed one short hole (100 m) for metallurgical samples at the Enma deposit. Drilling was completed by Rumi Drilling and under the same protocols as prevailed during the 2020 program. Drilling prior to December 31, 2021 were part of the previous Mineral Resource estimate.

Drilling done between 2022-2023 was provided to SRK to generate the updated Mineral Resource estimate, effective date of September 8, 2023. A total of 55 holes with 21, 838 m is summarized in Table 10-2.

Table 10-2: Drilling hole Summary of Condor Project (2022-2023)

Year	Area	Core Boreholes	Total Meters Drilled
2022	Camp Condor	13	4,695
	El Hito	4	2,418
	Los Cuyes	15	5,660
Sub Total		**32**	**12,773**
2023	Los Cuyes	16	7,990
	Prometedor	7	1,075
Sub Total		**23**	**9,064**
Grand Total		**55**	**21,838**

Sources: SRK, Summary from the drillhole database: CN_DH_Export_Database_8Sept2023.xlsx

Luminex Drilling Procedures

The exploration drilling procedure involves meticulous planning and execution to ensure accuracy and minimal environmental impact. Initially, diamond drilling using HQ and NQ diameter rods is the primary method, continuously monitored by Exploration Managers or their designees, with reverse-circulation (RC) drilling used occasionally as outlined in Lumina’s "Guidelines for Drilling and Trenching Contractors." Drilling contractors are responsible for mobilizing all necessary equipment to the site, controlling water usage and drilling mud, managing borehole progress, transporting core boxes, providing required pipes and consumables, and preventing spills of fuel and lubricants. They must also collect and transport all garbage or waste generated during the drilling process.

Contractors must construct drilling pads at specified borehole locations, taking care to separate and preserve topsoil for later reclamation. Geologists mark the positions in the field and assist drillers with marking azimuth and dip of the planned hole. Surveyors accurately measure collar locations with elevation using Total Station or GPS equipment with centimetre-scale accuracy. Drillers complete hole deviation surveys during drilling at systematic intervals using down-hole survey equipment.

Post-drilling, contractors are responsible for reclaiming drill pads by re-grading them to original contours to blend with the surrounding ground surface. The disturbed sites are covered with reserved topsoil and revegetated with native species. Drilling mud pits are backfilled, covered with reserved topsoil, and revegetated; all geosynthetic pit liner material and any other debris from drilling operations must be properly disposed of. Any residual water in the mud pits is tested for pH and adjusted with lime to pH 5-7 before release into the environment.

Drill collars are reclaimed by pouring an approximately 0.5 m^2 concrete monument around the casing. The monument is inscribed with the hole number and date of the borehole. The casing stub is cut off about 0.5 m to 0.75 m above the monument surface, fitted with a PVC slip cap, and marked with reflective tape. These detailed procedures ensure precise data collection and uphold environmental stewardship throughout the exploration process.

Core handling and sample preparation protocols used by Luminex for the 2019–2021 drill program mirrored those of Lumina with a few modifications. Drillers initially extracted the core from the drill onto a 4 m long angle iron installed at waist height at the rig site and orientated the last core run segment using a digital Reflex ACT II core orientation device. The orientation line was scribed on the re-assembled core before it was placed in slotted plastic core boxes, each having four rows for a total of approximately 2.5 m per box. Annotated plastic core tags, marked with the downhole depth, were placed inside the box. A 25 mm thick foam liner was then placed inside the boxes to prevent core segments from moving, and plastic lids were placed on each box and strapped shut.

The core was then transported to the nearest road and trucked to Luminex's core handling facility at the Luminex exploration camp. Once unloaded on core inspection racks, Luminex field assistants checked the depth and core recovery and recorded the "FROM and TO" intervals on the outside of the box. The core was washed, and photos were taken of whole core in dry and wet conditions under a table-mounted camera using consistent artificial light. Luminex geologists examined the whole core first and prepared geotechnical and geological logs. The geotechnical log recorded RQD, core recovery, fracture and vein quantity. Core was re-assembled on an angle iron in order to recheck the orientation lines. If deemed satisfactory, geologists measured the alpha and beta angles of all veins, faults, contacts, foliations and flow banding. Point-load and specific gravity measurements using paraffin-coating were taken at 10 m intervals. Whole core was measured for magnetic susceptibility at every assay sample.

10.1.3 Silvercorp Metals Inc. (2024-Present)

After Silvercorp acquired the Condor project in 2024, a small drill campaign was completed from May to early July 2025 that consisted of six drillholes with a total of 2,250.37 m, summarized in Table 10-3. The targeted program was to test the geologic interpretation of Los Cuyes West vein structure and the wide disseminated mineralization within the volcanic tuff host. The program concluded that the current geologic interpretation is well understood.

Table 10-3: Silvercorp Drilling Summary at the Condor Project (2024-2025)

Year	Core Boreholes	Total Meters Drilled	Area
2025	6	2,250	Los Cuyes
Grand Total	**6**	**2,250**	

Silvercorp Drilling Procedures

Pre-drilling preparation begins with pegging the planned drill collar in the field. Geologists use handheld GPS units to locate the position of the planned drill hole collar. Once located, a half-meter wooden stake is placed to mark the collar, and the hole ID and all relevant specifications are written on the stake. A front-sight stake is then positioned approximately five meters ahead of the collar in the drill azimuth direction, while a back-sight stake is placed five meters behind the collar, opposite to the drill azimuth.

The drill pad is prepared either manually or using machinery such as a backhoe or bulldozer. The prepared pad typically measures approximately 8 m by 10 m and must be aligned with the drill azimuth direction. Once the pad is ready, geologists repeat the pegging procedure, including GPS location of the collar, installation of a wooden collar stake with hole information, and placement of the front- and back-sight stakes at the required distances along the azimuth and reverse azimuth directions.

Rig alignment follows the preparation of the pad. The drill rig is moved onto the pad and aligned with the planned azimuth using a compass or rig-aligner. Geologists adjust the rig so that the drill rod is oriented within half a degree of the planned azimuth and dip angle. A second geologist double-checks the rig alignment, including coordinates, azimuth, and dip, before drilling begins.

Coring is conducted by contractor drillers under the supervision of New Pacific Metals Corp. technical staff. All cores must be carefully collected in correct order and orientation and placed in core boxes at the end of each drill run. Core placement begins at the far-left corner of the box, proceeding left to right in each row from the upper part of the hole downward. Every piece of core is numbered with the drill-run number and piece sequence using a permanent marker. When long core segments must be broken to fit into the box, drillers make a small red "X" (the driller's mark) at the break point. Start and end direction arrows are applied to each box, and hole ID and box number are written on the exterior of each core box.

A core block indicating hole ID and depth is placed at the end of each run. No markings other than orientation marks, driller's break marks, and run-piece numbers may be added. All cores must be clean and free from drilling fluids, oil, grease, and other contaminants. If core loss is suspected, drillers must insert a wooden block at the interval and mark it "LC" along with the estimated lost length; drillers may not fill lost-core zones with cuttings. Core boxes are numbered sequentially from the start of each hole. To avoid damaging the core, drillers may not use metal tools to remove core from twin-tube inner tubes; rubber mallets may only be used when all other non-impact methods have failed.

Once full, core boxes are covered and transported to the Company's core processing facilities by Company staff or contracted trucks under strict supervision. No more than ten core boxes (40 m of core) may remain on site, and cores may never be left unattended. Before transport, cores must be stored safely near the drill site under constant supervision by drillers or Company personnel.

After drilling is completed, the drill collar is cemented for protection. A plastic or steel pipe, one to two meters long, is inserted into the hole at the original azimuth and dip before cementing. Drill information is inscribed in the cement for identification.

Quick geological logging is performed at the drill site prior to transporting cores to camp. Geologists identify major rock types and intervals of mineralization, alteration, and structural features. Significant portable XRF readings are marked directly on the core. A photograph of each full core box is taken at a resolution greater than 3 megapixels and stored on the Company's data server. Preliminary interpretation of structures and mineralized zones is completed upon return to the camp.

Core transportation occurs daily under Company supervision. All full core boxes are moved from the rig site to the core processing facilities by end of dayshift. Tissue paper is placed between broken pieces of mineralized core to prevent loss of mineralized material, and in some cases broken pieces are sealed in plastic bags and returned to the core box. Core boxes are carefully loaded onto transport trucks, tightly secured to prevent shifting during transport, and monitored throughout the journey by Company staff. Upon arrival at camp, a core-handover form documenting hole number, number of boxes, and box sequence is completed and signed by both the driver and the receiving geologist. Any damage noted during handover is immediately addressed.

Cores are washed, reassembled, and their recovered length measured for each run. If oriented, the bottom-side line is marked. Depth marks are added at one-meter intervals using a permanent marker. Geologists or geological assistants record core recovery and RQD following Company procedures, entering the data into standardized templates.

Geologists then perform detailed logging, describing lithology, alteration, mineralization, and structure. Data are entered into standardized logging templates using Company coding systems, either on paper or directly into MX Deposit software.

Geologists define sampling intervals, normally 1.0 m to 1.5 m depending on geological boundaries such as lithology, alteration, mineralization, and structures. Sample intervals and cut lines are marked on the core using permanent markers, and sample IDs are recorded in Company sampling tables.

Before cutting, photographs of both dry and wet core are taken. Additional close-up photos with scale may be taken at the geologist's discretion. All images exceed 3 megapixels and are archived on the Company's cloud database (Dropbox) and local server.

10.2 Drilling Pattern and Density

The rugged terrain over the project area makes the siting of drill holes more challenging than projects with a flat topography. As a result, many of the holes are drilled in a fan pattern from a single collar location. As many as 20 holes are drilled from some collar locations and with the exception of Camp, there is not a typical drilling orientation. At Camp the holes are drilled as fans in fences spaced between 50 and 80 m apart on azimuths averaging either approximately 30° or approximately 210°. In the fans at Camp the vertical spacing varies from very short near the collars to approximately 100m near the ends of the holes.

At Enma and Los Cuyes the hole spacing is variable due to the multiple orientations and dips of the holes drilled in the fans, but the collar locations are spaced between 25 and 50 m apart over the core of the modeled deposits. The hole spacing at the toes of the drill holes are of the order of 100 m to 150 m, but in the intersected veins at Los Cuyes the intersections spacing is variable.

At Soledad there are three collar locations with large numbers of holes drilled from them spaced 150 to 250 m apart, and with a range of collar locations with one or two holes spaced between 50 and 150 m apart. The northern part of Soledad is drilled using a very dense grid of holes with collars between 10 and 50 m apart.

10.3 SRK Comments

In the authors' opinion, the current core handling, logging, sampling and core storage protocols on the Condor Project are consistent with common industry standards, and the authors are not aware of any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of these results.

All database records should be assigned a consistent year and area.

The authors of this report recommend that Silvercorp take additional bulk density measurements on samples for Los Cuyes, Soledad, and Enma to improve the confidence in the estimation of the bulk density in these deposits.

11 Sample Preparation, Analyses, and Security

11.1 Sample Preparation and Analyses

TVX used Bondar Clegg (now ALS Chemex) which has ISO/IEC 17025:2017 accreditation and SGS Canada Inc. which has ISO/IEC 17025 and ISO 9000 accreditation. EGX and Goldmarca/Ecometals used Acme Labs in Santiago, Chile which then had ISO 9001:2000 accreditation at the time the work was done. Lumina analyzed its samples using ALS Analytical Laboratories in Lima, Peru which has ISO/IEC 17025:2017 accreditation. Luminex used MSALABS in Vancouver, Canada which had ISO/IEC 17025:2005 accreditation.

ALS, MSALABS, Acme Labs, Bondar Clegg and SGS are commercial geochemical laboratories independent of TVX, Goldmarca, Ecometals, Lumina, Luminex and Silvercorp.

11.1.1 TVX Gold Inc. (1994-2000)

There is no detailed description of TVX's sampling procedures or security measures for its drill programs on the Condor North and Central areas. From 1994 to 2000, drill core was cut in half using a diamond saw, with one half sent for analysis and the other half stored securely in core boxes at the project site.

The first eight holes on the Chinapintza veins were continuously sampled at 1.0-m intervals, but, in subsequent holes, only potentially mineralized core was sampled. These samples had variable lengths, sometimes less than 10 cm. At the Enma, Los Cuyes, San Jose and Soledad Breccias, the entire hole was sampled with sample intervals ranging from 1.0 m to 2.5 m. Core was cut in half using a diamond saw. One half was sent for analysis, and the other half was returned to the core box.

TVX sent its samples to Bondar Clegg or SGS in Ecuador for sample preparation. A sample of 100 g of pulverized material was sent for analysis to the SGS laboratories in Canada. From 1994 until 1996, SGS used a 30 g sample to analyze for gold using a fire assay with an atomic absorption finish. In February 1996, the sample size was increased to 50 g. In 1999, TVX used ALS Chemex to analyze the drill samples from Santa Barbara. Gold was analyzed by fire assaying a 30 g sample. Copper and 33 other elements were analyzed using ICP (Easdon and Oviedo, 2004).

11.1.2 Goldmarca Ltd. (2004-2007) and Ecometals Ltd. (2007-2008)

During the Goldmarca/Ecometals drill programs, the entire hole was sampled at 2 m intervals using a diamond saw. Half the core was put into a marked sample bag which was sealed with tape and put into a rice bag. The other half of the core was returned to the core box and stored in the warehouse facility.

Samples were taken by truck to Loja and then shipped to the ALS Chemex preparation lab in Quito or Acme's preparation lab in Cuenca, Ecuador. When broken sample bags arrived at the lab, the sample was taken out of the process stream, Goldmarca was notified, and the sample was retaken.

The Acme samples were shipped to Vancouver, Canada for analysis. Gold and silver were analyzed by fire assay with an ICP finish on a 30 g sample. Zinc, copper and lead were analyzed using atomic absorption.

11.1.3 Ecuador Gold and Copper Corp. (2012-2014)

During the EGX drill programs, the drillers put core into core boxes, and intervals were marked with wooden blocks and permanent markers. The boxes stored in EGX's secure core-logging facility located at its Luminex exploration camp.

At the core facility, the core boxes were marked with intervals and hole numbers. Core was cleaned and then photographed in two box sets, and then it was examined by EGX geologists and technicians who prepared geotechnical (RQD, core recovery, hardness, fracture density) and geological logs. Specific gravity measurements were taken every 10 m to 15 m.

Sample intervals were determined by the geologist. The core was sampled at regular 1.0 m, 2.0 m or 2.5 m intervals. The core was cut in half using a diamond saw. Half of the core was put in a labelled plastic sample bag along with a numbered sample tag, and the bag was secured with a tamper-proof zip tie. The other half was returned to the core box and stored in a secure warehouse adjacent to the logging facility. Individual samples were packaged into large containers or sealed poly woven bags and transported by EGX employees or a bonded courier to Acme Lab's sample preparation facility in Cuenca, Ecuador.

At the preparation lab, each sample was crushed so that >80% passed through a 10-mesh screen. A 250 g split was pulverized so that >85% passes a 200-mesh screen. This was then shipped to the Acme Lab in Santiago, Chile for analysis. All samples were analyzed for gold using a fire assay technique with an AA finish on a 30 g sample. Any sample with >10 g/t Au was re-assayed using a gravimetric method. Samples were analyzed for silver and copper by ICP-ES after a four-acid digestion.

11.1.4 Lumina Gold Corp. (2017-2018)

Core was cut at the core cutting facility in the Luminex exploration camp using a diamond saw at 2 m intervals. For each sample, half the core was put into a plastic bag with a bar-coded sample ticket and then secured with a tamper-proof plastic zip-tie. A duplicate sample tag was stapled into the core box. The other half of the core was returned to the core box and stored on site. Certified reference standards purchased from CDN were inserted into the sample stream after every six core samples. These included three certified standards (high, medium and low gold grades), a blank and a coarse and fine duplicate. Sample bags were then packed into larger mesh sacks which were also tied with a numbered, tamper-proof plastic zip-tie.

Drill core samples from the 2017–2018 drill program were assayed by MSALABS in Vancouver, Canada. Sample shipments were picked up from the Luminex exploration camp by representatives of Lac y Asociados Cia. Ltda. (MSALAB's preparation lab in Cuenca, Ecuador) and delivered directly to the lab in Cuenca. The secure tamper-proof plastic tags were checked against a list that had been e-mailed to the prep labs upon arrival of the samples. (Note: No irregularities were detected in any sample shipments.) The samples were then digitally registered, dried, crushed and pulverized.

For each sample, approximately 250 g of pulverized material was separated by riffle splitter, placed in a paper craft bag and shipped to MSALABS in Vancouver for analysis. All samples were analyzed for gold using a fire assay technique on a 30 g charge and a 34-element ICP-MS analysis was completed using a four-acid digestion.

Remaining reject and pulp material from the drill programs have been returned to Lumina and stored at its secure warehouse in Quito, Ecuador.

During Lumina Gold Corp.'s 2017-2018 drilling program of Condor Project, samples were sent to MSALABS in Vancouver, Canada, for analysis. MSALABS conducted gold assays using fire assay techniques and ICP-MS analysis with four-acid digestion.

11.1.5 Luminex Resources (2019-2023)

Core was cut at the core-cutting facility in the Luminex exploration camp. The sampling intervals were proportional to the geology and mineralization over 1 m to 2 m intervals. Sample lengths varied with respect to geological boundaries and veins. Certain intervals devoid of visible mineralization, as quantified from previous assays, were subsequently sampled as 2 m composites every 10 m or 20 m. In rare cases, no samples were taken from visually unaltered and unmineralized sections. For each sample, half the core was put into a plastic bag with a bar coded sample ticket and then tied with a zip-tie. A duplicate sample tag was stapled into the core box. The other half of the core was returned to the core box and stored on site. Sample bags were sealed and secured with tamper-proof zip-ties and then packed into larger mesh sacks which tied with a numbered, tamper-proof nylon tie.

Sample shipments were picked up from the Luminex exploration camp by representatives of ALS Laboratories and delivered to their preparation lab in Quito. The secure tamper-proof plastic tags were checked against a list e-mailed to the prep labs upon arrival of the samples along with other chain of custody paperwork. The samples were then digitally registered, dried, crushed and pulverized. For each sample, approximately 250 g of pulverized material was separated by riffle splitter, placed in a paper craft bag and shipped to ALS Laboratories in Lima for analysis. All samples were analyzed for gold using a fire assay technique on a 50 g charge, and a 34-element ICP-MS analysis was completed using a four-acid digestion.

Remaining reject and pulp material from the prep lab was returned to Luminex and stored at its secure warehouse in Quito.

11.1.6 Silvercorp Metals Inc. (2024-Present)

Logged drillcore are transported to the cutting area where equipment is inspected prior to use. Cutting technicians must wear full PPE including ear protection, masks, waterproof gloves, and protective coats. Cores are cut in half along pre-marked cut lines. One half is bagged, tagged, and sealed for shipment to ALS Quito, or other accredited laboratories, while the remaining half is stored in core boxes for reference. Samples are tracked using three-part tickets: one stapled into the core box, one accompanying the sample, and one retained by the geologist. Sample bags are stored in a locked room until shipment.

Individual sample bags are grouped into poly-weave bags, typically holding ten samples each. These are secured with security ties and labeled with sample ID ranges. Each bag is photographed for records. Shipments typically include around 800 samples per truckload. Company staff supervise all loading, transport, and unloading at the preparation lab.

Prepared pulp samples are shipped from the preparation laboratory to the analysis laboratory ALS Laboratories in Lima, via commercial couriers. Each shipment typically includes 300 or more samples. Prior to shipment, Company geologists visit the preparation lab to insert Control Samples (standards, pulp duplicates, and blanks every eight intervals) without laboratory staff present to maintain anonymity.

All samples were analyzed for gold by fire assay (Au-AA25), and silver, lead, zinc, copper, arsenic and sulfur by aqua regia digestion, ICP-AES or AAS finish (OG46). (Figure 11-1).

11.2 Sample Shipment and Security

Drill core is stored in a clean and well-maintained core shack in the Luminex exploration camp. To avoid welling and caking, the sample pulps are stored inside a refrigerator in a sealed plastic bag.

Stringent sample shipment and security measures were consistently implemented to maintain the integrity of the samples throughout the Project.

Samples collected by TVX between 1994 and 2000 were shipped to SGS in Ecuador for preparation, and then to Canada for analysis. Samples were also sent directly to ALS Chemex. The other half of the core was stored securely.

Between 2004 and 2007, Goldmarca and Ecometals transported samples by truck to Loja and then shipped them to ALS Chemex in Quito or Acme in Cuenca. If broken sample bags were encountered, the samples were retaken and reshipped to ensure their integrity throughout transportation.

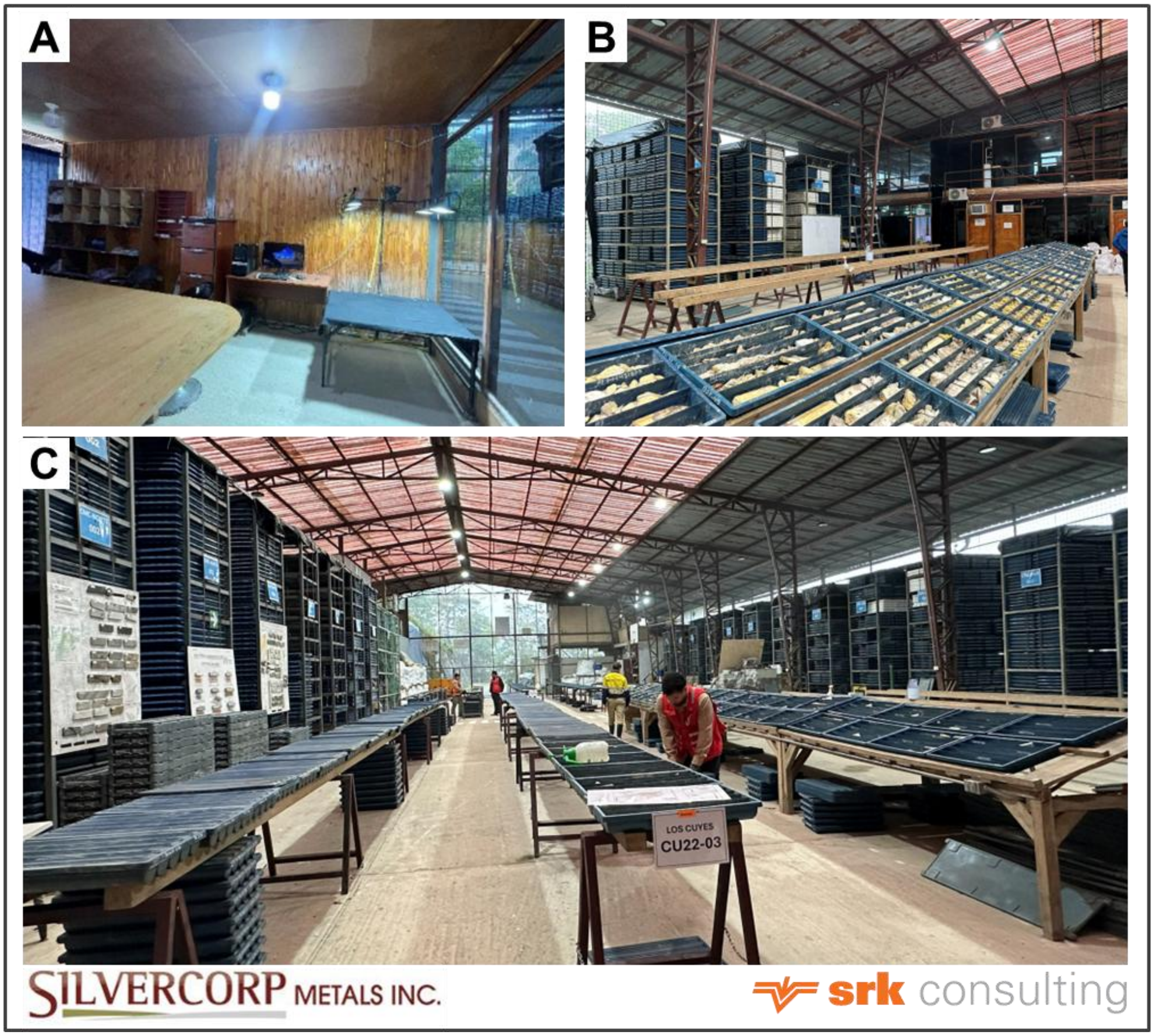
Ecuador Gold and Copper Corp. used their employees or bonded couriers to transport samples to Acme Lab's preparation facility in Cuenca between 2012 and 2014.

Between 2017 and 2019, Lumina Gold shipped samples to MSALABS preparation facility in Cuenca. Samples were bagged using secure tamper-proof tags, which were checked upon arrival to ensure their integrity.

From 2019-2023, Luminex had representatives from ALS collect samples from their exploration camp and deliver them to their preparation lab in Quito. The samples were transported in secure bags using tamper-proof tags, which helped to ensure the integrity of the samples throughout the process.

In 2025, Silvercorp utilized commercial couriers for transport and Company staff to monitor the loading, transport and unloading of all samples from the exploration camp to the preparation lab, ALS Laboratories in Quito. The lab's internal QAQC and security procedures apply for the transfer of pulps for analysis to the ALS Laboratories in Lima.

Figure 11-1: Drill Core at the Logging Areas of the Condor Project



11.3 Specific Gravity Data

Specific gravity (SG) data are only available for the Los Cuyes and Camp areas. SG measurements were determined using the water immersion method (weight in air versus weight in water). The SG data was collected by TVX between 1994 and 1995, Goldmarca / Ecometals between 2004 and 2007, EGX in 2012, Luminex between 2019 and 2023.

Typically, SG measurements were conducted on samples spaced at 10 m intervals down each drill hole. During the EGX drill program, SG measurements were conducted every 10 m to 15 m.

Between 2017 and 2018, every 10th specific gravity measurement taken by Lumina Gold was shipped to MSALABS in Vancouver for a second density measurement using paraffin-coated samples. Similaryly, every 10th specific gravity sample taken by Luminex was submitted to ALS Laboratories in Lima, Peru to validate measurements. The results were then checked and compiled in an Access database for each hole.

The volume and distribution of SG data are considered sufficient to support calculation of average densities per rock type in the block models at Los Cuyes and Camp. Table 11-1 summarizes the density data available per simplified logged lithology units within the Camp and Los Cuyes areas. These average densities are applied to the block model for each modelled lithology unit.

Selected core intervals are weighed in air and water to measure specific gravity, with measurements taken approximately every 10–20 m and more frequently in mineralized zones. All data and calculations are entered into a standard SG template. Porous or vuggy core is wax coated (≤1 mm thickness) before water immersion.

In 2025, Silvercorp followed the same SG sampling protocol. Porous or vuggy core is wax coated (≤1 mm thickness) before water immersion. These data were not used in the 2025 MRE and are not listed in Table 11-1.

Table 11-1: Density Data for Camp and Los Cuyes per Lithology Code

Lithology	Los Cuyes		Camp	
	Count	Average SG	Count	Average SG
Dacite	329	2.74	275	2.69
Granodiorite	308	2.74	1,240	2.70
Greenstone	34	2.75	315	2.85
Rhyodacite	57	2.64	568	2.61
Rhyolite lapilli tuff	461	2.63	-	-
Rhyolite North West	97	2.65	2	2.58
Rhyolite welded tuff	6	2.68	83	2.64

11.4 Quality Assurance and Quality Control Programs

QA/QC programs are typically set in place to ensure the reliability and trustworthiness of exploration data. They include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures are important as safeguard for the project data and form the basis for the quality assurance program implemented during exploration. Analytical quality control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation and assaying. They are also important to prevent sample mix-up and monitor the voluntary or inadvertent contamination of samples. Assaying protocols typically involve regular duplicate and replicate assays and insertion of quality control samples. Check assaying is typically performed as an additional test of reliability of assaying results. This typically involves re-assaying a set number of rejects ad pulps at a second umpire laboratory.

Operators of the Project that conducted drilling programs between 2004 and 2025 employed analytical quality control measures that included the routine insertion of blanks, certified reference materials and duplicate analysis. A total of five standards blended from in-house materials prepared by Inspectorate Services Peru, and 35 commercially sourced reference materials from Ore Research and Exploration Pty Ltd. (OREAS) and CDN Resource Laboratories Ltd (CDN) were used between 2004 and 2025 (Table 11-2).

A variety of commercially and locally sourced blank materials were used between 2004 and 2025 (Table 11-3).

Table 11-2: Specifications of Control Samples Used Between 2004 and 2025

Material	Period	Au (g/t)		Cu (%)		Ag (ppm)		Count[3]	Source[4]
		Exp. Value[1]	SD[2]	Exp. Value[1]	SD[2]	Exp. Value[1]	SD[2]		
STD-0	2004-2006	Unknown						39	Inspectorate
GEO-184 STD-1	2005-2007	1.05						132	Inspectorate
GEO-269 STD-2	2004-2007	2.23						175	Inspectorate
GEO-273 STD-3	2004-2007	3.19						164	Inspectorate
GEO-309 STD-4	2005-2007	3.82						108	Inspectorate
18Pb	2007-2008	3.63	0.07					27	OREAS
61Pa	2007-2008	4.46	0.06			8.54	0.19	27	OREAS
62Pa	2007-2008	9.64	0.14					21	OREAS
15Pa	2007-2012	1.02	0.026					38	OREAS
15Pc	2007-2013	1.61	0.05					39	OREAS
17Pb	2007-2013	2.56	0.17					37	OREAS
53P	2007-2013	0.38	0.009	0.413	0.009			31	OREAS
7Pb	2007-2013	2.77	0.055					28	OREAS
12a	2012	11.79	0.24					8	OREAS
152a	2012	0.116	0.005	0.385	0.009			10	OREAS
15g	2012	0.527	0.023					54	OREAS
15h	2012	1.02	0.025					34	OREAS
19a	2012	5.49	0.1					3	OREAS
52c	2012	0.346	0.017	0.344	0.009			7	OREAS
62d	2012	10.36	0.33			8.37	0.68	7	OREAS
15d	2012-2013	1.56	0.042					14	OREAS
2Pd	2012-2013	0.885	0.03					16	OREAS
67a	2012-2013	2.24	0.096	0.0325	0.001	33.6	2	8	OREAS
68a	2012-2013	3.89	0.15	0.0392	0.0015	42.9	1.7	10	OREAS
503	2013	0.687	0.024	0.566	0.15	1.63	0.124	50	OREAS
54Pa	2013	2.9	0.11	1.55	0.02			2	OREAS
CDN-CM-14	2012-2013	0.792	0.039	1.058	0.031			45	CDN
CDN-CM-25	2012-2013	0.228	0.015	0.191	0.003			84	CDN
CDN-CM-26	2013	0.372	0.024	0.246	0.008			168	CDN
CDN-CM-30	2013	1.3	0.06	0.73	0.017	15.9	0.65	88	CDN
CDN-CM-36	2017-2018	0.316	0.017	0.23	0.005	2.1	0.1	18	CDN
CDN-CM-28	2017-2019	1.38	0.085	1.36	0.04			62	CDN
CDN-CM-27	2017-2022	0.636	0.034	0.592	0.015			224	CDN
CDN-CM-43	2019-2020	0.309	0.02	0.233	0.006			127	CDN
504c	2019-2023	1.48	0.045	1.11	0.03	4.22	0.288	189	OREAS
505	2021-2022	0.555	0.014	0.321	0.008	1.53	0.072	56	OREAS
503d	2022, 2025	0.666	0.015	0.524	0.01	1.34	0.066	43	OREAS
501d	2022-2025	0.232	0.011	0.272	0.009	0.664	0.053	124	OREAS
507	2022-2025	0.176	0.006	0.622	0.013	1.34	0.081	112	OREAS
504d	2023	1.46	0.035	1.1	0.024	2.69	0.114	7	OREAS
Total								**2,436**	

Notes:

[1] Exp. Value = expected value

[2] SD = standard deviation

[3] Totals include data from Camp, Condor, Los Cuyes, Chinapintza, San Jose I, Guaya, Soldedad, Soledad Baja, Enma, Coguime, El Hito, Santa Barbara, Prometedor, and Nayumbi.

[4] OREAS = Ore Research and Exploration Pty Ltd, CDN = CDN Resource Laboratories Ltd, Inspectorate = Inspectorate Services Peru

Table 11-3: Summary of Blank Material Used Between 2004 and 2021

Blank ID	Description	Period	Value			Count
			Au (g/t)	Cu (%)	Ag (ppm)	
Glass-LAC	Crushed glass	2017-2023	<0.001	0.0001	0.25	1,147
Silice	Silica	2023	<0.001			43
BLK		2004-2013	0.002			596
OREAS 22P	Quartz	2007-2008	<0.002			202
OREAS 22b	Quartz	2012	<0.002	0.00089	<0.1	122
OREAS 23a	Granodiorite	2012-2013	0.003	0.00421	0.1	320
OREAS 22d	Quartz sand	2013	<0.001	0.000923	<0.1	192
Total						**2,579**

11.4.1 TVX Gold Inc (1994-2000)

There is no information about the implementation of an analytical quality control program by TVX.

11.4.2 Goldmarca Ltd. (2004-2007) and Ecometals Ltd. (2007-2008)

From 2004 to August 2007, Goldmarca's analytical quality control program involved the insertion of standard reference materials, blanks and ¼ core duplicates. The standards were sourced in-house, with some using mine waste material. However, due to high variability in analysis, these were discontinued.

Between July 2007 to 2011, the analytical program procedure involved inserting a blank every 6 samples, a standard after 7 samples, a duplicate after 6 samples, followed by another blank. The standards and blanks used for this period were sourced from Ore Research and Exploration Pty Ltd (OREAS).

11.4.3 Ecuador Gold and Copper Corp. (2012-2014)

The analtycial quality control program employed by EGX between 2012 to 2014 involved the insertion of standards, blanks and ¼ core duplicates. The insertion rate for these materials was 1 in 20. The standards were certified reference materials sourced from CDN Resource Laboratories Ltd. (CDN) or OREAS. Blank material was sourced from OREAS.

11.4.4 Lumina Gold Corp. (2017-2018)

Certified reference standards were inserted after every six to ten core samples. These included three certified reference standards from CDN and OREAS (high, medium and low gold grades), a blank, and a coarse and pulp duplicate. Blank material was comprised of crushed glass.

11.4.5 Luminex Resources (2019-2023)

Luminex followed similar analytical quality control procedures as Lumina. The resultant insertion rates for blanks, standards and coarse and pulp duplicate materials was between 2% and 4% between 2019 and 2021, and between 1% to 4% between 2022 and 2023.

11.4.6 Silvercorp (2024-Present)

Every eight samples a certified reference sample was inserted. These included three certified reference material from OREAS (high, medium and low gold grades), a blank, and three duplicates (core, coarse and fine). Blank material was comprised of crushed glass.

11.5 Qualified Person Comments

In the opinion of SRK, the sampling preparation, security and analytical procedures used by Silvercorp and previous operators are consistent with generally accepted industry best practices and are, therefore, adequate. Silvercorp should aim to employ consistency in the analytical quality control program procedures to ensure adequate number of samples for all drilling programs.

The review of analytical quality control data should be a continuous process in order to ensure corrective actions are taken, however this should be combined with reviews over longer periods of time to observe the long-term trends of data.

12 Data Verification

12.1 Verifications by Historical Operators

Between 2003 to 2004, Goldmarca re-assayed 1,219 samples of TVX core from Los Cuyes, San Jose and Santa Barbara and analyzed for gold using a screen fire assay method on a 50 g sample. Goldmarca reported good correlation with the original assay results (Easdon and Oviedo, 2004).

Lumina completed a resampling of the TVX holes from Los Cuyes as described in the 2018 Technical Report (Sim and Davis, 2018). Drill programs from 2004–2007 had a higher failure rate for gold in certified reference standards than would normally be acceptable; however, duplicate samples validated original assays. The failure rate for the 2007–2008 program was also higher than acceptable. Failures were found to be related to sample labelling errors rather than repeatability in resampled assays. Quality control failures for programs from 2012–2015 were addressed with programs of remedial assay analysis. Following this extensive check program, quality control issues with drill programs carried out by previous operators were deemed by the authors to have been adequately addressed.

For the Lumina/Luminex drill programs, a review of the QAQC protocols was conducted prior to drilling and formalized in a detailed QAQC manual developed by Lumina/Luminex. Each drilling phase was reviewed by a QP who was on site during the drill program. The procedures for core processing and the insertion of blanks and standards were examined. The QAQC program was deemed to have been conducted in accordance with industry best practices.

As part of the historical analytical quality control programs implemented by historical operators, the analytical quality control failures were addressed with programs of remedial assay analysis.

12.2 Verifications by SRK

12.2.1 Site Visit

The SRK team conducted the site inspections to the Condor project from June 19-20, 2024. The following verification steps were undertaken by this team:

- Site inspection of the project area.
- Meeting with Company representatives.
- Discussions with geologists regarding sample collection, sample preparation, sample storage, QAQC, geological interpretation.
- Review of the outcrop, mineralization, faults (Figure 12-1 C).
- Inspection of drillhole sealing mark (Figure 12-1 A & B).
- Visually checking stratigraphy against interpreted drilling sections.
- Visit the drill core store and core catalog room of Condor Project, to understand the company’s core storage protocols and procedures.

Figure 12-1: SRK Site Visit Photos (2024)

Source: SRK Site Visit, 2024

12.2.2 Verifications of Analytical Quality Control Data

The QP analyzed the analytical quality control data produced for the Condor Project between 2004 and 2023 by previous operators. A particular focus was placed on the Camp, Los Cuyes, Soledad and Enma deposits areas, since this data was included in the Mineral Resource Estimation contained herein.

Silvercorp provided the QP with the external analytical control data containing the assay results fo the quality control samples used for the Condor Project. All data were provided to the QP in Microsoft Excel spreadsheets. SRK aggregated the assay results for further analysis. Control samples (blanks and standard reference materials) were summarized on time series plots to highlight their performance. Duplicate samples were analyzed using bias charts, quantile-quantile and relative precision plots. For this period, Silvercorp did not submit samples to an umpire laboratory.

The external analytical quality control data produced by previous operators for the Camp, Los Cuyes, Soledad and Enma deposits between 2004 and 2023 are summarized in Table 12-1 and presented in graphical format in Appendix A.

Table 12-1: Summary of Analytical Quality Control Data Produced on the Condor Project Between 2004 and 2023

Material	Period	Inserts	Percentage
Sample Count		**66,280**	
Blanks		**1,811**	2.73%
Glass-LAC	2017-2023	971	
BLK	2004-2013	534	
OREAS 22P	2007-2008	187	
OREAS 22b	2012	78	
Silica	2023	41	
QC samples		**1,554**	2.34%
STD-0	2004-2006	31	
STD-1	2005-2007	117	
STD-2	2004-2007	157	
STD-3	2004-2007	147	
STD-4	2005-2007	101	
12a	2012	8	
15g	2012	49	
15h	2012	3	
15Pa	2007-2012	35	
15Pc	2007-2013	28	
17Pb	2007-2013	29	
18Pb	2007-2008	25	
2Pd	2012-2013	5	
501d	2022-2023	82	
503D	2022	1	
504c	2019-2023	165	
504d	2023	7	
505	2021-2022	30	
507	2022-2023	81	
53P	2007-2013	26	
61Pa	2007-2008	25	
62d	2012	7	
62Pa	2007-2008	20	
7Pb	2007-2013	25	
CDN-CM-27	2017-2022	180	
CDN-CM-28	2017-2019	44	
CDN-CM-43	2019-2020	126	
Field Duplicates	2012	78	0.12%
Coarse Duplicates	2004-2023	984	1.48%
Pulp Replicates	2019-2023	664	1.00%
Total QC Samples		**5,091**	**7.68%**

Notes: Totals include samples from Camp, Los Cuyes, Soledad and Enma datasets and are compared to the Mineral Resource database, as reported herein.

Although the target insertion rates for analytical quality control materials were defined throughout the various drilling programs since 2004, the actual insertion rates of standard samples has varied significantly over the projects history. Overall, the coverage of analytical quality control materials amounts of 8% since 2004. Silvercorp should ensure an insertion rate of 1 in 20 for each type (blank, standard, duplicate) for all programs moving forward.

Blanks

The source of blank material has varied over different periods, with a total of six different blank material types used across the Condor Project since 2004. Not all of these materials are considered true blanks, as some carry low-grade values for the elements of interest. It is recommended that the blank material selected reflect the detection limit and analytical methods selected for the elements of interest, as best as possible.

In general, the blank samples performed acceptably for silver, copper and gold analyses, with results consistently falling below the threshold of 10 times the detection limit for all periods. There is little to no systematic contamination observed for the Camp, Los Cuyes, Soledad and Enma deposits.

Standard Reference Materials

The standards from Inspectorate Services were prepared from blended ore from the Condor project. The reported from 3.5% to 10.5% analytical variance of the standard material provided by Inspectorate was considered too high for use as reference material on the Condor project and was discontinued since 2007.

It is noted that since there have been many material types used over the various drilling programs, a few of these materials did not have enough data to draw meaningful conclusions for trends observed over time.

The review of standard performance focused on the Camp, Los Cuyes, Soledad and Enma deposits. The performance of gold and copper was generally acceptable, with opportunity for improvement in the precision of some material analyses where results fell outside of two standard deviations of the expected value. In general, there appears to be a slight negative bias on some materials analyzed between 2019 and 2023, which presents an opportunity to improve the accuracy (e.g. Au analyses in material 501d, 505 and 507).

The performance of reference materials for silver analyses was variable. Poorer performances were observed for lower-grade materials is due to the detection limit of the analyses method during that period. This can be observed in materials 61Pa, 501d and 507 for example. For more recently used materials, such as 504c, there is a discrete period of bias observed during 2023, indicating a potential sample mix-up or analytical bias, which should be discussed with the lab to address mitigation efforts for future analysis.

Duplicates

The Condor Project analytical quality control protocols have included the insertion of field duplicates, coarse reject duplicates, and pulp duplicates between 2004 and 2023. These material types allow for a comparison at different stages of the preparation and analytical process. Paired datasets were analyzed for the Camp, Soledad, Los Cuyes and Enma deposits.

The field duplicates show a lower repeatability than the coarse and pulp duplicates, which is expected. The precision between these samples has approximately 8% of the pairs having a Half Absolute Relative Difference (HARD) value of less than 20%. This is an indication of a higher nugget effect in the data.

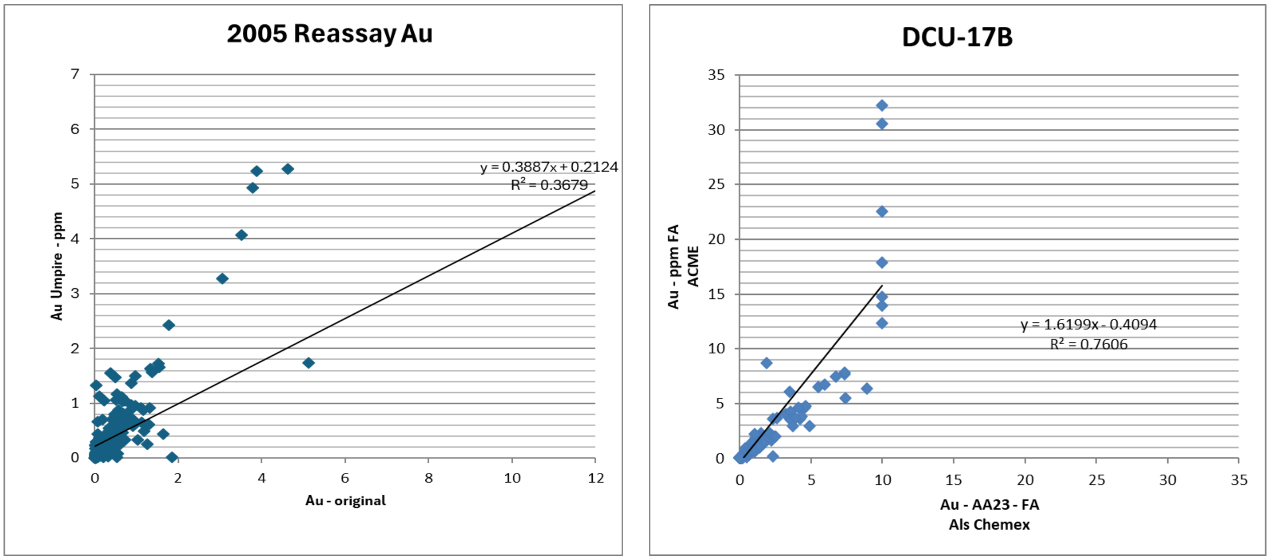
Paired analysis performed on coarse reject material between 2004 and 2023 have shown relatively good performance and repeatability for gold, copper and silver. There is a relatively low scatter round the deal correlation line for gold, and the HARD plot shows that more than 80% of the pairs have a HARD value of less than 20%. There is a small population of coarse duplicate pairs from 2007 that show a lower result in the original sample than the reanalyzed group, which may have been due to a lack of homogeneity in the prepared material at that time. This small bias population was limited and did not persist.

The performance of the pulp duplicates is considered acceptable for a deposit with coarse gold with good reproducibility. There was no obvious evidence of analytical bias observed between samples.

Umpire Duplicates

There were only two umpire programs conducted on the Condor Project. They were designed as validation for gold assay results, conducted in 2005 for drill hole DCU-17B, and in 2011 for drilling completed on the Los Cuyes Deposit. The results from this program are shown in Figure 12-2. For drill hole DCU-17B the correlation is generally good below 10 g/t. The ALS Chemex data appears to be reported to an upper limit of 10 g./t, and the over limit values are not reported.

Figure 12-2: Umpire Pulp Samples for the Los Cuyes Deposit



12.3 Conclusions and Recommendations

A comprehensive review of QAQC from drilling and trench sampling programs prior to 2014 is provided in Maynard and Jones (2011 and 2014) and Hastings (2013). As indicated in previous reviews of historical data, although no analytical quality control data was available drilling completed by TVX, the proportion of data assigned to this period amounts to only 24% of the total assay database used for Mineral Resource estimation. There is no observable bias between the TXV data and the data from other operators.

The QP is of the opinion that core sampling, logging and storage procedures are standardized, and that the analytical methods and processes generally comply with industry standards and as such the data used for the purposes of this report are adequate. The exploration team at the Condor Project demonstrated competence with respect to the managing, assessment and correction of analytic quality control data. SRK recommends that analytical quality control materials are inserted blind using a systematic approach at a rate of 1 in 20 samples (5%).

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Gold, silver, lead (galena) and zinc (sphalerite) are the four valuable elements in the Condor deposit. The gold and silver account for about 94% of total in-situ value. A large amount of metallurgical testwork was carried out by Plenge laboratory in Peru between 2020 and 2023 using the mineralized samples from the domains of Camp, Los Cuyes, Soledad, and Enma. The metallurgical testwork includes mainly the gravity concentration, whole-ore cyanide leach, bulk flotation, cyanide leach of the bulk flotation concentrate, and sequential selective flotation of gold/silver/lead/zinc from the cyanide leached residue. Some early testwork was completed by Goldmarca Mining Peru, Independent Metallurgical Laboratories and Lehne & Associates Applied Mineralogy. The results of these testwork programs are presented in the reports listed below:

- Goldmarca Mining Peru S.A.C., Breccias–San Jose–Ecuador, Direct Cyaniding Metallurgical Testwork, BX-Cuyes, Cuyes Dike and San Jose Samples, May 2004
- Independent Metallurgical Laboratories Pty Ltd., San Jose Ore Evaluation Testwork – Condor Gold Project for Goldmarca Limited, Project No. 2418, May 2006
- Lehne & Associates Applied Mineralogy, Microscopic Investigation of Drill Core Sections from the Condor Au-Ag-Cu Project, Southeast Ecuador, 22 March 2020
- Plenge Laboratory, Luminex Condor Project, Base Camp Samples, Report of Investigation No. 18525, Progress Report, 24 July 2020
- Plenge Laboratory, Luminex Condor Project, Base Camp Samples, Report of Investigation No. 18525, Progress Report, 27 July 2020
- Plenge Laboratory, Luminex Condor Project, Base Camp Samples, Report of Investigation No. 18525, Progress Report, 24 August 2020
- Plenge Laboratory, Luminex Condor Project, Camp, Los Cuyes and Enma Samples, Metallurgical Investigation No.18525-73-89, Progress Report, 26 May 2021
- Plenge, Luminex Gold Condor Project, Los Cuyes West (High Grade, Low Grade) and Breccia Pipe Samples, Report of Investigation No. 18702, 29 August 2023

13.2 Head Grades, Natural pH, and Specific Gravity

The head assays, natural pH, and specific gravity of sixteen mineralised samples are listed in Table 13-1. The four mineralised samples from the Camp domain contain 1.72 to 6.37 g/t gold, 11 to 60 g/t silver, 0.03 to 0.39% lead and 0.70 to 1.54% zinc. Total sulfur content ranges from 2.48 to 7.04%. This relatively high sulfur content results in a lower gold/sulfur ratio between 0.49 and 2.15. At a gold/sulfur ratio of 1.02 for the Master composite sample, it will not be possible to generate a high-grade concentrate containing 50 g/t gold when the bulk flotation is applied. The comparison between sulfide content and carbonate content indicates that there is a surplus amount of sulfide, and as a result, the net acid generation is expected after all sulfide minerals are oxidized. Arsenic, bismuth, cadmium, and antimony are common penalty elements when the gold concentrate is sold. The contents of these four elements are relatively low in the Camp domain with ranges between 129 to 273 ppm arsenic, 5 to 22 ppm bismuth, 46 to 101 ppm cadmium, and 20 to 38 ppm antimony. Mercury is a common penalty element for the concentrate sales, but its content was not assayed for these samples.

Table 13-1: Head Grade, Natural pH and Specific Gravity of Mineralized Feed Samples

Domain			Camp				Los Cuyes				Enma	Los Cuyes West				Soledad		
Program			18525 Plenge				18573 Plenge				18589 Plenge	18702 Plenge				2004 GMP		
Sample			L.G.	M.G.	H.G.	Master	L.G.	M.G.	H.G.	Master	Master	L.G.	H.G.	Breccia Pipe	Master	BX-Cuyes	Cuyes Dike	San Jose
Specific Gravity		t/m3	3.03	2.87	2.98	2.96	2.76	2.78	2.79	2.77	2.76	/	/	/	/	2.45	2.47	2.58
Natural pH			7.95	8.10	7.60	7.92	7.10	7.50	8.20	7.60	8.10	/	/	/	/	4.88	6.48	3.08
Gold	Au	g/t	1.72	5.33	6.37	4.28	0.53	0.58	1.05	0.75	0.90	0.73	3.65	1.06	1.81	1.02	3.13	2.11
Gold/Sulfur Ratio	Au/S		0.49	2.15	0.90	1.02	0.20	0.28	0.44	0.33	0.12	0.23	0.67	0.33	0.46	/	/	/
Silver	Ag	g/t	14	11	60	29	4	2	10	6	37	12	42	11	22	18	66	28
Copper	CuT	%	0.02	0.03	0.07	0.04	0.02	0.02	0.06	0.03	0.08	0.02	0.06	0.04	0.04	0.04	0.03	0.02
Lead	Pb	%	0.07	0.03	0.39	0.18	0.04	0.00	0.02	0.03	0.05	0.05	0.22	0.02	0.10	0.03	0.24	0.05
Zinc	Zn	%	0.70	0.71	1.54	0.98	0.20	0.05	0.08	0.13	0.25	0.40	0.91	0.43	0.58	0.07	0.25	0.08
Total Sulfur	ST	%	3.52	2.48	7.04	4.20	2.63	2.09	2.41	2.29	7.48	3.11	5.42	3.22	3.92	/	/	/
Sulfide	S2-	%	3.11	2.02	6.33	3.60	2.00	1.60	1.79	1.80	6.08	/	/	/	/	/	/	/
Total Carbon	CT	%	0.83	0.84	0.41	0.70	0.39	0.30	0.25	0.31	0.38	0.46	0.48	0.48	0.47	/	/	/
Carbonate Carbon	CCO3	%	0.73	0.74	0.33	0.60	0.35	0.26	0.21	0.27	/	/	/	/	/	/	/	/
Organic Carbon	CORG	%	0.10	0.10	0.08	0.10	0.04	0.04	0.04	0.04	<0.01	/	/	/	/	/	/	/
Aluminum	Al	%	7.1	7.7	6.6	7.1	5.8	5.8	5.7	6.0	7.9	6.7	6.0	6.4	6.4	/	/	/
Arsenic	As	ppm	129	178	272	/	67	60	98	70	296	33	518	35	195	33	95	59
Barium	Ba	ppm	289	530	335	371	533	676	666	639	300	277	225	112	204	/	/	/
Bismuth	Bi	ppm	5	8	22	7	<5	<5	<5	<5	17	9	41	7	19	/	/	/
Calcium	Ca	%	1.1	1.9	0.5	1.2	0.3	0.3	0.2	0.3	0.7	0.5	0.2	0.1	0.3	/	/	/
Cadmium	Cd	ppm	48	46	101	67	12	4	5	8	17	23	54	26	34	/	/	/
Cobalt	Co	ppm	16	17	15	14	7	7	8	8	21	4	5	4	5	/	/	/
Chromium	Cr	ppm	58	51	67	53	45	39	49	31	50	47	29	22	33	/	/	/
Iron	Fe	%	4.4	4.2	5.8	5.4	3.2	2.8	2.9	3.0	6.6	5.4	7.2	4.9	5.8	/	/	/
Potassium	K	%	3.3	2.9	3.3	3.1	4.2	4.5	4.2	4.6	3.9	3.3	3.0	3.4	3.2	/	/	/
Magnesium	Mg	%	0.70	0.78	0.36	0.60	0.19	0.17	0.16	0.17	0.54	0.33	0.25	0.23	0.27	/	/	/
Manganese	Mn	%	0.55	0.27	0.46	0.4	0.68	0.34	0.51	0.51	0.52	0.52	0.65	0.55	0.57	/	/	/
Sodium	Na	%	0.15	1.39	0.30	0.70	0.06	0.06	0.05	0.06	<0.01	0.03	0.03	0.03	0.03	/	/	/
Nickel	Ni	ppm	14	11	11	11	7	7	8	3	10	5	8	3	5	/	/	/
Phosphorus	P	ppm	344	525	305	399	234	228	198	244	663	312	250	252	271	/	/	/
Antimony	Sb	ppm	21	20	35	38	<5	<5	7	<5	6	<5	8	<5	8	/	/	/
Tin	Sn	ppm	12	12	13	11	7	8	8	<5	8	5	8	5	6	/	/	/
Strontium	Sr	ppm	38	173	45	86	25	28	28	28	11	11	9	5	8	/	/	/
Vanadium	V	ppm	54	72	43	60	28	20	14	18	102	34	24	34	30	/	/	/

The four mineralised samples from the Los Cuyes domain are lower grade for gold, silver, lead and zinc compared with the Camp domain. These samples contain 0.53 ~ 1.05 g/t gold, 2 ~ 10 g/t silver, 0.00 ~ 0.04% lead and 0.05 ~ 0.20% zinc. Total sulfur content is also lower, which is between 2.09% and 2.63%. Because of the significantly lower gold grade, the gold/sulfur ratio is only 0.20 ~ 0.44. At a gold/sulfur ratio of 0.33 for the Master composite sample, it will not be possible to generate a gold concentrate with over 25 g/t gold when the bulk flotation is applied. The contents of arsenic, bismuth, cadmium and antimony are even lower than the Camp domain.

The one mineralised sample from the Enma domain is undesirable due to its lower gold grade and high sulfur content. It contains only 0.90 g/t gold, but 7.48% sulfur. This results in a very low gold/sulfur ratio of 0.12. At this gold/sulfur ratio, it will not be possible to generate a gold concentrate with over 10 g/t gold when the bulk flotation is applied.

The four mineralized samples from the Los Cuyes West domain are somewhat favourable than those samples from the Los Cuyes domain in terms of the gold grade, gold/sulfur ratio and zinc content. Ranges include:

- Gold grades from 0.73 g/t to 3.65 g/t compared with 0.53 to 1.05 g/t gold for the Los Cuyes domain. The Master composite sample contains 1.81 g/t compared with 0.75 g/t for the Los Cuyes domain
- Zinc content from 0.40% to 0.91% compared with 0.05 to 0.20% zinc for the Los Cuyes domain. The Master composite sample contains 0.58% zinc compared with 0.13% zinc for the Los Cuyes domain.
- Gold/sulfur ratio from 0.23 to 0.67 compared with 0.20 to 0.44 for the Los Cuyes domain. The gold/sulfur ratio for the Master composite sample is 0.46 compared with 0.33 for the Los Cuyes domain.

The three mineralized samples from the Soledad domain contain a higher grade gold (1.02 to 3.13 g/t) compared with the Los Cuyes domain (0.53 to 1.05 g/t) and Enma domain (0.90 g/t), but these gold grades are lower than the Camp domain (1.72 to 6.37 g/t). The Soledad domain contains a low lead content (0.03 to 0.24%) and zinc content (0.07 ~ 0.25%). It is worth noting that the mineralized samples from the Soledad domain are slightly acidic with natural pH ranging from 3.08 to 6.48. This may imply that some natural oxidation of sulfide minerals might have taken place. The natural pHs for the Camp domain (7.60 to 8.10), Los Cuyes domain (7.10 to 8.20) and Enma domain (8.10) are all slightly alkaline.

13.3 Mineralogy and Liberation

The quantitative bulk mineralogy as determined by XRD method is shown in Table 13-2 for ten mineralized samples. Among sulfide minerals, pyrite (FeS_2) is most dominant, followed sphalerite (ZnS). Lead is present as galena (PbS). Copper is minor and present as chalcopyrite ($CuFeS_2$). Among non-sulfide gangue minerals, quartz (SiO_2) is most dominant, followed by muscovite [$KAl_2(AlSi_3O_{10})(O,F)_3$] and orthoclase ($KAlSi_3O_8$). For the Master composite sample from the Camp domain (Plenge 18525):

- Gold is 97% liberated at grind size of 80% passing 75 µm.
- Sphalerite is 89% liberated at grind size of 80% passing 74 µm.
- Galena is 80% liberated at grind size of 80% passing 149 µm.

Based on the study by Lehne & Associates in 2020 for nineteen drill core samples from the Camp domain, the mineralization is composed of varying amounts of pyrite, sphalerite and chalcopyrite which often appear together with pyrrhotite, marcasite and galena. Arsenopyrite, magnetite and tennantite are less common. Native gold is widespread and contains a substantial silver content. Gold is mainly associated with pyrite in which it occurs as minute inclusions rarely exceeding 50 µm. Isolated gold particles are also observed in sphalerite, chalcopyrite, galena and gangue. Sphalerite often carries numerous droplets of chalcopyrite. Where pyrrhotite appears as a main constituent, it is frequently replaced either by a fine-grained mixture of pyrite and magnetite or by a microscopically undefined “intermediate product” that can eventually transform into marcasite.

Table 13-2: Bulk Mineralogy of Mineralized Samples Measured by XRD Method

Domain		**Camp**			**Los Cuyes**			**Enma**	**Los Cuyes West**		
Program		**18525 Plenge**			**18573** Plenge			**18589 Plenge**	**18702 Plenge**		
Sample ID		L.G.	M.G	H.G.	L.G.	M.G.	H.G.	Master	L.G.	H.G.	Breccia Pipe
Actinolite	%	1.0	1.0	/	/	/	/	/	/	/	/
Albite	%	1.3	15.7	3.2	/	/	/	/	/	/	/
Alunite	%	/	/	/	/	/	/	/	/	/	1.2
Amorphous	%	/	/	/	8.2	8.2	11.2	/	/	/	/
Andesine	%	1.0	1.0	/	/	/	/	/	/	/	/
Anorthite	%	/	3.5	3.0	2.1	2.1	/	/	/	/	/
Butlerite	%	6.1	2.7	4.5	/	/	/	/	/	/	/
Calcite	%	2.3	3.2	/	/	/	/	/	/	/	/
Chamosite	%	1.3	6.0	1.0	/	/	/	/	5.6	3.9	/
Clinochlore	%	1.0	1.0	1.0	/	/	/	/	/	/	/
Illite	%	/	/	/	/	/	/	7.1	1.1	1.0	2.4
Kaolinite	%	/	1.7	/	1.0	1.0	1.0	/	/	1.0	1.7
Labradorite	%	/	/	1.0	/	/	/	/	/	/	/
Microcline	%	/	/	5.3	1.4	2.1	1.3	/	/	/	/
Montmorillonite	%	/	/	/	/	/	/	1.0	/	/	/
Muscovite	%	23.9	11.3	16.5	20.9	18.0	21.4	31.1	33.4	33.7	27.5
Orthoclase	%	6.2	12.1	5.8	13.9	17.5	13.3	/	/	/	/
Phlogopite	%	3.0	1.2	1.0	/	/	/	2.3	/	/	/
Pyrite	%	4.3	3.1	8.9	4.9	3.8	4.5	13.6	5.3	5.3	7.1
Quartz	%	40.2	29.4	39.5	42.0	41.8	40.0	36.2	47.2	48.1	52.1
Rhodonite	%	1.4	/	1.2	/	/	/	/	/	/	/
Sphalerite	%	1.1	1.1	2.4	/	/	/	/	0.7	0.7	1.4
Wollastonite	%	/	/	/	/	/	/	2.0	/	/	/
Others	%	6.0	5.9	5.8	5.6	5.5	7.3	6.7	6.7	6.3	6.6

13.4 Comminution

A limited amount of comminution testwork has been completed for twelve mineralized samples. The available testwork data are summarized in Table 13-3. For the mineralized samples from the Camp domain, only the Bond ball mill work index (BWi) was measured. The measured values are between 13.8 kW.h/t and 16.6 kW.h/t with an average of 14.9 kW.h/t. This implies that the mineralized material from the Camp domain is moderately hard.

Table 13-3: Comminution Testing Results of Mineralized Feed Samples

Domain	Program	Sample	SMC				BWi	Ai
			A	b	A*b	SCSE		
						kW.h/t	kW.h/t	g
Camp	18525 Plenge	L.G.	/	/	/	/	14.2	/
		M.G.	/	/	/	/	16.6	/
		H.G.	/	/	/	/	13.8	/
Los Cuyes	18573 Plenge	L.G.	68.0	0.70	47.6	8.99	13.3	0.105
		M.G.	68.2	0.80	54.6	8.49	12.2	0.087
		H.G.	64.1	1.07	68.6	7.77	12.1	0.072
Enma	18589 Plenge	Master	63.7	1.27	80.9	7.45	11.7	0.089
Soledad (San Jose)	2021 NI43-101	SJ-1A	/	/	/	/	12.1	/
		SJ-1B	/	/	/	/	12.7	/
	2004 GMP	BX-Cuyes	/	/	/	/	11.9	/
		Cuyes Dike	/	/	/	/	13.1	/
		San Jose	/	/	/	/	12.0	/

For the mineralized samples from the Los Cuyes domain, the SMC Test®, Bond ball mill work index and Bond abrasion index were determined.

The values of “A×b” are between 47.6 and 68.6 with an average of 56.9. This implies that the material from the Los Cuyes domain is relatively soft with respect to the SAG mill grinding.

The SCSE (SAG circuit specific energy) values range between 7.77 kW.h/t and 8.99 kW.h/t with an average of 8.42 kW.h/t.

The values of Bond ball mill work index (BWi) are between 12.1 kW.h/t and 13.3 kW.h/t with an average of 12.5 kW.h/t. This indicates that the mineralized material from the Los Cuyes domain is moderately hard with respect to the ball mill grinding.

The values of Bond abrasion index (Ai) vary between 0.072 and 0.105 with an average of 0.088. This implies that the mineralized material from the Los Cuyes domain has a relatively low abrasion property.

The mineralized sample from the Enma domain is similar to the Los Cuyes domain. The “A×b” value is 80.9. This means a relatively soft property with respect to the SAG mill grinding with the SCSE value being 7.45 kW.h/t. The value of Bond ball mill work index is 11.7 kW.h/t and the Bond abrasion index value is 0.089.

The values of Bond ball mill work index for the mineralized samples in the Soledad (San Jose) domain vary between 11.9 kW.h/t and 13.1 kW.h/t with an average of 12.4 kW.h/t. These values are similar to the domains of Los Cuyes and Enma, but somewhat lower than the domain of Camp.

13.5 Gravity Concentration

Gravity concentration tests were completed for ten mineralized samples from the domains of Camp, Los Cuyes, Los Cuyes West and Enma. This was a single-stage gravity concentration test at a fixed grind size of 80% passing 210 µm. If the gravity concentration test is carried out in multiple stages at varied grind sizes, more gold is expected to be recovered. The available gravity concentration testwork data are summarized in Table 13-4.

Table 13-4: Results of Gravity Concentration Testing

Domain	Program	Sample	Test ID	Head Grade		Gravity Concentrate				
						Mass Pull	Content		Recovery	
				Au	Ag		Au	Ag	Au	Ag
				g/t	g/t	%	g/t		%	
Camp	18525 Plenge	Master	MC-CG-1	4.50	31	0.29	574	1,320	36.5	12
			MC-CG-7	4.69	29	0.29	571	1,299	35.0	13
			MC-CG-12	4.41	32	0.29	446	942	29.4	9
			MC-CG-29	4.71	32	0.28	600	1,377	35.2	12
			Average	4.58	31	0.29	547	1,232	34.1	11
		L.G.	LG-CG-3	1.83	22	0.47	112	481	28.7	10
		M.G.	MG-CG-4	4.41	11	0.33	628	576	46.6	17
		H.G.	HG-CG-5	7.37	62	0.30	524	1,331	21.1	6
Los Cuyes	18573 Plenge	Master	MC-CG-4	0.76	6	0.32	55	188	23.1	10
Los Cuyes West	18702 Plenge	Master	MC-CG-7	1.79	21	0.20	134	550	14.9	5
Enma	18589 Plenge	Master	MC-CG-3	0.99	35	0.32	16	284	5.2	3

Note: Falcon centrifugal concentrator SB40. Grind size 80% passing 210 µm.

For the four mineralized samples from the Camp domain:

- Four gravity concentration tests were carried out for the Master composite sample. On average, the feed contained 4.58 g/t gold and 31 g/t silver. At 0.29% mass pull, the concentrate contained 547 g/t gold and 1,232 g/t silver with corresponding recoveries of 34.1% for gold and 11% for silver.
- For the Low-Grade sample (1.83 g/t gold and 22 g/t silver), the gravity concentrate had 0.47% mass pull with 28.7% gold recovery and 10% silver recovery and contained 112 g/t gold and 481 g/t silver.
- For the Medium-Grade sample (4.41 g/t gold and 11 g/t silver), the gravity concentrate had 0.33% mass pull with 46.6% gold recovery and 17% silver recovery and contained 628 g/t gold and 576 g/t silver.

- The High-Grade sample (7.37 g/t gold and 62 g/t silver) achieved 0.30% concentrate mass pull with corresponding recoveries of 21.1% for gold and 6% for silver, and the concentrate contained 524 g/t gold and 1,331 g/t silver.
- One gravity concentrate sample contained 82.9% pyrite (FeS_2), 9.5% galena (PbS), 3.9% sphalerite (ZnS), 1.2% magnetite (Fe_3O_4) and 1.0% quartz (SiO_2).

The amount of gravity recoverable gold from the Los Cuyes domain was smaller compared with the Camp domain, partially due to the lower head grade. The mineralized sample from the Los Cuyes domain contained 0.76 g/t gold and 6 g/t silver. At 0.32% mass pull, the gravity concentrate contained 55 g/t gold and 188 g/t silver with corresponding recoveries of 23.1% for gold and 10% for silver.

The amount of gravity recoverable gold from the Los Cuyes West domain further decreased. The sample this domain contained 1.79 g/t gold and 21 g/t silver. At 0.20% mass pull, the gravity concentrate contained 134 g/t gold and 500 g/t silver with corresponding recoveries of 14.9% for gold and 5% for silver.

The amount of gravity recoverable gold from the Enma domain decreased significantly in comparison with the Los Cuyes West domain. The sample from this domain contained 0.99 g/t gold and 35 g/t silver. At 0.32% mass pull, the gravity concentrate contained 16 g/t gold and 284 g/t silver with corresponding recoveries of 5.2% for gold and 3% for silver.

13.6 Cyanide Leach

Four different types of materials were used in cyanide leach testing, that is, (1) cyanide leach of the feed samples, (2) cyanide leach of the tail after gravity concentrate is removed, (3) cyanide leach of the flotation concentrate and (4) cyanide leach of the gravity concentrate.

13.6.1 Cyanide Leach of the Feed Samples

A number of cyanide leach tests were completed for the mineralized feed samples to determine gold recovery and silver recovery. The variables included grind size, cyanide concentration and retention time. The comparison between DCN (direct cyanide leach) and CIL (carbon-in-leach) cyanide leach was also made. The conditions and results of the completed cyanide leach tests are presented in Table 13-5. Four mineralized feed samples from the Camp domain were tested in cyanide leach testing, i.e., the Master composite sample, Low-Grade sample, Medium-Grade sample and High-Grade sample.

For the Master composite sample, average values are as follows:

- head grades were 4.38 g/t gold and 31 g/t silver
- recoveries were 97.5% for gold and 45% for silver after 24 hours of cyanide leach
- reagent consumptions were 1.17 kg/t for sodium cyanide and 0.7 kg/t for lime
- preg-robbing was not visible
- the impact of grind size on gold recovery was negligible at grind size (80% passing) between 38 and 75 µm.

Table 13-5: Conditions and Results of Cyanide Leach Testing for the Mineralized Feed Samples

Domain	Program	Sample	Test ID	Process	Grind Size (P80)	Pulp Density	Cyanide Concentration	Retention Time	Head Grade		Recovery		Reagent Consumption	
									Au	Ag	Au	Ag	Cyanide	Lime
					µm	% solid	g/L NaCN	h	g/t		%		kg NaCN/t	kg CaO/t
Camp	18525 Plenge	M.C.	CN-47	CIL	38	40	1.00	24	4.31	31	97.9	44	1.13	0.9
			CN-48	CIL	53	40	1.00	24	4.36	31	97.4	47	1.09	0.9
			CN-2	DCN	75	40	1.00	24	4.48	32	97.2	44	1.30	0.4
		L.G.	CN-1	DCN	75	40	1.00	24	1.70	20	91.8	50	0.80	0.7
			CN-2	CIL	75	40	1.00	24	1.77	22	92.1	42	0.98	0.7
		M.G.	CN-1	DCN	75	40	1.00	24	4.69	13	96.5	43	0.70	0.7
			CN-2	CIL	75	40	1.00	24	4.18	12	96.6	40	0.79	0.7
		H.G.	CN-1	DCN	75	40	1.00	24	7.54	65	96.0	43	1.10	0.7
			CN-2	CIL	75	40	1.00	24	8.14	61	96.7	49	1.07	0.7
Los Cuyes	18573 Plenge	M.C.	MC-CN-9	DCN	45	40	1.00	48	0.75	5	91.2	55	1.30	1.1
			MC-CN-8	DCN	75	40	1.00	24	0.76	5	87.1	49	0.70	0.6
			MC-CN-7	CIL	75	40	1.00	24	0.78	5	89.1	48	0.75	0.8
		L.G.	LG-CN-1	DCN	75	40	1.00	24	0.52	6	87.5	27	0.80	0.6
		M.G.	MG-CN-2	DCN	75	40	1.00	24	0.59	3	90.8	38	0.70	0.7
		H.G.	HG-CN-3	DCN	75	40	1.00	24	1.07	9	87.9	57	0.70	0.5
Los Cuyes West	18702 Plenge	L.G.	LG-CN-1	DCN	75	40	1.00	24	0.73	13	87.2	34	0.70	1.0
			LG-CIL-2	CIL	75	40	1.00	24	0.74	12	91.1	34	1.25	1.0
		H.G.	HG-CN-3	DCN	75	41	1.00	24	3.83	42	90.5	39	1.30	1.3
			HG-CIL-4	CIL	75	40	1.00	24	3.86	40	94.5	32	1.93	1.1
		Breccia Pipe	BP-CN-5	DCN	75	40	1.00	24	1.02	12	85.3	32	0.80	1.2
			BP-CIL-6	CIL	75	40	1.00	24	1.01	11	87.1	34	1.56	1.0
Enma	18589 Plenge	M.C.	MC-CN-1	DCN	75	39	1.00	24	0.97	37	74.2	68	0.70	0.6
			MC-CN-2	DCN	45	39	1.00	48	0.98	35	76.4	71	1.60	1.0
Soledad (San Jose)	2004 GMP	BX Cuyes	No.1	DCN	P90 75 µm	33	2.00	72	1.01	18	82.2	75	1.23	3.7
			No.4	DCN	P100 75 µm		1.50	96	1.03	18	98.3	94	1.84	4.0
		Cuyes Dike	No.2	DCN	P90 75 µm	33	2.00	72	3.06	71	92.5	84	1.89	3.0
			No.5	DCN	P100 75 µm		1.50	96	3.19	60	97.9	95	2.04	3.4
		San Jose	No.3	DCN	P90 75 µm	33	2.00	72	2.27	27	91.6	82	1.96	4.7
			No.6	DCN	P100 75 µm		1.50	96	1.94	30	98.3	86	2.10	5.7
	2006 IML	San Jose	No.1.2	DCN	106	40	0.5 ~ 0.3	48	4.17	9	72.7	43	1.64	6.6
			No.1.3	CIL	106	40	0.50-0.25	48	3.97	7	63.4	40	1.76	6.1

The Low-Grade sample contained 1.74 g/t gold and 21 g/t silver. Cyanide leach achieved 92.0% gold recovery and 46% silver recovery with reagent consumptions of 0.89 kg/t for sodium cyanide and 0.7 kg/t for lime.

The Medium-Grade sample contained 4.44 g/t gold and 12 g/t silver. Cyanide leach achieved 96.6% gold recovery and 42% silver recovery with reagent consumptions of 0.75 kg/t for sodium cyanide and 0.7 kg/t for lime.

The High-Grade sample contained 7.84 g/t gold and 63 g/t silver. After 24 hours of cyanide leach, 96.4% gold and 46% silver were dissolved, and the reagent consumptions were 1.09 kg/t for sodium cyanide and 0.7 kg/t for lime.

Four mineralized samples from the Los Cuyes domain were tested for cyanide leach, that is, the Master composite sample, Low-Grade sample, Medium-Grade sample and High-Grade sample. The mineralized samples from the Los Cuyes domain had the lower head grades and the cyanide leachable gold recoveries were also lower compared with the Camp domain.

- For the Master composite sample from the Los Cuyes domain, the preg-robbing was not clearly visible. Average head grades were 0.76 g/t gold and 5 g/t silver. Cyanide leach recoveries at grind size of 80% passing 75 µm were 88.1% for gold and 49% for silver with the reagent consumptions of 0.73 kg/t for sodium cyanide and 0.7 kg/t for lime. When grind size (80% passing) was reduced to 80% passing 45 µm, cyanide leach recoveries increased slightly to 91.2% for gold and 55% for silver.
- The Low-Grade sample (0.52 g/t gold and 6 g/t silver) achieved 87.5% gold recovery and 27% silver recovery with the reagent consumptions of 0.80 kg/t for sodium cyanide and 0.6 kg/t for lime.
- The Medium-Grade sample (0.59 g/t gold and 3 g/t silver) achieved 90.8% gold recovery and 38% silver recovery, and the corresponding reagent consumptions were 0.70 kg/t for sodium cyanide and 0.7 kg/t for lime.
- The High-Grade sample (1.07 g/t gold and 9 g/t silver) achieved 87.9% gold recovery and 57% silver recovery with the corresponding reagent consumptions of 0.70 kg/t for sodium cyanide and 0.5 kg/t for lime.

For the Los Cuyes West domain, three mineralized samples were tested for cyanide leach, i.e., the Low-Grade sample, High-Grade sample and Breccia Pipe sample. Overall, the materials from the Los Cuyes West show a minor preg-robbing phenomenon.

- For the Low-Grade sample (0.74 g/t gold and 13 g/t silver), cyanide leach achieved 87.2% gold recovery and 34% silver recovery along with reagent consumptions of 0.70 kg/t for sodium cyanide and 1.0 kg/t for lime. When cyanide leach was carried out in the presence of activated carbon (i.e., CIL cyanide leach), gold recovery increased to 91.1%.
- The High-Grade sample (3.85 g/t gold and 41 g/t silver) achieved 90.5% gold recovery and 39% silver recovery with the corresponding reagent consumptions of 1.30 kg/t for sodium cyanide and 1.3 kg/t for lime. When cyanide leach was carried out in the presence of activated carbon (i.e., CIL cyanide leach), gold recovery increased to 94.5%.
- The Breccia Pipe sample (1.02 g/t gold and 12 g/t silver) achieved 85.3% gold recovery and 32% silver recovery with the corresponding reagent consumptions of 0.80 kg/t for sodium cyanide and 1.2 kg/t for lime. When cyanide leach was carried out in the presence of activated carbon (i.e., CIL cyanide leach), gold recovery increased to 87.1%.

One mineralized sample (0.98 g/t gold and 36 g/t silver) from the Enma domain was tested for cyanide leach. Average gold recovery was 75.3%, which is lower than the domains of Camp and Los Cuyes. Average silver recovery was 69%, which is higher than the domains of Camp and Los Cuyes.

Four mineralized samples from the Soledad (San Jose) were tested for cyanide leach, three of which were tested in 2004 by Goldmarca Mining Peru (GMP) and one of which was tested in 2006 by Independent Metallurgical Laboratories (IML). It is worth noting that the samples from the Soledad (San Jose) were naturally acidic, which consumed more cyanide and lime during cyanide leach. Furthermore, it seems that the fine grind size and longer retention time are needed to achieve satisfactory gold recovery from cyanide leach.

- The "BX Cuyes" sample contained 1.02 g/t gold and 18 g/t silver. At grind size of 90% passing 75 µm, cyanide leach recoveries were 82.2% for gold and 75% for silver. When grind size was reduced to 100% passing 75 µm and cyanide leach retention time was extended to 96 hours, recoveries increased to 98.3% for gold and 94% for silver.
- The "Cuyes Dike" sample contained 3.12 g/t gold and 66 g/t silver. At grind size of 90% passing 75 µm, cyanide leach recoveries were 92.5% for gold and 84% for silver. When grind size was reduced to 100% passing 75 µm and cyanide leach retention time was extended to 96 hours, recoveries increased to 97.9% for gold and 95% for silver.
- The "San Jose" sample, which was tested in 2004 by GMP, contained 2.11 g/t gold and 29 g/t silver. At grind size of 90% passing 75 µm, cyanide leach recoveries were 91.6% for gold and 82% for silver. When grind size was reduced to 100% passing 75 µm and cyanide leach retention time was extended to 96 hours, recoveries increased to 98.3% for gold and 86% for silver.
- The "San Jose" sample, which was tested in 2006 by IML, contained a higher gold head grade (4.07 g/t) than the "San Jose" sample (2.11 g/t) tested in 2004 by GMP. Probably due to coarser grind size, lower cyanide concentration and shorter retention time, cyanide leach gold recoveries for this higher grade sample were only 72.7% from direct cyanide leach and 63.4% from CIL cyanide leach. The lower gold recovery from CIL cyanide leach might be caused by the breakage of activated carbon.

Figure 13-1 and Figure 13-2 show the relationship between recoveries and head grades of gold and silver based on the available cyanide leach testwork data. Although the low-grade sample generally results in lower recovery, more testwork data are needed to define a reliable equation between recovery and head grade. For the time being, the following average recoveries may be used for different domains.

- 95.8% gold recovery and 45% silver for the Camp domain
- 88.9% gold recovery and 46% silver recovery for the Los Cuyes domain
- 89.3% gold recovery and 34% silver recovery for the Los Cuyes West domain
- 87.1% gold recovery and 75% silver recovery for the Soledad (San Jose) domain
- 75.3% gold recovery and 69% silver recovery for the Enma domain

The consumption of sodium cyanide (NaCN) is shown in Figure 13-3 as a function of gold head grade. Average values are 1.00 kg/t for the Camp domain, 0.83 kg/t for the Los Cuyes domain, 1.26 kg/t for the Los Cuyes West domain, 1.15 kg/t for the Enma domain and 1.81 kg/t for the Soledad (San Jose) domain.

Figure 13-1: Relationship between Gold Recovery and Gold Head Grade from the Cyanide Leach of Feed Samples

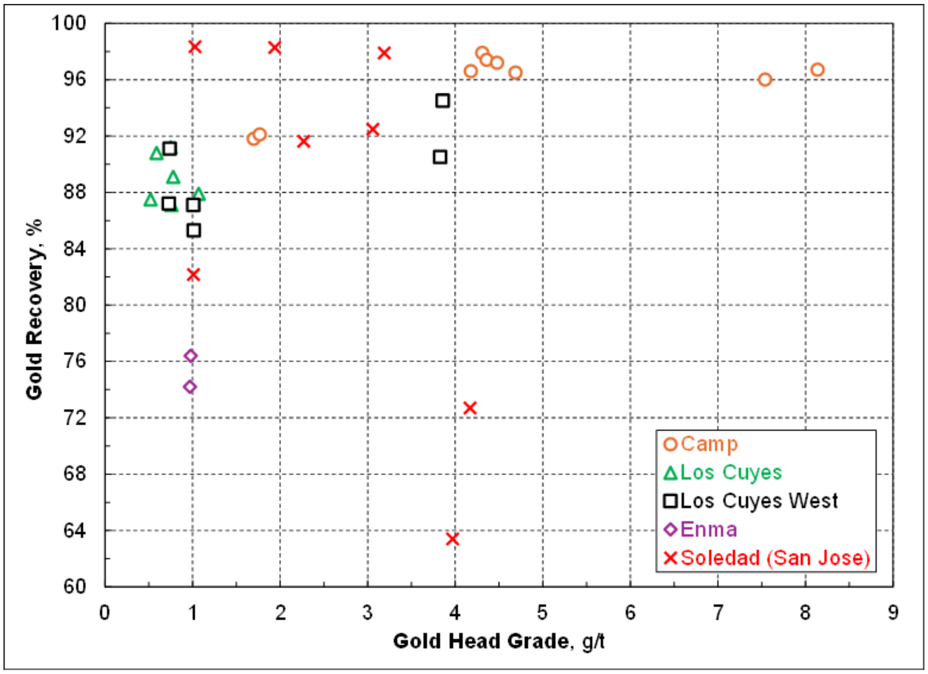


Figure 13-2: Relationship between Silver Recovery and Silver Head Grade from the Cyanide Leach of Feed Samples

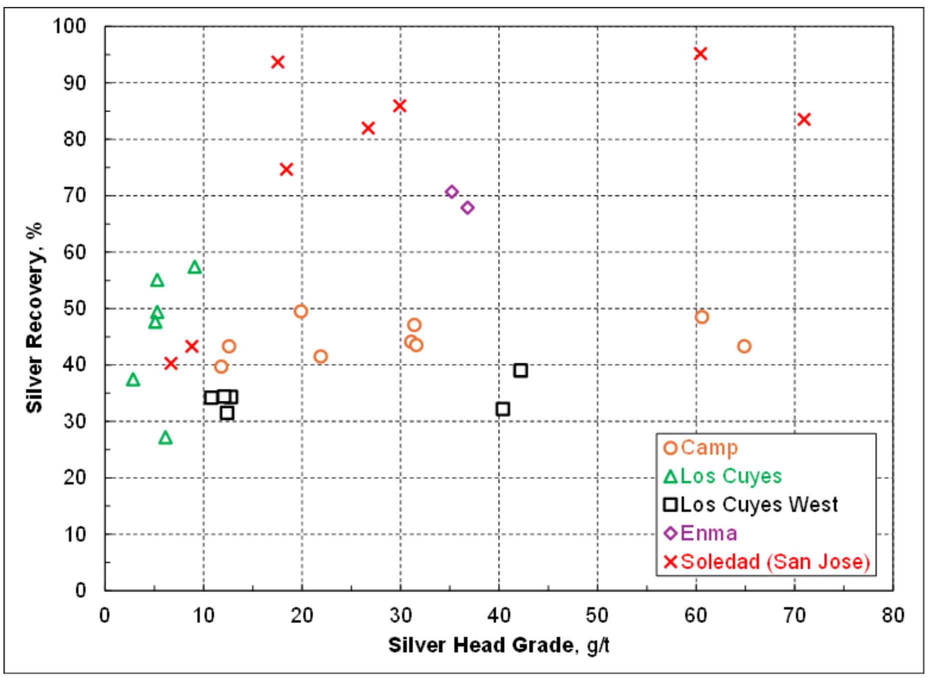


Figure 13-3: Relationship between Cyanide Consumption and Gold Head Grade from the Cyanide Leach of Feed Samples

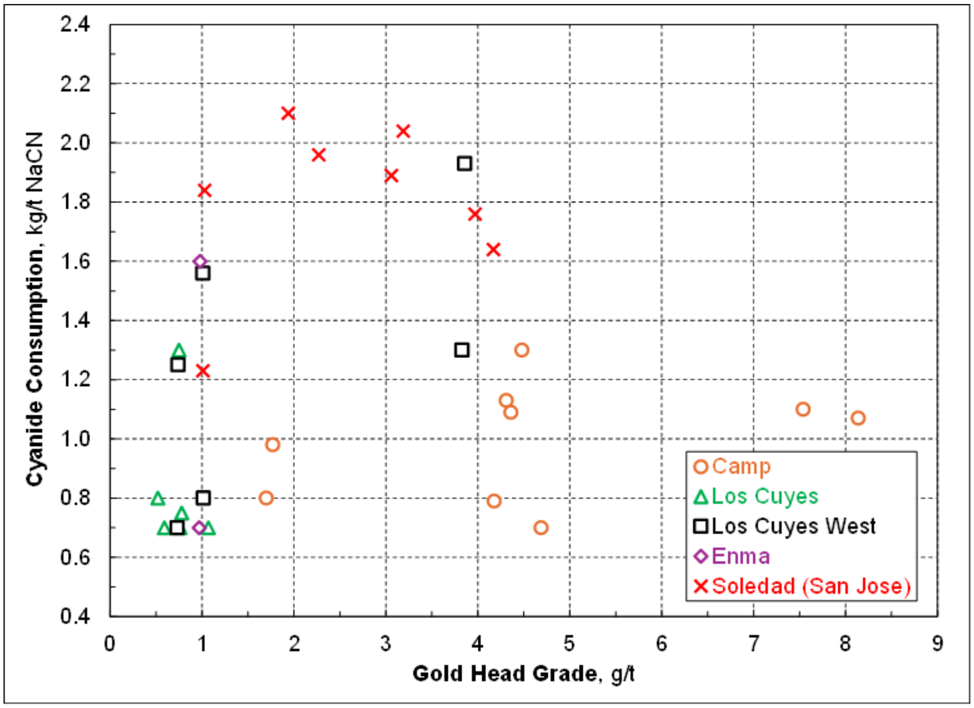
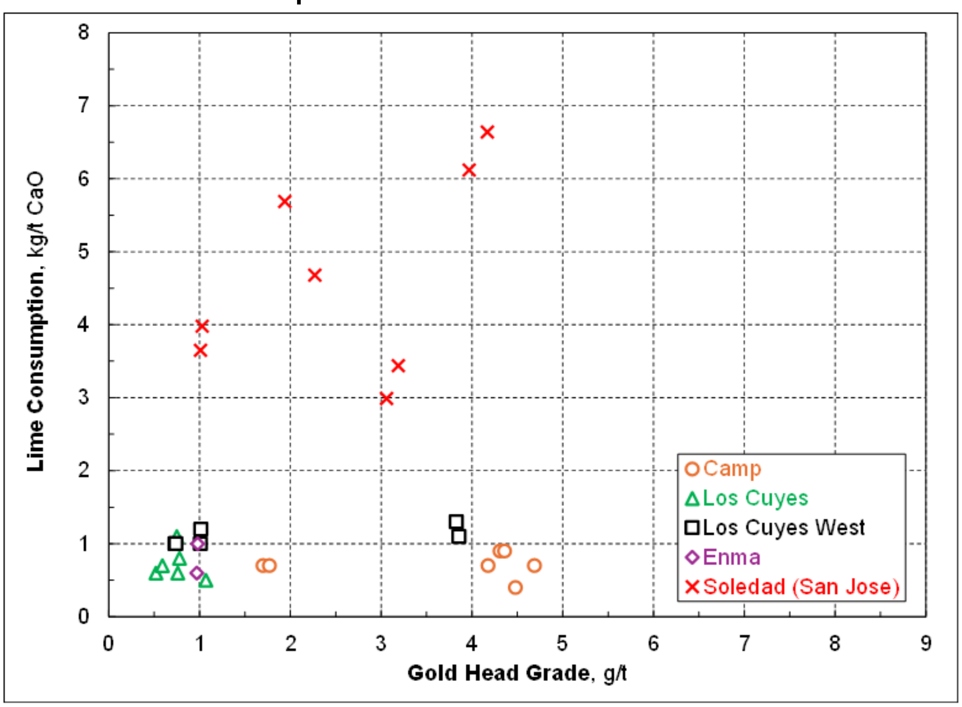


Figure 13-4: Relationship between Lime Consumption and Gold Head Grade from the Cyanide Leach of Feed Samples



The consumption of lime (CaO) is presented in Figure 13-4 as a function of gold head grade. Average values are 0.7 kg/t for the Camp domain, 0.7 kg/t for the Los Cuyes domain, 1.1 kg/t for the Los Cuyes West domain, 0.8 kg/t for the Enma domain and 4.6 kg/t for the Soledad (San Jose) domain.

Because of the presence of sphalerite and chalcopyrite, the cyanide tail solution contained the elevated levels of dissolved zinc and dissolved copper (Table 13-6). When the process water is recycled, the levels of dissolved zinc and dissolved copper are expected to increase. Some of these high-level dissolved zinc and especially dissolved copper will adsorb onto the activated carbon. Therefore, proper acid wash is required for the loaded carbon before gold/silver are stripped off.

Table 13-6: Composition of Cyanide Leach Tail Solutions

Domain	Program	Sample	Test ID	Grind Size P80	As	Ca	Cu	Fe	K	Pb	S	Zn
				µm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Camp	18525 Plenge	M.C.	CN-2	74	<1	8	17	18	52	<1	203	86
			CN-5	75	<1	11	11	<1	15	<1	94	42
			CN-6	150	<1	4	14	<1	9	<1	70	43
			CN-47	38	<1	54	12	<1	83	<1	255	87
			CN-48	53	<1	52	11	<1	71	<1	229	79
Los Cuyes West	18702 Plenge	L.G.	CN-1	75	<1	101	11	<1	47	<1	274	66
			CIL-2	75	<1	71	10	<1	67	<1	291	53
		H.G.	CN-3	75	<1	209	20	<1	57	<1	518	216
			CIL-4	75	<1	113	16	<1	70	<1	520	160
		Breccia Pipe	CN-5	75	<1	204	22	<1	60	<1	380	71
			CIL-6	75	<1	102	18	<1	78	<1	375	57

13.6.2 Cyanide Leach of the Gravity Tail Samples

Several tail samples from gravity concentration testing were subjected to cyanide leach at pulp density of 37 ~ 41% solid, 1.0 g/L NaCN cyanide concentration and 24-hour retention time (Table 13-7). Because a portion of free gold has been removed, the cyanide leach of these gravity tail samples resulted in slightly lower gold recovery compared with their corresponding feed samples.

For the Master composite sample from the Camp domain, the grind size of 80% passing 75 µm is desirable because gold recoveries at coarser grind size of 80% passing 150 µm and 210 µm were apparently lower.

At the grind size of 80% passing 75 µm, the gravity tail samples from the Camp domain resulted in average gold recoveries of 95.2% for the Master composite sample, 91.2% for the Low-Grade sample, 94.8% for the Medium-Grade sample and 94.9% for the High-Grade sample.

For the Los Cuyes domain, the gravity tail sample resulted in 84.0% gold recovery. For the Enma domain, the gravity tail sample achieved 70.9% gold recovery. Sodium cyanide consumption ranged from 0.63 kg/t to 0.95 kg/t.

Table 13-7: Results of Cyanide Leach of the Gravity Tail Samples

Domain	Program	Sample	Process	Grind Size (P80)	Head Grade		Recovery		Reagent Consumption	
					Au	Ag	Au	Ag	Cyanide	Lime
				µm	g/t		%		kg NaCN/t	kg CaO/t
Camp	18525 Plenge	M.C.	DCN	75	2.51	23	95.3	38	0.90	0.8
			CIL	75	2.59	25	95.1	29	0.95	0.4
			CIL	150	3.06	25	92.6	36	0.63	2.1
			DCN	210	2.90	27	88.2	36	0.70	0.4
			CIL	210	2.93	22	89.0	29	0.80	0.4
		L.G.	DCN	75	1.31	20	91.2	43	0.80	0.8
		M.G.	DCN	75	2.36	10	94.8	37	0.80	0.7
		H.G.	DCN	75	5.83	58	94.9	42	0.80	0.6
Los Cuyes	18573 Plenge	M.C.	DCN	75	0.58	5	84.0	55	0.70	0.8
Enma	18589 Plenge	M.C.	DCN	75	0.94	35	70.9	67	0.80	0.6

Note: Cyanide concentration 1.0 g/L NaCN pulp density 37% to 41% solid, retention time of 24 hours.

13.6.3 Cyanide Leach of the Flotation Concentrate

A number of cyanide leach tests were carried out using the bulk flotation concentrate samples. Average gold recovery was over 90%. Although further testwork is required, it appears that 2 to 3 g/L NaCN cyanide concentration and 24-hour retention time seem adequate. The details of cyanide leach of the flotation concentrates are presented in Table 13-8. Figure 13-5 shows the relationship between gold recovery and gold grade in the flotation concentrate. Figure 13-6 shows the relationship between silver recovery and silver content in the flotation concentrate.

The flotation concentrate was produced from the Master composite sample in the Camp domain at the grind size of 80% passing 106 ~ 180 µm. Gold recovery from cyanide leach of the flotation concentrate changed slightly from 94.7% at grind size of 106 µm to 93.2% at grind size of 180 µm. Silver recovery was 54% on average. Average cyanide consumption was 4.5 kg/t NaCN and average lime consumption was 0.6 kg/t CaO.

The flotation concentrate from the Low-Grade sample in the Los Cuyes West domain resulted in 92.6% gold recovery and 52% silver recovery on average. Cyanide consumption was 3.1 kg/t NaCN when 2.0 g/L NaCN cyanide concentration was used.

The flotation concentrate from the High-Grade sample in the Los Cuyes West domain resulted in 95.8% gold recovery and 48% silver recovery on average. Cyanide consumption was 5.5 kg/t NaCN when 2.0 g/L NaCN cyanide concentration was used.

The flotation concentrate from the Breccia Pipe sample in the Los Cuyes West domain results in 90.9% gold recovery and 49% silver recovery on average. Cyanide consumption is 5.6 kg/t NaCN when 2.0 g/L NaCN cyanide concentration is used.

The flotation concentrate from the Master Composite sample in the Los Cuyes West domain resulted in 93.5% gold recovery and 50% silver recovery. Cyanide consumption was 4.5 kg/t NaCN.

Table 13-8: Results of Cyanide Leach of the Flotation Concentrate Samples

Domain	Program	Sample	Grind Size (P80)	Cyanide Conc'n	Retention Time	Head Grade		Recovery		Reagent Consumption	
						Au	Ag	Au	Ag	Cyanide	Lime
			µm	g/L NaCN	h	g/t		%		kg NaCN/t	kg CaO/t
Camp	18525 Plenge	M.C.	106	3.0	48	16.3	147	94.7	55	4.5	0.6
			125	3.0	48	16.4	148	95.0	54	4.7	0.5
			150	3.0	48	17.2	149	93.8	53	4.3	0.8
			180	3.0	48	16.5	144	93.2	53	4.6	0.6
Los Cuyes West	18702 Plenge	L.G.	75	10.0	24	6.62	115	90.1	52	6.8	2.4
			75	10.0	24	8.35	105	94.7	64	9.5	1.0
			75	2.0	24	8.08	119	93.1	42	3.1	1.8
		H.G.	75	10.0	24	23.7	225	94.2	48	10.6	1.9
			75	10.0	24	28.7	256	96.3	53	14.9	1.4
			75	2.0	24	26.9	248	96.8	43	5.5	1.7
		Breccia Pipe	75	10.0	24	8.99	102	89.9	52	10.7	2.0
			75	10.0	24	9.95	106	91.8	52	16.4	1.7
			75	2.0	24	9.40	102	91.1	42	5.6	1.7
		M.C.	75	10.0	24	24.4	248	93.5	50	4.5	0.7

Note: direct cyanide leach, pulp density 18% to 31% solid

Figure 13-5: Relationship between Gold Recovery and Gold Content in the Flotation Concentrate from the Cyanide Leach of Flotation Concentrate Samples

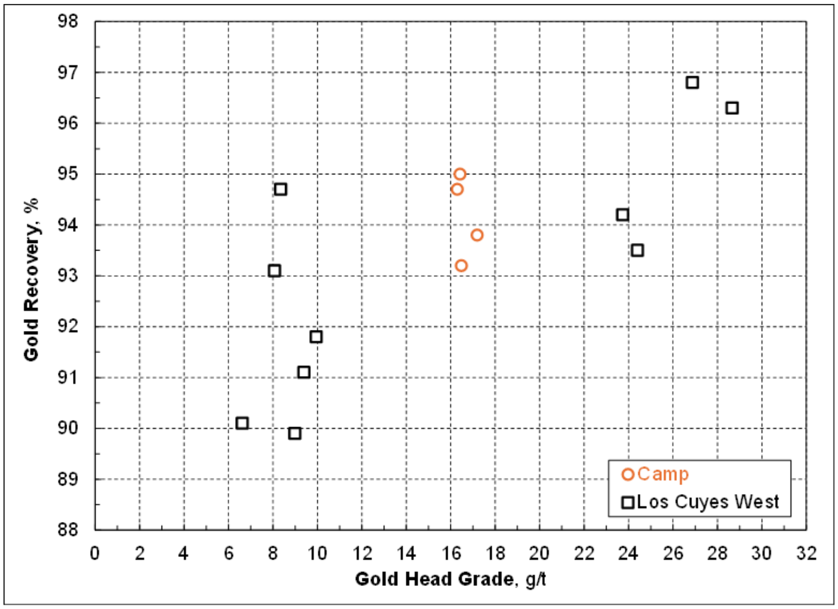
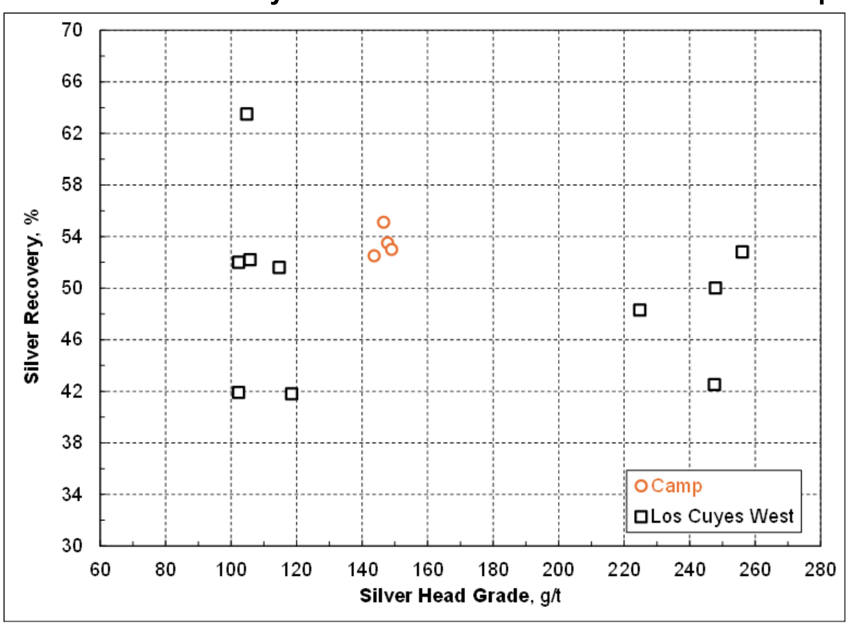


Figure 13-6: Relationship between Silver Recovery and Silver Content in the Flotation Concentrate from the Cyanide Leach of Flotation Concentrate Samples



13.6.4 Cyanide Leach of the Gravity Concentrate Samples

Gravity concentrate samples produced from the samples in the domains of Camp, Los Cuyes and Enma were tested for cyanide leach. Cyanide leach tests were carried out under the conditions of 10 g/L NaCN cyanide concentration for 24 hours. The obtained results are presented in Table 13-9. The gravity concentrates from the Camp domain leached well with 98.9% gold recovery and 66% silver recovery on average. Average sodium cyanide consumption was 17.3 kg/t.

Table 13-9: Results of Cyanide Leach of the Gravity Concentrate Samples

Domain	Program	Sample	Head Grade		Recovery		Reagent Consumption	
			Au	Ag	Au	Ag	Cyanide	Lime
			g/t		%		kg NaCN/t	kg CaO/t
Camp	18525 Plenge	M.C.	571	1,299	99.7	60	16.9	0.8
		L.G.	112	481	97.0	71	19.2	0.6
		M.G.	628	576	99.4	77	16.7	0.6
		H.G.	524	1,331	99.3	57	16.4	0.6
Los Cuyes	18573 Plenge	M.C.	55	188	92.3	46	13.2	0.5
Enma	18589 Plenge	M.C.	16	284	82.8	61	13.2	0.5

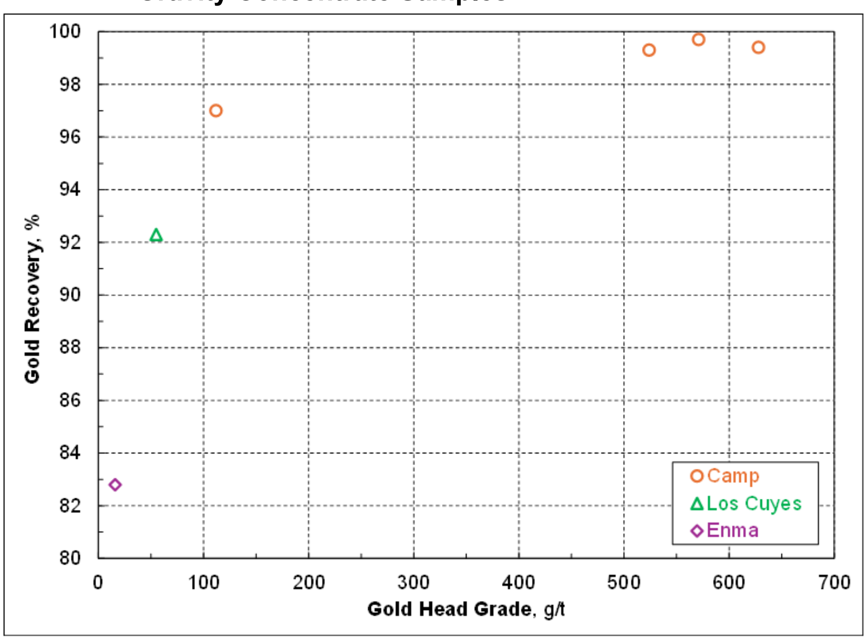
Note: Direct cyanide leach, pulp density 10% solid, cyanide concentration 10 g/L NaCN, retention time 24 hours, grind size P80 210 µm.

The gravity concentrate from the Los Cuyes domain behaved poorly during cyanide leach with 92.3% gold recovery and 46% silver recovery. The poor gold recovery might be due to the lower gold grade in the gravity concentrate.

The gravity concentrate from the Enma domain did not leach well with gold recovery being only 82.8%. This was a very low grade concentrate with 16 g/t gold. Also, some of the gold in this gravity concentrate might be refractory.

When gold recovery from cyanide leach of the gravity concentrate is plotted against gold content in the concentrate, a positive correlation is clearly visible (Figure 13-7).

Figure 13-7: Relationship between Gold Recovery and Gold Head Grade from Cyanide Leach of Gravity Concentrate Samples



13.7 Bulk Flotation and Sequential Selective Flotation

Three types of samples were investigated for flotation, i.e., the mineralized feed samples, the cyanide leached feed samples, and the cyanide leached flotation concentrate samples.

13.8 Flotation of the Feed Samples

The mineralized samples from the domains of Camp and Los Cuyes West were used in flotation testing, most of which was carried out as bulk flotation. The bulk flotation means that gold, silver and all sulfide minerals (such as pyrite, sphalerite, galena and chalcopyrite) are floated together into a single concentrate. Because of the presence of high-level pyrite, the gold grade in the produced bulk flotation concentrate will be relatively low grade, and thus it is not suitable for sales to a third party during the future commercial operation. The purpose of producing a bulk flotation concentrate is to reduce the solid mass for the subsequent cyanide leach to recover gold and silver. Another benefit is that the flotation tail is free from cyanide.

One of the flotation tests was carried out as sequential selective flotation by adding zinc sulfate and sodium cyanide as depressants to reject pyrite and sphalerite while gold, silver and galena were floated. For this selective flotation test (FT-02, Los Cuyes West), the gold grade in the concentrate was significantly increased, but gold recovery was relatively low.

Table 13-10 shows the details of conditions and results for the flotation tests using the mineralized samples from the domains of Camp and Los Cuyes West. For the Master composite sample from the Camp domain, average results are as follows:

- Head grades were 4.43 g/t gold, 19 g/t silver, 0.10% lead and 0.87% zinc.
- The bulk concentrate at 13.2% mass pull contained 33.2 g/t gold, 132 g/t silver, 0.61% lead and 4.8% zinc with the corresponding recoveries of 98.4% for gold, 97% for silver, 76% for lead and 72% for zinc.
- After the bulk concentrate was produced, the tail was floated again to produce a zinc concentrate after activation with the addition of copper sulfate. The zinc concentrate at 4.1% mass pull contained 0.82 g/t gold, 5 g/t silver, 0.02% lead and 5.6% zinc with the corresponding recoveries of 0.8% for gold, 1% for silver 1% for lead and 27% for zinc. Thus, total zinc recovery in two flotation concentrates was 72% + 27% = 99%.
- After a portion of free gold was removed, the tail was floated. Average results are as follows.
  - Head grades were 3.06 g/t gold, 28 g/t silvers, 0.12% lead and 1.01% zinc.
  - The bulk concentrate at 14.6% mass pull contained 20.8 g/t gold, 186 g/t silver, 0.73% lead and 5.7% zinc with the corresponding recoveries of 97.2% for gold, 96% for silver, 89% for lead and 82% for zinc.
  - After the bulk concentrate was produced, the tail was floated again to produce a zinc concentrate after activation with the addition of copper sulfate. The zinc concentrate at 4.3% mass pull contained 0.94 g/t gold, 13 g/t silver, 0.02% lead and 4.2% zinc with the corresponding recoveries of 1.2% for gold, 2% for silver 1% for lead and 17% for zinc. Thus, total zinc recovery in two concentrates was 82% + 17% = 99%.

For the mineralized samples from the Los Cuyes West domain, average results are as follows:

- For the Master composite sample (1.72 ~ 1.93 g/t gold, 18 ~ 21 g/t silver 0.08 ~ 0.10% lead, 0.54 ~ 0.60% zinc), the bulk concentrate at 16.5% mass pull contained 10.6 g/t gold, 118 g/t silver, 0.6% lead and 3.5% zinc with corresponding recoveries of 90.8% for gold, 94% for silver, 92% for lead and 96% for zinc.
- When the flotation test was carried out selectively by adding zinc sulfate and sodium cyanide as depressant to reject pyrite and sphalerite while gold/silver/lead were floated, the Au/Ag/Pb concentrate mass pull decreased significantly to 3.6% and the concentrate contained 39.5 g/t gold, 347 g/t silver, 1.7% lead and 6.6% zinc with corresponding recoveries of 83.0% for gold, 71% for silver, 77% for lead and 44% for zinc. After activation with the addition of copper sulfate, the tailing was floated again to produce a zinc concentrate. The zinc concentrate at 6.7% mass pull contained 2.9 g/t gold, 43 g/t silver, 0.1% lead and 3.9% zinc with corresponding recoveries of 11.4% for gold, 16% for silver, 11% for lead and 48% for zinc. Thus, total recoveries were 83.0% + 11.4% = 94.4% for gold, 71% + 16% = 87% for silver, 77% + 11% = 88% for lead, and 44% + 48% = 92% for zinc.

Table 13-10: Results of Rougher Flotation Tests Using the Feed Samples and Gravity Tail Samples

Domain	Program	Sample	Test ID	Conditions			Head Grade					Gold Concentrate											Zinc Concentrate										
				Grind Size P80	Gold Flotation	Zinc Flotation						Mass	Composition					Recovery					Mass	Composition					Recovery				
							Au	Ag	Pb	Zn	S		Au	Ag	Pb	Zn	S	Au	Ag	Pb	Zn	S		Au	Ag	Pb	Zn	S	Au	Ag	Pb	Zn	S
				µm			g/t	g/t	%	%	%	%	g/t	g/t	%	%	%	%					%	g/t	g/t	%	%	%	%				
Camp	18525 Plenge	M.G.	FT-1	60	natural pH, 20 g/t PAX, 10 g/t A208, 10 g/t F2044, 15 g/t Z11, 6 minutes	pH 11.0, 50 g/t $CuSO_4$, 10 g/t Z11, 3 minutes	4.17	10.5	0.04	0.72	2.4	11.9	34.3	79	0.2	3.9		98.3	98.3	57.6	65.3		3.5	0.94	6	0.02	6.9		0.8	0.8	1.7	33.5	
		M.C.	FT-11	150	37% solid, natural pH, 20 g/t PAX, 10 g/t A208, 10 g/t F2044, 18 g/t MIBC, 8 minutes	pH 11.0, 50 g/t $CuSO_4$, 20 g/t Z11, 5 g/t MIBC, 6 minutes	4.69	27.9	0.15	1.03	3.9	14.4	32.1	187	1.0	5.6	25.9	98.4	96.3	94.5	78.9	96.7	4.7	0.70	5	0.01	4.3	2.3	0.7	0.8	0.3	19.5	2.8
		Gravity Tail from Test MC-CG-7	FT-10	74	37% solid, natural pH, 20 g/t PAX, 10 g/t A208, 10 g/t A2044, 18 g/t MIBC, 8 minutes	pH 11.0, 50 g/t CuSO4, 20 g/t Z11, 5 g/t MIBC, 6 minutes	2.97	26.2	0.11	0.93	3.6	17.7	16.5	143	0.6	4.7	19.7	98.2	96.4	92.4	88.6	97.7	4.5	0.52	4	0.01	2.2	1.7	0.8	0.6	0.4	10.6	2.1
			FT-9	106			3.05	26.4	0.11	1.00	3.6	15.5	19.3	164	0.6	5.3	22.5	98.4	96.3	92.2	83.0	96.9	4.6	0.54	4	0.01	3.3	2.1	0.8	0.7	0.4	15.4	2.7
			FT-8	150			2.99	29.6	0.11	1.02	3.6	16.0	18.1	178	0.6	5.4	21.8	97.0	96.3	92.5	83.9	97.2	4.7	0.69	6	0.01	3.1	2.0	1.1	1.0	0.4	14.5	2.6
		Gravity Tail from Test MC-CG-12	FT-13	106	37% solid, natural pH, 10 g/t PAX, 10 g/t A208, 10 g/t F2044, 15 g/t MIBC, 4 minutes	pH 11.0, 50 g/t CuSO4, 20 g/t Z11, 5 g/t MIBC, 4 minutes	3.09	28.6	0.12	1.04	3.9	13.6	22.0	195	0.8	5.8	26.8	97.2	93.0	86.2	75.7	94.0	4.2	0.88	36	0.02	5.6	3.6	1.2	5.3	0.5	22.7	3.9
			FT-14	125			3.13	28.0	0.12	1.04	3.9	12.4	24.3	216	0.9	6.7	29.4	96.4	95.4	85.8	79.3	94.2	3.3	1.59	16	0.02	6.1	4.0	1.7	1.9	0.6	19.0	3.4
			FT-15	150			3.16	28.6	0.12	1.01	3.8	12.4	24.3	219	0.9	6.4	28.6	95.7	95.4	85.9	78.6	93.5	4.2	1.43	13	0.02	4.7	3.3	1.9	1.9	0.7	19.8	3.7
Los Cuyes West	18702 Plenge	M.C.	FT-01	75	natural pH, 100 g/t CuSO4, 10 g/t A208, 40 g/t Z6, 6 minutes	/	1.93	20.7	0.10	0.60	3.8	16.5	10.6	118	0.6	3.5	22.0	90.8	94.0	91.7	95.8	95.0	/	/	/	/	/	/	/	/	/	/	/
			FT-02	106	pH 9.0, 100 g/t ZnSO4, 5 g/t NaCN, 5 g/t AP3418A, 10 g/t MIBC, 4 minutes	pH 11.0, 100 g/t CuSO4, 20 g/t Z11, 15 g/t MIBC, 3 minutes	1.72	17.6	0.08	0.54	3.8	3.6	39.5	347	1.7	6.6	20.8	83.0	71.0	77.4	43.6	19.7	6.7	2.9	43	0.13	3.9	33.9	11.4	16.4	11.2	48.3	60.0
		L.G.	FT-05	75	natural pH (7.1), 10 g/t A208, 20 g/t Z6, 10 g/t MIBC, 4 minutes	/	0.80	12.5	0.06	0.48	3.4	10.7	6.8	108	0.5	3.4	27.5	91.7	92.2	85.3	76.2	87.3	/	/	/	/	/	/	/	/	/	/	/
			FT-07	75			0.90	12.5	0.05	0.45	3.0	10.2	8.4	105	0.4	2.6	25.8	95.2	85.7	81.6	58.5	88.8	/	/	/	/	/	/	/	/	/	/	/
			FT-07A	75			0.85	13.9	0.06	0.42	3.1	9.8	8.1	119	0.5	2.7	26.0	93.4	83.9	85.0	64.1	83.3	/	/	/	/	/	/	/	/	/	/	/
		H.G.	FT-04	75	natural pH (7.3), 10 g/t A208, 20 g/t Z6, 10 g/t MIBC, 4 minutes	/	3.97	41.3	0.22	0.96	5.3	16.1	23.6	228	1.2	4.7	29.3	95.6	88.9	92.3	79.3	88.7	/	/	/	/	/	/	/	/	/	/	/
			FT-08	75			4.21	40.9	0.21	0.93	5.0	13.9	28.7	256	1.3	4.8	29.6	95.0	87.4	83.9	71.9	82.2	/	/	/	/	/	/	/	/	/	/	/
			FT-08A	75			4.22	43.0	0.21	0.94	5.4	15.1	26.9	248	1.3	3.9	29.7	95.9	86.8	92.5	63.1	83.0	/	/	/	/	/	/	/	/	/	/	/
		Breccia Pipe	FT-03	75	natural pH (7.5), 10 g/t A208, 20 g/t Z6, 10 g/t MIBC, 4 minutes	/	1.10	11.7	0.03	0.49	3.0	10.8	9.4	97	0.2	3.4	23.6	92.4	89.3	71.9	73.6	84.5	/	/	/	/	/	/	/	/	/	/	/
			FT-06	75			1.08	12.4	0.03	0.46	3.0	10.5	10.0	106	0.2	3.2	25.1	96.8	89.3	67.2	73.5	88.3	/	/	/	/	/	/	/	/	/	/	/
			FT-06A	75			1.09	12.2	0.03	0.45	3.1	11.3	9.4	102	0.2	3.1	26.0	97.8	95.0	65.6	78.5	96.2	/	/	/	/	/	/	/	/	/	/	/

For the Low-Grade sample (0.85 g/t gold, 13 g/t silver, 0.06% lead, 0.45% zinc), the bulk concentrate at 10.2% mass pull contained 7.8 g/t gold, 110 g/t silver, 0.5% lead and 2.9% zinc with corresponding recoveries of 93.4% for gold, 87% for silver, 84% for lead and 66% for zinc.

For the High-Grade sample (4.13 g/t gold, 42 g/t silver, 0.21% lead, 0.94% zinc), the bulk concentrate at 15.0% mass pull contained 26.4 g/t gold, 244 g/t silver, 1.3% lead and 4.5% zinc with corresponding recoveries of 95.5% for gold, 88% for silver, 90% for lead and 71% for zinc.

For the Breccia Pipe sample (1.09 g/t gold, 12 g/t silver, 0.03% lead, 0.47% zinc), the bulk concentrate at 10.9% mass pull contained 9.58 g/t Au, 102 g/t Ag, 0.18% Pb and 3.2% with corresponding recoveries of 95.7% for gold, 91% for silver, 68% for lead and 75% for zinc.

Several graphs are plotted to reveal any potential underlying relationships.

- Figure 13-8 shows the relationship between gold recovery and concentrate mass pull. As expected, a positive correlation is visible between gold recovery and concentrate mass pull. Over 10% concentrate mass pull is required in order to achieve over 90% gold recovery.
- Figure 13-9 shows the relationship between gold recovery and gold head grade. The higher head grade generally results in higher gold recovery. For the high-grade samples with over 3.0 g/t gold, over 95% gold recovery was consistently achieved. For the low-grade samples, gold recoveries varied between 91% and 98%.
- Figure 13-10 shows the relationship between gold grade in the concentrate and the gold/sulfur ratio in the feed. As expected, a positive correlation is clearly visible between them.
- Figure 13-11 compares silver recovery with gold recovery. Overall, silver flotation performance was poorer than gold, especially for the Los Cuyes West domain.
- As with silver flotation, lead flotation performance (Figure 13-12) and zinc flotation performance (Figure 13-13) were also poorer than gold flotation performance.
- As with gold recovery, silver recovery was strongly dependent on the concentrate mass pull (Figure 13-14).
- Silver flotation recovery was independent of the silver head grade (Figure 13-15).

Both gold recovery and silver recovery are expected to increase during bulk flotation after the grind size, activators, collectors and concentrate mass pull are properly selected.

Figure 13-8: Relationship between Gold Recovery and Concentrate Mass Pull from the Flotation of Feed Samples and Gravity Tail Samples

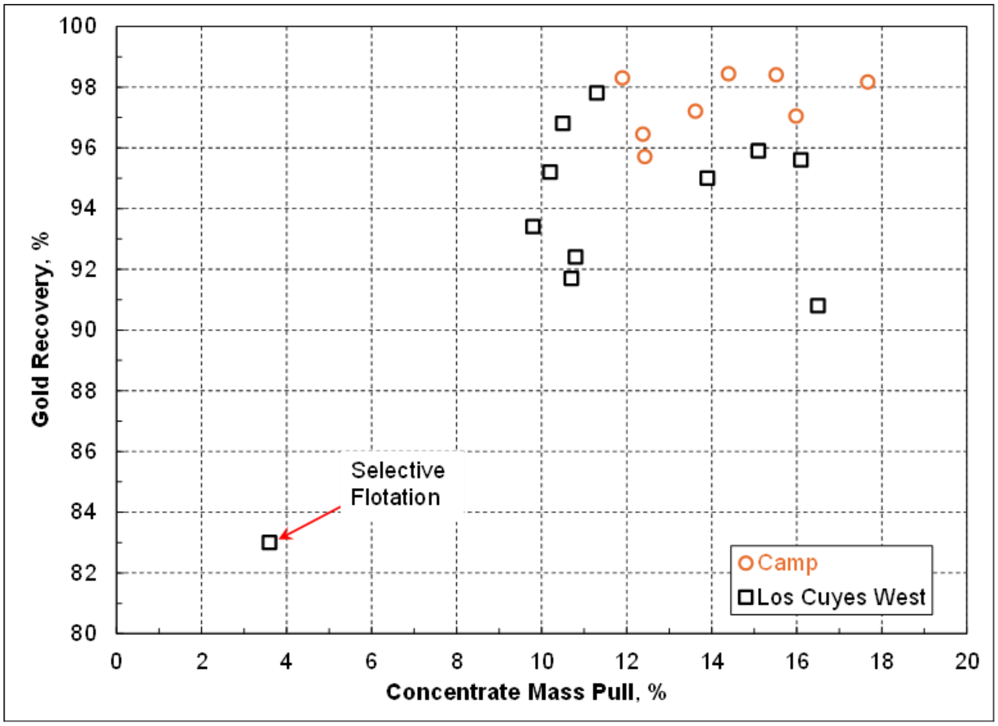


Figure 13-9: Relationship between Gold Recovery and Gold Head Grade from the Flotation of Feed Samples and Gravity Tail Samples

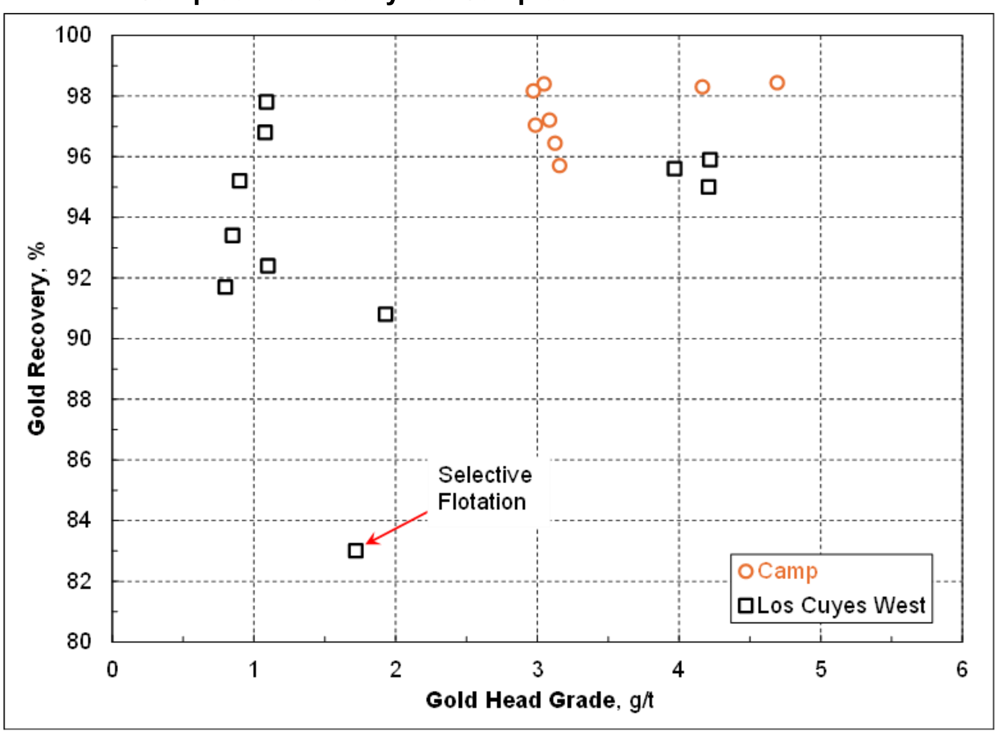


Figure 13-10: Relationship between Gold Grade in the Concentrate and the Gold/Sulfur Ratio in the Feed from the Flotation of Feed Samples and Gravity Tail Samples

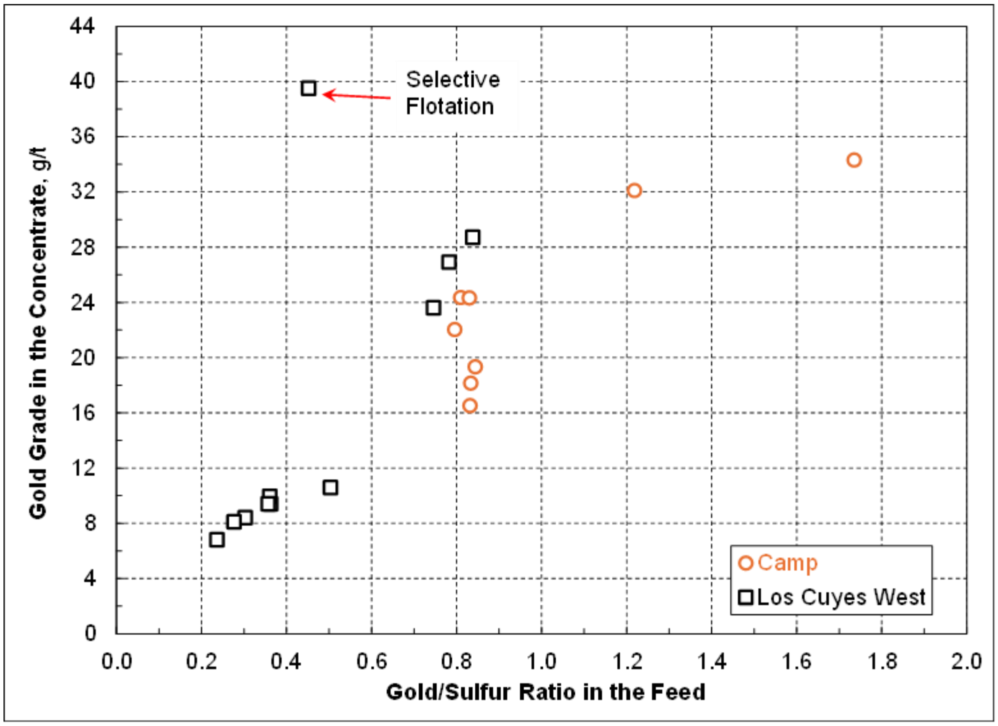


Figure 13-11: Relationship between Silver Recovery and Gold Recovery from the Flotation of Feed Samples and Gravity Tail Samples

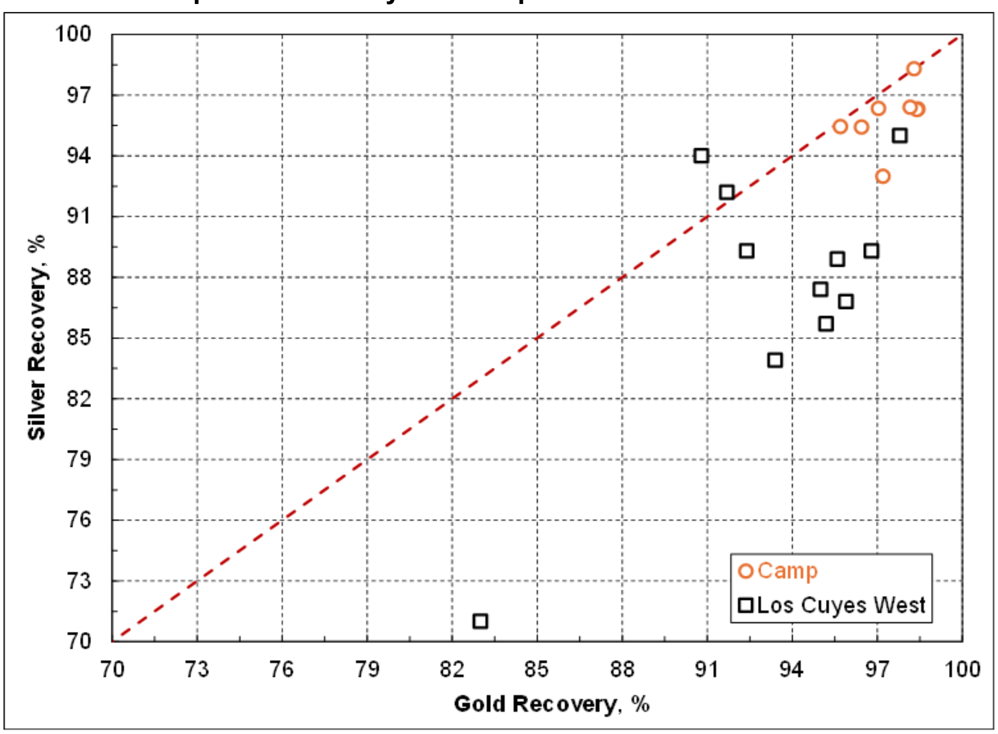


Figure 13-12: Relationship between Lead Recovery and Gold Recovery from the Flotation of Feed Samples and Gravity Tail Samples

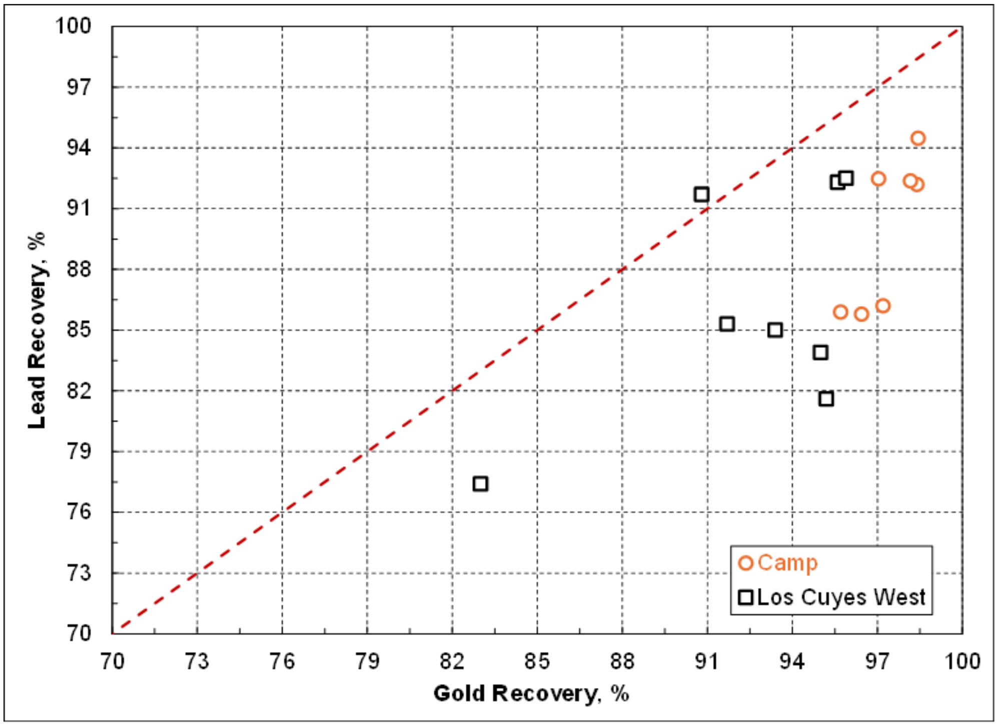


Figure 13-13: Relationship between Zinc Recovery and Gold Recovery from the Flotation of Feed Samples and Gravity Tail Samples

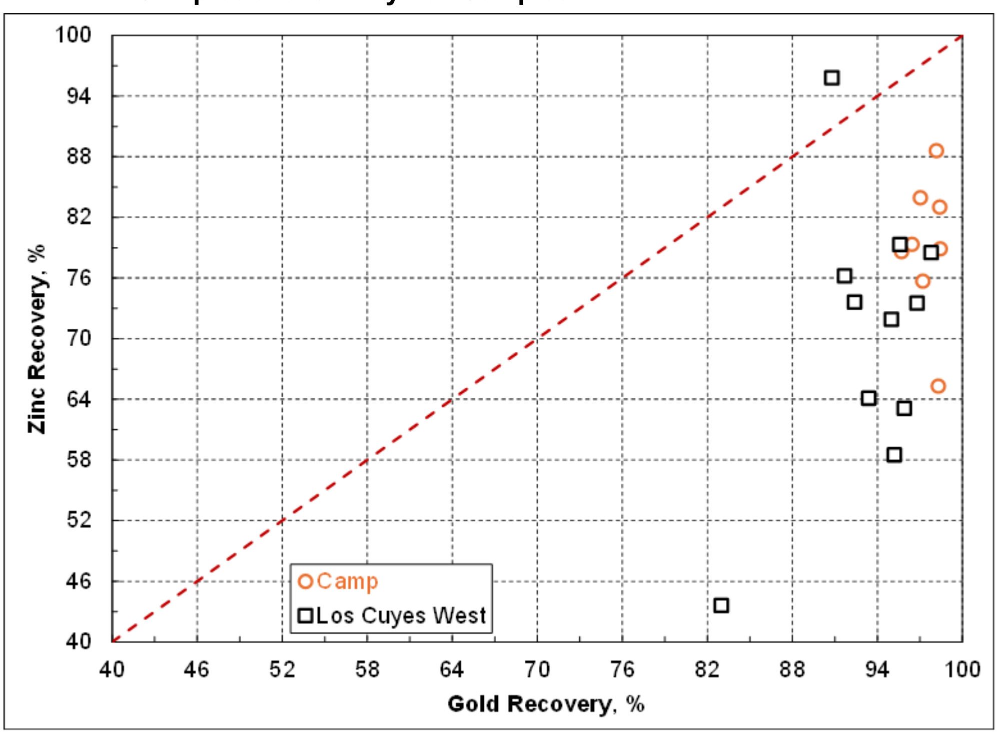


Figure 13-14: Relationship between Silver Recovery and Concentrate Mass Pull from the Flotation of Feed Samples and Gravity Tail Samples

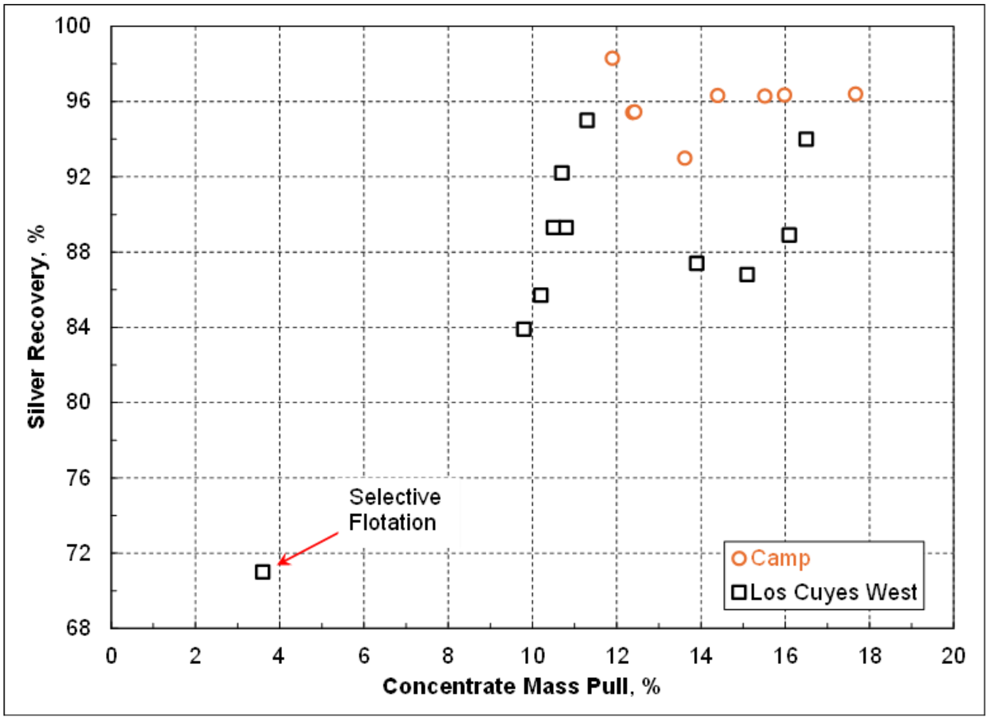
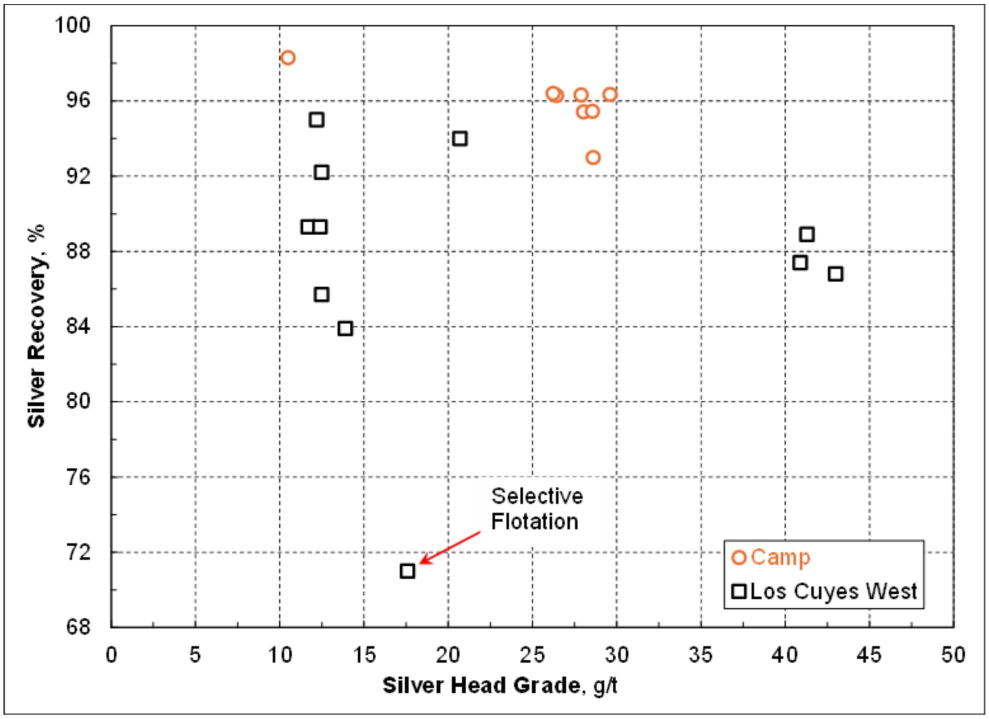


Figure 13-15: Relationship between Silver Recovery and Silver Head Grade from the Flotation of Feed Samples and Gravity Tail Samples



13.9 Flotation of the Cyanide Leached Bulk Flotation Concentrate Samples

After the cyanide leached bulk concentrate was filtered and then repulped, the slurry was conditioned and then floated selectively to produce the lead/silver concentrate while pyrite and sphalerite were rejected. Then, the tail slurry after lead/silver flotation was re-conditioned and activated followed by flotation to produce the zinc/silver concentrate. The details of conditions and results for these selective flotation tests are presented in Table 13-11.

For the cyanide leached bulk concentrate produced from the Master composite in the Camp domain, the average values are:

- The head grades were 0.91 g/t gold, 63 g/t silver, 0.59% lead and 4.83% zinc on average.
- The mass pull of lead/silver concentrate was 0.30%.
- The lead/silver concentrate contained 0.99 g/t gold, 2429 g/t silver, 41% lead and 9.3% zinc. Lead content in the lead/silver concentrate for each individual flotation test was 52.4% at grind size P80 of 106 µm, 42.9% at grind size P80 of 125 µm, 39.1% at grind size P80 of 150 µm and 29.4% at grind size P80 of 180 µm. These data clearly indicate that the lead content in the lead/silver concentrate increased with decreasing grind size.
- Metal recoveries were 0.33% for gold, 10.9% for silver, 19.7% for lead and 0.53% for zinc. Lead recovery of each individual flotation test was 24.8% at grind size P80 of 106 µm, 18.6% at grind size P80 of 125 µm, 18.5% at grind size P80 of 150 µm and 16.9% at grind size P80 of 180 µm. These data indicate a higher lead recovery with decreasing grind size.
- After the lead/silver concentrate was produced, the tail slurry was re-conditioned and activated with the addition of copper sulfate followed by flotation to produce the zinc/silver concentrate. When zinc was activated properly and the concentrate mass pull was controlled at a low level, the quality of zinc concentrate was good (Test F-26 and F-28). The zinc/silver concentrate contained 1.98 ~ 3.19 g/t gold, 268 ~ 417 g/t silver, 2.3 ~ 4.2% lead and 40.3 ~ 47.8% zinc with the corresponding recoveries of 6.8 ~ 16.8% for gold, 20.2 ~ 22.2% for silver, 21.3 ~ 21.7% for lead and 26.2 ~ 55.4% for zinc.
- For the penalty elements of arsenic, cadmium and antimony, the lead/silver concentrate (of Test F-26) contained 0.09% arsenic, <0.01% cadmium and 0.18% antimony (18525 Plenge). These levels are below the typical thresholds of penalty charges. Mercury content was not analysed. The zinc/silver concentrate (of Test F-28) contained 0.06% arsenic, <0.01% cadmium and 0.02% antimony. These levels are below the typical thresholds of penalty charges. Mercury content was not analysed.

For the cyanide leached bulk concentrates produced from the Los Cuyes West domain, the results are as follows:

- The cyanide leached bulk concentrate from the Low-Grade sample contained 42 g/t silver, 0.38% lead and 2.36% zinc.
  - The lead/silver concentrate contained 1748 g/t silver, 23.2% lead and 10.7% zinc with the corresponding recoveries of 33% for silver, 47% for lead and 4% for zinc.
  - The zinc/silver concentrate contained 128 g/t silver, 0.9% lead and 49.0% zinc with corresponding recoveries of 9% for silver, 7% for lead and 63% for zinc.

Table 13-11: Results of Lead Flotation and Zinc Flotation from the Cyanide Leached Bulk Flotation Concentrate Samples

Deposit	Program	Sample	Test ID	Conditions			Head Grade				Final Lead Concentrate									Final Zinc Concentrate								
				Grind Size P80	Lead Flotation	Zinc Flotation					Mass	Composition				Recovery				Mass	Composition				Recovery			
							Au	Ag	Pb	Zn		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
				µm			g/t	g/t	%	%	%	g/t	g/t	%	%	% (stage)				%	g/t	g/t	%	%	% (stage)			
Camp	18525 Plenge	M.C. - Test CN-21	FT-25	106	CN residue filtered and then pH dropped to ~9. 5 with H_2SO_4. Zinc sulfate addition 43 g/t for Ro, 43 g/t for 1st Cl and 29 g/t for 2nd Cl. 7 g/t NaCN for 3rd Cl. 14 g/t AP3418A and 14 g/t MIBC for rougher. 3 minutes for Ro, 3 minutes for 1st Cl, 2 minutes for 2nd Cl and 2 minutes for 3rd Cl	pH 11.0, 143 g/t $CuSO_4$, 29 g/t Z11, 14 g/t MIBC, 5 minutes regrind, 4 minutes for rougher and 3 minutes for 1st Cl	0.68	58.2	0.56	4.77	0.30	1.01	3,324	52.4	3.9	0.40	15.2	24.8	0.2	1.00	0.83	317	4.3	2.3	1.2	5.5	7.7	0.5
		M.C. - Test CN-22	FT-26	125	CN residue filtered and then pH dropped to ~9. 5 with H_2SO_4. Zinc sulfate addition 43 g/t for Ro, 43 g/t for 1st Cl and 29 g/t for 2nd Cl. 7 g/t NaCN for 3rd Cl. 14 g/t AP3418A and 14 g/t MIBC for rougher. 3 minutes for Ro, 3 minutes for 1st Cl, 2 minutes for 2nd Cl and 2 minutes for 3rd Cl	pH 11.0, 286 g/t $CuSO_4$, 57 g/t Z11 and 14 g/t MIBC for rougher. 5 minutes regrind, 4 minutes for Ro, 4 minutes for 1st Cl, 3 minutes for 2nd Cl and 2 minutes for 3rd Cl	0.92	65.8	0.61	4.90	0.30	1.01	2,472	42.9	6.6	0.30	10.0	18.6	0.4	3.20	1.98	417	4.2	40.3	6.8	20.2	21.7	26.2
		M.C. - Test CN-23	FT-27	150	CN residue filtered and then pH dropped to ~9. 5 with H_2SO_4. Zinc sulfate addition none to rougher, 29 g/t for 1st Cl and 29 g/t for 2nd Cl. 7 g/t NaCN for 3rd Cl. 14 g/t AP3418A and 14 g/t MIBC for rougher. 4 minutes for Ro, 3 minutes for 1st Cl, 2 minutes for 2nd Cl and 2 minutes for 3rd Cl	pH 11.0, 439 g/t $CuSO_4$, 57 g/t Z11 and 14 g/t MIBC for rougher. 5 minutes regrind. 20 g/t Z11 for 1st Cl. 4 minutes for Ro, 3 minutes for 1st Cl	1.03	63.4	0.58	4.96	0.30	1.17	2,405	39.1	23.6	0.30	10.4	18.5	1.3	0.70	5.75	169	2.3	1.6	4.1	2	3	0.2
		M.C. - Test CN-24	FT-28	180	CN residue filtered and then pH dropped to ~9. 0 with H_2SO_4. Zinc sulfate addition 43 g/t to rougher, 43 g/t for 1st Cl, 29 g/t for 2nd Cl and 14 g/t for 3rd Cl. 7 g/t NaCN for 3rd Cl. 14 g/t AP3418A and 14 g/t MIBC for rougher. 3 minutes for Ro, 3 minutes for 1st Cl, 2 minutes for 2nd Cl and 2 minutes for 3rd Cl	pH 11.0~11.5, 429 g/t $CuSO_4$, 57 g/t Z11 to rougher and 14 g/t Z11 to 1st Cl. 8 minutes regrind. 4 minutes for rougher, 4 minutes for 1st Cl, 3 minutes for 2nd Cl and 2 minutes for 3rd Cl	1.04	65.7	0.59	4.70	0.30	0.76	1,513	29.4	3.3	0.30	7.9	16.9	0.2	5.40	3.19	268	2.3	47.8	16.8	22.2	21.3	55.4
Los Cuyes West	18702 Plenge	L.G.	LG-FT-11	75	pH ~9.0, 13 g/t AP3418A + 13 g/t A208 for rougher, 13 g/t AP3418A + 13 g/t A208 for 1st Cl. 500 g/t NaCN for 1st Cl. 25 g/t MIBC for rougher	pH 11.0~11.5, 500 g/t $CuSO_4$ for rougher and 200 g/t $CuSO_4$ for 1st Cl. 3 minutes regrind. 50 g/t Z11 for rougher and 25 g/t Z11 for 1st Cl. 3 minutes rougher, 3 minutes for 1st Cl and 2 minutes for 2nd Cl	/	41.6	0.38	2.36	0.80	/	1,748	23.2	10.7	/	33.0	47.6	3.5	3.00	/	128	0.9	49.0	/	9.3	7.2	62.8
		H.G.	HG-FT-12	75			/	129	1.05	4.79	1.50	/	2,653	32.3	9.7	/	30.8	46.0	3	4.80	/	161	0.9	45.2	/	6.0	4.3	45.4
		Breccia Pipe	BP-FT-10	75			/	49.3	0.18	3.17	0.90	/	1,216	5.8	22.8	/	22.0	28.5	6.4	3.50	/	73	0.2	48.0	/	5.2	3.3	53.2

- The cyanide leached bulk concentrate from the High-Grade sample contained 129 g/t silver, 1.05% lead and 4.79% zinc.
  - The lead/silver concentrate contained 2653 g/t silver, 32.3% lead and 9.7% zinc with the corresponding recoveries of 31% for silver, 46% for lead and 3% for zinc.
  - The zinc/silver concentrate contained 161 g/t silver, 0.9% lead and 45.2% zinc with corresponding recoveries of 6% for silver, 4% for lead and 45% for zinc.
- The cyanide leached bulk concentrate from the Breccia Pipe sample contained 49 g/t silver, 0.18% lead and 3.17% zinc.
  - The lead/silver concentrate contained 1216 g/t silver, 5.8% lead and 22.8% zinc with the corresponding recoveries of 22% for silver, 29% for lead and 6% for zinc.
  - The zinc/silver concentrate contained 73 g/t silver, 0.2% lead and 48.0% zinc with corresponding recoveries of 5% for silver, 3% for lead and 53% for zinc.

For the penalty elements of arsenic, cadmium and antimony, the lead/silver concentrate contained 1.52% arsenic, 0.83% cadmium and 0.50% antimony (18702 Plenge). These levels are above the typical thresholds of penalty charges. Mercury content was not analysed. The zinc/silver concentrate contained 0.57% arsenic, 2.41% cadmium and 0.01% antimony. The levels of arsenic and cadmium are above the typical thresholds of penalty charges. The cadmium contents look suspiciously high and thus further investigations are required in the future. Mercury content was not analysed.

13.10 Flotation of the Cyanide Leached Feed Samples

The limited amount of testwork data on the flotation of the cyanide leached feed samples are shown in a 2021 report by Plenge laboratory (project number 18525-73-89). Some of the details are reproduced in Table 13-12. For the sample from the Camp domain, the cyanide leached feed samples was floated more favorably for both lead/silver concentrate and zinc/silver concentrate than the cyanide leached bulk flotation concentrate.

- Cyanide leach achieved 95.7% gold recovery and 44.6% silver recovery.
- The cyanide leached residue contained 0.21 g/t gold, 18.6 g/t silver, 0.14% lead and 1.01% zinc.
- The lead/silver concentrate at 0.10% mass pull contained 3.28 g/t gold, 3348 g/t silver, 44% lead and 20% zinc with corresponding recoveries of 1.9% for gold, 22% for silver, 38% for lead and 2% for zinc.
- The zinc/silver concentrate contained 2.52 g/t gold, 219 g/t silver, 1% lead and 45% zinc with the corresponding recoveries of 15.9% for gold, 15.7% for silver, 9% for lead and 60% for zinc.

For the sample from the Los Cuyes domain,

- Cyanide leach achieved 87.1% gold recovery and 49.4% silver recovery.
- The cyanide leached residue contained 0.09 g/t gold, 2.4 g/t silver, 0.04% lead, 0.12% zinc and 0.02% copper.
- The lead/silver concentrate contained 3.8 g/t gold, 650 g/t silver, 16.2% lead, 0.5% zinc and 5.9% copper. When this concentrate is further upgraded to about 45% lead content, the lead recovery is expected to drop significantly below 61.9%.

Table 13-12: Results of the Cyanide Leached Feed Samples

Domain				Camp	Los Cuyes
Program				18525-73-89 Plenge	18525-73-89 Plenge
Head Grade		Au	g/t	0.21	0.09
		Ag	g/t	18.6	2.4
		Pb	%	0.14	0.04
		Zn	%	1.01	0.12
Lead/Silver Concentrate	Mass Pull		%	0.1	0.1
	Composition	Au	g/t	3.28	3.81
		Ag	g/t	3,348	650
		Pb	%	44.2	16.2
		Zn	%	20.0	0.5
	Recovery	Au	% (stage)	1.9	6.0
		Ag		22.2	36.8
		Pb		38.1	61.9
		Zn		2.4	0.6
Zinc/Silver Concentrate	Mass Pull		%	1.3	/
	Composition	Au	g/t	2.52	/
		Ag	g/t	219	/
		Pb	%	1.0	/
		Zn	%	45.3	/
	Recovery	Au	% (stage)	15.9	/
		Ag		15.7	/
		Pb		9.3	/
		Zn		59.7	/

13.11 Preferred Flowsheet and Forecast of Metallurgical Performance

Based on the results obtained from the completed metallurgical testwork, there are two competing flowsheets for the future process plant. One key difference between these two flowsheets is whether the cyanide leach should be carried out before or after flotation. The benefits with the flowsheet of “bulk flotation cyanide leach of the bulk concentrate selective flotation” are as follows:

- The cyanide leach circuit is smaller.
- The cyanide destruction circuit is smaller.
- The tailing from the bulk flotation, which accounts for about 85% of mill feed by the solid mass, is free from cyanide and will not generate acid because nearly all sulfide minerals have been removed.

However, there are a few disadvantages with this flowsheet.

- Overall gold recovery will be lower.

- Cyanide leach of the bulk flotation concentrate is usually problematic metallurgically when the concentrate contains high levels of sphalerite, chalcopyrite and pyrite.
- A second tailing pond is needed to contain the flotation tail which is generated from the flotation of the cyanide leached bulk concentrate. This flotation tail will contain a small amount of cyanide and will generate acid over time.
- A second flotation circuit is needed to recover the lead/silver concentrate and zinc/silver concentrate by floating the cyanide leached bulk flotation concentrate.
- There are two different types of process water. The first one is for the bulk flotation and the second one is for the cyanide leach and selective flotation of the cyanide leached bulk flotation concentrate. The second process water will contain a small amount of cyanide. If the process water in the bulk flotation circuit is contaminated with cyanide, gold recovery in the bulk flotation circuit will likely decrease. This is a serious risk.
- The separation among gold/silver, galena, sphalerite and pyrite is expected to be more difficult with the cyanide leached bulk flotation concentrate.

The flowsheet of “whole-ore cyanide leach selective flotation” has the following benefits:

- Overall gold recovery is consistently higher.
- The flowsheet is simpler.
- The impact of process water on gold recovery in the cyanide leach circuit is negligible.
- Only one tailing pond is needed.
- There is only one process water system.

However, a few disadvantages are present with this flowsheet.

- The cyanide leach circuit is larger.
- The cyanide detox circuit is larger.
- All of the final tailing may contain a small amount of cyanide. Potentially, the tailing will generate acid over time. Because of the small amount of cyanide and the potential acid generation, the tailing dam will be expensive to build.

Because the combined in-situ value of gold and silver accounts for about 94% of total value and the high gold recovery is achieved from the whole-ore cyanide leach, the flowsheet of “whole-ore cyanide leach followed by selective flotation of the cyanide leached residue” is more favorable economically and metallurgically. A simple block flow diagram for this flowsheet is shown in Figure 13-16.

Because a significant amount of coarse gold is present, a gravity concentrator will be installed in the grinding circuit to treat a portion of cyclone underflow. The resultant gravity concentrate is then leached with cyanide to dissolve gold and silver. The resultant pregnant leach solution is then sent to an electrowinning circuit. The gold sludge on the cathode will be collected, filtered, dried and smelted with fluxes to produce gold dore. This gold dore will contain a significant amount of silver.

Preg-robbing is expected to be very weak or does not exist. Thus, the gold and silver will be leached first in cyanide solution, and then the dissolved gold and silver will be adsorbed onto the activated carbon in a CIP circuit.

Figure 13-16: Preferred Flowsheet to Produce Gold Dore, Lead/Silver Concentrate and Zinc/Silver Concentrate

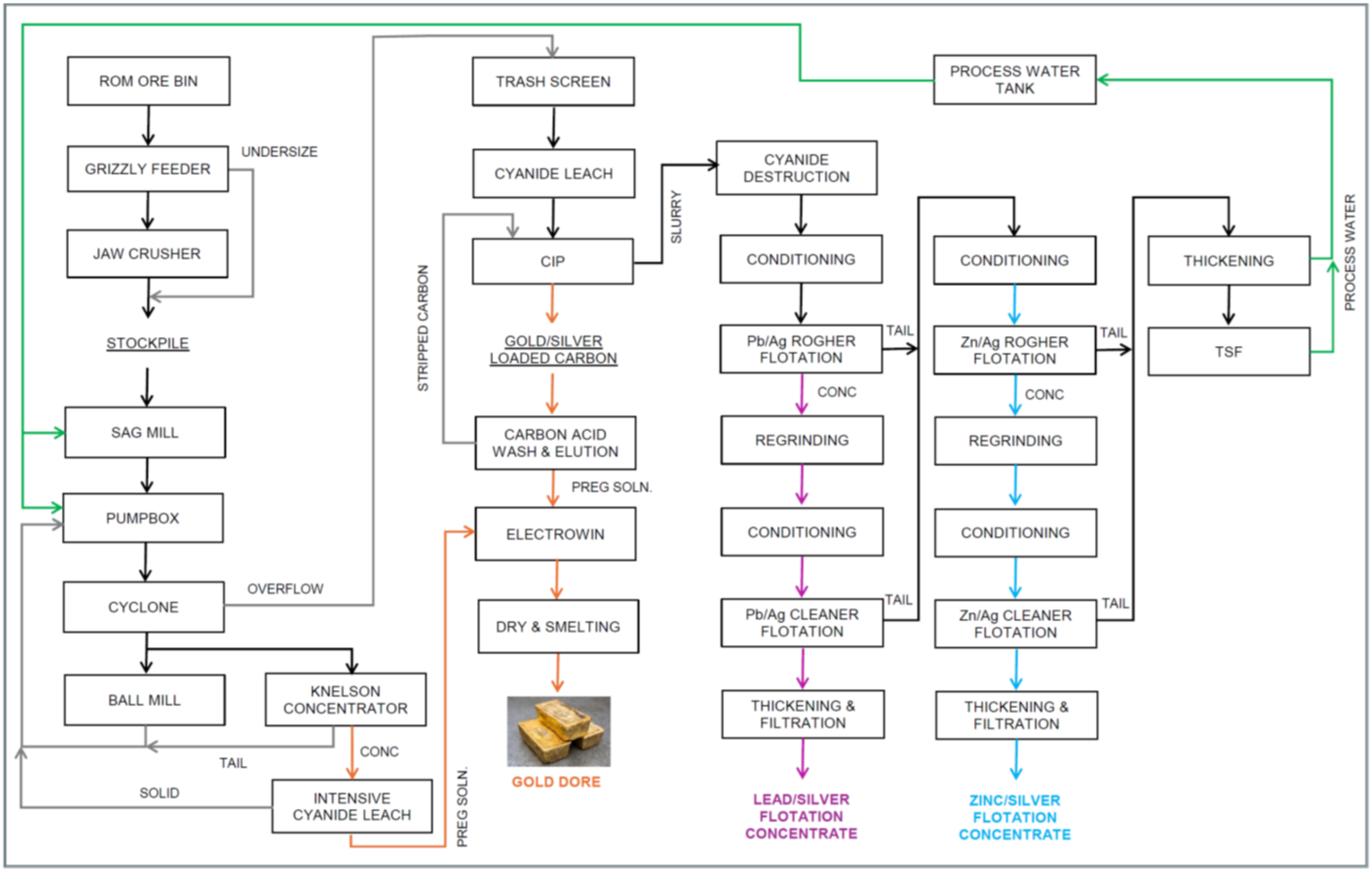


Table 13-13: Forecast of Metal Recoveries and Concentrate Grades

Domain					Camp	Los Cuyes	Seledad (San Jose)	Enma
Gravity + Cyanide Leach	Gold/Silver Dore	Recovery	Gold	%	95.8	88.9	87.1	75.3
			Silver		45	46	75	69
Flotation of Detoxed Cyanide Leached Tail	Silver/Lead Concentrate	Mass Pull		%	38% x Pb (%) / 45%	34% x Pb (%) / 28%	/	/
		Recovery	Gold	% (net)	0.08	0.09	/	/
			Silver		12.2	10.0	/	/
			Lead		38	34	/	/
		Content	Gold	g/t	0.08% x Au (g/t) / M.P. (%)	0.09% x Au (g/t) / M.P. (%)	/	/
			Silver	g/t	12.2% x Ag (g/t) / M.P. (%)	10.0% x Ag (g/t) / M.P. (%)	/	/
			Lead	%	45	28	/	/
	Zinc/Silver Concentrate	Mass Pull		%	59% x Zn (%) / 45%	53% x Zn (%) / 45%	/	/
		Recovery	Gold	% (net)	0.67	0.67	/	/
			Silver		8.6	3.7	/	/
			Zinc		59	53	/	/
		Content	Gold	g/t	0.67% x Au (g/t) / M.P. (%)	0.67% x Au (g/t) / M.P. (%)	/	/
			Silver	g/t	8.6% x Ag (g/t) / M.P. (%)	3.7% x Ag (g/t) / M.P. (%)	/	/
			Zinc	%	45	45	/	/

Note: Au (g/t), Ag (g/t), Pb (%) and Zn (%) refer to the mill feed head grade for gold, silver, lead and zinc, respectively. M.P.(%) stands for the concentrate mass pull

The gold/silver-loaded carbon from the CIP circuit will be harvested periodically, acid washed and then eluted. The resultant eluate will be sent to an electrowinning circuit to reduce the dissolved gold/silver onto the cathode as metallic gold/silver. The sludge on the cathode will then be collected, filtered, dried and smelted with fluxes to produce the gold/silver doré.

The tailing from the CIP circuit will be treated in a cyanide destruction circuit to reduce the amount of excess cyanide and to decrease the pH before the lead/silver concentrate is recovered via selective flotation. As an alternative, the CIP tailing may be thickened first before cyanide destruction so that a portion of residual cyanide in the CIP tailing can be recovered and recycled.

Depressants and collectors will be selected properly to selectively float the lead/silver while as much zinc (sphalerite) and pyrite are rejected. In order to effectively upgrade the lead/silver rougher concentrate to a high-grade product attractive to sell on the market, the regrinding will likely be required for the lead/silver rougher concentrate.

The tailing from the lead/silver flotation circuit will be conditioned with the addition of lime to depress pyrite and then with the addition of copper sulfate to activate zinc (sphalerite). The subsequent zinc/silver flotation will be carried out at a high pH so that majority of pyrite is rejected. The regrinding will likely needed to the zinc/silver rougher concentrate before it is upgraded to a high-grade product attractive to sell on the market.

After the lead/silver concentrate and zinc/silver concentrate are produced, the tailing will be thickened and/or filtered before it is disposed of at a properly designed/constructed TSF.

The process water from the tailing thickener overflow and reclaimed from the TSF will be recycled to the grinding circuit and other circuits in the process plant as a dilution water.

Table 13-13 shows the expected metal recoveries and concentrate grades based on this desirable flowsheet. The combined gold recoveries from gravity concentration and cyanide leach are reasonably reliable based on the available testwork data. However, those recoveries and concentrate grades related to the lead/silver concentrate and the zinc/silver concentrate are approximate and thus should be treated with caution due to inadequate amount of available testwork data. A series of flotation tests are required to verify and improve these flotation performance estimates.

13.12 Conclusions and Recommendations

Gold, silver, lead (galena) and zinc (sphalerite) are four valuable components for the Condor project. At the average mill feed head grades of 2.15 g/t gold, 14.2 g/t silver, 0.06% lead and 0.54% zinc, the in-situ values for each tonne of mill feed at metal prices of US$3,500/oz gold, US$40/oz silver, US$0.90/lb lead and US$1.35/lb zinc are US$242/tonne for gold, US$18/tonne for silver, US$1/tonne for lead and US$16/tonne for zinc. These numbers indicate that gold and silver represent 87.2% and 6.6% of total in-situ value, respectively. Therefore, it is necessary to maximize the recoveries of gold and silver into the doré product.

Based on the available metallurgical testwork data, a few important observations can be summarized below:

- The gold is generally free milling. The whole-ore cyanide leach achieved gold recovery on the order of 96% for the Camp domain, 89% for the Los Cuyes domain, 87% for the Soledad domain and 75% for the Enma domain.

- The whole-ore cyanide leach results in poor silver recovery on the order of 45% for the Camp domain, 46% for the Los Cuyes domain, 75% for the Soledad domain and 69% for the Enma domain. Most of the unrecovered silver is probably associated with galena.
- The results of gravity concentration testwork show a significant amount of gravity recoverable gold around 34% for the Camp domain, but a less amount of gravity recoverable gold (23%) for the Los Cuyes domain and a further less amount of gravity recoverable gold (5%) for the Enma domain.
- Because gold and silver account for about 94% of total in-situ value in the mill feed, the flowsheet of gravity concentration followed by cyanide leach is preferred so that the final will be the gold/silver doré.
- Subject to the metal prices and operating cost, the remaining gold, silver, lead and zinc in the cyanide leached residue may be recovered by selective flotation. Although further flotation testwork is needed, the completed testwork has indicated that the marketable lead/silver concentrate and zinc/silver concentrate can be produced.
- The mineralized materials have a moderate hardness and a low abrasion property.
- As with the high gold recovery achieved from the whole-ore cyanide leach, the bulk flotation also resulted in the high gold recovery.
  - For the Camp domain, average gold recovery was 97.5% at 14.2% concentrate mass pull. Average silver recovery was 95.9%.
  - For the Los Cuyes West domain, average gold recovery was 94.5% at 12.5% concentrate mass pull. Average silver recovery was 89.3%.
  - Because of the high sulfide (pyrite) content, the bulk flotation will not generate a high-grade gold concentrate attractive to sell. Nevertheless, the bulk flotation concentrate is amenable to cyanide leach with gold recovery on the order of 94% for the Camp domain and 93% for the Los Cuyes West domain. Thus, the net gold recovery is 97.5% x 94% = 91.7% for the Camp domain, and 94.5% x 93% = 87.9% for the Los Cuyes West domain. If the bulk flotation concentrate is reground prior to cyanide leach, the gold recovery from cyanide leach may be higher. For the Camp domain, this net gold recovery is about 4% lower than the whole-ore cyanide leach.
  - The cyanide leached residue was tested for selective flotation to generate the lead/silver concentrate and zinc/silver concentrate. Although further flotation testwork is needed, it appears that the marketable lead/silver concentrate and zinc/silver concentrate can be produced by floating the cyanide leached flotation concentrate.
  - The flowsheet of “bulk flotation followed by cyanide leach of the flotation concentrate” is an alternative to the whole-ore cyanide leach. This alternative would be attractive in a situation where it is problematic and expensive to dispose of the cyanide leached tailing.

Although the completed metallurgical testwork has demonstrated that the mineralized materials from the Condor project are amenable to the whole-ore cyanide leach and to the bulk flotation, further investigations are needed to maximize the gold/silver recoveries and to generate a series of process parameters necessary for engineering design of the process plant.

- More representative samples from each domain should be selected for the comminution testing, including the crusher work index, SMC or drop weight test, rod mill work index, ball mill work index and abrasion index.

- The single-stage gravity concentration at grind size of 80% passing 210 µm was exclusively carried out. The multi-stage gravity concentration testwork is strongly recommended for each domain. The gravity recoverable gold will likely increase with the multi-stage gravity concentration. After the testwork data from the multi-stage gravity concentration are obtained, a series of simulations are recommended to forecast the expected gold recovery from the future commercial operations.
- For the whole-ore cyanide leach, optimization testwork is recommended to fine tune the operating conditions, including the pulp density, grind size, cyanide concentration, pH, dissolved oxygen and retention time. Previous cyanide leach testwork showed a weak preg-robbing with some materials, and this should be verified by carrying out the parallel CIL cyanide leach and CIP cyanide leach, and then compare their gold recoveries.
- The adsorption behavior of dissolved gold and silver on the activated carbon should be determined using the actual pregnant leachate. The dissolved silver does not adsorb strongly on the activated carbon and thus some of the dissolved silver may be lost to the CIP tail in the future commercial operation. Also, some dissolved copper and zinc are present in the pregnant leachate, and they may adversely impact the loading of gold and silver on the activated carbon. When the process water is recycled, the dissolved copper and zinc will build up in the process water.
- After representative CIP tail samples become available, the continuous cyanide destruction testwork is recommended.
- The cyanide leach tail after cyanide destruction has never been tested for the sequential selective flotation to generate the lead/silver concentrate and zinc/silver concentrate. The oxidizing nature during cyanide destruction may deteriorate the subsequent flotation performance. Although the economic contribution by these two flotation concentrates will be marginal, a series of flotation tests are needed to verify the marketable lead/silver concentrate and zinc/silver concentrate can indeed be produced cconsistently. Previous testwork data showed the concentrates produced from the Los Cuyes West domain contained high levels of arsenic, cadmium and antimony. The assays of these penalty elements plus mercury, chloride and fluoride, etc should be repeated when the representative flotation concentrates become available.
- Because of the high sulfide (pyrite) content, the cyanide leached tail may generate acid in the tailing pond when the sulfide minerals are oxidized over time. This potential acid generation may remain even after the cyanide leached tail is floated again to produce the lead/silver and zinc/silver concentrate. Therefore, several environmental tests, including ABA, SPLP, TCLP, column leach and humidity cell, are recommended for the representative tail samples.
- The mineralized material from the Soledad (San Jose) domain seems acidic in-situ. As a result, the in-situ pH of all future mineralized samples should be measured. The in-situ acidity will cause some corrosion issue to the mining equipment and process equipment.
- As for the cyanide leach of the bulk flotation concentrate, gold and silver recoveries will likely increase if the bulk flotation concentrate is reground. Such testing is recommended. Also, the addition of lead nitrate to the cyanide leach of flotation concentrate may be beneficial to gold recovery, and thus, some testing is recommended.
- The thickening and filtration testwork for the final tail, lead/silver concentrate and zinc/silver concentrate are required for the engineering design of the process plant.

14 Mineral Resource Estimates

14.1 Introduction

The Mineral Resource Statement presented herein represents the second Mineral Resource evaluation prepared for the Condor project in accordance with the Canadian Securities Administrators' National Instrument 43-101. Previous mineral resources were generated in 2018, and later restated with updated metal prices in 2020.

The Mineral Resource model prepared by SRK considers 343 core boreholes drilled during the period of 1994 to 2023. The resource estimation work was completed by Mark Wanless (Pr.Sci.Nat, FGSSA), and Ms. Yanfang Zhao (MAusIMM), who are Principal Geologists from SRK. Mr. Mark Wanless, an appropriate independent Qualified Person as this term is defined in National Instrument 43-101 is Registered with the South African Council for Natural Scientific Professionals as Pr.Sci.Nat, 400178/05, Fellow of the Geological Society of South Africa, Member of the Geostatistical Association of South Africa and a Member of the South African Institute for Mining and Metallurgy (SAIMM). The effective date of the Mineral Resource Statement is November 30, 2025.

This section describes the resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold, silver, lead, zinc Mineral Resources found in the Condor project at the current level of sampling. The Mineral Resources have been estimated in conformity with generally accepted CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines* (November 2019) and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve.

The Mineral Resource estimates contained herein were utilized in preparing the PEA, however the PEA is preliminary in nature and includes Inferred Mineral Rresources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.

The database used to estimate the Condor project Mineral Resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold mineralization and that the assay data are sufficiently reliable to support Mineral Resource estimation.

Leapfrog® Geo software was used to create lithology models, grade shells and estimation domains for all four of the deposits. Leapfrog® Edge was used to prepare assay data for geostatistical analysis, construct the block model, estimate metal grades, and tabulate Mineral Resources at Camp and Enma. Isatis.Neo was used to prepare assay data for geostatistical analysis, construct the block model, estimate metal grades, and tabulate Mineral Resources at Soledad and Los Cuyes.

14.2 Resource Estimation Procedures

The Mineral Resource evaluation methodology involved the following procedures:

- Database compilation and verification
- Geological interpretation for estimation domain
- Definition of resource domains
- Data conditioning (compositing and capping) for geostatistical analysis and variography
- Block modelling and grade interpolation
- Resource classification and validation
- Assessment of “reasonable prospects for eventual economic extraction” (“RPEEE”) and selection of appropriate AuEq cut-off values for potential open pit Mineral Resources fore Soledad and Enma and underground for Camp and Los Cuyes
- Preparation of the Mineral Resource Statement

14.3 Resource Database

The data provided for the Condor Project include the database in CSV format, including the collar locations, downhole survey results, geologic information, SG data, assay, grade shell wireframes, lithology wireframes, QAQC data, block models in csv format, Topo file, etc., and Resource Database summary is presented in Table 14-1, and the drillhole locations are shown in Figure 10-1.

Table 14-1: Resource Database Summary for the Condor Project

Deposit	Number of Drill Holes	Total Length of Drilling (m)	Total Length of Samples in Drilling (m)	Number of samples
Camp	56	27,805	15,803	24,022
Los Cuyes	44	7,613	3,980	7,576
Soledad	117	37,513	23,607	36,263
Enma	126	20,725	12,888	20,419

Notes: The summary has been sourced by SRK from the database provided by Silvercorp.

Subsequent to the compilation of the Mineral Resource database, Silvercorp has drilled an additional six holes at the Los Cuyes deposit as described in chapter 10.1.3. The six holes were aimed to intersect the modelled shear zones within the relatively densly drilled areas, with two holes (CU25-35 and CU25-32) designed to text the lateral extension of some of the NW shears in realtively poorly drilled areas. The holes were generally very successful at intersecting the shear zones at the modelled locations, with elevated grades intersected where modelled. These holes generally confirm the existing interpretation.

14.4 Domain Modelling

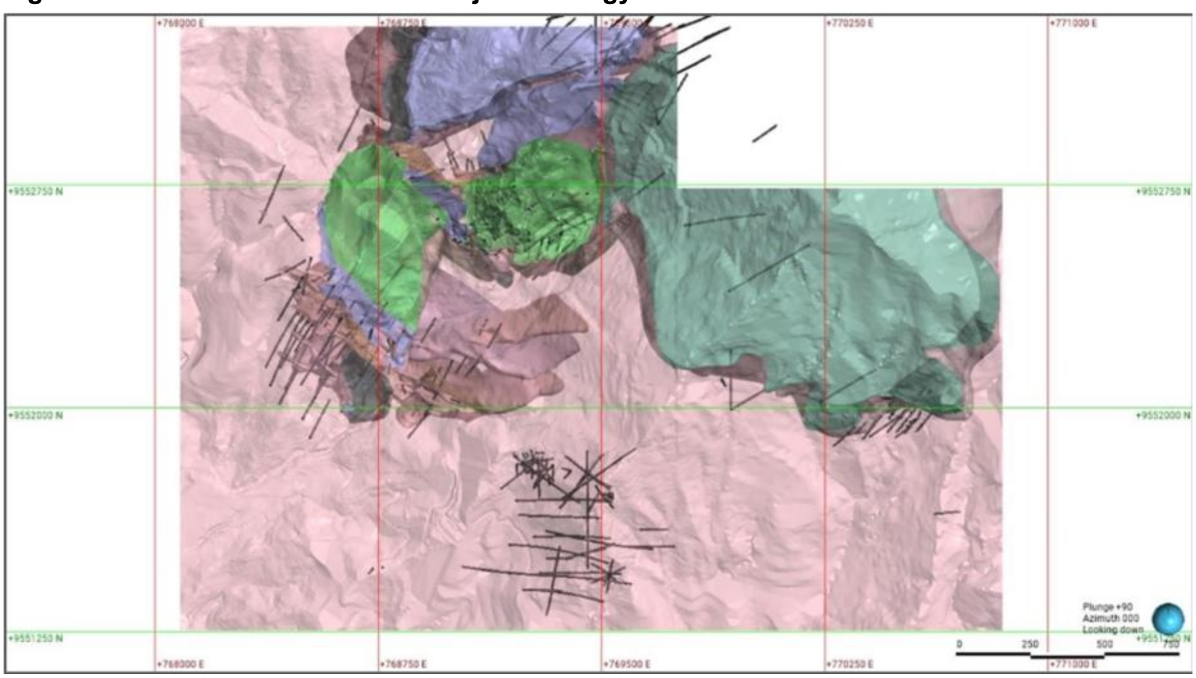
14.4.1 Camp

The Camp deposit host rock types are a series of intrusive units. The structure and geology model were created by Silvercorp using a combination of geological logs and surface mapping (see also Figure 7-4). The primary rock types included in the model are:

- Rhyolite
- Vent Rhyolitic welded tuff
- Granodiorite
- Andesite-Dacite
- Diorite
- Greenstone
- Rhyodacite

Figure 14-1 shows a plan view of the surface geology below the saprolite weathering. There is a northeast-striking fault (Piedras Blancas Fault) on the southeast side of the deposit that appears to cutoff or bound mineralization.

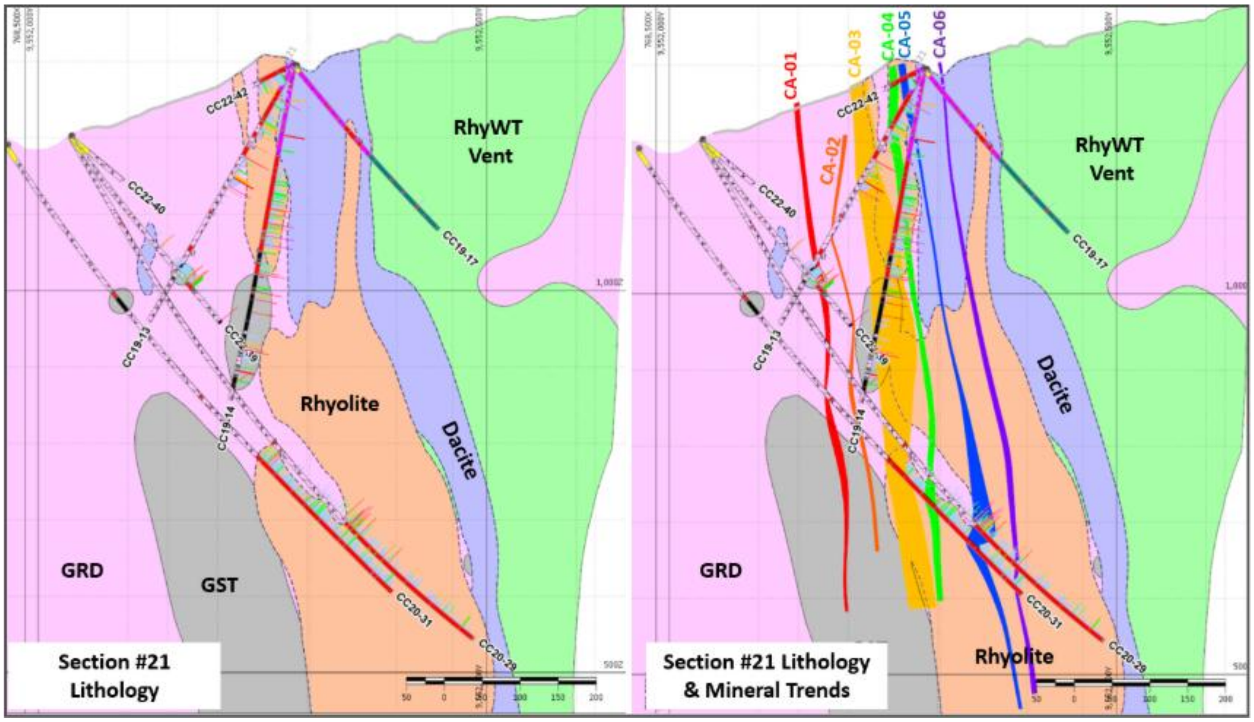
Figure 14-1: Plan View of Condor Project Geology at Surface



Source: Silvercorp, 2024

The gold, silver, copper, and zinc mineralization is not confined by rock type, but there are distinct grade zones that form relatively cohesive vein-like geometries that run parallel to the footwall of the rhyolite. A total of six major mineralization trends were defined and named CA-01 to CA-06 by Silvercorp with a clear relation to Rhyolite intrusions through veins-stringers, dissemination, and contacts (Figure 14-2).

Figure 14-2: Cross Sections of the Veins CA-01 to CA-06 looking Northwest



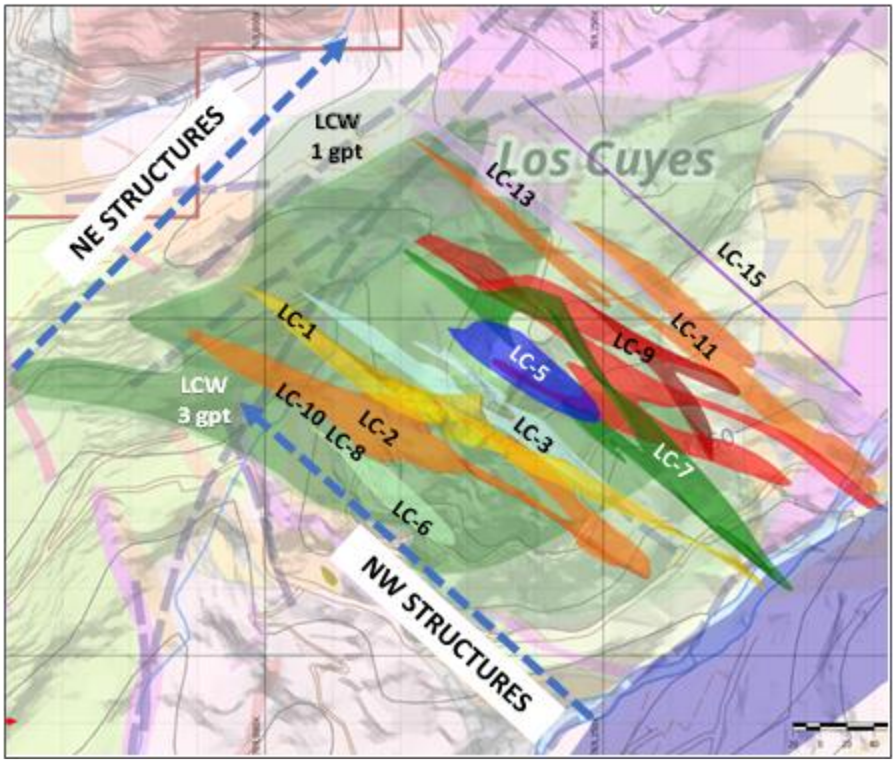
Source: Silvercorp, 2024

14.4.2 Los Cuyes

The Los Cuyes mineralization is not directly lithologically controlled but is focused around the rhyolite lapilli tuff vent in a series of shear structures and a lower grade disseminated halo of mineralization. Two faults play an important role in the mineralization, the northeast striking Piedras Blancas fault, which cuts off the mineralization to the southeast, and the northeast striking Los Cuyes west (LCW) structure in the northwest near the contact between the rhyolite lapilli tuff and the granodiorite.

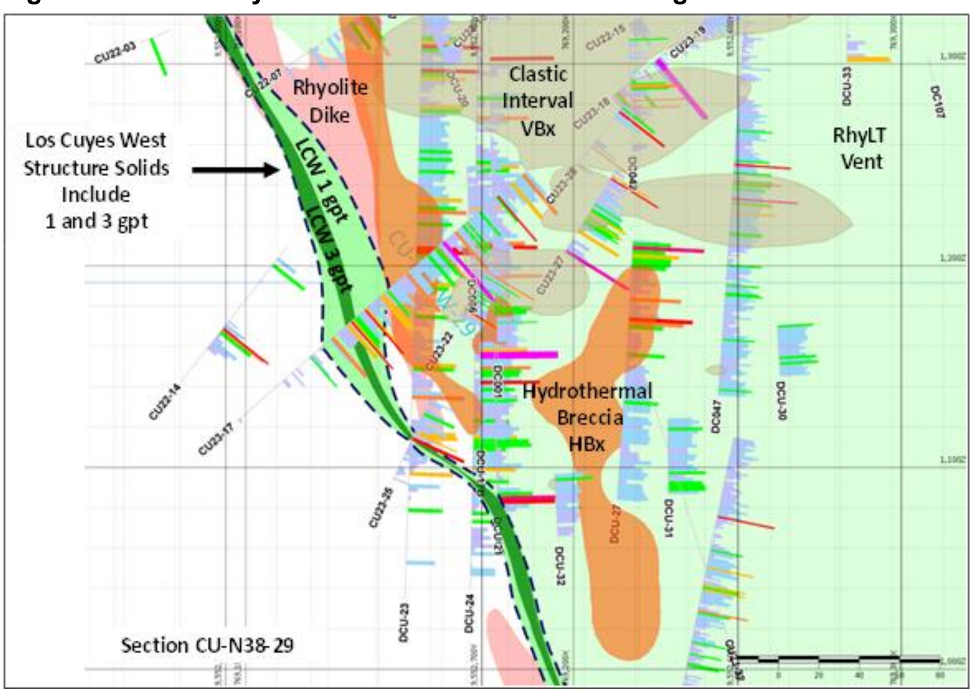
Within the rhyolite lapilli tuff twelve north west striking shears which dip steeply to the northeast have been modelled (Figure 14-3). These NW striking structures are cut off against the LCW structure which also hosts significant mineralization (Figure 14-4). A lower grade halo of disseminated mineralization has been modelled surrounding these higher-grade shears, also constrained by the bounding structures, and predominantly within the rhyolite lapilli tuff.

Figure 14-3: Plan View of the Los Cuyes Shear Hosted Mineralization Models



Source: Silvercorp, 2024
Notes: NW structures modelled using a 3 g/t gold cutoff, the LCW mineralization is modelled above a 1 g/t gold cutoff.

Figure 14-4: Los Cuyes Vertical Cross Section looking Northwest



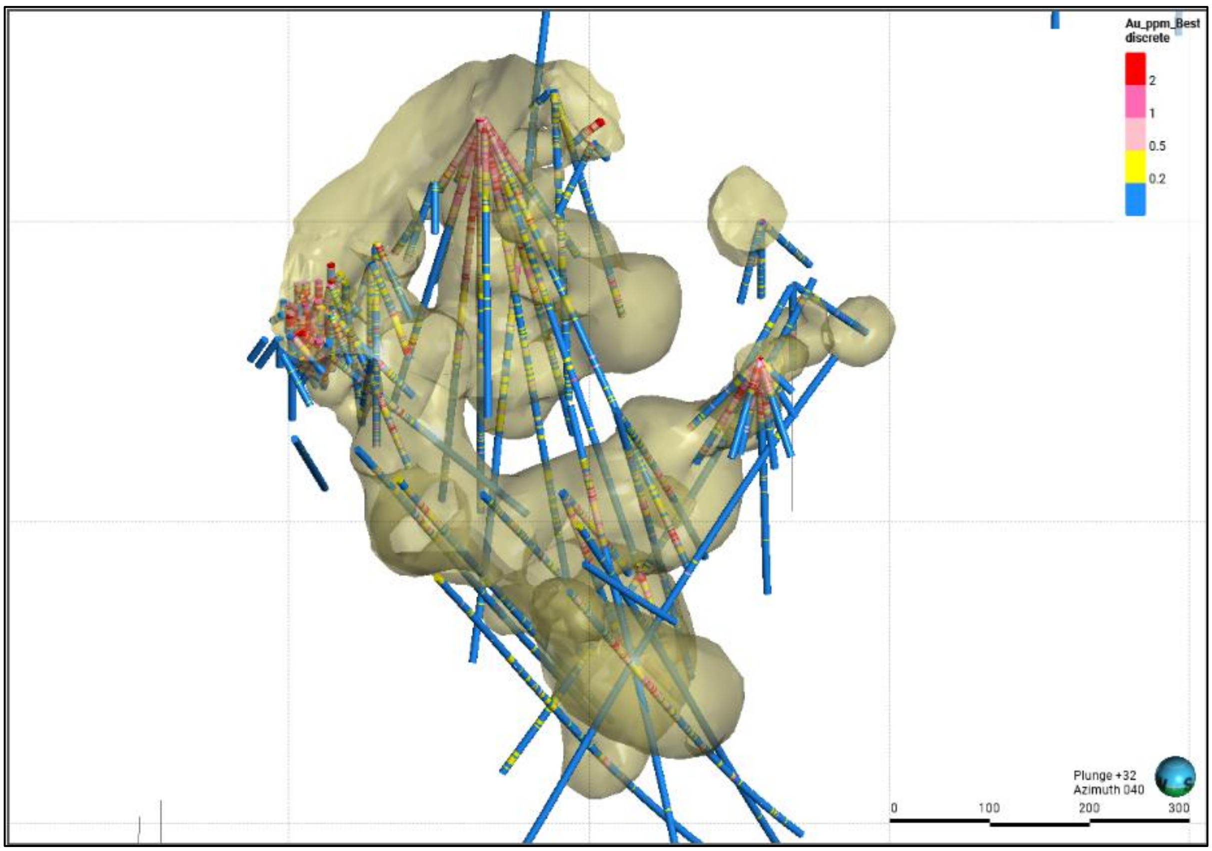
Sources: Silvercorp, 2024
Notes: Only the 1 g/t LCW shell is used in the estimates.

14.4.3 Soledad

Mineralization at Soledad is related to a felsic (rhyolitic) diatreme intrusion and associated breccias. No detailed lithological model was created for this area. The drilling database contains underlying geologic information, including lithology code designations derived from observations during core logging. Series of grade shell domains were interpreted for zones of continuous mineralization a set of intervals of Au g/t. At the time of the previous resource statement, a detailed geological interpretation was not available. A 0.2 g/t Au grade shell was chosen to constrain the mineralization, based on an assessment of several grade shell intervals, assessing the continuity of the mineralization (Figure 14-5).

The lack of understanding of the geological controls on the mineralization will limit the estimation confidence a grade shell based on 0.2g/t Au Cutoff was generated by SRK Leapfrog® Geo and Edge and used for the estimation domain of Soledad (Figure 14-5).

Figure 14-5: Soledad Deposit 0.2 g/t Constraining Gold Grade Shell



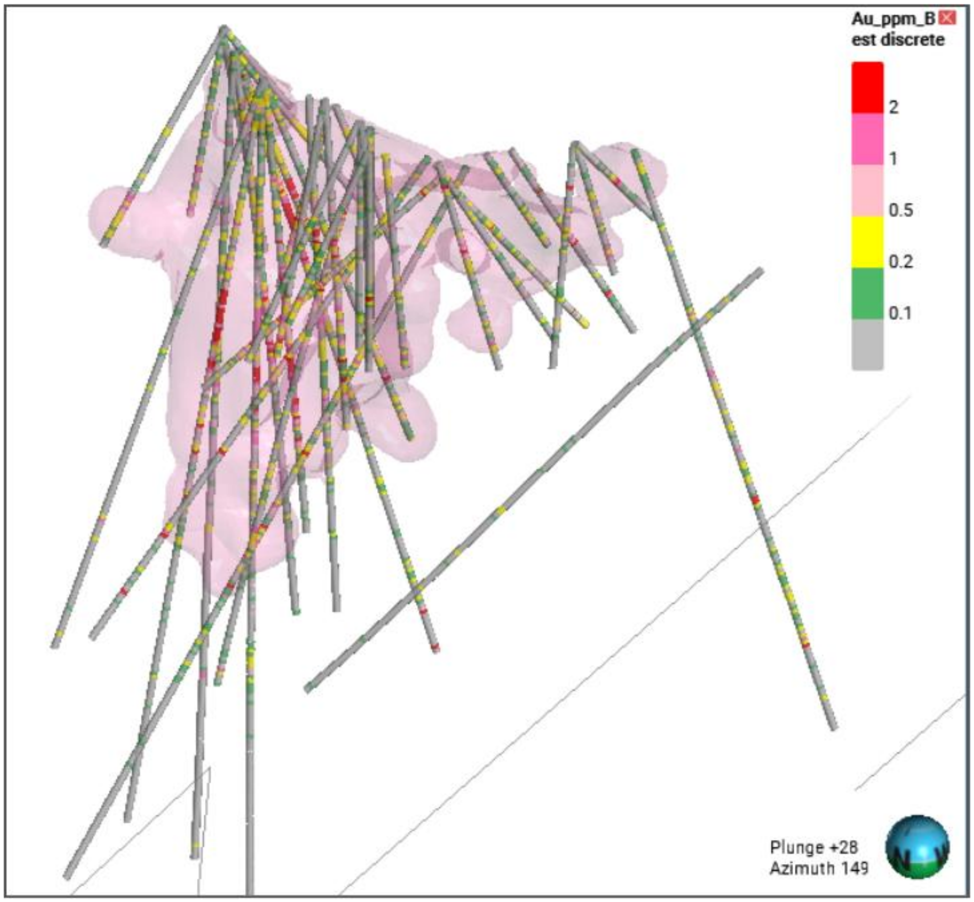
Source: Silvercorp, 2024

14.4.4 Enma

Gold and silver mineralization at Enma is hosted in a west-northwest-trending rhyolitic breccia that occurs at the contact between andesite lapilli tuffs and the Zamora batholith. No detailed lithological model was created for this area. The deposit has dimensions of 280 m west-northwest, is approximately 20-75 m wide, and has a vertical extent of 350 m.

Similar to Soledad, a grade shell based on 0.1g/t Au Cutoff and ISO of 0.5 was generated by SRK Leapfrog® Geo and Edge and used for the estimation domain of Enma (Figure 14-6).

Figure 14-6: Enma Deposit 0.1 g/t Constraining Gold Grade Shell



Sources: Silvercorp data and SRK Model

14.5 Specific Gravity

Specific gravity (SG) data is only available for drill holes in the Los Cuyes and Camp areas. SG measurements are determined using the water immersion method (weight in air versus weight in water). SG measurements are undertaken on whole pieces of core spaced at approximately 10 m intervals down each drill hole.

Table 14-2 summarizes the density data available per simplified logged lithology units within the Camp and Los Cuyes areas. These average densities are applied to the block model for each modelled lithology unit which covers all four of the deposits.

Table 14-2: Density Data for Camp and Los Cuyes per Lithology Code

	Los Cuyes		Camp	
Lithology	**Count**	**Average SG**	**Count**	**Average SG**
Dacite	329	2.74	275	2.69
Granodiorite	308	2.74	1,240	2.70
Greenstone	34	2.75	315	2.85
Rhyodacite	57	2.64	568	2.61
Rhyolite lapilli tuff	461	2.63	-	-
Rhyolite North West	97	2.65	2	2.58
Rhyolite welded tuff	6	2.68	83	2.64

Sources: Silvercorp
Notes: Some simplifications of lithology units have been undertaken by SRK for average density calculations.

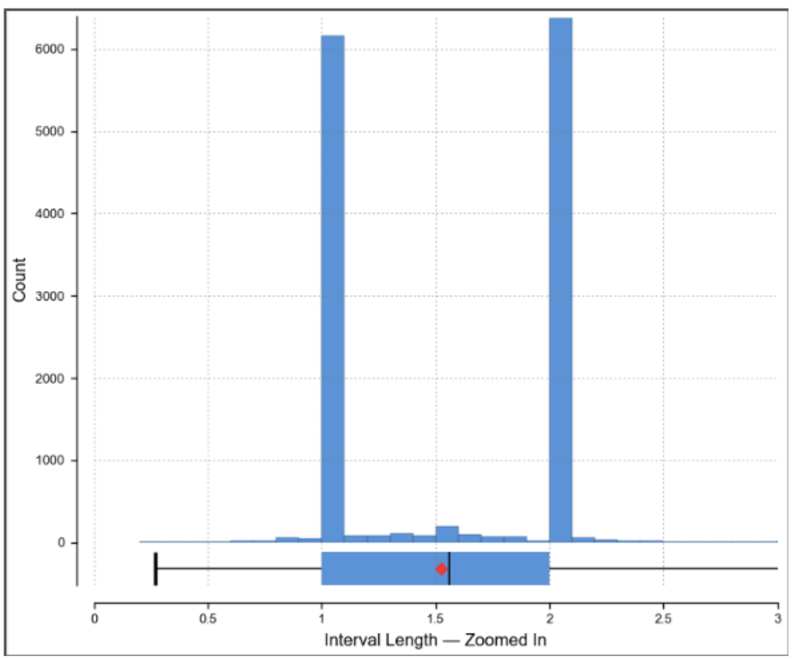
14.6 Compositing

Compositing the drill hole samples helps standardize the database for further statistical evaluation. This step ensures that the data has consistent support and can aid in reducing the high variance that may be introduced through short samples.

14.6.1 Camp

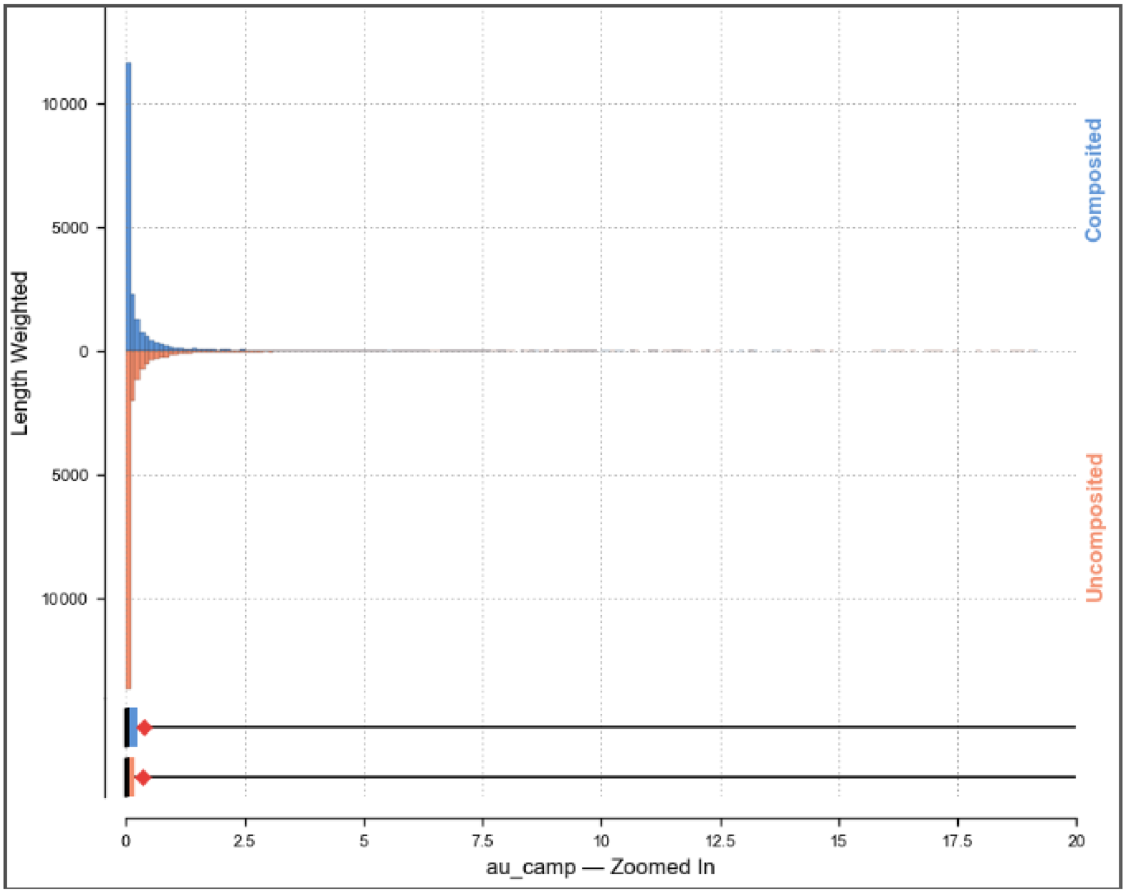
Within the assay database of Camp, 45% of intervals are 1 m long, and 47% are 2 m long (Figure 14-7).

Figure 14-7: Interval Length Histogram for the Camp Deposit



A composite length of 2 m and minimum coverage of 50% was selected for Camp. Composites were created within the mineralization wireframe domains beginning at the upper contacts. The intersection thickness encountered by any given drill hole, however, is not an even multiple of the composite length. If the remaining length was less than 1 m, the composite was distributed equally. The elimination of the small composites did not affect the overall integrity of the composited database. The compositing of samples before and after does not affect the overall distribution of the samples (Figure 14-8).

Figure 14-8: Gold Grades Before and After Compositing (Camp)



The average grades of composite datasets of each domain are shown in Table 14-3.

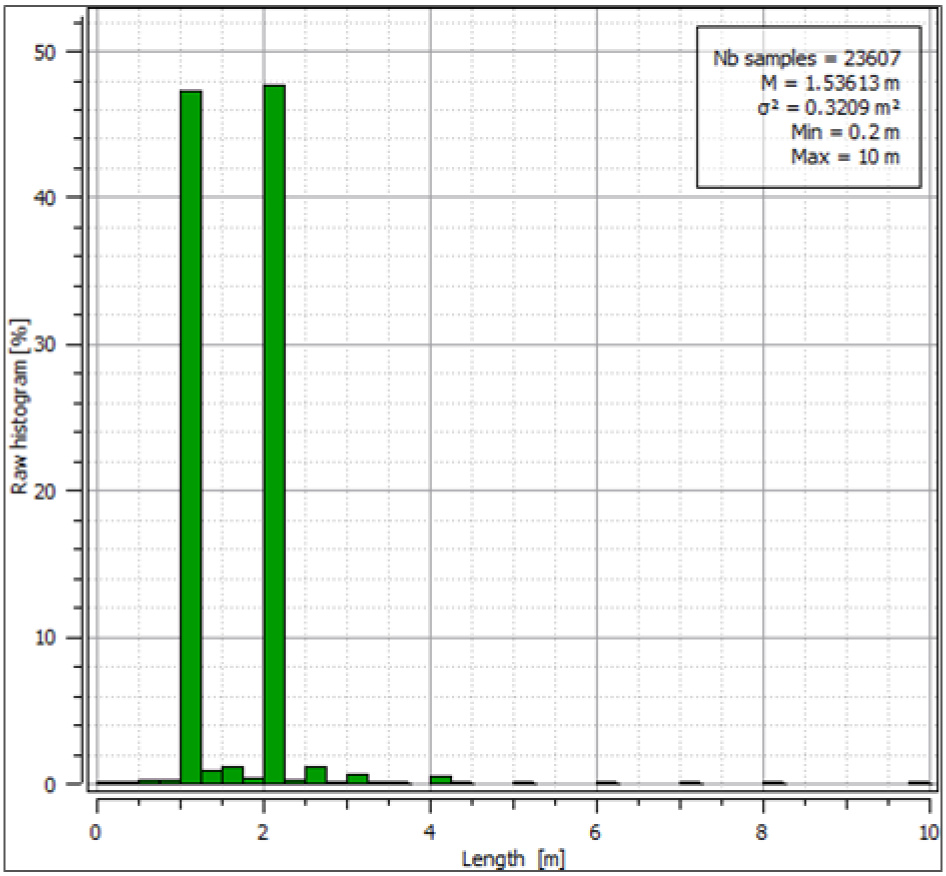
Table 14-3: Camp Composites for Each Domain

Domain	Count	Ag g/t	Au g/t	Cu %	Pb %	Zn %	As ppm	S %
CA-01	101	22.53	2.31	0.02	0.08	0.76	163.96	2.73
CA-02	77	7.00	1.18	0.01	0.04	0.44	38.65	1.65
CA-03	1,266	13.06	1.54	0.02	0.05	0.53	113.19	2.96
CA-04	188	13.42	1.38	0.02	0.05	0.60	116.50	3.04
CA-05	198	14.94	1.40	0.02	0.07	0.52	96.93	2.60
CA-06	159	9.41	0.70	0.01	0.02	0.17	43.66	1.24

14.6.2 Los Cuyes

The Los Cuyes domains can be separated into the shear style domains and the surrounding disseminated halo domain. The consideration of the appropriate composite length is different for each of these since the dimensions of the domains are significantly different. For the shear domains the dimensions are similar to that discussed for camp, while for the disseminated halo the mineralization dimensions do not impact the choice of composite length. As with Camp the sample length distribution at Los Cuyes is bimodal with common values of 1 m (47% of samples) and 2 m 47% of samples with a minor population of variable samples as shown in Figure 14-9.

Figure 14-9: Interval Length Histogram of Los Cuyes



SRK undertook a composite optimization considering a range of composite lengths for each shear and halo domain. For the shear domains composite lengths of longer than 2 m resulted in a number of composites shorter than the target length due to the dimensions of the mineralized domain and did not materially reduce the coefficient of variation. For the shear domains a composite length of 2 m was selected. For the halo domain a length of 3 m was selected as the coefficient of variation stabilised for composite lengths above this value. The compositing did not materially affect the average grades as is illustrated in Table 14-4 which shows the sample and composite Au g/t values, along with the value of the remaining residual samples at the margins of the domains.

Table 14-4: Los Cuyes Domain Sample and Composite Au g/t Grades with Residual Sample Grades

Domain	Sample	Composite	Residual
LCW_1gpt	3.14	3.14	11.04
NW01_3gpt	2.81	2.81	1.79
NW9_3gpt	5.89	5.89	9.51
NW7_3gpt	3.20	3.20	3.06
NW3_3gpt	11.30	11.30	3.92
NW2_3gpt	2.83	2.82	4.05
NW5_3gpt	8.52	8.52	4.53
NW10_3gpt	4.09	4.09	2.28
NW8_3gpt	5.17	5.17	3.12
NW6_3gpt	6.42	6.42	27.90
NW13_3gpt	6.61	6.61	7.54
NW11_3gpt	4.04	4.04	6.47
NW15_3gpt	6.73	6.74	2.93
Halo	0.65	0.65	0.62

As the halo domain residual value is not materially different to the mean for the domain, the residual sample was ignored. For the shear domains however the residual value can be significantly different to the composite values. Therefore for the shear domains the residual composite was merged with the previous composite to ensure the grade is not biased.

The declustered average grades of the 2 m and 3 m composite datasets are shown in Table 14-5.

Table 14-5: Los Cuyes Declustered Average Values for Estimated Variables in Each Domain

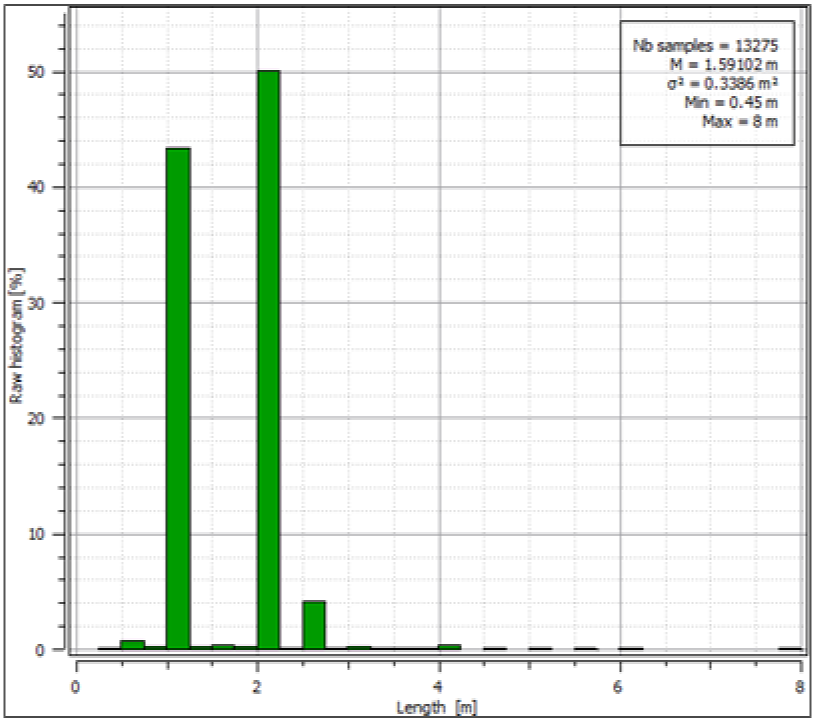
Domain	Count	Ag g/t	Au g/t	Cu %	Pb %	Zn %	Count[1]	As ppm	S %
LCW	246	27.7	5.43	0.04	0.24	0.78	158	521	4.46
NW1	245	36.6	2.76	0.06	0.12	0.49	66	2,670	4.41
NW9	66	18.2	8.93	0.05	0.06	0.38	36	51	3.13
NW7	40	23.0	3.25	0.04	0.08	0.31	21	89	3.37
NW3	28	43.4	10.42	0.1	0.21	0.97	12	1,152	6.85
NW2	71	14.8	5.20	0.03	0.07	0.78	41	41	2.01
NW5	9	63.6	7.68	0.06	0.11	1.1	3	1,692	6.3
NW10	5	30.3	3.89	0.01	0.51	0.54	2	91	0.8
NW8	12	28.5	5.37	0.02	0.67	0.47	6	107	2.73
NW6	3	30.8	10.00	0.01	0.11	0.36	3	181	6.52
NW13	7	55.7	6.24	0.04	0.07	1.22	0	-	-
NW11	13	19.9	5.26	0.02	0.05	0.35	0	-	-
NW15	4	35.7	5.68	0.03	0.15	0.85	0	-	-
Halo[2]	3,459	6.0	0.69	0.02	0.02	0.24	842	33	2.14

[1] As and S are not assayed for all drillholes resulting in different numbers of composites.
[2] Composite length of 3 m.

14.6.3 Soledad

A similar approach of compositing at Los Cuyes was done at Soledad for the disseminated halo domain, since the domain definition using a grade shell has defined a similar kind of domain. The distribution of sample length is similarly grouped around 1 m (44%) and 2 m (50%) samples (see Figure 14-10).

Figure 14-10: Interval Length Histogram of Soledad



A 2-m composite length was chosen for Soledad based on the composite optimisation results. Table 14-6 shows that the compositing did not impact the mean grades. As the residual samples were quite different, they were merged with the previous composite.

Table 14-6: Soledad Sample and Composite Grades with Residual Sample Grades

Variable	Sample	Composite	Residual
Ag_ppm	6.96	6.97	2.5
As_ppm	43.99	43.99	71.2
Au_ppm	0.96	0.96	0.19
Cu_pct	0.02	0.02	0.01
Pb_pct	0.05	0.05	0.02
S_pct	2.34	2.34	2.06
Zn_pct	0.51	0.51	0.19

Table 14-7 describes the declustered average grades of the 2-m composite data.

Table 14-7: Soledad Declustered Average Values for Estimated Values

Domain	Count	Ag g/t	Au g/t	Cu %	Pb %	Zn %	Count[1]	As ppm	S %
All	5276	6.7	0.59	0.02	0.05	0.38	349	45	2.18

[1] As and S are not assayed for all drillholes resulting in different numbers of composites.

14.6.4 Enma

Within the assay database, the average sample length is 1.9 m, about 11% of samples are 1 m long, and 83% are exactly 2 m long (Figure 14-11). A composite length of 2 m and minimum coverage of 50% was selected for Enma. The intersection thickness encountered by any given drill hole, however, is not an even multiple of the composite length. if the remaining length was less than 1 m, the composite was distributed equally. The compositing of samples before and after does not affect the overall distribution of the samples (Figure 14-12).

Figure 14-11: Interval Length Histogram for Enma

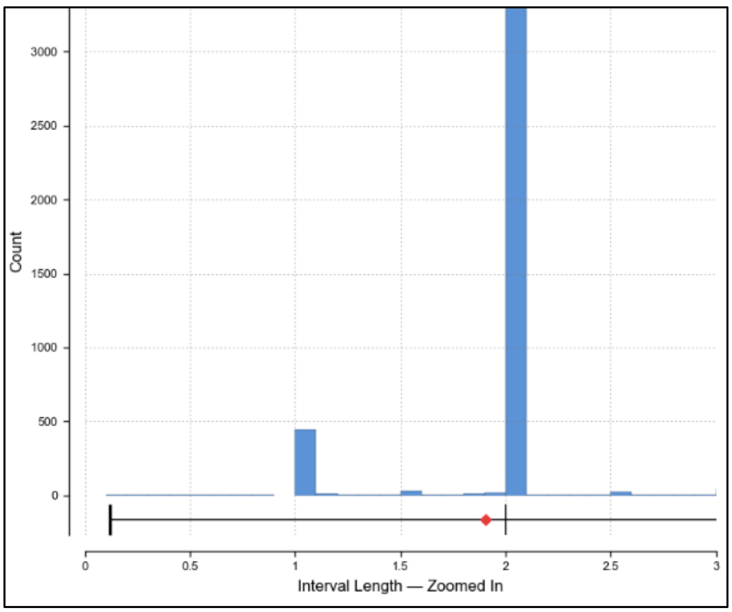
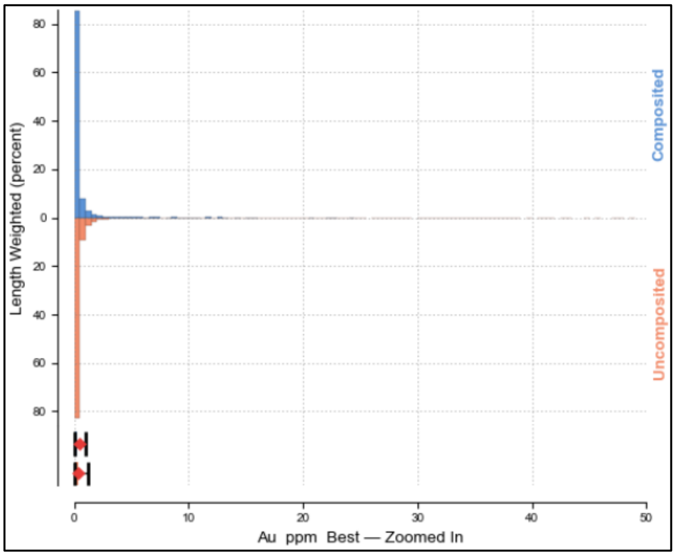


Figure 14-12: Before and After Composition (Enma)



The average grade of the 2 m composite data of Enma are shown in Table 14-8.

Table 14-8: Enma Average Values for Estimated Variables

Domain	Count	Ag g/t	Au g/t	Cu %	Pb %	Zn %	Count[1]	As ppm	S %
Enma	1,615	15.39	1.07	0.02	0.08	0.31	44	188.58	5.36

[1] As and S are not assayed for all drillholes resulting in different numbers of composites.

14.7 Evaluation of Outliers

14.7.1 Camp

Assay capping for the variables was applied after compositing for the mineralized domains of Camp. Capping values were selected based on the visual assessment of the variable histogram. No distance-based capping was determined to be required at Camp. The capping values and the impact of applying outlier capping are presented in Table 14-9.

Table 14-9: Summary of Grade Capping Applied to Camp

Domain Group	Variable	Mean	Cap Value	Capped mean	Composite Count	No Capped	Metal loss %
Camp	Au g/t	1.47	32	1.47	1,816	3	0.25
	Ag g/t	13.29	330	13.11	1,816	1	1.41
	Pb %	0.05	1	0.05	1,816	12	7.43
	Zn %	0.52	5.9	0.51	1,816	8	1.19
	Cu %	0.02	-	-	1,816	-	-
	As ppm	105.72	1800	94.94	1,816	13	10.19
	S %	2.73	-	-	1,816	-	-

14.7.2 Los Cuyes

For each of the variables to be estimated the need for treatment of outlier values was assessed. For deleterious variables such as Arsenic, the assessment and treatment of outlier values can be different to that of the economically valuable variables. The estimation domains are grouped according to the mineralization style for this exercise, the LCW shear, halo, and northwest striking shears (NW shears). An initial assessment of each of the NW shears individually revealed that the distributions were materially similar where there is sufficient data to reasonably assess these.

The assessment considers the variable histogram, normal probability plot, and charts where samples are sorted from low to high and the impact of adding samples one by one to the dataset from low to high on the mean, standard deviation and coefficient of variation. The capping values and the impact of applying outlier capping is summarised in Table 14-10.

Table 14-10: Summary of Grade Capping Applied to Los Cuyes

Domain Group	Variable	Mean	Cap Value	Capped mean	Composite Count	No Capped	Metal loss %
LCW	Au g/t	3.37	40	3.01	246	2	10.62
	Ag g/t	15.21	130	14.46	246	4	4.93
	Pb %	0.09	1	0.08	246	5	8.99
	Zn %	0.52	4	0.50	246	4	4.35
	Cu %	0.04	-	-	246	-	-
	As ppm	195.5	-	-	156	-	-
	S %	3.89	-	-	156	-	-
NW	Au g/t	4.01	40	3.83	503	5	4.57
	Ag g/t	25.23	300	24.06	503	4	4.26
	Pb %	0.1	1.5	0.1	490	4	2.74
	Zn %	0.71	6	0.69	503	5	3.69
	Cu %	0.06	-	-	490	-	-
	As ppm	211.4	-	-	190	-	-
	S %	3.58	-	-	190	-	-
Halo	Au g/t	0.7	10	0.69	3,459	3	1.68
	Ag g/t	6.23	100	6.03	3,459	8	0.23
	Pb %	0.02	-		3,443		
	Zn %	0.24	2	0.24	3,443	8	0.51
	Cu %	0.02	-	-	3,443	-	-
	As ppm	33.08	-	-	842	-	-
	S %	2.14	-	-	842	-	-

Notes: Distance capping is also applied in some instances

For some domains, there remain isolated high values that can have a significant impact on the estimates. In these situations, and additional distance-based capping is applied. Where a composite is above the selected threshold and is also beyond a selected distance from the block, this secondary capping is applied.

The secondary capping is not applied when the composite is closer to the estimating block than the selected distance. For the domains NW2, NW9 and LCW for gold a distance-based capping was also applied of 20 g/t, 20 g/t and 14 g/t respectively. The distance beyond which this capping is applied is 20 m in the plane of mineralization in the case of the NW group of domains, and 50 m for the LCW domain. For the LCW domain for arsenic only a distance-based capping at 1000 ppm with a distance of 20 m was applied.

14.7.3 Soledad

The assessment of outliers used for Los Cuyes was also applied for Soledad. Only three variables had distributions which required capping at Soledad, and the capping parameters and effect are summarised in Table 14-11.

Table 14-11: Summary of Grade Capping Applied to Soledad

Domain Group	Variable	Mean	Cap Value	Capped mean	Composite Count	No Capped	Metal loss %
All	Au g/t	0.96	-		5,276		
	Ag g/t	6.92	60	6.79	5,276	26	1.95
	Pb %	0.05	-		4,711		
	Zn %	0.5	-		5,258		
	Cu %	0.02	0.5	0.02	4,711	10	1.7
	As ppm	44.06	-		349		
	S %	2.34	2.4	0.5	349	2	0.04

No distance-based capping was determined to be required at Soledad. In most instances, the high value composites which might otherwise have required distance-based capping are in densely sampled areas, where there is sufficient data to limit the range of influence of these high value composites.

14.7.4 Enma

Assay capping for the variables was applied after compositing for the mineralized domains of Enma. Capping values were selected based on the visual assessment of the variable histogram, and the capping parameters and effect are summarised in in Table 14-12.

Table 14-12: Summary of Grade Capping Applied to Enma

Domain Group	Variable	Mean	Cap Value	Capped mean	Composite Count	No Capped	Metal loss %
Enma	Au g/t	1.47	32	1.47	1,816	3	0.25
	Ag g/t	13.29	330	13.11	1,816	1	1.41
	Pb %	0.05	1	0.05	1,816	12	7.43
	Zn %	0.52	5.9	0.51	1,816	8	1.19
	Cu %	0.02	-	-	1,816	-	-
	As ppm	105.72	1800	94.94	1,816	13	10.19
	S %	2.73	-	-	1,816	-	-

No distance-based capping was determined to be required at Enma. In most instances, the high value composites which might otherwise have required distance-based capping are in densely sampled areas, where there is sufficient data to limit the range of influence of these high value composites.

14.8 Statistical Analysis and Variography

For each deposit and domain, SRK undertook an assessment of the continuity of each variable, considering the understanding of the geological controls on the mineralization, using tools such as the semi-variogram map, directional semi-variograms, swath plots, histograms and correlation plots to understand the relationships between variables, and their spatial continuity.

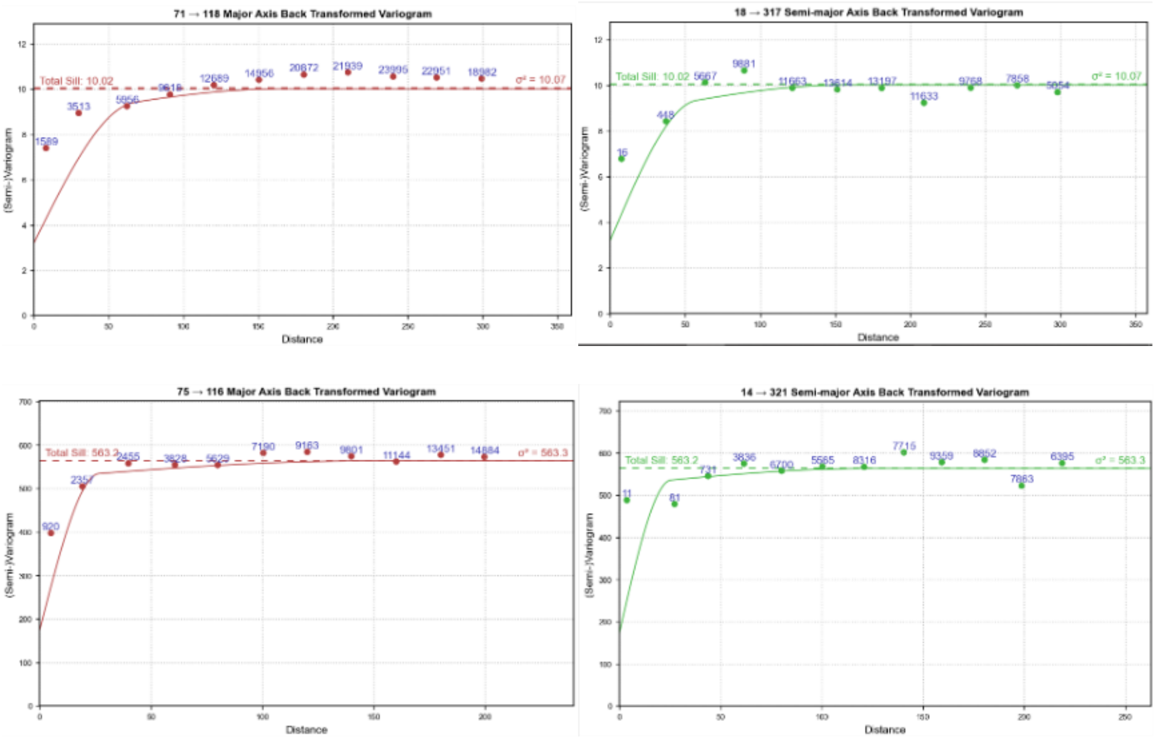
14.8.1 Camp

Many of the Camp domains showed no sign of interpretable structure in the experimental semi-variograms assessment, likely due to a limited number of composites with relatively widely spaced intersections. Sufficiently robust structures were found in domain CA-03 to produce a semi-variogram for gold, silver, copper, lead and zinc. Semi-variograms were modelled onto normal score transforms of the variables with a spherical structure and back transformed into real space. The real space back transformed sermi-variogram models are shown in Figure 14-13 and examples of the semi-variogram models for gold and silver for the CA-03 domain are shown in Table 14-13.

Table 14-13: Camp Semi-Variogram Model Parameters

		Dip				Structure 1				Structure 2			
Domain	Variable	Dip (°)	Azimuth (°)	Pitch (°)	Nugget	Sill	Major	Int	Minor	Sill	Major	Int	Minor
CA-03	Ag_ppm	84	49	18.37	174.3	349	26.7	25.4	5.9	39.9	147.7	122.8	12.1
	Au_ppm	84	45	18.37	3.21	5.39	67.1	60.7	13.7	1.42	166.0	158.0	28.9
	Cu_pct	84	45	18.37	0.0003	0.0004	92	81.3	6.8	0.0002	168.3	138.8	21.3
	Pb_pct	84	45	18.37	0.0022	0.008	71.6	67.7	11.6	0.0021	165.2	155	32.8
	Zn_pct	84	45	18.37	0.1088	0.3751	81.8	73.2	13	0.0646	152.4	142.2	26.5

Figure 14-13: Camp CA-03 Domain Gaussian Space Semi-Variogram Models



Notes: CA-03 domain semi-variograms for gold (top) and silver (bottom).

14.8.2 Los Cuyes

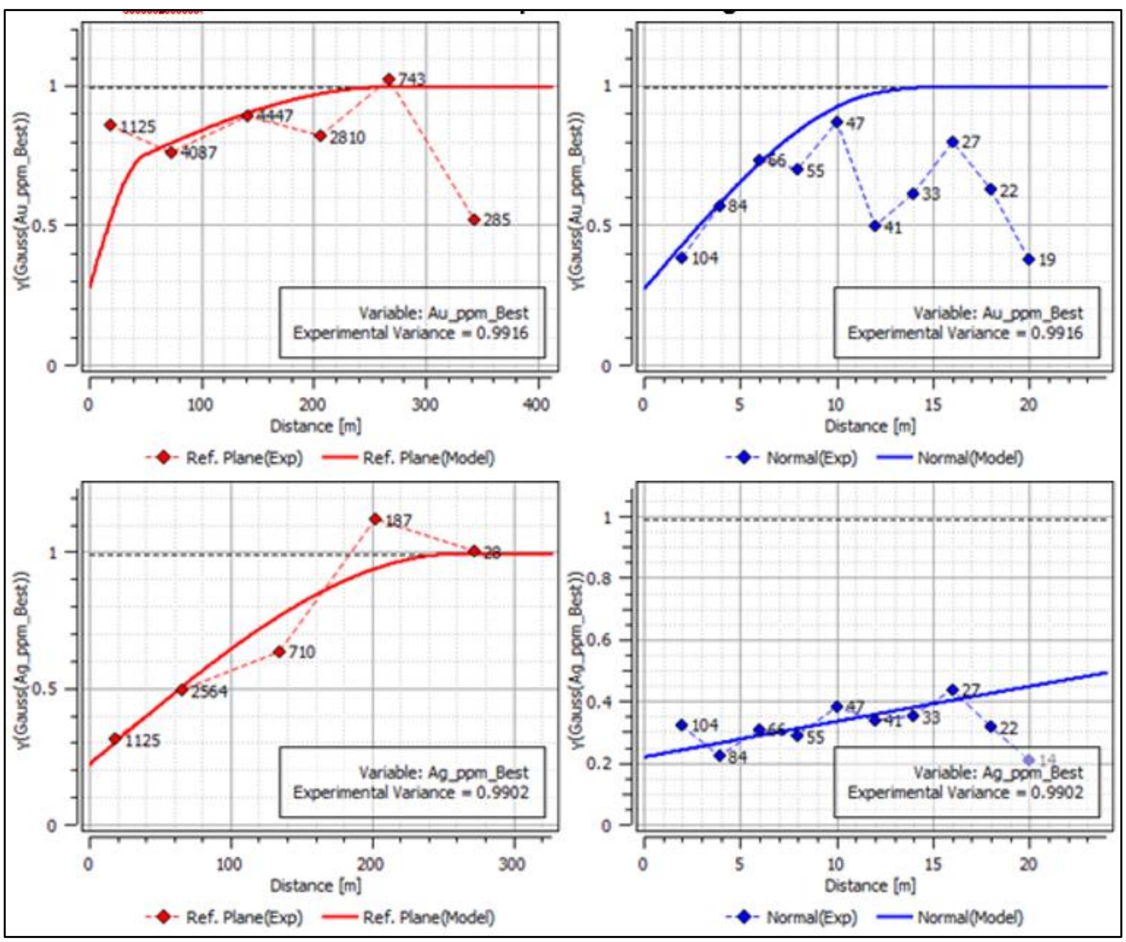
Most of the NW domains showed no sign of interpretable structure in the experimental semi-variogram assessment, likely due to a limited number of composites with relatively widely spaced intersections. Sufficiently robust structures were found in domains NW9 and LCW to produce semi-variogram models for gold, silver, copper, lead and zinc.

Semi-variograms were modelled onto normal score transforms of the variables with a spherical structure and back transformed into real space. The real space back transformed semi-variogram models are shown in Table 14-14 and examples of the gaussian space semi-variogram models for gold and silver for the LCW domain are shown in Figure 14-14. The models are isotropic in the plane of mineralization.

Table 14-14: Los Cuyes Semi-Variogram Model Parameters

						Structure 1				Structure 2			
Domain	Variable	Dip (°)	Dip Azimuth (°)	Pitch (°)	Nugget	Sill	Major	Int	Minor	Sill	Major	Int	Minor
LCW	Ag_ppm	70	150	0	429.1	800.8	180.3	180.3	102.6	261.6	522.1	522.1	49.5
	Au_ppm	70	150	0	66.6	80.1	398.1	398.1	687.4				
	Cu_pct	70	150	0	0.0005	0.0014	121.6	121.6	8.5				
	Pb_pct	70	150	0	0.0463	0.1256	171.9	171.9	25.4				
	Zn_pct	70	150	0	0.0930	0.4908	15.6	15.6	13.8	0.4	205.2	205.2	37.2
NW9	Ag_ppm	80	20	0	147.5	306.3	57.0	57.0	16.2				
	Au_ppm	80	20	0	149.4	49.8	61.8	61.8	17.0				
	Cu_pct	80	20	0	0.0008	0.0016	57.3	57.3	16.2				
	Pb_pct	70	30	320	0.0057	0.0015	42.3	33.6	27.5	0.0041	109.3	107.5	3.4
	Zn_pct	80	20	0	0.0867	0.0228	14.9	14.9	5.0	0.2044	52.4	52.4	16.9

Figure 14-14: Los Cuyes LCW Domain Gaussian Space Semi-Variogram Models



Notes: LCW domain semi-variograms for gold (top) and silver (bottom)

14.8.3 Soledad

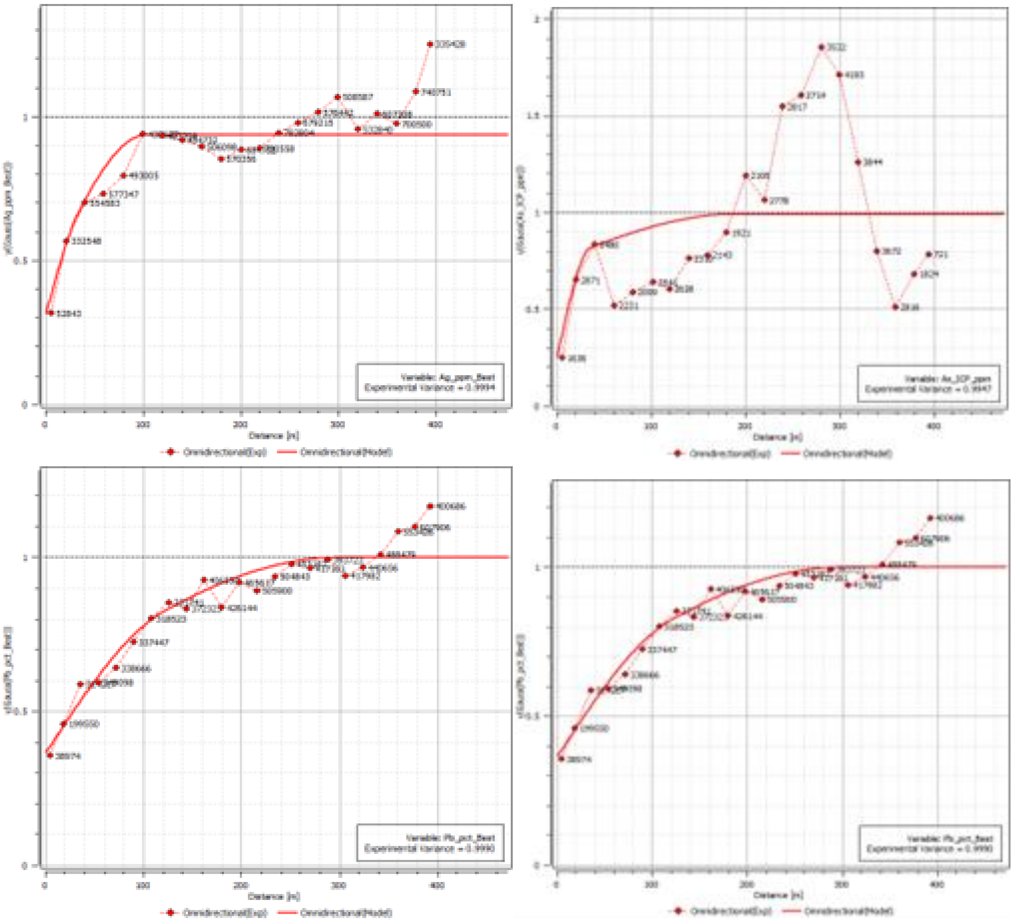
At Soledad within the grade shell the continuity assessment did not show any discernible anisotropy for the variables for which semi-variograms could be modelled. Only omni-directional experimental data showed sufficiently robust structures for semi-variogram modelling in this domain. For arsenic and sulphur there is insufficient data to model semi-variograms. The real space back transformed semi-variogram models are shown in Table 14-15 and examples of the gaussian space semi-variogram models for silver, gold, lead and zinc are shown in Figure 14-15. The structure in the experimental semi-variogram for gold is relatively poorly defined – the shape and ranges of continuity of the other variables were considered in the modelling of the gold semi-variogram.

Table 14-15: Soledad Semi-Variogram Model Parameters

			Structure 1		**Structure 2**	
Domain	Variable	Nugget	Sill	Range	Sill	Range
LCW	Ag_ppm	51.5	17.3	27.1	41.0	95.6
	Au_ppm	0.965	0.282	39.0		
	Cu_pct	0.0014	0.0018	40.6	0.0012	231.0
	Pb_pct	0.0041	0.0016	85.1	0.0025	262.8
	Zn_pct	0.0381	0.0340	23.1	0.0494	108.8

Note: Back-transformed models

Figure 14-15: Soledad Gaussian Space Semi-Variogram Model



Notes: Gaussian space semi-variograms for silver (top left), gold (top right), lead (bottom left) and zinc (bottom right)

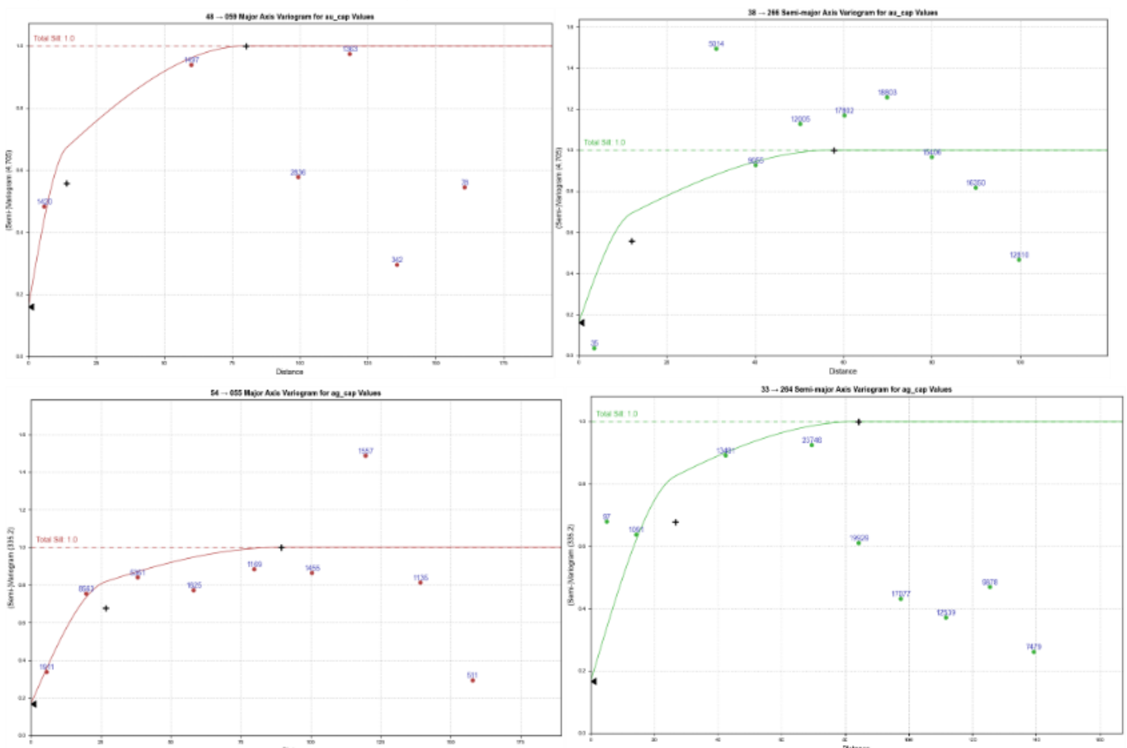
14.8.4 Enma

The semi-variogram structure and examples of the models of Enma are shown in Table 14-16 and Figure 14-16.

Table 14-16: Enma Semi-Variogram Model Parameters

						Structure 1				Structure 2			
Domain	Variable	Dip (°)	Dip Azimuth (°)	Pitch (°)	Nugget	Sill	Major	Int	Minor	Sill	Major	Int	Minor
Enma	Ag_ppm	76	345	128	0.17	0.52	30	26.5	5.2	0.31	89.5	84.1	22.6
	Au_ppm	76	345	128	0.16	0.4	14	12	10.6	0.44	80.1	57.8	18.1
	Cu_pct	76	345	128	0.05	0.95	52.8	43.5	7.9				
	Pb_pct	76	345	128	0.35	0.33	31.3	20.4	2.4	0.32	99.3	40.7	39.5
	Zn_pct	76	345	128	0.16	0.42	32.5	28.6	2.4	0.68	61.3	50.8	10.3

Figure 14-16: Enma Semi-Variogram Models



Notes: semi-variograms for gold (top) and silver (bottom)

14.9 Block Model and Grade Estimation

The models with different block model origins, dimensions and rotations for each deposit were generated by SRK in the first quarter of 2025. A block model parameter summary is presented in Table 14-17 for each deposit. Drill hole spacing, dimensions of the mineralization domains and consideration to possible mining methods dictated block size and sub-block size.

Table 14-17: Block Model Summary

Deposit	Axis	Camp	Los Cuyes	Soledad	Enma
Rotation	Z	No rotation	No rotation	No rotation	No rotation
Origin	X	768,260	768,660	768,990	770,220
	Y	9,551,790	9,552,330	9,551,170	9,55,1820
	Z	400	600	975	1,200
Extent	X	769,150	769,690	769,890	770,670
	Y	9,552,740	9,553,070	9,552,050	9,552,170
	Z	1,600	1,770	1,805	1,920
Block Size (m)	X	10	10	20	10
	Y	10	10	20	10
	Z	10	10	10	10
Min Sub-cell	X	1	0.25	0.5	2
	Y	1	0.25	0.5	2
	Z	1	0.25	0.5	2

14.9.1 Camp

Ordinary Kriging (OK) was used for grade estimation of CA-03, For the remaining domains the Au and Ag estimates were interpolated using inverse distance cubic (ID3) while Cu, Pb, Zn, S and As estimates were interpolated using inverse distance squared (ID2). The search parameters were selected based on a kriging neighbourhood analysis for the domains with semi-variograms. For the domains estimated using ID2 and ID3 the optimised parameters selected for the kriged domains were used as the basis for selecting the search ranges and sample selection criteria. Variable orientation based on the footwall of the veins was applied to align to the mineralization model wireframe.

Generally, a three-pass search strategy is applied (Table 14-18), with the first search radius, where practical, selected to approximate the first structure in the semi-variogram or for single structure models to a range which approximated two thirds of the full semi-variogram range. The second search is generally aimed to align with the full semi-variogram range, while a third search is added to populate estimates for all blocks in the domain. The structure of the gold semi-variogram model is the primary determinant for the search ranges, however the structure of the other variables is also considered. The shorter first pass is used with the aim of generating high confidence local estimates where there is sufficient closely spaced data with a low degree of smoothing.

Table 14-18: Camp Search Parameters

Domain	Dip	Dip Azimuth	Pitch	Major	Int	Minor	No Sectors	Min Samples	Max Samples
CA-01 Pass 1	84	49	122	60	60	15	1	4	16
CA-01 Pass 2	84	49	122	120	120	30	1	3	16
CA-01 Pass 3	84	49	122	180	180	40	1	1	16
CA-02 Pass 1	89	45	90	60	60	20	1	4	16
CA-02 Pass 2	89	45	90	120	120	40	1	3	16
CA-02 Pass 3	89	45	90	180	180	60	1	1	16
CA-03 Pass 1	84	48	7	60	60	20	1	5	16
CA-03 Pass 2	84	48	7	120	120	40	1	3	36
CA-03 Pass 3	84	48	7	180	180	60	1	1	16
CA-04 Pass 1	81	41	90	60	60	15	1	4	16
CA-04 Pass 2	81	41	90	120	120	30	1	3	16
CA-04 Pass 3	81	41	90	180	180	40	1	1	16
CA-05 Pass 1	79	36	90	60	60	15	1	4	16
CA-05 Pass 2	79	36	90	120	120	30	1	3	16
CA-05 Pass 3	79	36	90	180	180	40	1	1	16
CA-06 Pass 1	84	46	90	60	60	15	1	4	16
CA-06 Pass 2	84	46	90	120	120	30	1	3	16
CA-06 Pass 3	84	46	90	180	180	40	1	1	16

Notes: Search parameters listed applied to Au, Ag, Cu, Pb, Zn, S and As estimates.
Larger search ranges are applied after the third search pass where needed to inform all blocks in the domain.
Local search orientations are applied based on the domain wireframe orientation.

14.9.2 Los Cuyes

Ordinary Kriging (OK) was used for grade estimation in the domains where semi-variograms were modelled. Remaining domains were estimated with inverse distance squared (ID2). Search parameters were selected based on a kriging neighbourhood analysis for the domains with semi-variograms.

The search parameters were selected based on a kriging neighbourhood analysis for the domains with semi-variograms. For the domains estimated using ID2 the optimised parameters selected for the kriged domains were used as the basis for selecting the search ranges and sample selection criteria. The orientation of the search parameters is modified for each block to align to the mineralization model wireframe.

Generally, a three-pass search strategy is applied (Table 14-19), with the first search radius, where practical, selected to approximate the first structure in the semi-variogram or for single structure models to a range which approximated two thirds of the full semi-variogram range. The second search is generally aimed to align with the full semi-variogram range, while a third search is added to populate estimates for all blocks in the domain. The structure of the gold semi-variogram model is the primary determinant for the search ranges, however the structure of the other variables is also considered. The shorter first pass is used with the aim of generating high confidence local estimates where there is sufficient closely spaced data with a low degree of smoothing.

Table 14-19: Los Cuyes Search Parameters

Domain	Pass	Dip	Dip azimuth	Pitch	Major	Int	Minor	No sectors	Min	Max / Sector	Max / hole	Min holes
LCW	1	70	150	0	60	60	30	4	4	5	5	2
LCW	2	70	150	0	100	100	40	4	4	4	5	2
LCW	3	70	150	0	265	265	60	4	3	5	5	2
NW9	1	80	20	0	60	60	25	1	5	16	-	-
NW9	2	80	20	0	120	120	38	1	4	16	-	-
NW9	3	80	20	0	120	180	63	1	3	16	-	-
NW ID2	1	80	30	90	60	60	25	1	5	14	-	2
NW ID2	2	80	30	90	120	120	25	1	1	16	-	2
NW ID2	3	80	30	90	180	180	25	1	1	16	-	2
Halo	1	40	220	0	50	50	15	4	6	5	-	2
Halo	2	40	220	0	100	100	25	4	5	4	-	2
Halo	3	40	220	0	200	200	40	4	4	4	-	2

Notes: Search parameters listed applied to Au, Ag, Cu, Pb, and Zn. For all shear domains the NW ID2 parameters are applied for As and S estimates.
Larger search ranges are applied after the third search pass where needed to inform all blocks in the domain.
Local search orientations are applied based on the domain wireframe orientation.

14.9.3 Soledad

The Soledad variables were estimated using OK except for arsenic and sulphur which were interpolated using ID2. The search parameters were selected based on a kriging neighbourhood analysis for the variables with semi-variograms. For the variables estimated using ID2 the optimised parameters selected for the kriged domains were used as the basis for selecting the search ranges and sample selection criteria, however these are modified as not all drill holes have assays for these two variables.

A three-pass search strategy was employed for each variable, with the first search range being the full semi-variogram range for the precious metals, but a fraction of the full variogram range for the base metals. The search parameters applied are summarised in Table 14-20.

Table 14-20: Soledad Search Parameters

Search	Variable	Range	Min	Max per sector	No Sectors
Pass 1	Ag ppm	100	5	18	1
	Au ppm	100	5	26	1
	Cu%	100	5	16	1
	Pb%	100	5	16	1
	Zn%	100	5	5	4
	As ppm	100	5	22	1
	S%	100	5	14	1
Pass 2	Ag ppm	150	5	18	1
	Au ppm	150	5	24	1
	Cu%	150	5	14	1

Search	Variable	Range	Min	Max per sector	No Sectors
	Pb%	150	5	14	1
	Zn%	150	5	12	1
	As ppm	200	4	16	1
	S%	200	4	16	1
Pass 3	Ag ppm	300	3	16	1
	Au ppm	300	3	20	1
	Cu%	300	3	12	1
	Pb%	300	3	12	1
	Zn%	300	3	10	1
	As ppm	300	2	16	1
	S%	300	2	16	1

Notes: In the first search pass for gold, silver, copper, lead and zinc a minimum of two holes is required, and an optimum of four composites per hole is applied.

14.9.4 Enma

The Enma variables were estimated using OK for Au, Ag, Cu, Pb and Zn. The search parameters were selected based on a kriging neighbourhood analysis for the variables with semi-variograms. Due to relatively small numbers of composites the experimental semi-variogram for S and As was not defined, and S and As were interpolated using ID2.

A three-pass search strategy was employed for each variable. The search parameters applied are summarised in Table 14-21.

Table 14-21: Enma Search Parameters

Search	Variable	Range	Dip	Dip azimuth	Pitch	Min Samples	Max per sector	No Sectors
Pass 1	Ag ppm	30	76	345	124	5	10	3
	Au ppm	30	77	345	128	5	10	3
	Cu%	30	77	345	128	5	10	3
	Pb%	30	77	345	128	5	10	3
	Zn%	30	77	345	128	5	10	3
	As ppm	30	77	345	128	5	10	3
	S%	30	77	345	128	5	10	3
Pass 2	Ag ppm	50	76	345	124	4	10	3
	Au ppm	50	77	345	128	4	10	3
	Cu%	50	77	345	128	4	10	3
	Pb%	50	77	345	128	4	10	3
	Zn%	50	77	345	128	4	10	3
	As ppm	50	77	345	128	4	10	3
	S%	50	77	345	128	4	10	3
Pass 3	Ag ppm	100	76	345	124	2	10	1
	Au ppm	100	77	345	128	2	10	1

Search	Variable	Range	Dip	Dip azimuth	Pitch	Min Samples	Max per sector	No Sectors
	Cu%	100	77	345	128	2	10	1
	Pb%	100	77	345	128	2	10	1
	Zn%	100	77	345	128	2	10	1
	As ppm	100	77	345	128	2	10	1
	S%	100	77	345	128	2	10	1

14.10 Model Validation and Sensitivity

Model validation is a common approach for determining whether grade estimation has performed as expected. An acceptable or preferred validation result does not necessarily imply that the model is correct or derived from the right estimation approach. It suggests only that the model is a reasonable representation of the resource data used and of the estimation method applied. Other issues such as the relationship between the model-selectivity assumptions and mining practices are equally important when determining the appropriateness of the Mineral Resource estimate.

For each deposit SRK undertook a range of validations including visual validations of the estimates and informing data, comparisons of the mean values of the data and estimates per estimation domain and swath plots to assess the reproduction of the spatial variability of the variables. Selected examples of these are presented for each deposit to illustrate the conclusions drawn from analysing the validations.

14.10.1 Camp

For the domains of Camp, the validations are affected by irregularly spaced and relatively small number of intersections, changes in the thickness of the modelled domains, isolated high values, capping, as well as the higher variance associated with higher grades. The mean grades of the composites and the mean of the classified Mineral Resources are shown in Table 14-22.

Table 14-22: Camp per Domain Comparison Between Composites and Estimates

	Au g/t		Ag g/t		Pb %		Zn %		Cu %	
Domain	Comp	Est	Comp	Est	Comp	Est	Comp	Est	Comp	Est
CA-01	2.33	2.33	15.83	16.90	0.05	0.05	0.69	0.70	0.02	0.02
CA-02	1.26	1.36	7.58	7.73	0.04	0.04	0.50	0.58	0.01	0.01
CA-03	1.54	1.70	12.43	13.53	0.05	0.05	0.53	0.60	0.02	0.02
CA-04	1.41	1.32	13.54	13.68	0.05	0.05	0.61	0.54	0.02	0.02
CA-05	1.43	1.29	13.14	12.06	0.07	0.05	0.52	0.49	0.02	0.02
CA-06	0.73	0.59	9.52	8.13	0.02	0.02	0.17	0.19	0.01	0.01

The heterogenous nature of the vein mineralization, the spatial grade distribution and variable intersection spacing for Camp resulted in some differences in the comparison of the global mean of the composites and the block model. Figure 14-17 displays an example of the gold distribution in the largest vein, CA-03.

Figure 14-18 and Figure 14-19 show the swath plots for combined domains of Camp block models and composites. These plots show the block models and composites match reasonably well in all orthogonal directions in the central area of the domain, but with poor correlation in the areas with fewer samples. These areas are classified as Inferred or not classified as Mineral Resources and additional exploration is required to support the declaration of a Mineral Resource in these areas.

Figure 14-17: Vertical Section of the Camp CA-03 Domain Gold Distribution Looking North

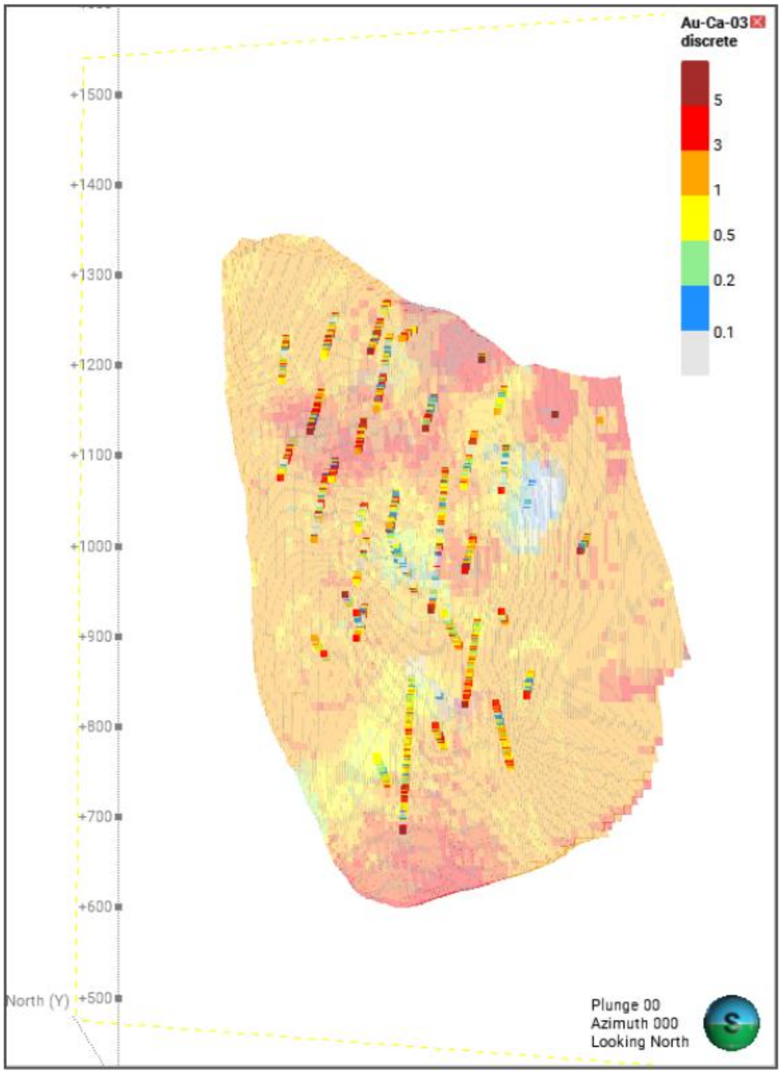


Figure 14-18: Camp X Swath Plots for Gold, Silver, Lead and Zinc

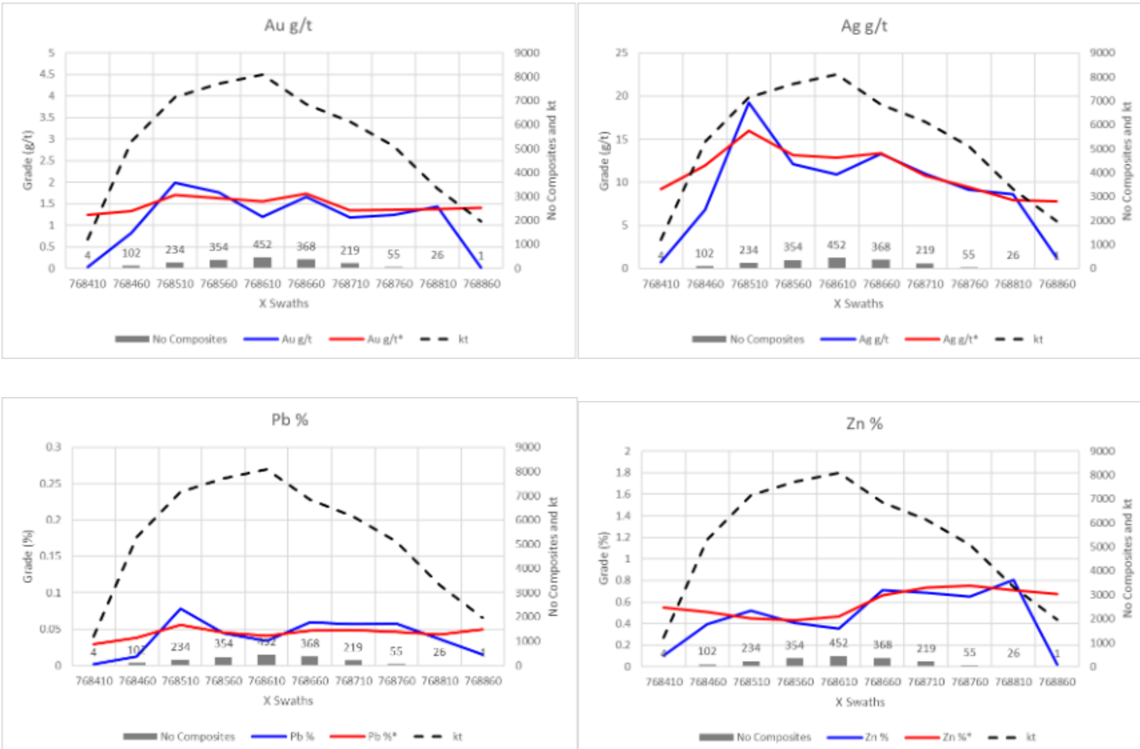
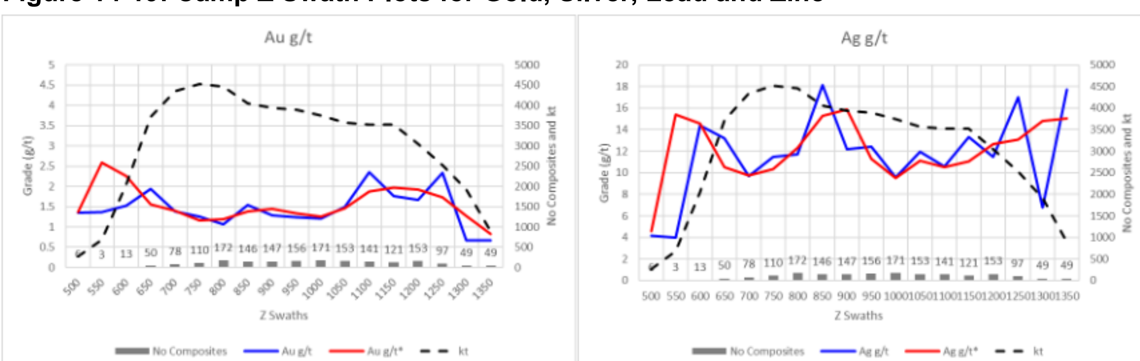
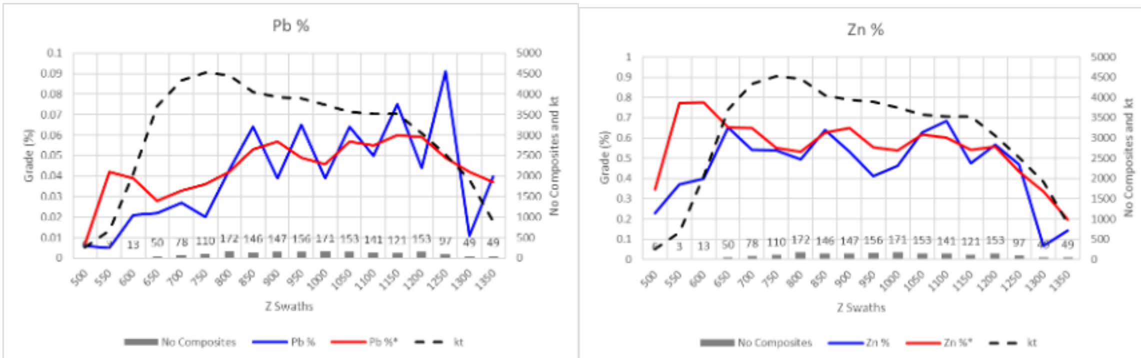


Figure 14-19: Camp Z Swath Plots for Gold, Silver, Lead and Zinc





14.10.2 Los Cuyes

The number of informing composites in the halo domain is significantly more than is available for any of the shear hosted domains. For the tabular shear hosted domains, the validations are affected by irregularly spaced and relatively small number of intersections, changes in the thickness of the modelled domains, isolated high values, capping, as well as the higher variance associated with higher grades. The comparison between estimates and composites in these domains shows quite variable correlations. The declustered mean grades of the composites and the mean of the classified Mineral Resources are shown in Table 14-23.

Some domains (such as NW6, NW7) show very close correlation between the composites and estimate for all variables, while others (such as NW1, NW2, NW15) show good correlations for some variables and weaker matching for others. In the LCW domain the estimates appear to underestimate the grades when compared to the composites. However, when considering the spatial grade distribution and variable intersection spacing, the reason for this is apparent as is illustrated in Figure 14-20. The widely spaced but very high-grade intersections on the western margin of the deposit are intentionally affected by capping to reduce the risk of over estimation, but these will have an impact on the mean grades. The central core of the domain is thicker than the margins and is also lower grade.

Table 14-23: Los Cuyes per Domain Comparison between Composites and Estimates

	Au g/t		Ag g/t		Pb %		Zn %	
Domain	**Comp**	**Est**	**Comp**	**Est**	**Comp**	**Est**	**Comp**	**Est**
LCW	4.95	2.81	26.17	16.61	0.208	0.100	0.75	0.55
NW1	2.77	2.67	36.33	27.23	0.111	0.095	0.49	0.68
NW2	4.32	3.27	14.37	10.62	0.068	0.065	0.78	0.87
NW3	10.10	10.28	40.41	44.56	0.215	0.199	0.84	1.01
NW5	7.69	9.90	63.61	79.38	0.108	0.159	1.10	1.51
NW6	10.00	10.10	30.78	30.85	0.110	0.110	0.36	0.37
NW7	3.25	3.21	23.03	23.01	0.076	0.060	0.31	0.25
NW8	5.37	4.18	28.49	28.57	0.325	0.664	0.47	0.53
NW9	7.22	4.47	18.15	13.40	0.056	0.040	0.38	0.49
NW10	3.89	4.12	30.32	33.43	0.315	0.576	0.55	0.59
NW11	5.26	4.02	19.92	18.31	0.050	0.065	0.35	0.28
NW13	6.24	6.71	39.63	57.81	0.073	0.108	1.22	0.70
NW15	5.69	5.40	35.71	33.42	0.153	0.185	0.85	0.75
Halo	0.69	0.75	5.40	5.71	0.020	0.024	0.23	0.21

Some domains (such as NW6, NW7) show very close correlation between the composites and estimate for all variables, while others (such as NW1, NW2, NW15) show good correlations for some variables and weaker matching for others.

In the LCW domain the estimates appear to underestimate the grades when compared to the composites. However, when considering the spatial grade distribution and variable intersection spacing, the reason for this is apparent as is illustrated in Figure 14-20. The widely spaced but very high-grade intersections on the western margin of the deposit are intentionally affected by capping to reduce the risk of over estimation, but these will have an impact on the mean grades. The central core of the domain is thicker than the margins and is also lower grade. This can also be observed in the swath plots in Figure 14-21 which highlights the higher tonnage and lower grades in the central core and elevated grades on the western margin informed by relatively few composites. Composite mean values are shown in blue (Au g/t) and estimated values in red (Au g/t*).

In aggregate, the visual validations indicate that there is a relatively good reproduction of the composite grades however the grade distribution is quite variable, and additional exploration will be required to support detailed mine planning.

Figure 14-20: Vertical Section of the Los Cuyes LCW Domain Gold Distribution Looking North

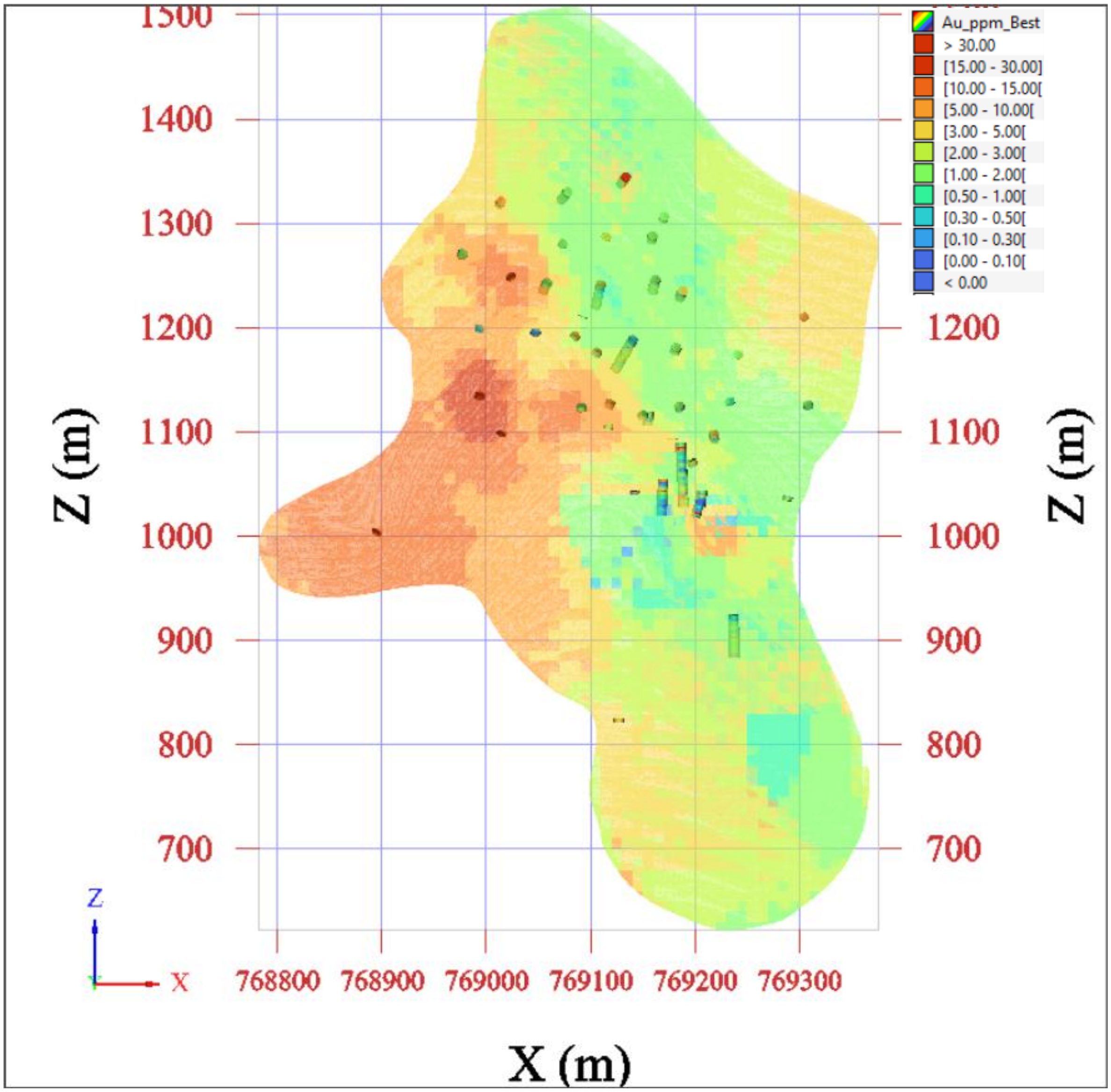
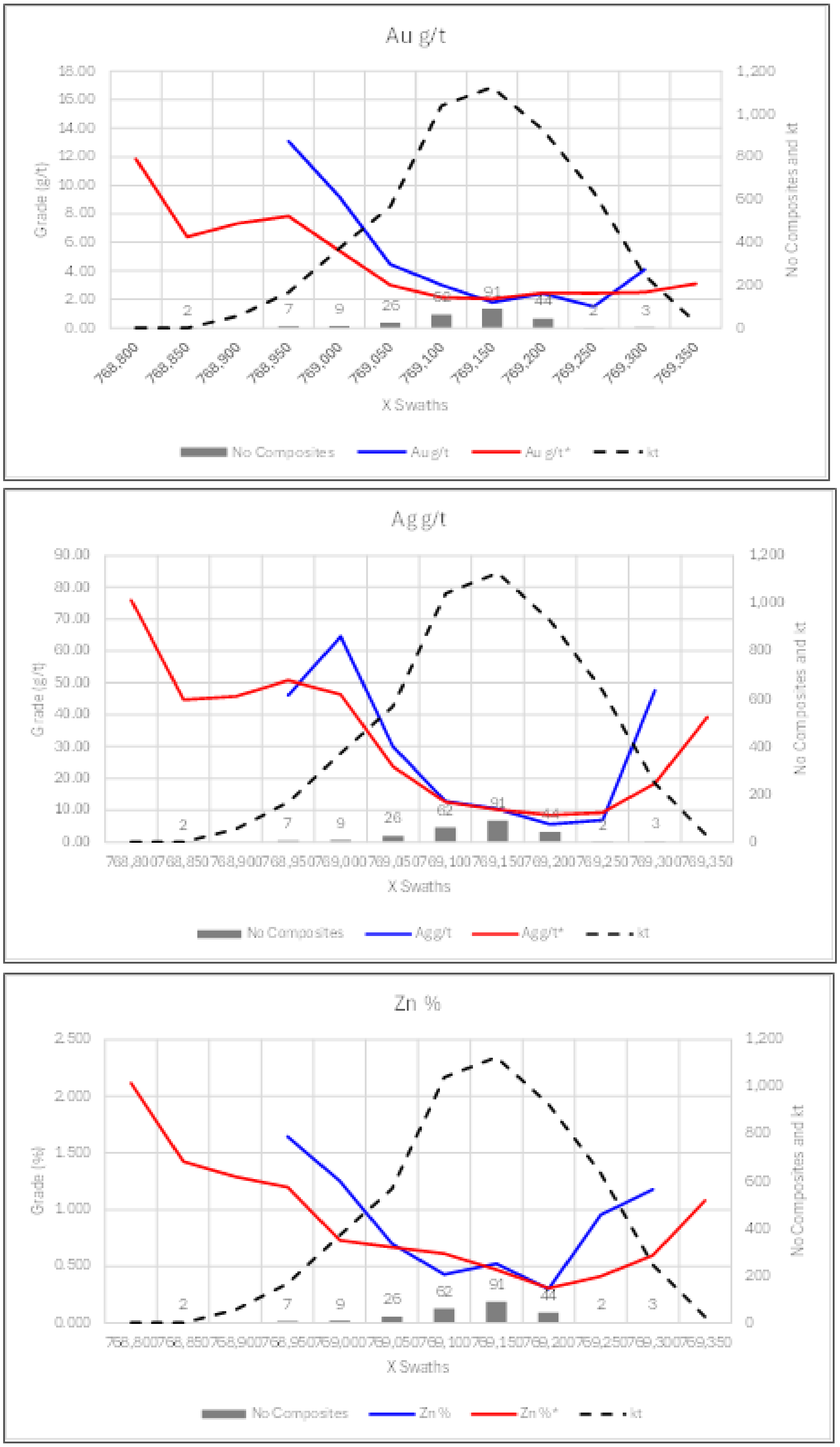


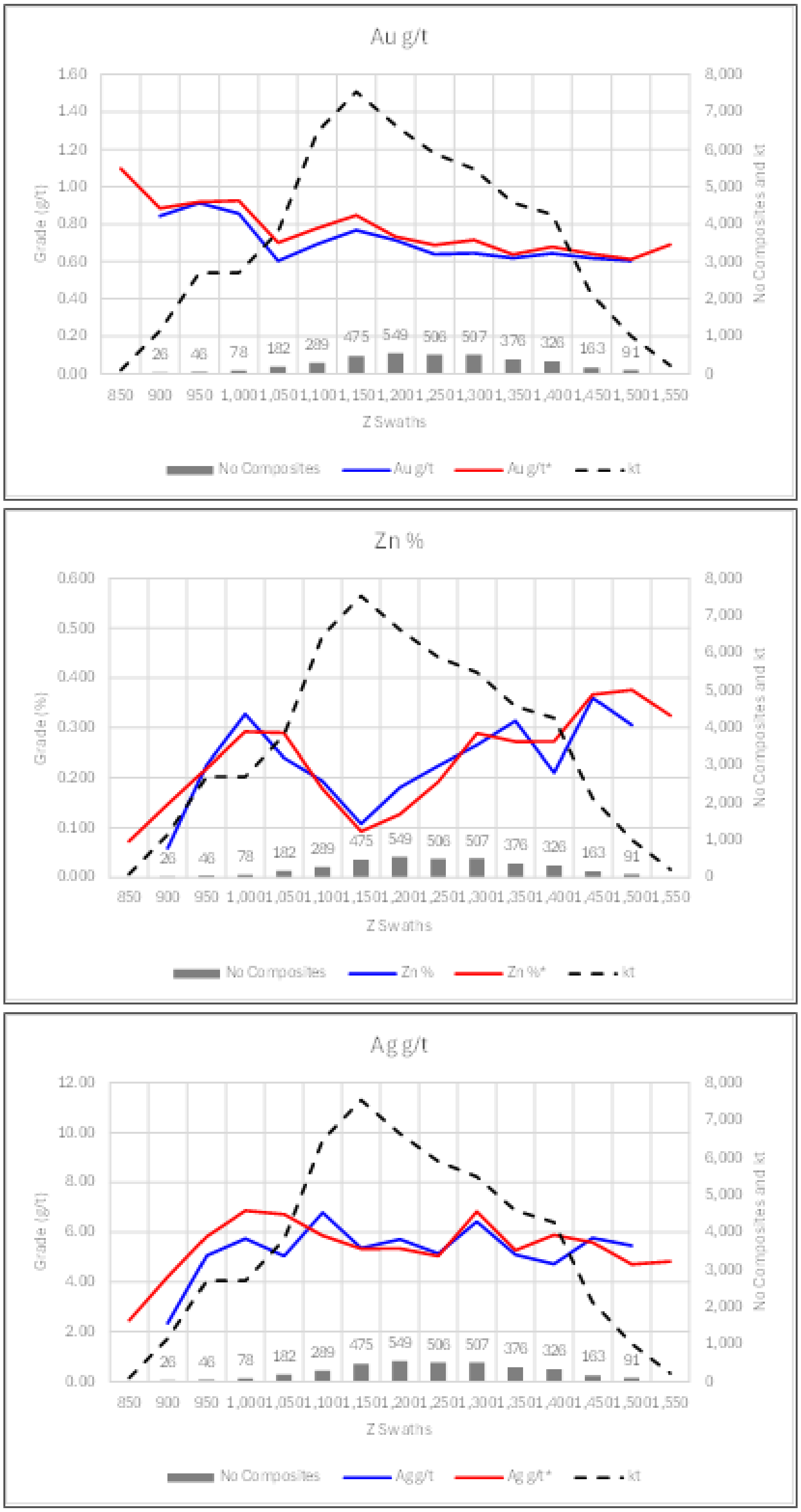
Figure 14-22 displays the correlation between estimates and information data in the swath plots is considered good. There is significantly more informing data in the halo domain, and relatively lower variance in the variable grades compared to the tabular domains.

Figure 14-21: Los Cuyes X Swath Plots for Gold, Silver and Zinc in the LCW Domain



Notes: Composite mean values are shown in blue (Au g/t) and estimated values in red (Au g/t*)

Figure 14-22: Los Cuyes Z Swath Plots for Gold, Silver and Zinc in the Halo Domain

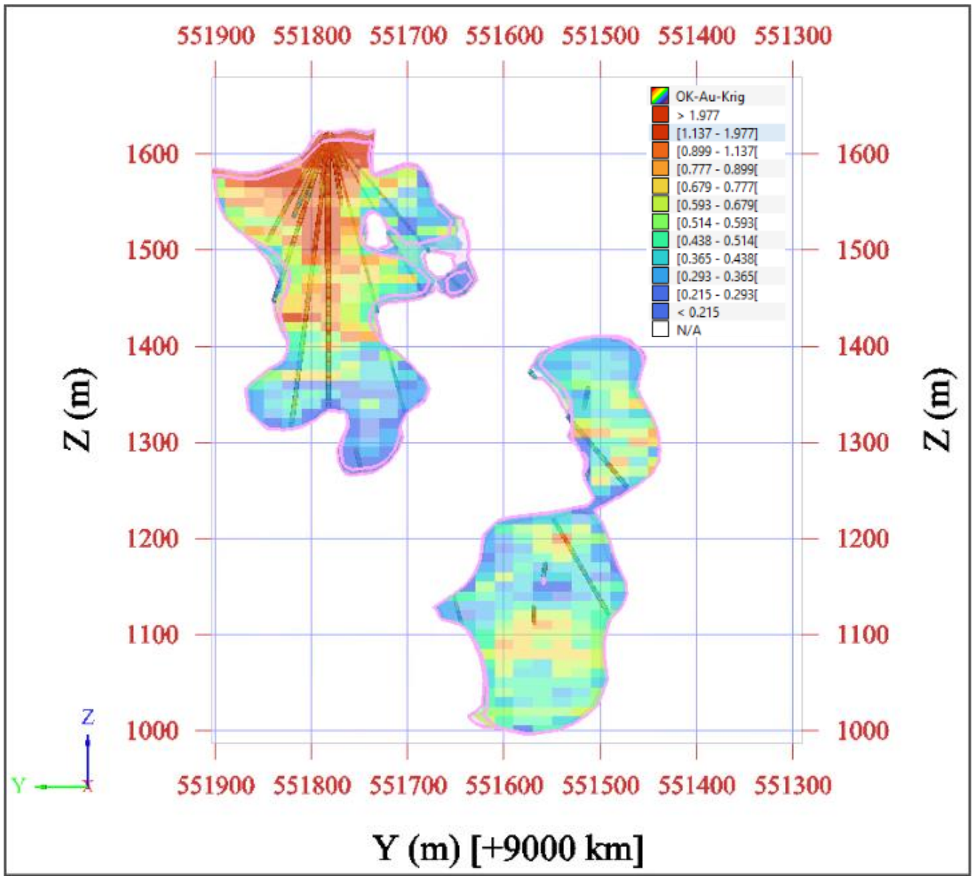


Notes: Composite mean values are shown in blue (Au g/t) and estimated values in red (Au g/t*)

14.10.3 Soledad

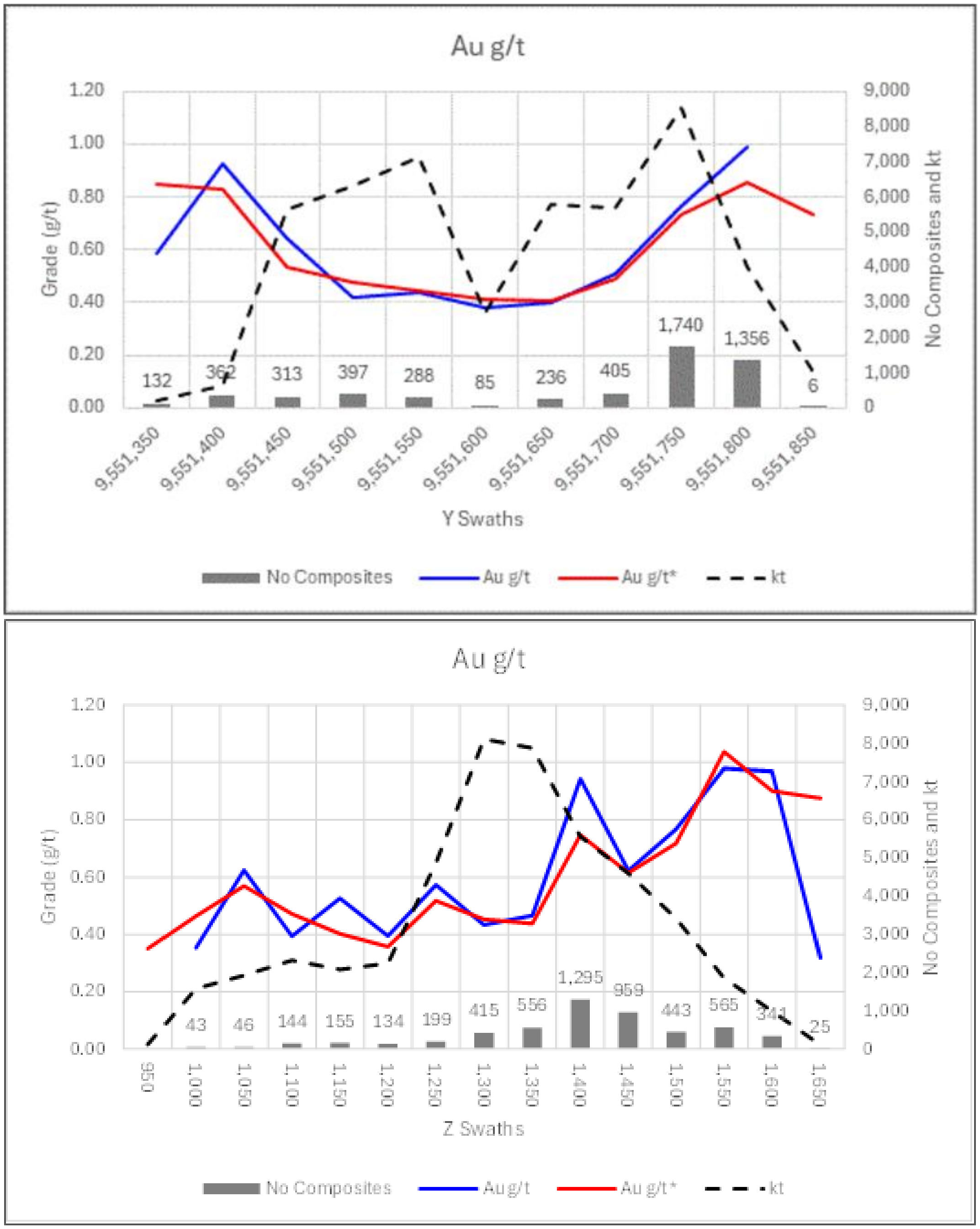
The grade distribution at Soledad is variable, with higher grades of gold close to surface at the core of the grade shells, transitioning to a lower grade disseminated style of mineralization with greater depth. The base metals are more evenly distributed within the grade shell. The gold distribution is illustrated in Figure 14-23. The estimated grades overall show a good correlation with the composite data.

Figure 14-23: Soledad Vertical Cross Section Looking West Showing Gold Grade



A relatively good reproduction of the composite grades in the estimates are displayed in the Soledad swath plots, Figure 14-24. A global comparison between composites and the estimate is shown in Table 14-24.

Figure 14-24: Soledad Y and Z Swath Plots for Gold



Notes: Composite mean values are shown in blue (Au g/t) and estimated values in red (Au g/t*)

Table 14-24: Soledad Global Comparison Between Composites and Estimates

Deposit	Au g/t	Ag g/t	Cu %	Pb %	Zn %
Composites	0.59	6.40	0.022	0.046	0.38
Estimates	0.56	6.70	0.024	0.042	0.41
% difference	-5.8%	4.8%	9.0%	-8.2%	8.2%

Notes: Composite grades are declustered, only Indicated and Inferred estimates are included.

14.10.4 Enma

The grade distribution at Enma is variable. The gold distribution is illustrated in Figure 14-25. The estimated grades overall show a good correlation with the composite data.

Figure 14-25: Enma Vertical Cross Section Looking South Showing Gold Grades

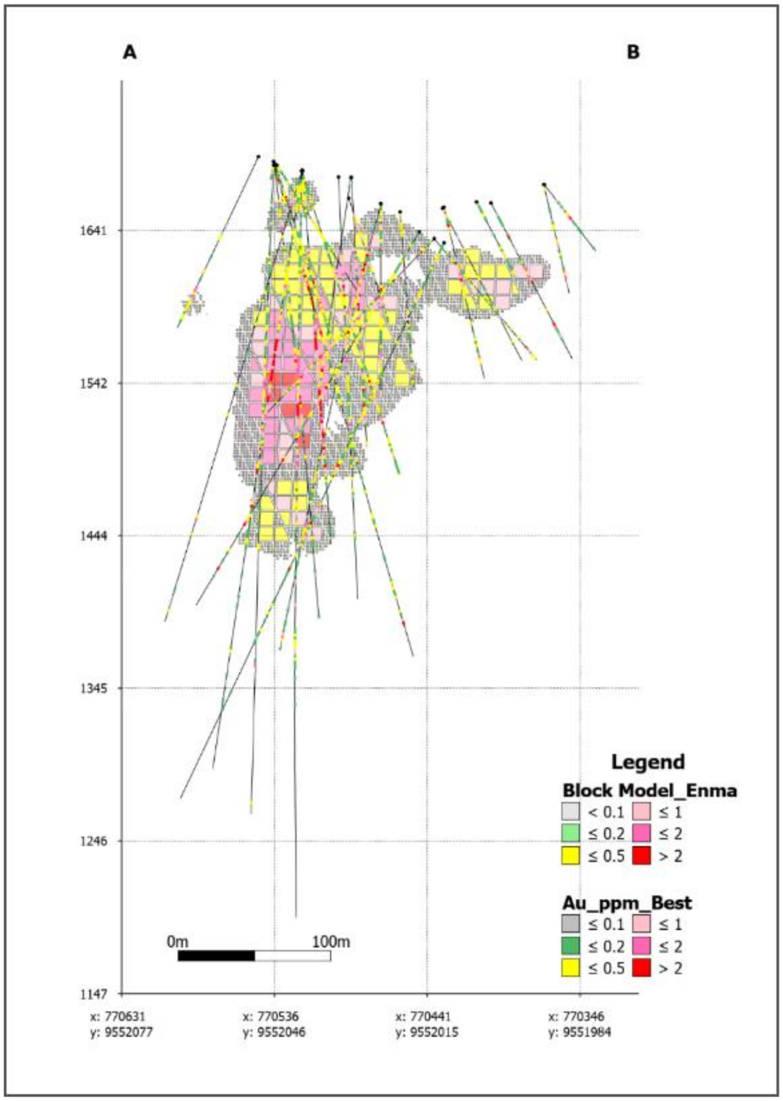
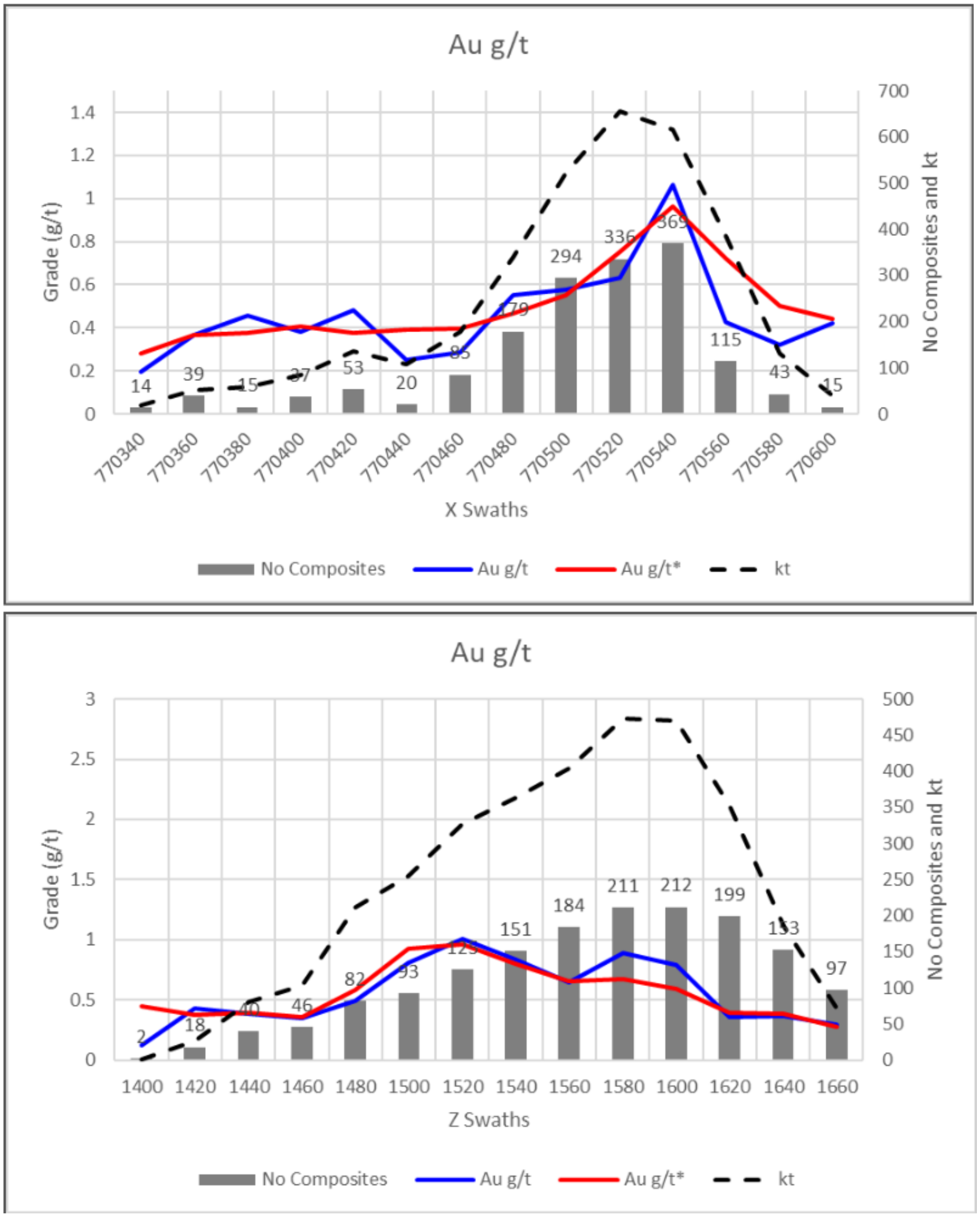


Figure 14-26 illustrates the relatively good reproduction of the composite grades in the estimates in swath plots. The global comparison between the composites and estimate is shown in Table 14-25.

Figure 14-26: Enma X and Z Swath Plots for Gold



Notes: Composite mean values are shown in blue (Au g/t) and estimated values in red (Au g/t*)

Table 14-25: Enma Global Comparison between Composites and Estimates

Deposit	Au g/t	Ag g/t	Cu %	Pb %	Zn %
Composites	0.64	12.43	0.02	0.08	0.31
Estimates	0.64	13.39	0.02	0.07	0.30
% difference	0.4%	7.7%	7.9%	-13.8%	-3.9%

14.11 Mineral Resource Classification

Block model quantities and grade estimates for the Condor project were classified according to the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (May 2014) by Mark Wanless, Pr.Sci.Nat, FGSSA (400178/05), an appropriate independent Qualified Person for the purpose of National Instrument 43-101.

Mineral Resource classification is typically a subjective concept, industry best practices suggest that Mineral Resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar resource classification.

In the QP's opinion, the applied core handling, logging, sampling, and core storage protocols on the Condor Project are consistent with industry standards, and the QP is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of these results. The analytical QAQC program has been in place over the duration of the exploration programs and has been used to monitor the accuracy and precision of the analytical laboratories. The QAQC data confirm that the analytical results have an acceptable accuracy and precision for use in Mineral Resource estimation, and do not represent a constraint in the classification of Mineral Resources. For Enma there is insufficient QAQC data from which to draw meaningful conclusions, however the quality of the assay results for Enma are expected to be consistent with that of the other deposits.

The exploration data and analytical results are of acceptable confidence and have been generated and managed by a competent team for the duration of the exploration programmes.

Mineral resource classification is typically a subjective concept. Industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

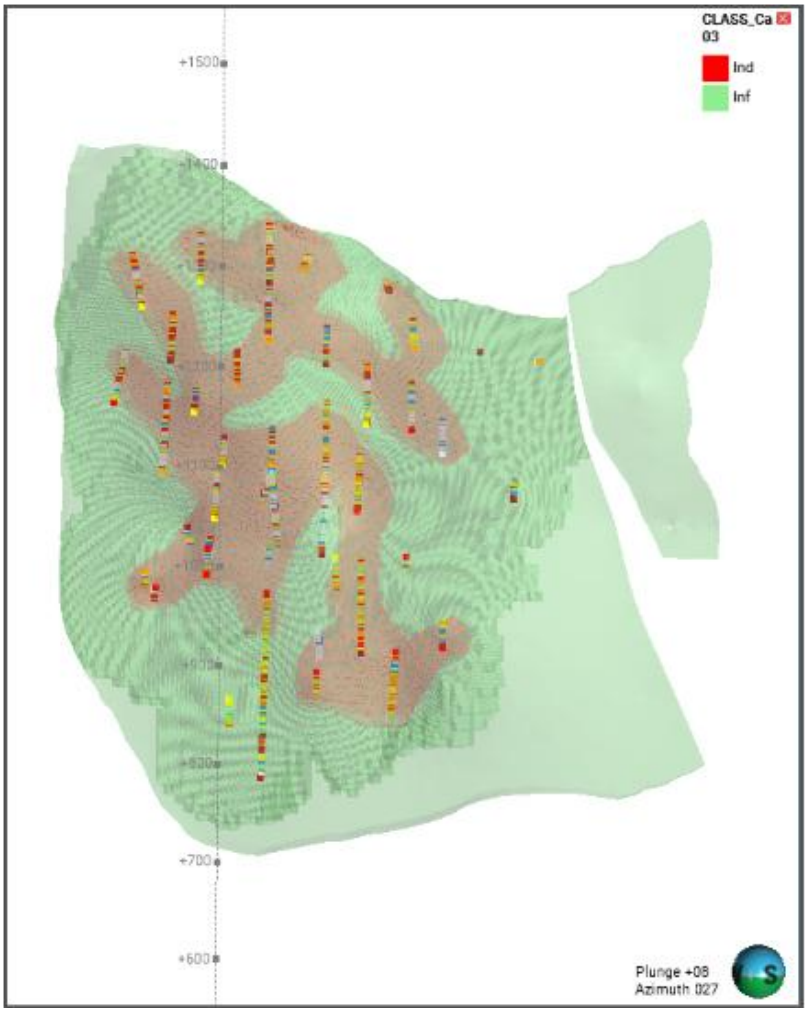
14.11.1 Camp

Silvercorp has a credible interpretation of the mineralization controls that inform the geological modelling. The lithological modelling is generally consistent with the geological logging data and presents a reasonable interpretation of the lithologies which hosts the mineralization.

No Measured Mineral Resources are classified at Camp.

The drilling density is variable over the extent of the Camp. The largest domain is the CA-03 domain. This domain has the most intersections and is relatively thicker than the other domains. The core of the deposit is relatively well drilled, and in the model densely drilled area the drill hole spacing approximates 30 to 100 m. For the CA-03 domain blocks which have a slope of regression of greater than 0.7, are estimated in the first or second search pass, have an average distance of 60 m to the informing composites and are estimated with at least 3 drillholes support an Indicated classification. Blocks which are estimated within search passes 1 to 3 with at least 2 drillholes and minimum samples distance no more than 120 m are classified as Inferred Mineral Resources. Blocks behind this are not classified as Mineral Resources (Figure 14-27).

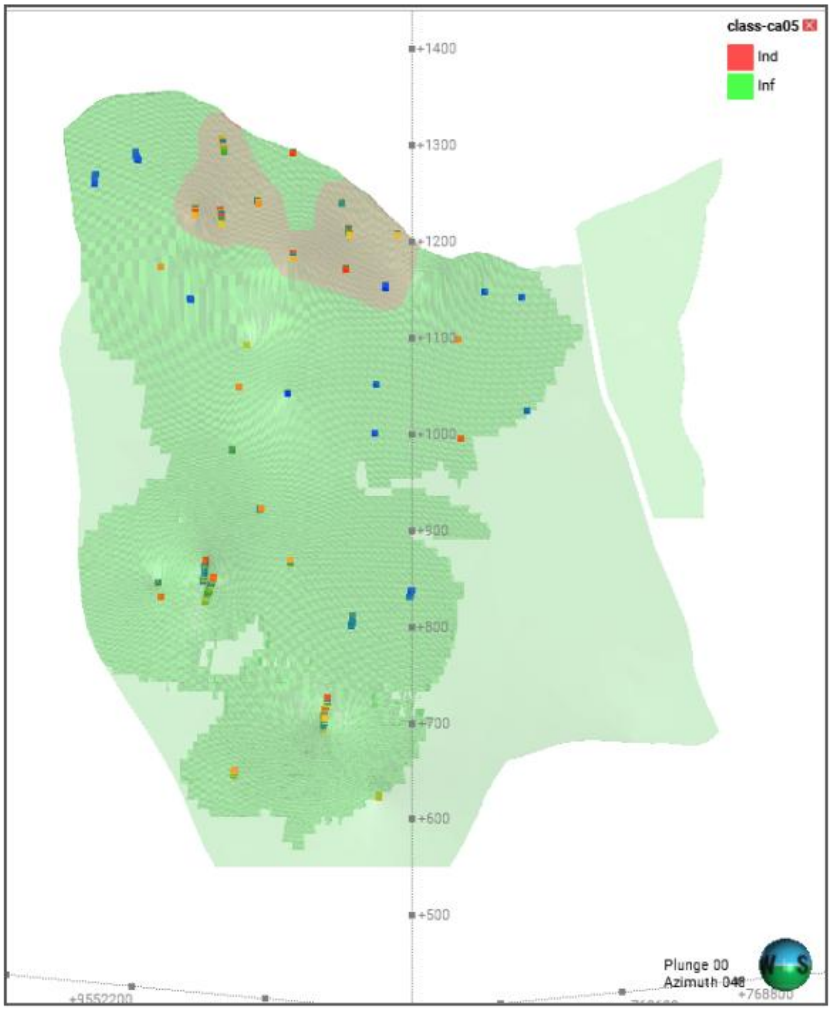
Figure 14-27: Plan Showing Camp Domain CA-03 Classification



Notes: Wireframe of domain CA-03 shown for context

Some area of CA-05 is intersected relatively closely spaced drill holes (<60 m). Block which are estimated in the first search pass, support an Indicated classification with at least 3 holes, with the remainder of the domain estimated in the second and third search pass with at least 2 drillholes and minimum samples distance no more than 120 m was classified as Inferred Mineral Resources (Figure 14-28).

Figure 14-28: Plan Showing Camp Domain CA-05 Classification



Notes: Wireframe of domain CA-05 shown for context.

For other domains, Since CA-01 are relative thinner, drilling density of CA-02, CA-04 and CA-06 is variable over the extent of the domain, the confidence in the continuity of the mineralization is lower than that of the more extensive and better-informed domains. For these domains, the confidence in the domain and grade continuity only supports the classification of Inferred Mineral Resources, for blocks estimated in search passes 1 to 3 with at least 2 drillholes and minimum samples distance no more than 120 m.

14.11.2 Los Cuyes

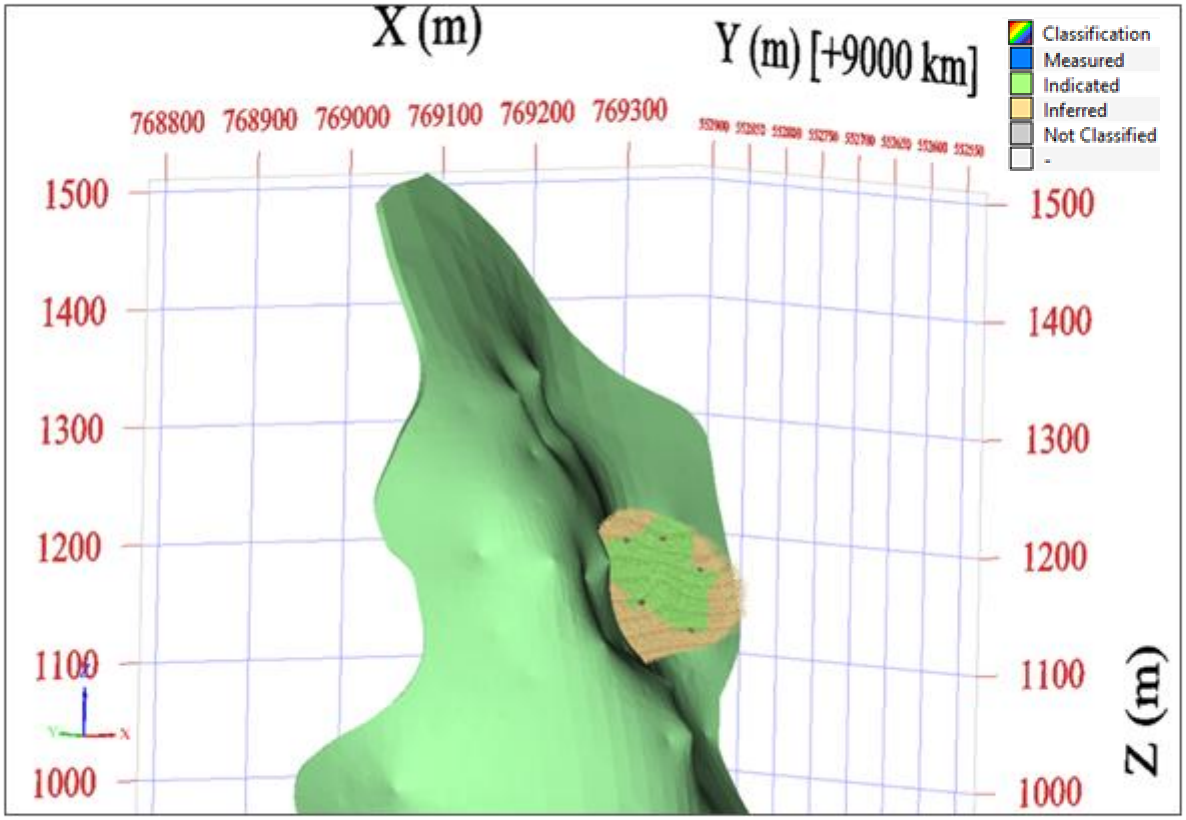
Silvercorp has a credible interpretation of the mineralization controls that inform the geological modelling. The lithological modelling is generally consistent with the geological logging data and presents a reasonable interpretation of the lithologies which hosts the mineralization.

No Measured Mineral Resources are classified at Los Cuyes.

For several of the NW domains there are only a small number of relatively widely spaced intersections. These are NW3, NW7, NW8, NW9, NW10, NW11, NW13 and NW15. The confidence in the continuity of the mineralization is lower than that of the more extensive and better-informed domains. For these domains, the confidence in the domain and grade continuity only supports the classification of Inferred Mineral Resources, for blocks estimated in the three search passes.

NW5 is a relatively small domain, which is intersected by five relatively closely spaced drill holes (<50 m). Block which are estimated in the first search pass, support an Indicated classification, with the remainder of the domain estimated in the second search pass classified as Inferred Mineral Resources. The NW5 classification is illustrated in Figure 14-29.

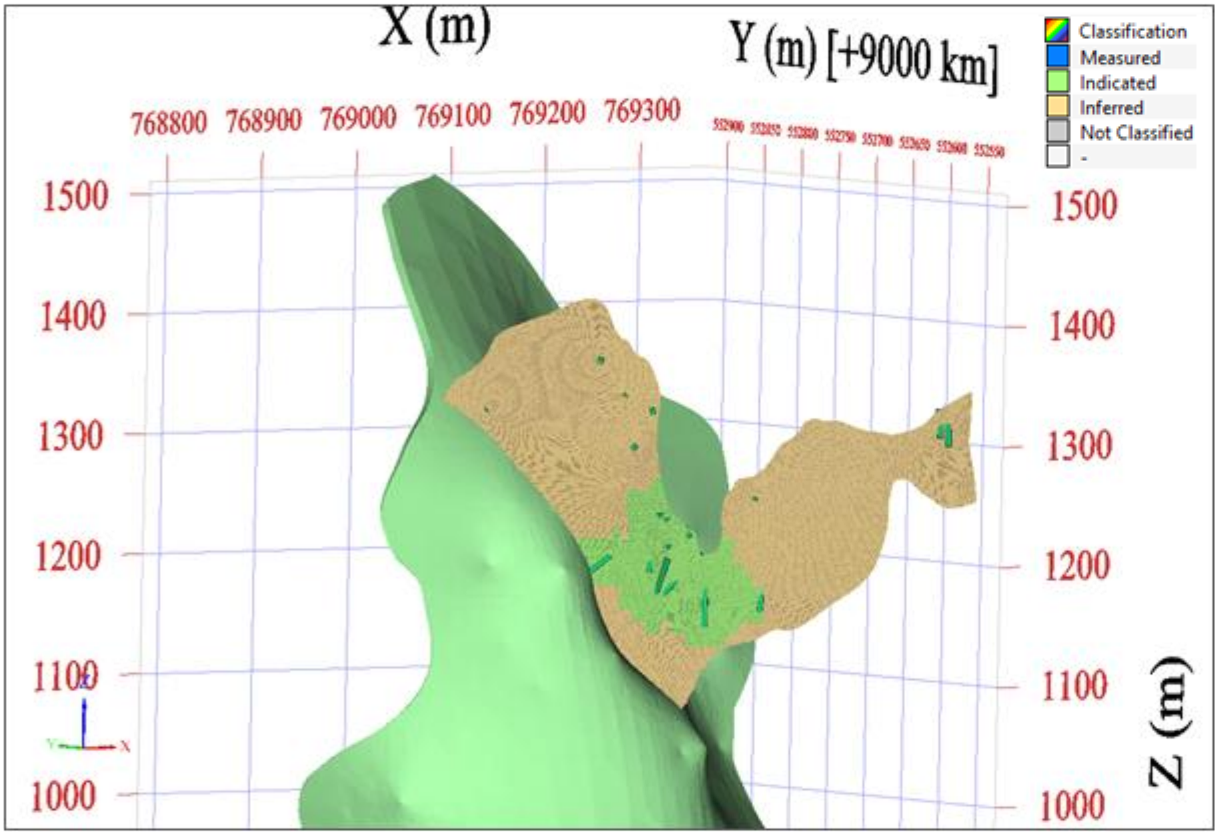
Figure 14-29: Section Showing Los Cuyes Domain NW5 Classification



Notes: Wireframe of domain LCW shown for context

NW1 is a larger domain, with nineteen intersections, several of which are in the thicker central part of the domain, resulting in a relatively large number of samples in the domain. The drilling density is variable over the extent of the domain, with some areas having very closely spaced data (< 10 m between intersections), and other areas with intersection spacings greater than 150 m. For NW1 the blocks which are estimated in the first search pass, which are informed by more than six composites, and for which the average distance to the informing composites is less than 40 m support classification as Indicated Mineral Resources. As is illustrated in Figure 14-30 there is a portion of the domain with closely spaced drilling in a limited area. Although there are block in this area that meet the above criteria, this area is otherwise poorly informed and does not support an Indicated classification. The remainder of the estimation domain is classified as an Inferred Mineral Resource.

Figure 14-30: Section Showing Los Cuyes Domain NW1 Classification

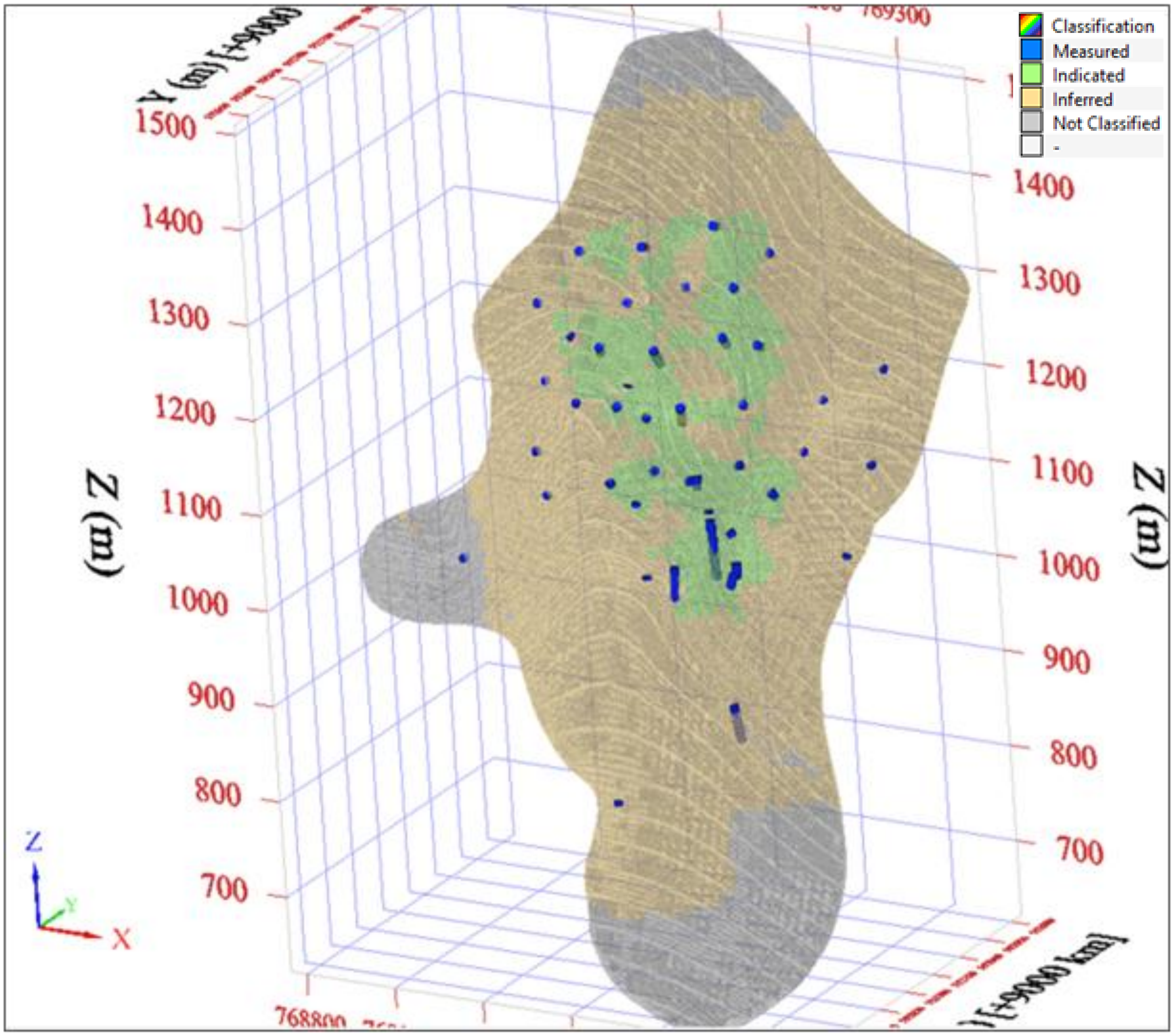


Notes: Wireframe of domain LCW shown for context

The same criteria discussed for domain NW1 are applied for domain NW2, which results in a central portion of the domain supporting an Indicated classification, with the majority of the wider spaced domain classified as an Inferred Mineral Resource.

The largest domain at Los Cuyes is the LCW domain. This domain has the most intersections and is relatively thicker than the NW group of domains. The core of the deposit is relatively well drilled, however the semi-variogram ranges are not long relative to the drill hole spacing (Figure 14-31). In the model densely drilled area the drill hole spacing approximates 30 to 60 m. For the LCW domain blocks which have a slope of regression of greater than 0.7, are estimated in the first or second search pass, have an average distance of 40 m to the informing composites and are estimated with at least 8 composites support an Indicated classification. Blocks which have a slope of regression of greater than 0.5 and are estimated within search passes 1 to 3 are classified as Inferred Mineral Resources. Blocks behind this are not classified as Mineral Resources (Figure 14-31).

Figure 14-31: Section Showing Los Cuyes Domain LCW Classification



Notes: Drill holes plotted in blue

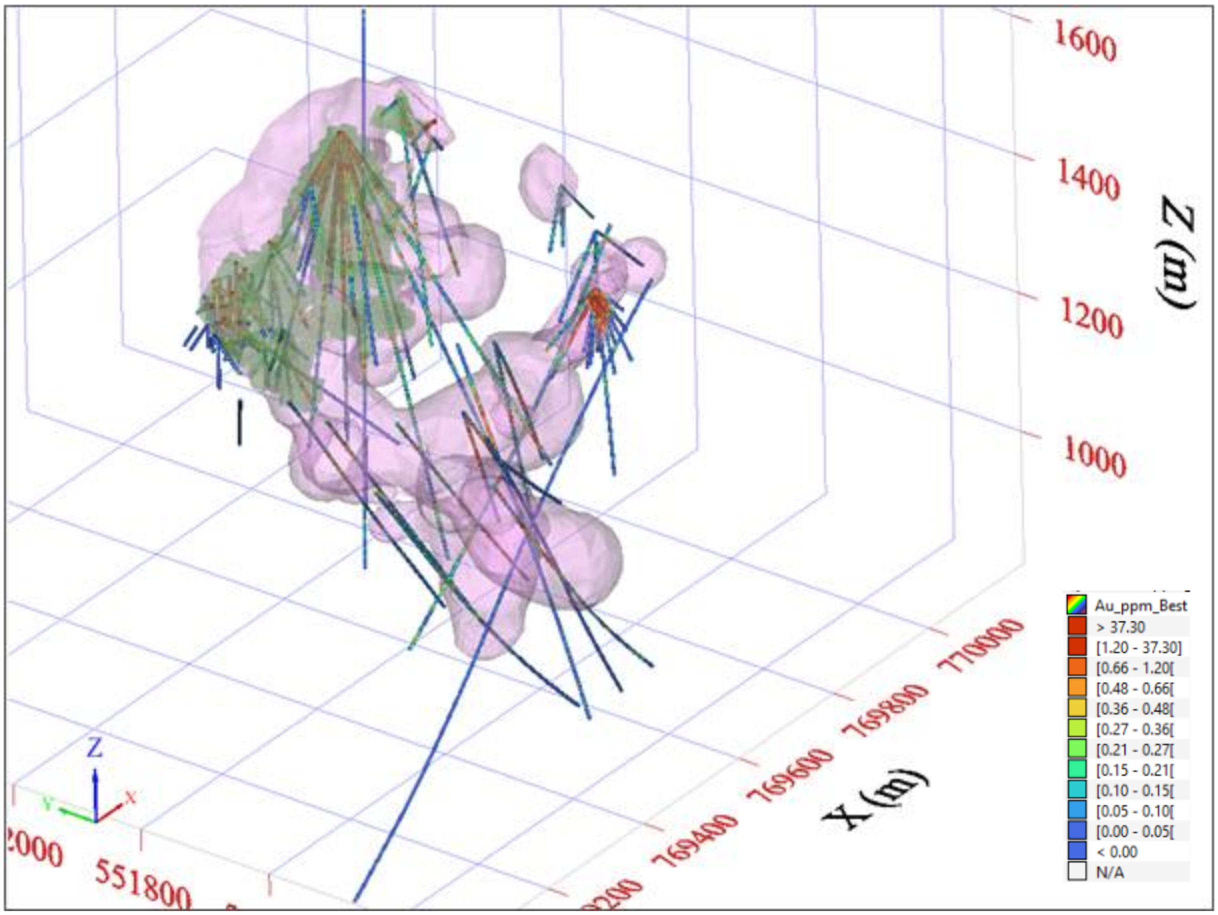
Finally, for the disseminated halo domain, blocks estimated in the first search pass, with an average distance to composites of 40 m or less and estimated with a minimum of six composites support an Indicated classification. Beyond these blocks, all blocks estimated in the first or second search pass are classified as Inferred Mineral Resources.

14.11.3 Soledad

The mineralization controls and geological framework at Soledad are not well understood at present. There is no detailed lithological model available for this area. The constraints on the mineralization are limited to the grade shell that is modelled by Silvercorp. The majority of the drilling is concentrated in three area, with a smaller number of wider spaced holes. The continuity modelled for gold is lower than that of the other variables modelled, particularly the base metals which have relatively longer ranges of continuity. In the densely drilled areas the drill hole spacing is in places as close as 10 m (Figure 14-32). There are areas where the grade is consistently elevated, which coincides with the densest drilling in many instances.

No Measured Mineral Resources are classified at Soledad. Blocks which have a slope of regression of greater than 0.6 for silver and greater than 0.5 for gold, have an average distance of less than 50 m to the informing composites, and are estimated in the first search pass support an Indicated classification. The Inferred classification is limited to blocks within 75m of a composite sample. The majority of the grade envelope is classified as either Indicated or Inferred. Only 14% of the volume within the grade envelope does not meet these criteria and is not classified.

Figure 14-32: Section Showing Soledad Indicated Mineral Resource Classification



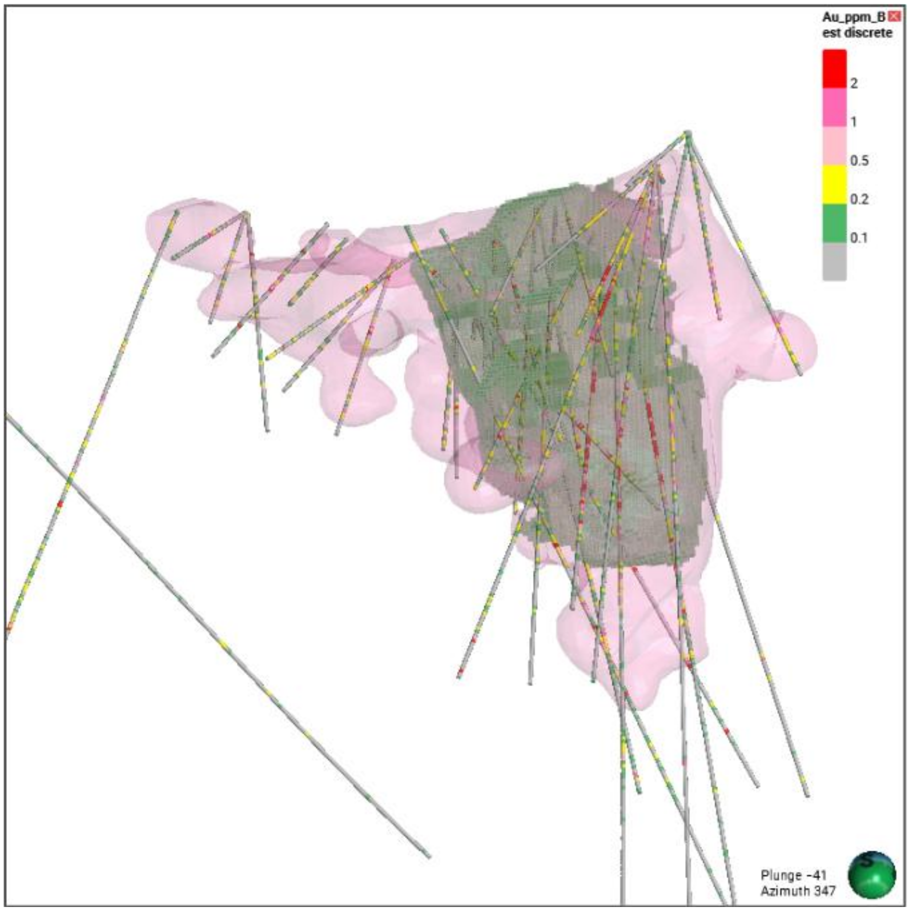
Notes: Drill holes are coloured according to gold grade. The grade envelope is shown in pink, and the blocks classified as Indicated Mineral Resources are shown in green

14.11.4 Enma

There is no detailed lithological model available for Enma. The constraints on the mineralization are limited to the grade shell that is modelled by SRK. In the densely drilled areas the drill hole spacing is in places as close as 10 m (Figure 14-33). There are areas where the grade is consistently elevated, which coincides with the densest drilling in many instances.

No Measured Mineral Resources are classified at Enma. Blocks which have a slope of regression of greater than 0.7 for gold, have an average distance of less than 30 m to the informing composites, and are estimated in the first search pass support an Indicated classification. The remainder of the estimation domain is classified as an Inferred Mineral Resource.

Figure 14-33: Section Showing Enma Indicated Mineral Resource Classification



Notes: Drill holes are colored according to gold grade. The grade envelope is shown in pink, and the blocks classified as Indicated Mineral Resources are shown in green

14.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a Mineral Resource as:

"A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. ("RPEEE").

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

For all the Condor projects, because the mineralization occurs relatively close to surface, the use of a pit optimisation shell is an acceptable standard approach used in industry for Mineral Resource reporting purposes to ensure that the Mineral Resource is tested for RPEEE. The Company considered future operation on Soledad and Enma using surface mining. However, at Camp and Los Cuyes Silvercorp consider underground mining to be a preferred approach due to the steep terrain, relative complexity, high grade tabular mineralization.

The pit optimization parameters reflect a conventional open pit operation with the cost and revenue assumptions on Soledad and Enma detailed in Table 14-26 below. Note that the parameters used are not related to any mine plan or financial analysis, they were used only to define the RPEEE envelope, and the figures were derived from current information.

The commodity prices are sourced from an independent analyst, Consensus Market Forecasts (CMF) for gold, silver, lead, and zinc. The projected outlook (in real USD) was issued by CMF in November 2025. A Resource premium of 15% was chosen over the long-term prices for the RPEEE.

The “RPEEE” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that the Soledad and Enma deposits are amenable for open pit extraction.

The optimization parameters were selected based on experience and benchmarking against similar projects. The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate Mineral Reserves. There are no Mineral Reserves on the Condor project. The results are used as a guide to assist in the preparation of a Mineral Resource Statement and to select an appropriate resource reporting cut-off grade.

Table 14-26: Pit Shell Optimization Inputs for RPEEE

Whittle Inputs	Unit	Enma	Soledad
Costs			
Mining Cost	USD/t Material	3	3
Processing Cost	USD/t ROM	20	20
General & Admin	USD/t ROM	12	12
Average Processing Recovery Rates			
Au	%	75	90
Ag	%	68	80
Zn	%	0	0
Pb	%	0	0
Payability			
Au	%	99.5	99.5
Ag	%	99.5	99.5
Zn	%	-	-
Pb	%	-	-
Commodity Prices			
Gold	USD/oz	3,000	3,000
Silver	USD/oz	40	40
Zinc	USD/t Metal	3,220	3,220
Lead	USD/t Metal	2,300	2,300
Royalty	**% of Revenue**	**3.00%**	**3.00%**
Overall Slope Angle	**degree**	**45**	**45**
Cut off grade (AuEq)	**g/t**	**0.5**	**0.4**

Sources: CMF metal price projections. SRK benchmarks and assumptions
Notes: Pit slope angle are assumed and are not based on a geochemical stability assessment.
Enma AuEq = Au (g/t) + Ag (g/t) x 0.01209
Soledad AuEq = Au (g/t) + Ag (g/t) x 0.01185

For the higher-grade and tabular domains at Camp and Los Cuyes, there is the opportunity using a bulk underground mining method such as long hole open stoping for extraction as detailed in section 16. For the underground Mineral Resources, SRK used a Mineable Shapes Optimiser (MSO) to outline areas of the mineralization domain that have suitable continuity and grade to sustain underground mining operations.

The block model quantities and grade estimates were also reviewed to determine the portions of the Camp and Los Cuyes deposits having “reasonable prospects for economic extraction” from an underground mine, based on parameters summarized in Table 14-27. Simplified MSO parameters, based on the mining study optimisations described in chapter 16, such as no pillars, no Equivalent Linear Overbreak / Sloughage, (ELOS), no distinguishment of P-S stopes, uniform stope length and maximizing stope width, were applied for the Mineral Resource optimisations at Camp and Los Cuyes.

Table 14-27: Underground Optimization Parameters for the Condor Project

Parameter	Product	Unit	Value	
Mine			**Camp**	**Los Cuyes**
Mill Recovery	Gold	%	98[1]	96[1]
	Silver	%	44[2]	50[2]
	Lead	%	38	34
	Zinc	%	60	35
Metal Price	Gold	US$/oz	3,000	
	Silver	US$/oz	40	
	Lead	US$/lb	1.05	
	Zinc	US$/lb	1.47	
Mining Cost		$/t milled	38.96	
Processing		$/t milled	19.11	
G&A		$/t milled	13.5	
Environmental and water		$/t milled	4.04	
Royalties		$/t milled	5.36	
Total offsite costs		$/t milled	2.51	

Notes:

[1] Cap values, recoveries are variable based on formulae in Table 16-14

[2] Values for Dore, Camp and Los Cuyes are 12.3 and 11.4 respectively in Pb conc and 8.7 and 1.6 respectively in Zn conc

Camp AuEq = Au (g/t) + Ag (g/t) x 0.00599 + Pb (%) x 0.2992 + Zn (%) x 0.66139

Los Cuyes AuEq = Au (g/t) + Ag (g/t) x 0.00694 + Pb (%) x 0.27328 + Zn (%) x 0.39385

Within the current mining license area, as of 30 November 2025, the Condor Project, Mineral Resources are constrained within mineable shapes for Camp and Los Cuyes planned for underground extraction; above a cut off of 0.5 g/t and 0.4 g/t for Enma and Soledad constrained with a conceptual pit, designed using Whittle software. The details of the estimated Mineral Resources are shown in Table 14-28 for Mineral Resources with underground mining potential, and in Table 14-29 for Mineral Resources with open pit mining potential.

Table 14-28: Underground Extraction Mineral Resource Statement for Condor Project as of 30 November 2025

		Average Grade					Contained Metal				
Deposit	Tonnes (Mt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (koz)	Au (koz)	Ag (koz)	Pb (lb'000)	Zn (lb'000)
Indicated											
Camp	5.93	2.46	1.94	15.51	0.06	0.61	468	370	2,956	7,914	79,864
Los Cuyes	4.22	2.07	1.84	11.06	0.05	0.36	280	249	1,500	4,301	33,067
Total	**10.15**	**2.30**	**1.90**	**13.66**	**0.05**	**0.50**	**748.9**	**620**	**4,456**	**12,215**	**112,931**
Inferred											
Camp	20.04	2.42	1.87	14.83	0.05	0.68	1,557	1,202	9,558	23,042	298,873
Los Cuyes	10.06	2.63	2.37	13.26	0.07	0.36	849	767	4,287	14,936	80,696
Total	**30.10**	**2.49**	**2.03**	**14.31**	**0.06**	**0.57**	**2406**	**1,969**	**13,846**	**37,978**	**379,569**

Notes:

1. Mineral Resources are reported within a MSO shape for Camp and Los Cuyes with no additional cut off value applied,. Including must take material. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. The Mineral Resources are reported on a 100% basis, and not the portion attributable to Silvercorp.
2. The resource statement does not include mineralization in the Halo domain of the Los Cuyes, and its economic potential remains to be further investigated in future studies. Optimisations are undertaken using a gold price of USD/oz 3,000, silver price of USD/oz 40, zinc price of USD/lb 1.47 and lead price of USD/lb 1.05.
3. 1 troy ounce = 31.1034768 metric grams.
4. 1 metric tonne = 2204.62 lb

Table 14-29: Open Pit Mineral Resource Statement for Condor Project, as of 30 November 2025

		Average Grade					Contained Metal				
Deposit	**Tonnes (Mt)**	**AuEq (g/t)**	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**AuEq (koz)**	**Au (koz)**	**Ag (koz)**	**Pb (lb'000)**	**Zn (lb'000)**
Indicated											
Soledad	4.63	1.06	0.98	6.86	0.05	0.54	158.0	146	1020	4,651	55,499
Enma	0.02	1.20	1.12	6.73	0.04	0.34	0.9	1	5	21	180
Total	**4.65**	**1.06**	**0.98**	**6.86**	**0.05**	**0.54**	**158.9**	**147**	**1025**	**4,672**	**55,679**
Inferred											
Soledad	19.99	0.73	0.66	5.97	0.04	0.46	467.8	422	3839	16,588	202,758
Enma	0.01	0.95	0.86	7.82	0.04	0.28	0.2	0	1	4	34
Total	**20.00**	**0.73**	**0.66**	**5.97**	**0.04**	**0.46**	**468.0**	**422**	**3841**	**16,592**	**202,792**

Notes:

1. Mineral Resources are reported in relation to a conceptual pit shell for Soledad and Enma. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. The Mineral Resources are reported on a 100% basis, and not the portion attributable to Silvercorp.
2. Open pit Mineral Resources are reported at a cut-off grade of 0.5 g/t AuEq for Enma and 0.4 g/t AuEq for Soledad. Open pit optimizations have been determined using a gold price of USD/oz 3,000, silver price of USD/oz 40, zinc price of USD/lb 1.47 and lead price of USD/lb 1.05.
3. 1 troy ounce = 31.1034768 metric grams.
4. 1 metric tonne = 2204.62 lb

14.13 Grade Sensitivity Analysis

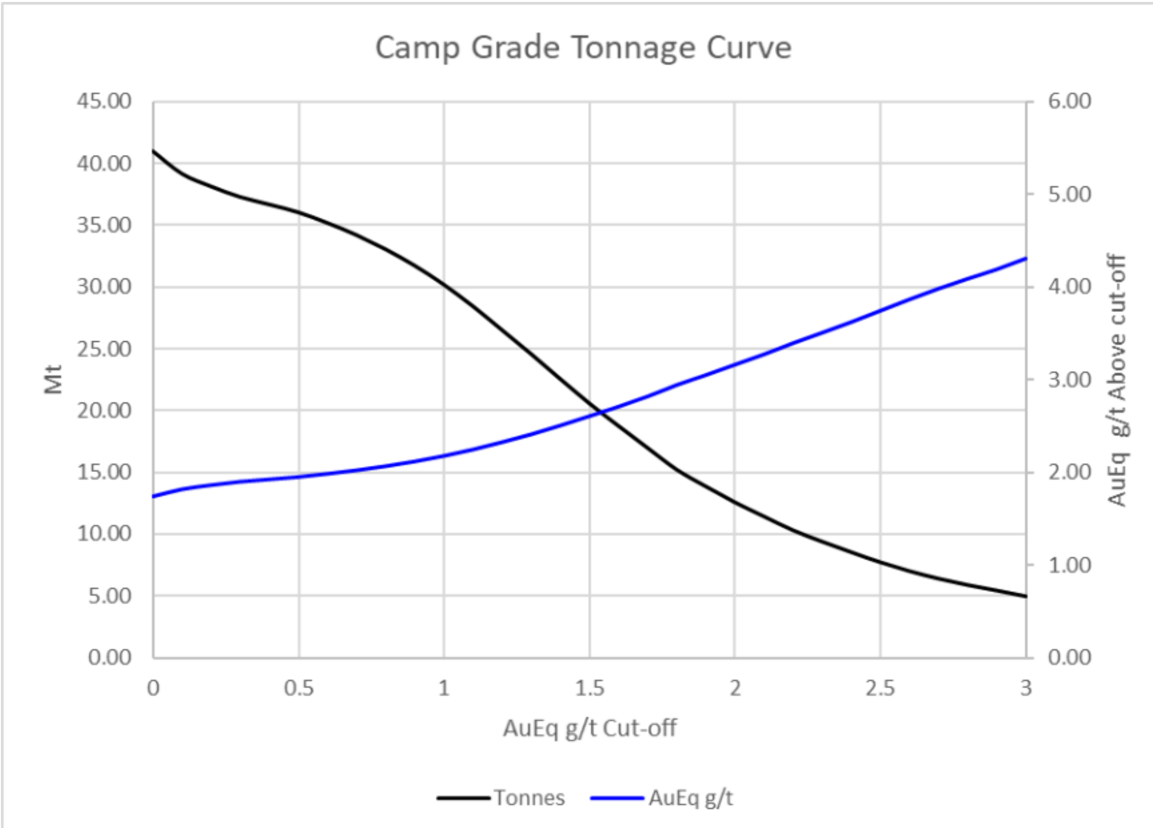
Mineral Resources are sensitive to the selection of cut-off grades. To illustrate this sensitivity, ore quantities and grade estimates at different cut-off grades are presented in Table 14-30 to Table 14-33. The reader is cautioned that the figures presented in this table should not be mistaken for a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. Figure 14-34 to Figure 14-37 represent this sensitivity as grade-tonnage curves.

Table 14-30: Global Block Model Quantities and Grade Estimates for Indicated and Inferred Category, Camp at Various cut-off Grades

Cut off	Tonnes	AuEq	Au	Ag	Pb	Zn
(AuEq g/t)	Mt	g/t	g/t	g/t	%	%
0	41.01	1.74	1.55	12.90	0.05	0.55
0.10	39.19	1.82	1.62	13.44	0.05	0.57
0.20	38.16	1.86	1.66	13.73	0.05	0.59
0.30	37.30	1.90	1.69	13.94	0.05	0.60
0.40	36.70	1.92	1.71	14.09	0.05	0.60
0.50	36.06	1.95	1.74	14.24	0.05	0.61
0.60	35.20	1.98	1.77	14.46	0.05	0.62
0.70	34.22	2.02	1.80	14.68	0.05	0.63
0.80	33.07	2.07	1.84	14.93	0.06	0.64
0.90	31.75	2.12	1.89	15.21	0.06	0.65
1.00	30.21	2.18	1.94	15.50	0.06	0.67
1.10	28.46	2.25	2.01	15.83	0.06	0.68
1.20	26.54	2.33	2.08	16.21	0.06	0.70
1.30	24.60	2.41	2.16	16.57	0.06	0.72
1.40	22.58	2.51	2.25	16.99	0.06	0.74
1.50	20.61	2.61	2.34	17.38	0.06	0.75
1.60	18.79	2.71	2.44	17.71	0.07	0.77
1.70	17.01	2.82	2.55	18.16	0.07	0.79
1.80	15.25	2.94	2.66	18.59	0.07	0.80
1.90	13.89	3.05	2.77	19.05	0.07	0.81
2.00	12.60	3.16	2.87	19.43	0.07	0.83
2.10	11.46	3.27	2.98	19.92	0.08	0.84
2.20	10.34	3.40	3.10	20.14	0.08	0.85
2.30	9.42	3.51	3.20	20.56	0.08	0.86
2.40	8.57	3.62	3.31	20.97	0.08	0.88
2.50	7.75	3.75	3.43	21.51	0.08	0.89
2.60	7.03	3.87	3.54	22.10	0.08	0.89
2.70	6.43	3.98	3.65	22.66	0.09	0.91
2.80	5.91	4.09	3.75	23.07	0.09	0.92
2.90	5.46	4.19	3.85	23.55	0.09	0.93
3.00	5.00	4.31	3.96	24.11	0.09	0.95

Notes: The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this tabulation are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource

Figure 14-34: Camp Deposit Global Grade Tonnage Curve



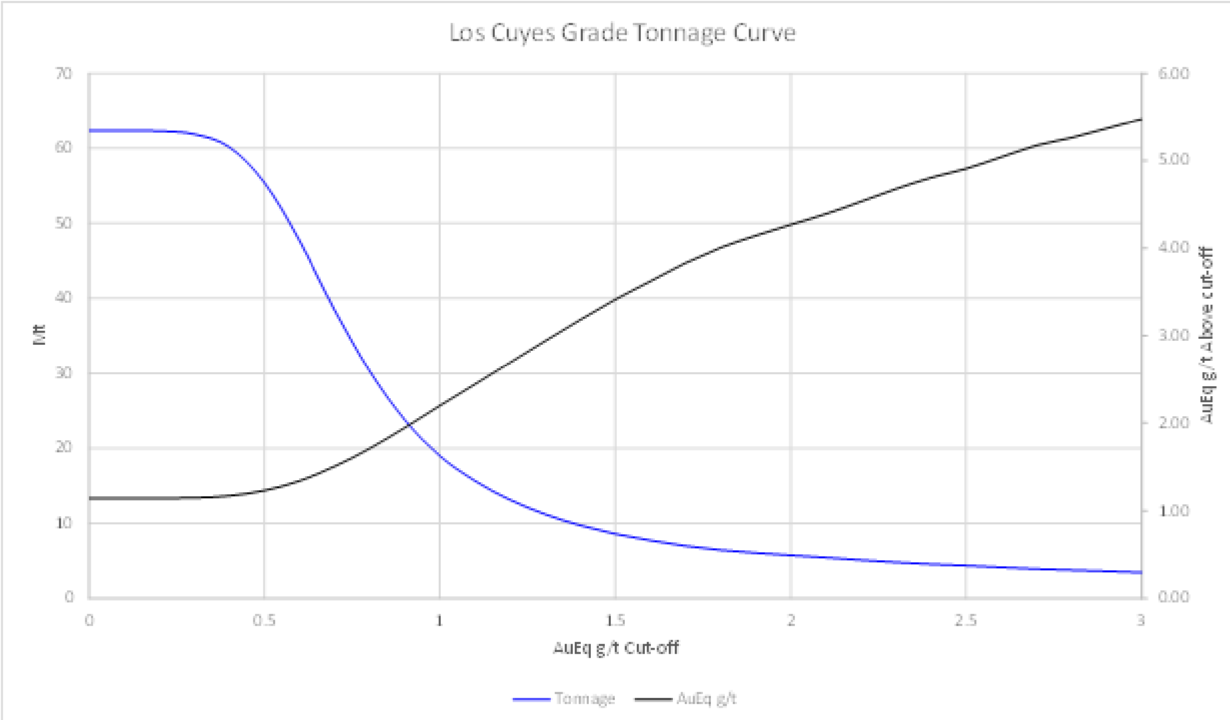
Notes: The reader is cautioned that the figures in this chart should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this chart are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource.

Table 14-31: Global Block Model Quantities and Grade Estimates for Indicated and Inferred Category, Los Cuyes at Various cut-off Grades

Cut off (AuEq g/t)	Tonnes Mt	AuEq g/t	Au g/t	Ag g/t	Pb %	Zn %
0	62.3	1.1	1.07	7.39	0.03	0.26
0.1	62.3	1.1	1.07	7.39	0.03	0.26
0.2	62.3	1.1	1.07	7.39	0.03	0.26
0.3	61.9	1.1	1.07	7.42	0.03	0.26
0.4	60.1	1.2	1.10	7.54	0.03	0.26
0.5	55.4	1.2	1.15	7.83	0.04	0.27
0.6	47.7	1.3	1.26	8.33	0.04	0.27
0.7	38.5	1.5	1.42	9.05	0.04	0.29
0.8	30.3	1.7	1.61	9.93	0.05	0.30
0.9	23.8	1.9	1.84	10.99	0.05	0.32
1	19.0	2.2	2.08	12.05	0.06	0.35
1.1	15.7	2.4	2.31	13.14	0.06	0.37
1.2	13.1	2.7	2.55	14.28	0.07	0.40
1.3	11.2	2.9	2.79	15.47	0.07	0.42
1.4	9.7	3.2	3.02	16.74	0.08	0.45
1.5	8.6	3.4	3.24	17.95	0.08	0.46
1.6	7.7	3.6	3.43	19.02	0.09	0.48
1.7	7.0	3.8	3.63	20.14	0.09	0.49
1.8	6.4	4.0	3.80	21.04	0.10	0.50
1.9	6.1	4.1	3.93	21.71	0.10	0.51
2	5.7	4.3	4.05	22.36	0.10	0.51
2.1	5.4	4.4	4.16	22.99	0.11	0.52
2.2	5.1	4.5	4.30	23.77	0.11	0.53
2.3	4.8	4.7	4.44	24.49	0.11	0.55
2.4	4.6	4.8	4.56	25.15	0.12	0.56
2.5	4.4	4.9	4.66	25.53	0.12	0.57
2.6	4.1	5.0	4.78	26.05	0.12	0.58
2.7	3.9	5.2	4.91	26.58	0.13	0.59
2.8	3.8	5.3	5.00	26.88	0.13	0.59
2.9	3.6	5.4	5.10	27.27	0.13	0.60
3	3.5	5.5	5.20	27.75	0.13	0.61

Notes: The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this tabulation are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource.

Figure 14-35: Los Cuyes Deposit Global Grade Tonnage Curve



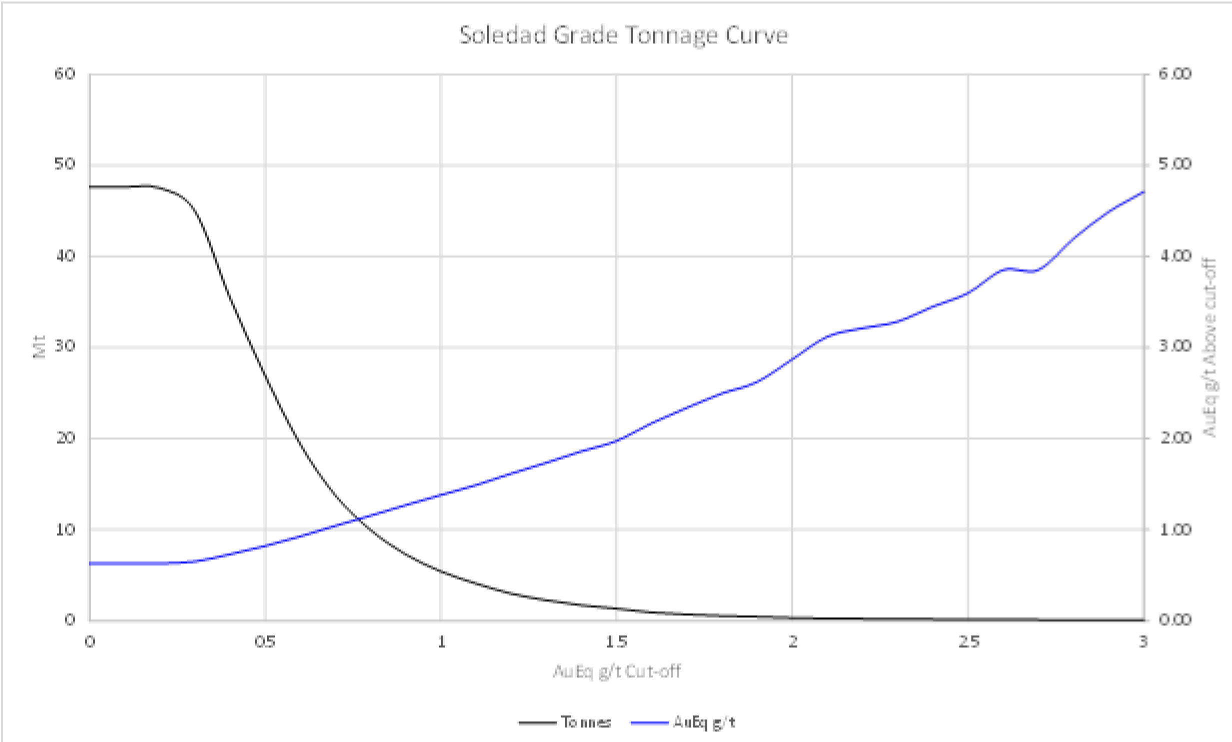
Notes: The reader is cautioned that the figures in this chart should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this chart are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource.

Table 14-32: Global Block Model Quantities and Grade Estimates for Indicated and Inferred Category, Soledad at Various Cut-off Grades

Cut off (AuEq g/t)	Tonnes Mt	AuEq g/t	Au g/t	Ag g/t	Pb %	Zn %
0	47.6	0.63	0.56	6.70	0.04	0.41
0.1	47.6	0.63	0.56	6.70	0.04	0.41
0.2	47.5	0.63	0.56	6.71	0.04	0.41
0.3	45.0	0.65	0.58	6.90	0.04	0.42
0.4	35.5	0.73	0.65	7.51	0.05	0.45
0.5	27.0	0.82	0.74	7.94	0.05	0.48
0.6	19.4	0.93	0.84	8.23	0.05	0.52
0.7	13.8	1.04	0.95	8.31	0.06	0.55
0.8	10.0	1.16	1.07	7.98	0.06	0.58
0.9	7.3	1.27	1.19	7.73	0.06	0.60
1	5.4	1.38	1.30	7.60	0.06	0.62
1.1	4.1	1.49	1.40	7.96	0.06	0.64
1.2	3.0	1.62	1.53	8.13	0.07	0.66
1.3	2.3	1.74	1.64	8.46	0.07	0.67
1.4	1.7	1.86	1.77	8.61	0.07	0.65
1.5	1.4	1.98	1.89	8.36	0.07	0.62
1.6	0.9	2.17	2.08	8.28	0.06	0.60
1.7	0.7	2.34	2.25	8.25	0.06	0.57
1.8	0.6	2.50	2.41	8.03	0.05	0.54
1.9	0.5	2.62	2.53	8.33	0.05	0.56
2	0.3	2.87	2.79	7.18	0.05	0.56
2.1	0.3	3.12	3.05	6.71	0.03	0.51
2.2	0.2	3.21	3.15	6.23	0.03	0.49
2.3	0.2	3.29	3.22	6.22	0.03	0.48
2.4	0.2	3.45	3.39	5.85	0.02	0.46
2.5	0.2	3.60	3.54	5.15	0.01	0.43
2.6	0.1	3.85	3.79	5.72	0.01	0.45
2.7	0.1	3.85	3.79	5.72	0.01	0.45
2.8	0.1	4.20	4.14	5.77	0.01	0.43
2.9	0.1	4.49	4.43	6.02	0.01	0.44
3	0.1	4.71	4.65	5.74	0.01	0.42

Notes: The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this tabulation are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource.

Figure 14-36: Soledad Deposit Global Grade Tonnage Curve



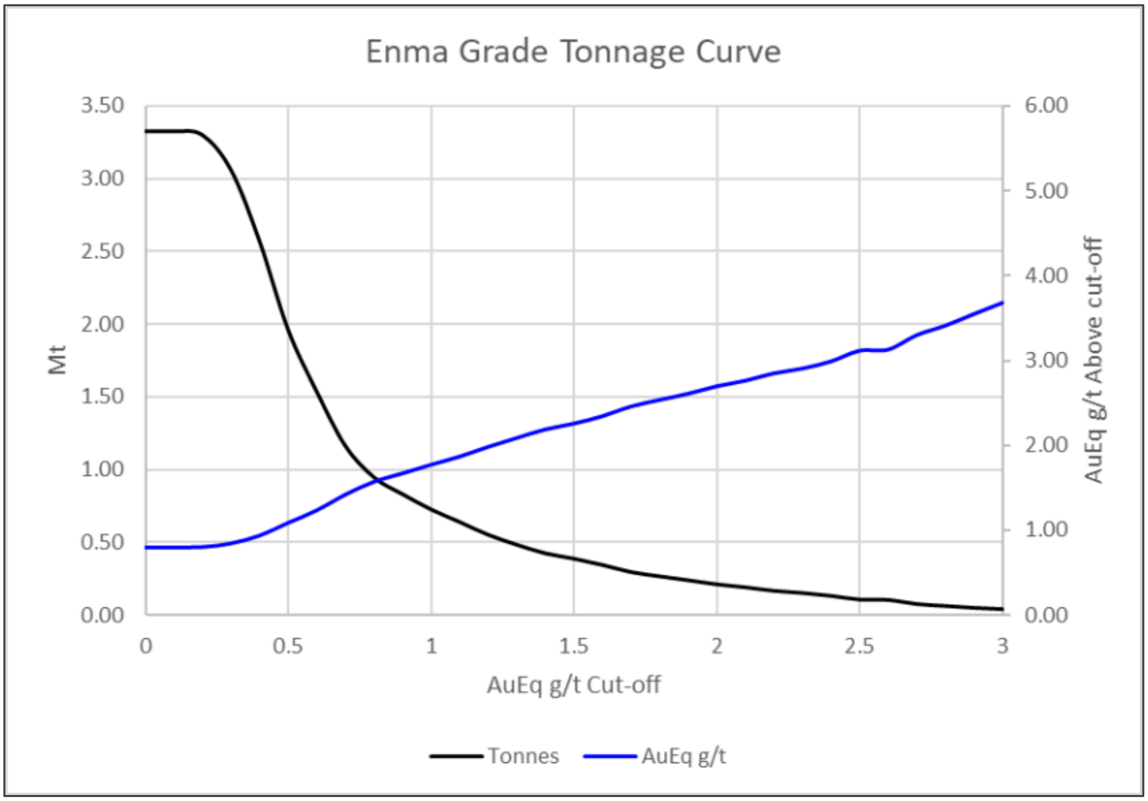
Notes: The reader is cautioned that the figures in this chart should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this chart are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource.

Table 14-33: Global Block Model Quantities and Grade Estimates for Indicated and Inferred Category, Enma at Various Cut-Off Grades

Cut off (AuEq g/t)	Tonnes Mt	AuEq g/t	Au g/t	Ag g/t	Pb %	Zn %
0	3.33	0.79	0.64	13.24	0.07	0.30
0.10	3.33	0.79	0.64	13.24	0.07	0.30
0.20	3.30	0.80	0.65	13.33	0.07	0.30
0.30	3.06	0.84	0.68	14.01	0.07	0.32
0.40	2.57	0.93	0.76	15.23	0.07	0.33
0.50	1.96	1.08	0.89	17.05	0.08	0.35
0.60	1.54	1.23	1.02	18.74	0.08	0.36
0.70	1.16	1.41	1.18	20.65	0.09	0.39
0.80	0.95	1.57	1.32	22.01	0.09	0.41
0.90	0.83	1.66	1.41	23.10	0.10	0.42
1.00	0.73	1.77	1.50	24.45	0.10	0.43
1.10	0.64	1.86	1.58	25.22	0.10	0.44
1.20	0.55	1.98	1.69	26.20	0.11	0.45
1.30	0.49	2.08	1.78	27.26	0.11	0.46
1.40	0.43	2.18	1.87	28.21	0.11	0.46
1.50	0.39	2.25	1.93	28.91	0.11	0.46
1.60	0.35	2.34	2.01	29.98	0.11	0.47
1.70	0.30	2.45	2.11	31.14	0.12	0.48
1.80	0.27	2.53	2.18	31.83	0.12	0.49
1.90	0.24	2.61	2.25	32.37	0.12	0.49
2.00	0.21	2.69	2.32	33.59	0.12	0.46
2.10	0.19	2.76	2.38	34.07	0.12	0.46
2.20	0.17	2.85	2.46	35.06	0.11	0.46
2.30	0.15	2.90	2.51	35.45	0.12	0.47
2.40	0.13	2.99	2.59	35.74	0.11	0.45
2.50	0.11	3.11	2.71	36.59	0.12	0.45
2.60	0.11	3.13	2.72	36.87	0.11	0.45
2.70	0.08	3.30	2.87	38.26	0.12	0.46
2.80	0.06	3.41	2.97	39.34	0.13	0.49
2.90	0.05	3.54	3.09	40.69	0.12	0.47
3.00	0.04	3.68	3.20	43.31	0.11	0.42

Notes: The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this tabulation are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource

Figure 14-37: Enma Deposit Global Grade Tonnage Curve



Notes: The reader is cautioned that the figures in this chart should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. The tonnes reported in this chart are not limited by the reasonable prospects of eventual economic extraction that must be applied to a Mineral Resource.

15 Mineral Reserve Estimates

At the current stage, there are no Mineral Reserves declared for the Condor Project. To support a Mineral Reserve estimate, a prefeasibility study or a feasibility study is required.

16 Mining Methods

This section summarizes the mine design and planning work completed to support the Condor PEA including the plant feed schedule. The underground mine planning work was prepared by Mr. Benny Zhang, MEng, PEng (PEO#100115459) of SRK, the Qualified Person taking professional responsibility, and Mr. Eric Wu, PEng (PEO#100604418). The rock geotechnical assessment for the proposed underground mine was undertaken by Ross Greenwood. The hydrogeological study for the planned underground mine was prepared by Dr. Tom Sharp, PEng (#36988), the Qualified Person taking professional responsibility. The ventilation modelling was undertaken by Brian Prosser, PEng (#15465). The underground mining infrastructure requirement was prepared by Mr. Sean Kautzman, PEng (PEO#100159892), the Qualified Person taking professional responsibility. All the Qualified Persons are SRK employees and associates.

The objective of this preliminary economic assessment is to determine the potential economic viability of the Condor project at a scoping level.

The Condor project consists of an underground mine with a mine life of 12-13 years and a processing plant. The maximum mill feed rate is set at 1.8 million tonnes per annum (Mtpa), or 5,000 tonnes per day (tpd).

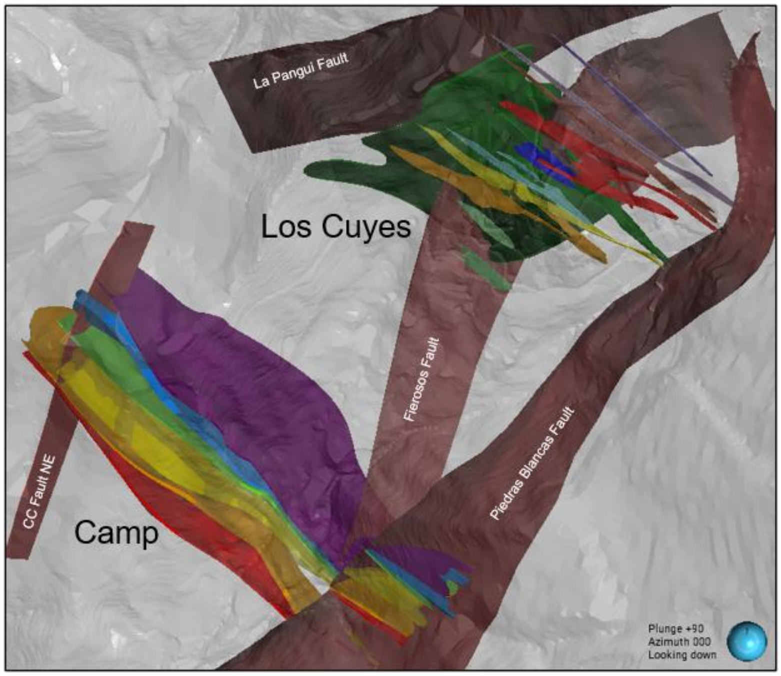
16.1 Mine Geotechnical

16.1.1 Geotechnical Context

Figure 16-1 shows the general layout of the Camp and Los Cuyes areas. Each area is made up of a series of veins, at Camp extending from near surface to approximately 1,000 m depth over a strike distance of 600 m, while at Los Cuyes extending from near surface to 730 m depth with a strike length of 470 m.

Four faults have been modelled in proximity to vein areas, which truncate the Camp area, and cross-cut and parallel the Los Cuyes area. A relatively shallow weathering profile exists across the area which is generally less than 30 m deep.

Figure 16-1: Plan View of the Camp and Los Cuyes Modelled Mineralized Vein Systems and Faults



16.1.2 Camp Evaluation

Geology

The Camp mining area comprises six parallel northwest trending near-vertical (~80° dip) dykes or vein type structures which have a relatively continuous and undulating geometry (named CA-01 to CA-06). The veins are concentrated at the contact between a volcanic/intrusive complex and a major granodiorite intrusion.

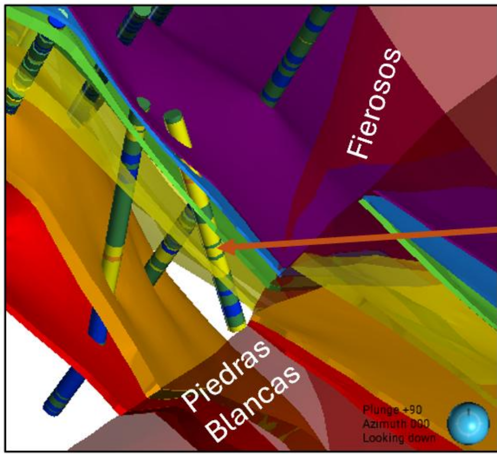
A saprolite-bedrock surface has been developed for the Camp area that generally indicates 30m of weathered soils and rock. Vein width at surface varies from 3mW to ~40 mW (CA03 vein, yellow).

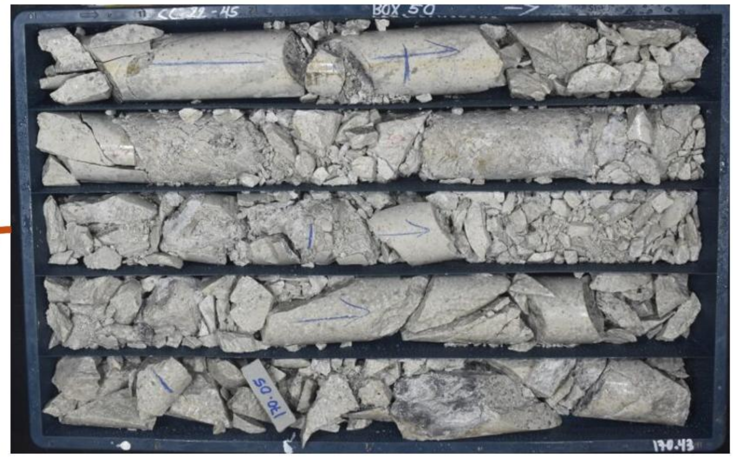
Major Structures

Structural mapping was completed by Specialized Geologic Mapping Ltd (2020) that indicates predominantly northeast trending faulting, dipping steeply to the east. Three major fault structures have been modeled which are approximately perpendicular to the CA vein system: the CC Fault NE and Piedras Blancas Fault form the approximate west and east extents of the vein systems; and the Fierosos Fault offsets the veins at the eastern extent.

Based on a review of drill core logging and core photos, there are not many damaged core intervals that would suggest there are more major fault structures intersecting the CA vein system. However it should be noted that the drilling orientation (~parallel to major faults) is not ideal for the identification of structures. As an example, drillhole CC22-45 was drilled oblique to vein orientation close to the intersection of the Fierosos and Piedras Blancas Faults, with generally poorer ground conditions observed in this hole (Figure 16-2). This area of the vein and fault system should be further reviewed for potential damage zones which could affect stoping performance.

Figure 16-2: Example of Poor Ground Conditions from Drillhole CC22-45 Drilled Oblique to the CA Fault System, and in Close Proximity to Major Fault Structures





Geotechnical Data

Geotechnical data has been collected along approximately 38 drillholes which are reasonably well distributed across the planned mining area, with detailed geotechnical parameters logged sufficient for the calculation of rock mass rating (RMR) (Bieniawski, 1989) and Q (Barton, 1990). Rock mass conditions are generally good with little variability observed between veins/mineralized zones, hangingwall, and footwall. Table 16-1 presents the calculated RMR89 parameters from drill core logging coded using the simplified lithology model.

Table 16-1: Camp Drill Core RMR89 Based on Simplified Lithology Model

Litho	Count (Ea.)	Length		RMR89	
		Metres	%	Mean	Std. Dev
GRD	5134	10736	49	79	12
RDCam	2236	4954	23	78	12
DACam	1517	2810	13	77	13
GST	1166	2373	11	82	10
RhyWT	537	985	4	74	15
DACNW	19	29	<1	75	19
		21,888		78	

Occasional altered and/or damaged zones are observed in drill core photos and geotechnical logging, however it is difficult to determine what the exact cause of this is based on the available data. A potential correlation between rock mass weaking alteration types and rock quality should be further investigated, and spatial geotechnical domains developed if possible. As an example, the damaged area proximal to the Fierosos Fault should be further evaluated during future studies to determine the cause.

A significant point load testing database is available for the Camp area, which indicates intact rock strengths for the dominant lithologies in the range 70 to 110 MPa when converted using a generic point load Is_{50} multiplier of x24 (Table 16-2). Specific correlation factors should be developed following the completion of a UCS testing program.

Table 16-2: Camp Point Load UCS Results Based on Simplified Lithology Model

Litho	Count		Point Load UCS (MPa)	
	Ea.	(%)	Mean	Std. Dev
GRD	1176	50	110	66
RDCam	495	21	68	47
DACam	279	12	75	40
GST	268	11	113	43
RhyWT	90	4	75	48
RhyNW	43	2	51	30
DACNW	3	<1	115	54
	2355		95	

Geotechnical Design Parameters

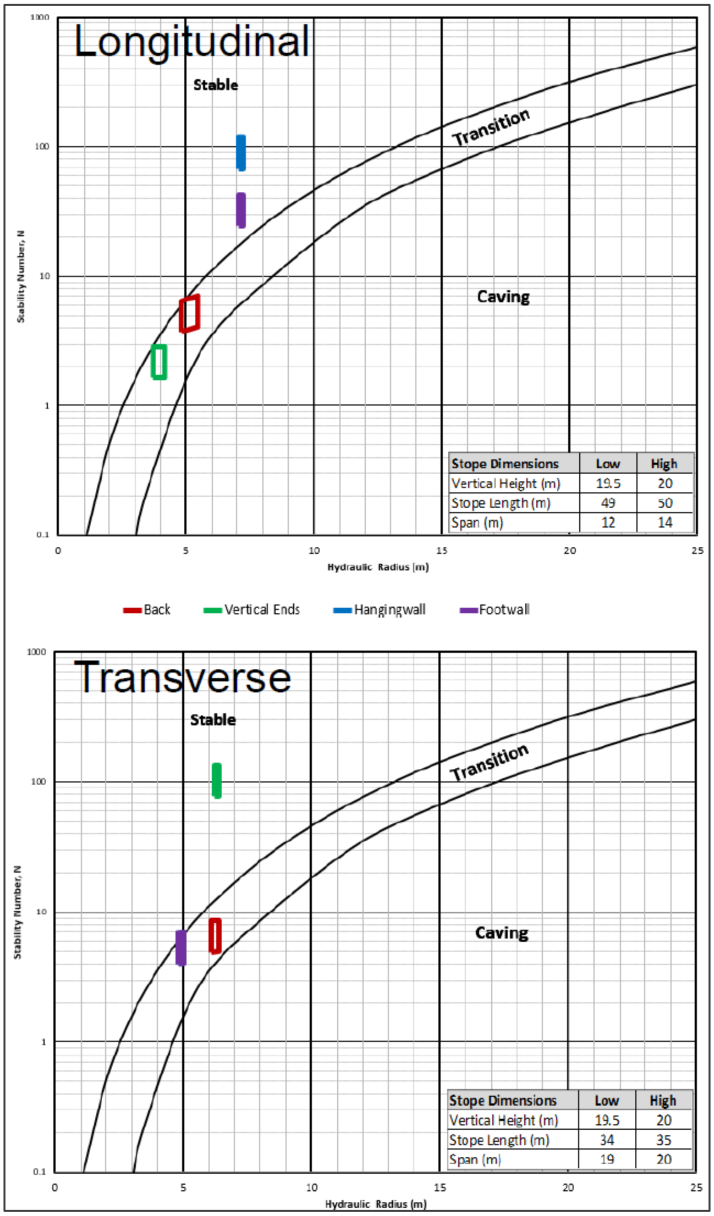
Geotechnical inputs for the development of mine design guidance include RMR89 data, joint systems assumed to be parallel to veins and major fault systems, point load test results, and estimated in situ stresses with horizontal stress k-ratios of 1.5 and 2.0 x sigma v.

- The design provided in Table 16-3 assumes that cable bolting will be required for all transverse stope backs, and for longitudinal stope backs where widths will exceed ~6 m.
- Backfill is assumed to be uncemented waste rock, and temporary rock pillars will be required between adjacent stopes to contain the waste rock.
- External dilution (represented as Equivalent Linear Overbreak / Sloughage, ELOS) is based on empirical stability graphs and rationalized against benchmark values from similar mining operations.

Table 16-3: Stope and Mine Design Geotechnical Guidance – Camp

Stope parameter	Longitudinal	Transverse
Level spacing (m)	20	20
Distance along strike (m)	35	20
HW-FW distance (m)	15	35
Surface crown pillar (mV)	40	40
Dilution ELOS (m)	0.9 (HW only)	0.5 (per long wall)
HR long wall (m)	6.3	6.3
Temporary Rib pillar (W:H ratio)	1:1 (minimum 3mW)	1:1 (minimum 5mW)
Permanent Sill Pillar (m)	Assume one level (20mH) with 50% recovery	

Figure 16-3: Stope Stability Graph Showing Preliminary Hydraulic Radii for Longitudinal (top) and Transverse (bottom) Stope Orientations



The design of rock pillars is based on the empirical pillar stability graph that includes zones defined by pillar yield and pillar failure but does not differentiate between temporary or permanent pillars. The current mine plan requires pillars that bridge between the HW and FW to contain uncemented waste rockfill. While these are common in longitudinal stoping scenarios where stope widths are generally less than ~8m, very few documented examples exist for using slender pillars in transverse stoping orientations where pillars could be 8mW × 15mH by up to 35mL, not considering overbreak. In these situations any persistent through-going structure exposed in the adjacent open stope poses a risk to the integrity and stability of the pillar. The risk is that the pillar fails allowing waste rock to enter the stope (additional dilution), potential for unstable stope back, and subsequently larger pillars.

Ground support requirements for lateral development have been recommended using the design guidelines presented by Potvin and Hadjigeoriou (2016) which relates ground conditions and tunnel span to bolt density, surface support type (and thickness), and support coverage based on the installed support systems at a wide range of mining operations.

For permanent development (>1yr lifespan) resin grouted rebar is recommended with wire mesh surface support, while inflatable type anchors (Swellex, Omega bolt or similar) and welded wire mesh are recommended for temporary headings. Support recommendations for permanent and temporary areas are provided in Table 16-4 and Table 16-5 .

Table 16-4: Ground Support for Permanent Lateral Development and Intersections

Aspect	Support Recommendation
5.0mW × 5.0mH up to 5.1mW × 5.4mH	2.4 m #7 (7/8") rebar in back, shoulders, upper walls with full resin encapsulation 2.4 m FS-39 galvanized split set in bottom row 1.2 m × 1.2 m spacing (offset rings) 6 Ga. 4" welded wire mesh across back and walls to 1.5 m from floor Fibrecrete: 50 mm applied across back and walls to 1.5 m from floor in 10% of development
Max. 9 m inscribed diameter	3-way intersections recommended 6.0 m twin-strand cable anchors 2.0 × 2.0 m spacing 9 anchor pattern, 3 rings of 3 bolts installed in center of intersection

Table 16-5: Ground Support for Temporary Development and Stoping

Aspect	Support Recommendation
Ore Drives (two profiles) & Longitudinal stopes	**Primary Support** Ore Drive 3.0mW: 1.8 m PM16 plain inflatable anchor in back and shoulders to 1.5 m above floor Ore Drive 4.5mW: 2.1 m PM16 plain inflatable anchor in back and shoulders to 1.5 m above floor 1.2 m × 1.2 m square pattern (offset rings) 6 Ga. galvanized welded wire mesh across back and walls to 1.5 m above floor
	Secondary Support (stope widths >6.0mW) 6.0m twin-strand cable anchor 2.4 m × 2.0 m square pattern 5 anchors across the back
Cross Cuts & Transverse stopes	**Primary Support** Cross Cut 4.5mW: 2.1 m PM16 plain inflatable anchor in back and shoulders to 1.5 m above floor 1.2 m × 1.2 m square pattern (offset rings) 6 Ga. galvanized welded wire mesh across back and walls to 1.5 m above floor
	Secondary Support (all transverse stopes) 8.0m twin-strand cable anchor 2.4 m × 2.0 m square pattern 5 anchors across the back

16.1.3 Los Cuyes Evaluation

Geology

The Los Cuyes mining area comprises twelve northwest trending subvertical (~70° dip) and roughly parallel veins, which are truncated by the NE trending LCW vein. The veins are variable in dimensions and continuity with waste holes within some veins, and are with a more complex geology (compared to Camp) where the veins are within a system of tuffs, breccias, and volcanic sandstone package.

Similar to Camp area, there does not appear to be a deep weathering surface with weathering logging indicating variable oxidation depth which is generally less than 20m deep. Core photos support the logging data however there are some more broken-rock zones closer to surface with oxidation on joint surfaces.

Major Structures

Three major fault structures are modeled in the Los Cuyes area all oriented sub-parallel to the LCW vein. There are other damaged zones recorded in drill core logs and observed in photos which could be major structures or coincident with lithology contacts. Structures that are oriented perpendicular to the LCW vein (parallel to the may be challenging to distinguish due to the litho contacts and NW vein sequence.

Due to the current drillhole density there are not many examples of damage zones associated to the Fieros fault. It is recommended that the major structures model is updated to include potential NW trending structures, including a structure description matrix.

Geotechnical Data

More recent drilling has provided a reasonable geotechnical dataset across most of the NW and LCW veins however no geotechnical data is available for the NW11, 13, and 15 veins. Detailed relogging or photo interpretation of historic holes (where RQD data was identified as unreliable) is recommended to infill this data gap.

The dominant vein host rocks characterized by generally fair to good rock mass conditions, as presented in Table 16-6. Throughout the geotechnical data there is evidence of adverse rock matrix alteration or matrix weakening associated with some (potentially) geological contacts which results in the presence of poor ground conditions; however these are not well represented in the geotechnical statistics compared to the distribution observed in core photos. When compared to Camp, RMR89 values are generally ten points lower, with a wider standard deviation.

Intervals of reduced rock quality are occasionally concurrent with the NW veins and may locally affect HW and FW stability. Additional domaining potentially could be completed using updated alteration/ structure wireframes to better identify areas of fair ground.

Table 16-6: Los Cuyes Drill Core RMR89 Based on Simplified Lithology Model

Litho	Count (Ea.)	Length		RMR_{89}	
		Metres	%	Mean	Std. Dev
RhyLT	2349	5326	40	71	14
GRD	1069	2777	21	76	14
DACNW	923	2514	19	73	15
DAC	249	714	5	81	13
RD_1	202	533	4	75	13
RhyNW	204	557	4	76	12
GST	145	352	3	74	15
RIII	232	422	3	62	14
RhyWt	11	34	<1	82	9
		13,233		**74**	

A significant point load testing database is also available for the Los Cuyes area, which indicates intact rock strengths for the dominant lithologies are in the range 80 to 90 MPa when converted using a generic point load Is_{50} multiplier of x24 (Table 16-7).

Table 16-7: Los Cuyes Point Load UCS Results Based on Simplified Lithology Model

Litho	Count		Point Load UCS (MPa)	
	Ea.	(%)	Mean	Std. Dev
RhyLT	445	37	80	38
GRD	272	23	91	58
DACNQ	246	21	90	54
DAC	67	6	86	51
RD_1	50	4	81	59
GST	30	3	119	54
RhyNW	50	4	50	29
RIII	29	2	58	39
RhyWt	4	<1	80	49
	1,193		84	

Geotechnical Design Parameters

Geotechnical inputs for the development of mine design guidance include RMR89 data, joint systems assumed to be parallel to veins and major fault systems, point load test results, and estimated in situ stresses with horizontal stress k-ratios of 1.5 and 2.0 x sigma v.

- The design provided in Table 16-8 assumes that cable bolting will be required for all transverse stope backs.
- Longitudinal stope backs where widths will exceed ~5 m will require longer primary support.
- Backfill is assumed to be uncemented waste rock, and temporary rock pillars will be required between adjacent stopes to contain the waste rock.

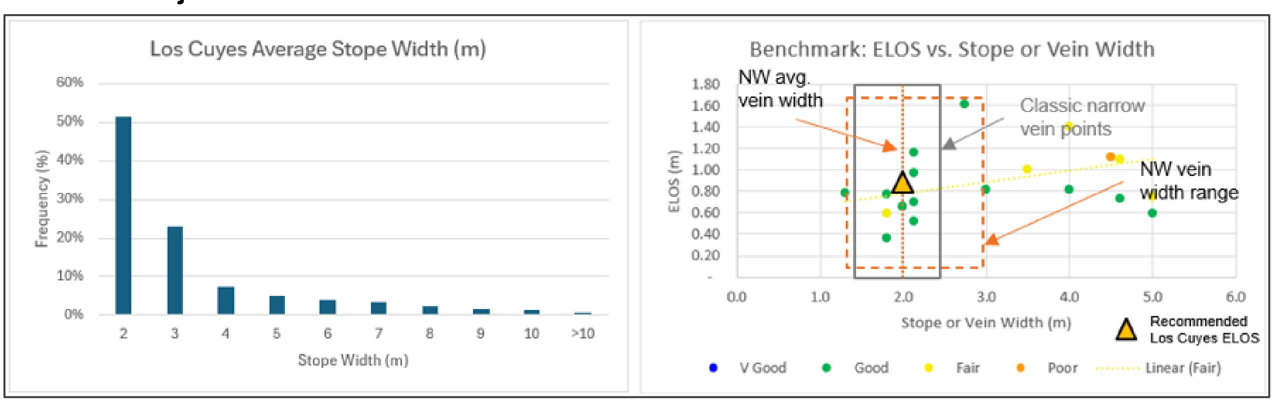
Table 16-8: Stope and Mine Design Geotechnical Guidance – Los Cuyes

Stope Parameter	Longitudinal	Transverse
Level spacing (m)	20	20
Distance along strike (m)	15	12
HW-FW distance (m)	<10	18
Surface crown pillar (mV)	40	40
Dilution ELOS (m)	0.9 (HW only)	0.5 (per long wall)
HR long wall (m)	4.2	4.7
Temporary Rib pillar (W:H ratio)	1:1.2 (minimum 3mW)	1:1.2 (minimum 5mW)
Permanent Sill Pillar (m)	Assume one level (20mH) with 40% recovery	

The average longitudinal stope width at Los Cuyes is 2.0m (70% all stopes are <3.0m wide); 9 of the 11 NW veins have an average width between 1.5m and 3.0m. The following points describe the benchmark considerations around dilution in this context (Figure 16-4).

- Classic narrow vein cases (<2.0m width) indicate a small amount of overbreak has a very significant impact to dilution; as veins/stope widths become wider they become less sensitive to dilution.
- Data from mines with good ground generally have lower dilution, fair ground conditions generally have higher dilution, but there is a wide spread of data.
- The best benchmark correlation is with fair ground data points, indicative of some of the near vein conditions at Los Cuyes.
- Hangingwall stability becomes increasingly sensitive to factors including blast design (drill pattern, charging), stope layout (location and dimensions of drill and mucking drives), drill deviation, and blast damage.
- The benchmark data set indicates dilution between 0.8 and 1.0m primarily from the hangingwall, is appropriate.

Figure 16-4: Distribution of Los Cuyes Stope Widths (left) and Comparison to Similar Benchmarked Projects



The ground support for permanent development (>1 year lifespan) includes a recommendation for resin grouted rebar with wire mesh surface support. Inflatable type anchors (Swellex, Omega bolt or similar) and welded wire mesh are recommended for temporary headings, and also includes an allocation of fibercrete for weaker ground areas. Support recommendations for permanent and temporary areas are provided in Table 16-9 and Table 16-10 .

Table 16-9: Ground Support for Permanent Lateral Development and Intersections

Aspect	Support Recommendation
5.0mW x 5.0mH up to 5.1mW x 5.4mH	2.4 m #7 (7/8") rebar in back, shoulders, upper walls with full resin encapsulation 2.4 m FS-39 galvanized split set in bottom row 1.2 m x 1.2 m spacing (offset rings) 6 Ga. 4" welded wire mesh across back and walls to 1.5m from floor Fibercrete: 50 mm applied across back and walls to 1.5 m from floor in 10% of development
Max. 9 m inscribed diameter	3-way intersections recommended 6.0 m twin-strand cable anchors 2.0 x 2.0 m spacing 9 anchor pattern, 3 rings of 3 bolts installed in center of intersection

Table 16-10: Ground Support for Temporary Development and Stoping

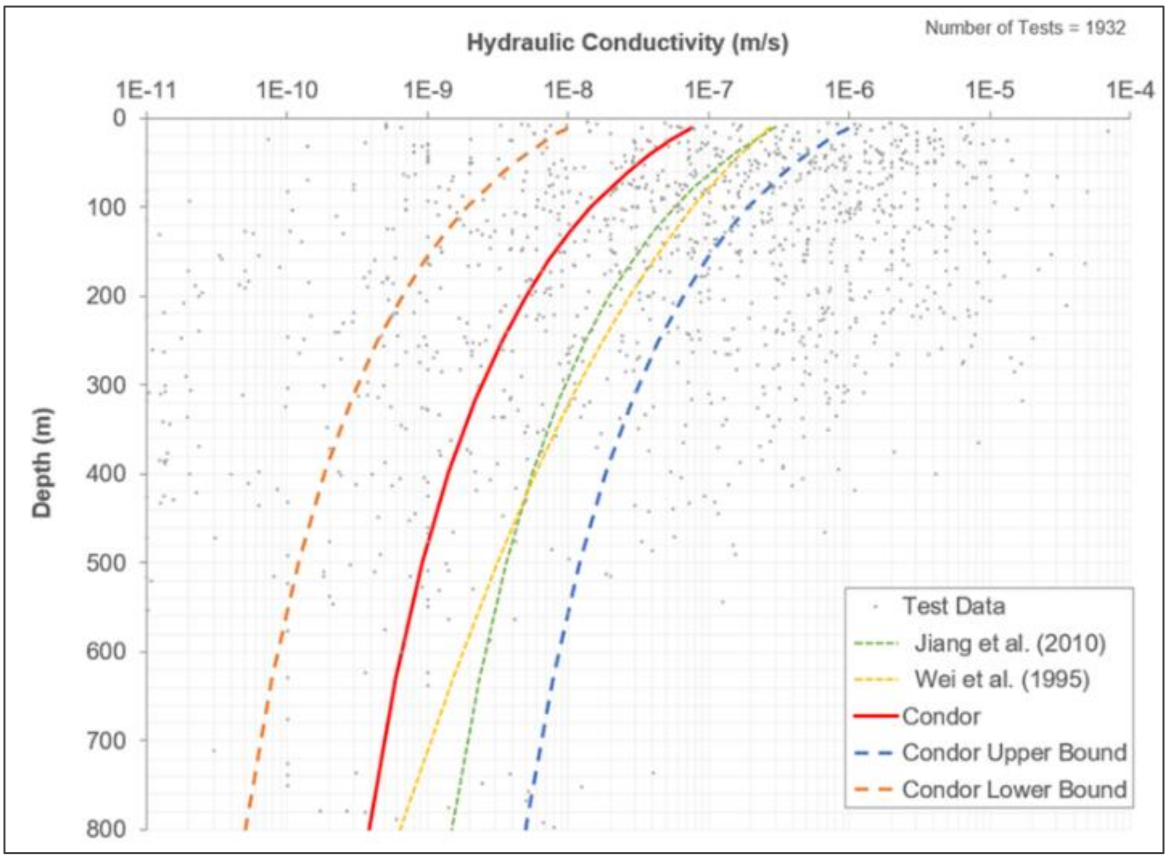
Aspect	Support Recommendation
Ore Drives (two profiles) & Longitudinal stopes	Primary Support Ore Drive 3.0mW: 1.8 m PM16 plain inflatable anchor in back and shoulders to 1.5 m above floor Ore Drive 4.5mW: 2.1 m PM16 plain inflatable anchor in back and shoulders to 1.5 m above floor 1.2 m x 1.2 m square pattern (offset rings) 6 Ga. galvanized welded wire mesh across back and walls to 1.5 m above floor Fibercrete: 50mm applied across back and walls to 1.5 m from floor in 15% of ore drives
	Secondary Support (stope widths >5.0mW) 6.0m twin-strand cable anchor 2.4 m x 2.0 m square pattern 5 anchors across the back
Cross Cuts & Transverse stopes	Primary Support Cross Cut 4.5mW: 2.1 m PM16 plain inflatable anchor in back and shoulders to 1.5 m above floor 1.2 m x 1.2 m square pattern (offset rings) 6 Ga. galvanized welded wire mesh across back and walls to 1.5 m above floor Fibercrete: 50mm applied across back and walls to 1.5 m from floor in 15% of ore drives
	Secondary Support (all transverse stopes) 8.0m twin-strand cable anchor 2.4 m x 2.0 m square pattern 5 anchors across the back

16.2 Hydrogeology

16.2.1 Available Data

No hydraulic conductivity (K) data exists for the deposits in the mine area. Values from literature for the primary rock type (granodiorite) in the area range from 2x10 -9 to 5x10-8 m/s (Singhal and Gupta, 2010). SRK has previously conducted groundwater investigations, including determinations of hydraulic conductivity values at the nearby Fruta del Norte mine, located approximately 30 km away. Bulk hydraulic conductivity values for similar geological units at Fruta del Norte mine generally ranged between 10-9 to 10-5 m/s (SRK 2016; NCL 2013). It is understood that the hydraulic conductivity of a rock mass will decrease with depth. To account for this, a hydraulic conductivity vs depth model was developed using an SRK database with the Jiang et al. (2010) and Wei et al. (1995) models, adjusted for hydraulic conductivity values observed at Fruta del Norte. This model is illustrated in Figure 16-5.

Figure 16-5: Hydraulic Conductivity vs Depth Model



Limited water level data is available in the mine area. Ausenco (2021a; 2021b) previously assumed that water levels in the mining areas were between 10 to 30 mbgs. This range was considered in the inflow estimates. It was assumed that the water table in the mining areas would decrease at a constant rate of 10 m per year to account for decreasing water levels caused by mine development draining the local groundwater.

16.2.2 Modelling Approach

Inflows of groundwater to the underground mine design were estimated using the Goodman (1965) approach and calculated using semi-transient conditions to account for increasing development length:

$$Q = \frac{2\pi K H_0}{2.3 log \frac{2H_0}{R}}$$

Where:

Q = Inflow per unit length

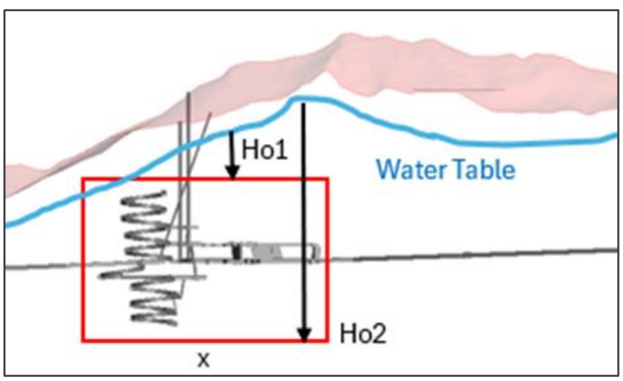
K = Hydraulic conductivity

H_0 = Hydraulic head above tunnel

R = Tunnel radius

A box was conceptualized around the mine workings for each year of development. The box dimensions were based on the footprint and approximate dimensions of the Camp and Los Cuyes deposits. An equivalent radius was calculated using dimension x as well as measurements from the estimated water table elevation to the approximate top and bottom limits of the mine workings (Ho1 and Ho2) as shown in Figure 16-6.

Figure 16-6: Inflow Conceptual Model



Recharge from precipitation was not considered in the inflow model calculations. At the nearby Fruta del Norte mine, precipitation records indicate relatively wet conditions, with average annual rainfall of 3,652 mm/year between 2008 and 2014 (SRK, 2016). Infiltration can conservatively be approximated at 15% of annual precipitation over the area of the mine. This would contribute approximately 747 m^3/day over an annual period.

The inflow was estimated using the upper and lower bounds of hydraulic conductivity as well as the median of hydraulic conductivity from the depth model.

16.2.3 Model Outcomes

The inflow estimates for the Camp Deposit (Table 16-11; Figure 16-7), Los Cuyes Deposit (Table 16-12; Figure 16-8), and for the total mine (Table 16-13, Figure 16-9) are shown below. Maximum inflow of 961 m^3/day is reached for the total mine by Year 10.

Table 16-11: Camp Inflow Estimates by Year

	Inflow (m^3/day)		
Year	**High K**	**Low K**	**Best Est K**
1	1,748	17	134
2	2,537	25	194
3	3,834	38	293
4	4,102	41	314
5	4,102	41	314
6	5,013	50	384
7	4,728	47	362
8	4,736	47	362
9	4,732	47	362
10	5,381	54	412
11	5,381	54	412
12	5,381	54	412
13	5,381	54	412
14	5,381	54	412

Figure 16-7: Camp Inflow Estimates by Year

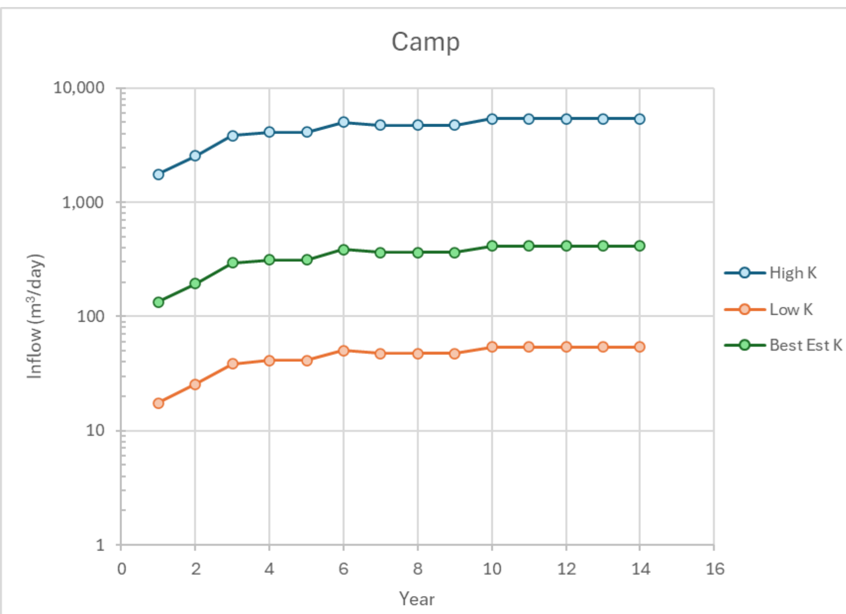


Table 16-12: Los Cuyes Inflow Estimates by Year

	Inflow (m^3/day)		
Year	High K	Low K	Best Est K
1	128	1	10
2	954	10	73
3	1,621	16	124
4	2,132	21	163
5	2,709	27	207
6	3,385	34	259
7	3,361	34	257
8	5,852	59	448
9	7,182	72	550
10	7,182	72	550
11	7,182	72	550
12	7,182	72	550
13	7,182	72	550
14	7,182	72	550

Figure 16-8: Los Cuyes Inflow Estimates by Year

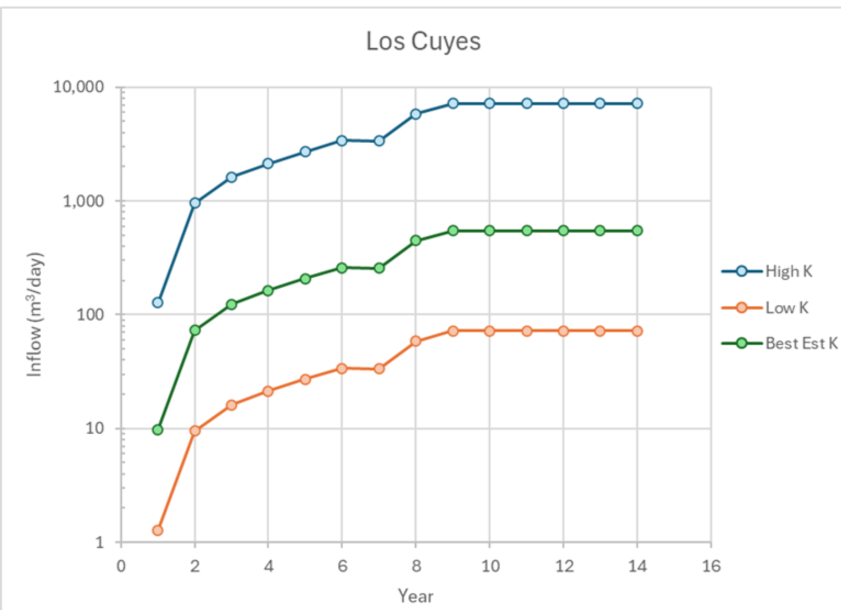
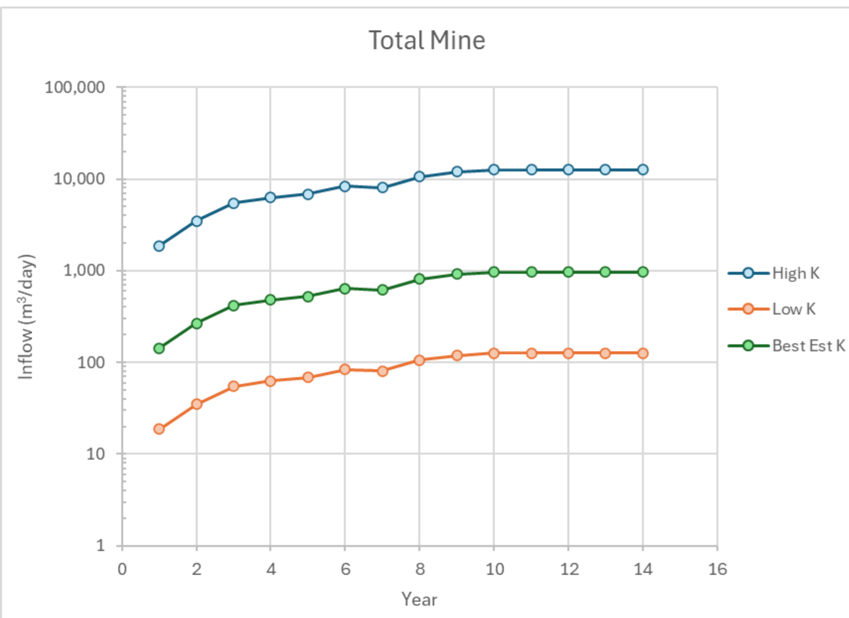


Table 16-13: Total Mine Inflow Estimates by Year

	Inflow (m^3/day)		
Year	High K	Low K	Best Est K
1	1,876	19	144
2	3,491	35	267
3	5,455	55	417
4	6,233	62	477
5	6,811	68	521
6	8,399	84	643
7	8,089	81	619
8	10,588	106	810
9	11,914	119	912
10	12,563	126	961
11	12,563	126	961
12	12,563	126	961
13	12,563	126	961
14	12,563	126	961

Figure 16-9: Total Mine Inflow Estimates by Year



16.3 Block Models and Net Smelter Return Estimation

16.3.1 Block Models Used in Mine Planning

There are two block models used in the Condor PEA mine planning, one for the Camp zone and another for Los Cuyes zone. The block model buildups have been discussed in detail in Section 13.8. The block models used in mine planning are:

- Los_Cuyes_0314_Rev02.dm
- Camp_0429_Rev02.dm

Both block models are subcelled Datamine block models with a parent cell size of 10 m × 10 m × 10 m.

16.3.2 NSR Calculation

In the PEA, SRK used a net smelter return (NSR, $/tonne) value as an indicator to determine if a mining shape/stope meet the economic cut-off criteria for inclusion into the mining plan. Table 16-14 shows the assumptions and parameters used in the initial NSR calculation which was incorporated into the resource block model for mine design.

Table 16-14: Parameters and Assumptions Used in NSR Calculations

Item	Metal Price		Mill Recovery*		Comment	Payable	Comment
	Unit	In USD	Camp	Los Cuyes			
Au	$/oz	2,450	2.5884* Ln(x) + 92.0696	2.5993* Ln(x) + 89.498	to Dore	99.8%	to Dore
			0.1%	0.1%	to Pb conc	95.0%	to Pb conc
			0.7%	0.7%	to Zn conc	96.5%	to Zn conc
Ag	$/oz	27.25	44.0%	50.0%	to Dore	90.0%	to Dore
			12.3%	11.4%	to Pb conc	95.0%	to Pb conc
			8.7%	1.6%	to Zn conc	70.0%	to Zn conc
Pb	$/lb	0.86	38.0%	34.0%	to Pb Conc	95.0%	to Pb Conc
Zn	$/lb	1.22	60.0%	35.0%	to Zn conc	85.0%	to Zn conc

* Gold recoveries to doré are capped at 98% and 96% for Camp and Los Cuyes, respectively

** TCRC and deducts are also applied based on benchmark international smelter terms and conditions

Source: SRK 2025

The derived initial NSR formulae are:

- Camp: NSR ($/t) = 71.7799*[Au] (g/t) + 0.4461*[Ag] (g/t) + 4.7647 *[Pb] (%) + 7.3026*[Zn] (%)
- Los Cuyes: NSR ($/t) = 70.4721*[Au] (g/t) + 0.4632*[Ag] (g/t) + 2.6748 *[Pb] (%) + 4.2595*[Zn] (%)

The NSR formulae above are used for stope assessment, mine design, and scheduling for this PEA. For reporting purposes, a separate set of commodity prices ($2,600/oz for gold, $31.00/oz for silver, $0.91/lb for lead, and $1.27/lb for zinc) was applied to generate updated NSR formulae.

The NSR formulae used for reporting are:

- Camp: NSR ($/t) = 76.1899*[Au] (g/t) + 0.5086*[Ag] (g/t) + 4.9809 *[Pb] (%) + 7.9303*[Zn] (%)
- Los Cuyes: NSR ($/t) = 74.8017*[Au] (g/t) + 0.5281*[Ag] (g/t) + 2.8562 *[Pb] (%) + 4.6269*[Zn] (%)

SRK assessed the impact of updated NSR to the shapes included in the LOM plan. With the updated NSR, some of the incremental stope shapes will updated to breakeven stopes, and the overall impact is not material; therefore, SRK kept the initial mine design and included shapes in the mine plan as they are.

16.4 Planned Mining Methods

16.4.1 Mining Context

The key characteristics of the Condor underground project relevant to mining method selection are summarized below:

- The deposit consists of two separate zones, namely Camp and Los Cuyes zones, located approximately 400-900 m apart in plan view.
- The Camp zone consists of six relatively continuous, steeply dipping, subparallel veins that generally dip toward the northeast.
- The Los Cuyes zone includes about twelve subparallel veins, also dipping northeast, which are truncated by a northeast-trending structure referred to as the LCW vein.
- Geologically, the Los Cuyes zone is more complex than the Camp zone.
- The Los Cuyes zone comprises predominantly steeply dipping, variable-width, narrow veins with locally higher gold grades, typically ranging from less than 1 m to 50 m in width.
- The Camp zone exhibits better geological continuity, with vein widths varying from narrow to moderate, also ranging from approximately 1 m to 50 m.
- The project is situated in a tropical Amazonian forest environment characterized by high annual rainfall.
- It is classified as a low- to medium-grade deposit with good continuity, based on an in-situ cut-off grade of 1.5 grams of gold per tonne (g/t Au).
- Approximately two-thirds of economic mining tonnage is located in Camp zone, with the remaining one-third in the Los Cuyes zone.
- At the proposed main portal elevation (1,100mEL), approximately one-third of economic tonnage lies above the adit level and two-thirds below, in both zones.
- Both zones are characterized by steep mountainous terrain with a thin to medium saprolite overburden.
- The weathered layer above the bedrock surface varies in thickness from less than 5 m to 30 m.
- No artisanal workings are present within the Camp or Los Cuyes zones; however, extensive artisanal mining has occurred in nearby deposits.

16.4.2 Mining Methods

The underground mine will be accessed through an adit and a system of ramps. Ore extraction will utilize a combination of Transverse Longhole Open Stoping (TOS), Sublevel Retreat Longhole Open Stoping (SLOS), and Uppers with the method applied according to the geometry of the orebody and geotechnical conditions. Production rates are expected to vary between deposits throughout the mine life, with a planned maximum total mining rate of 5,000 tonnes per day for the operation.

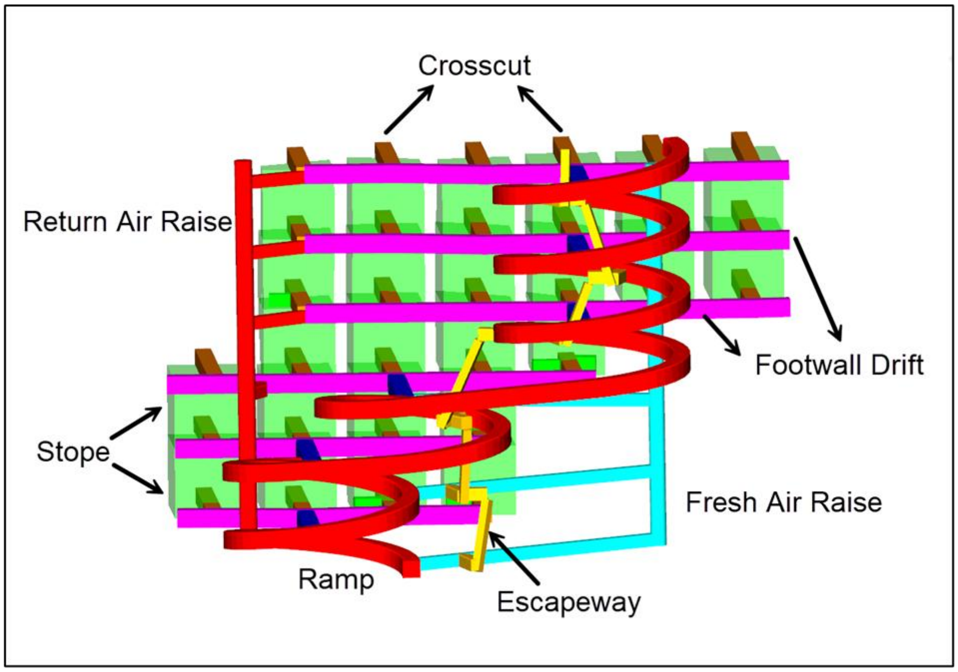
The underground mine will adopt a bottom-up mining sequence, employing a combination of TOS and SLOS, depending on orebody geometry and geotechnical conditions. A 5-m thick temporary rib pillar will be left between adjacent stopes on the same sublevel, eliminating the need for cemented backfill. The rib pillars are not designed to provide structural support but rather to contain the backfill material and minimize backfill dilution. All mined-out stopes, except for upper stopes, will be backfilled with waste.

Transverse Longhole Open Stoping (TOS)

TOS will be applied where mineralization zones are wider, with stopes oriented perpendicular to the strike of mineralization. Crosscuts will be driven perpendicular to strike, with drill drifts developed to the top of the stope and mucking drifts established at the bottom. Stopes will be mined in a bottom-up sequence, starting from the lowest level of a mining block. A center-out mining sequence will be used, whereby stopes are mined and backfilled with waste prior to extraction of adjacent stopes.

Stopes will be initiated by establishing a slot raise, typically developed via a drop raise and blasted in lifts to provide a void or free space. Production rings will then be blasted retreating from the slot towards the stope entrance. For the stopes above, only the drill drift is required, as the mucking drift will have already been established by the stope below. Figure 16-10 demonstrates the typical TOS layout adopted for the project.

Figure 16-10: Illustration of Transverse Longhole Open Stoping

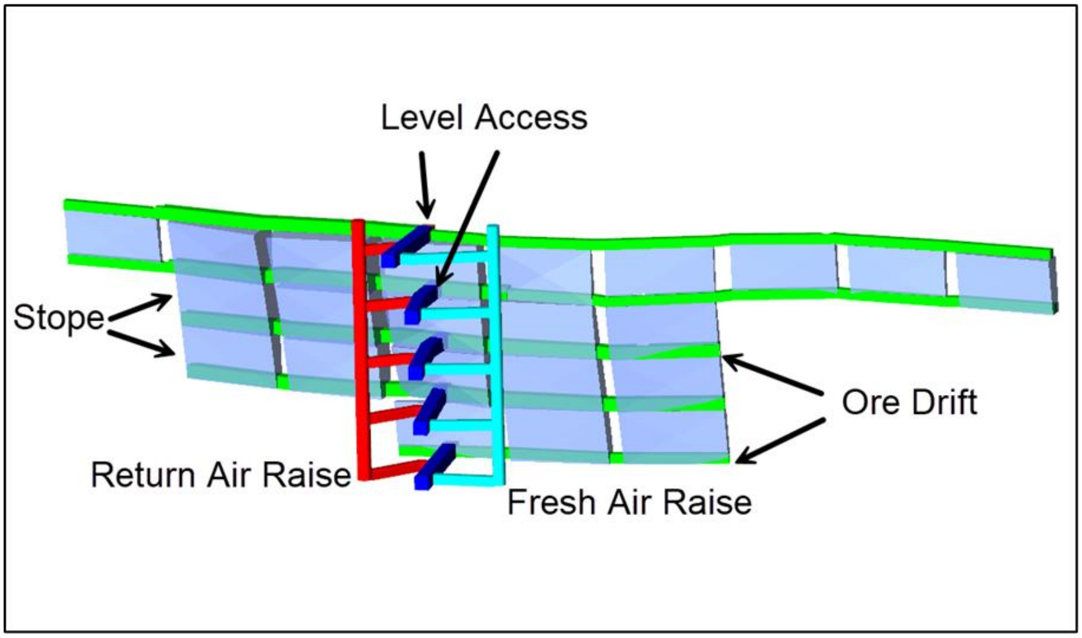


Source: SRK 2025

Sublevel Retreat Longitudinal Longhole Open Stoping (SLOS)

SLOS will be used where ore zones are narrower. In this method, development drifts are driven along the strike, with separate drill and mucking drifts established. Stopes will be mined sequentially along the drift, retreating from the far end of the orebody back toward the access crosscut (Figure 15 3). Each stope will be initiated with a slot raise in a similar manner to TOS, followed by retreat blasting. Upon completion, voids will be backfilled with waste. Figure 16-11 demonstrates the typical SLOS layout adopted for the project.

Figure 16-11: Illustration of Sublevel Retreat Longhole Open Stoping



Source: SRK 2025

16.5 Potential Run-of-Mine Material Estimate

16.5.1 Initial Cut-off Value (COV)

The cut-off values for the project were determined based on projected mining cost, mineral processing cost, G&A cost, and sustaining capital applicable at the PEA level of study, seeTable 16-15. The cut-off value represents the minimum NSR value required to cover operating cost, with allowances for sustaining capital where applicable. These values were established to distinguish potentially economic material from waste.

The break-even cut-off value of US$95/t corresponds to the minimum NSR value required to cover total operating and sustaining capital costs for longhole stoping production areas.

The incremental cut-off value of US$65/t represents the minimum NSR required to justify the cost to extract lower-grade stopes once access has been established.

The development cut-off value of US$40/t represents the minimum NSR required to recover the costs associated with in-ore development headings.

Table 16-15 summarizes the breakdown of costs used to derive the cut-off values for the break-even cut-off, incremental cut-off, and development cut-off.

Table 16-15: Initial Estimations of Cut-off Value

Item	Unit	Break-even COV	Incremental COV	Development COV
U/G Mining	$/t	44.22	38.32	8.00
Mineral Processing	$/t	26.36	26.36	26.36
G&A	$/t	13.50	0.68	0.68
Total Operating Cost	$/t	84.08	65.36	35.04
Total Sustaining Capital	$/t	10.00	-	5.00
Total AISC	$/t	94.08	65.36	40.04
Plant Feed Cut-off Value	$/t	95.00	65.00	40.00

Source: SRK 2025

16.5.2 Stope Design

Stope shapes were generated using Deswik Stope Optimizer (DSO), applying geotechnical, geometrical, and economic parameters defined for each mining zone. The design parameters incorporated in the optimization process varied according to the geotechnical conditions of the host rock and the selected mining method for each deposit. Both stope length and stope width were determined based on the geotechnical parameters and stability assessments presented in Section 16.1. The cut-off value applied in the optimization represents the in-situ cut-off value that incorporates external dilution factors. Table 16-16 summarizes the DSO input parameters.

Table 16-16: Deswik Stope Optimizer Input Parameters

Parameters	Unit	Camp		Los Cuyes	
		Longitudinal	Transverse	Longitudinal	Transverse
Optimization Field	$/t	NSR	NSR	NSR	NSR
Default Value	$/t	0	0	0	0
Default Density	t/m³	2.71	2.71	2.61	2.61
Stope Height	m	20	20	20	20
Stope Length	m	35	20	15	12
Min Width	m	1.5	1.5	1.5	1.5
Max Width	m	15	35	10	18
Stope Pillar	m	5	5	5	5
Rib Pillar	m	5	5	5	5
Breakeven Cut-off[1]	$/t	105	105	100	100
Incremental Cut-off[1]	$/t	60	60	55	55

[1]The Los Cuyes deposit has a higher dilution grade compared to Camp, resulting in lower in-situ break-even COV and in-situ incremental COV values

Source: SRK 2025

To maintain ground stability and eliminate the need for cemented rock fill (CRF), 5 m-wide rib pillars were designed to remain in situ between adjacent stopes across the mine. In accordance with geotechnical recommendations, all stopes located within the 40 m crown-pillar zone were excluded from the stope inventory to preserve overall stability of the uppermost mining horizon. Because both the Camp and Los Cuyes deposits are subdivided into five mining fronts, 10 m sill pillars were maintained at the uppermost level of each mining front except for the topmost mining front, where the crown pillar provides the required separation from surface. These sill and crown pillars and the backfill collectively ensure adequate vertical and lateral support for stability and allow for progressive extraction sequencing without compromising safety. The configuration and spacing of rib and sill pillars were selected to balance extraction ratio, ground stability, and operational practicality.

Following the completion of preliminary stope designs, mine development strings were digitized to provide access to the stoping areas, ensuring logical connection of ore drives, ventilation raises, and haulage drifts. The preliminary stope inventory was subsequently subjected to an economic evaluation. Stopes that did not meet the economic threshold or that breached geotechnical constraints were removed from the design, resulting in an optimized and economically viable stope shapes for inclusion in the mine plan.

16.5.3 Dilution Assessment and Mining Recovery Parameters

The dilution factor for each stope was estimated using the stope length and stope width parameters derived from the DSO output along with the corresponding ELOS value. External dilution was applied by incorporating an ELOS value based on expected wall stability and mining method geometry.

For longitudinal stopes, an ELOS of 0.9 m was applied along the hanging wall and 0.5 m along each side wall to represent localized sloughing and overbreak. For transverse stopes, an ELOS of 0.5 m was applied uniformly on all four sides.

A dilution grade study was completed to estimate the default dilution grade based on mining zone and mining method. For this study, three representative stope shapes were selected for each combination of deposit and mining method. The dilution skins surrounding these stopes were interrogated to estimate the grade of the diluted material, which was subsequently adjusted to derive the default dilution grade applicable to each stope type. The resulting dilution grades were then incorporated into Deswik Sched to determine the diluted grade of each stope for production scheduling. Table 16-17 summarizes the applied dilution factors, dilution grades, and mining recovery parameters for each combination of deposit and mining method.

Table 16-17:Summary of Dilution and Mining Recovery Parameters

Parameters	Unit	Camp		Los Cuyes	
		Longitudinal	Transverse	Longitudinal	Transverse
Dilution	%	7~63	8~16	16~68	14~41
Dilution Grade - NSR	$/t	37	93	53	109
Dilution Grade - Au	g/t	0.52	1.3	0.75	1.54
Mining Recovery	%	92	95	92	95

1 Mining recovery for upper stopes is 85%

2 Only the gold grade is considered in the dilution grade estimation, as gold represents the primary contributor to the economic value of the ore

The dilution factor for sill development was set at 10%, while a factor of 12% was applied to waste lateral development. For vertical development, dilution was assumed to be 0% where excavation is completed by raise boring, and 12% where development is conducted using Alimak or drop-raise methods. The dilution grade for all development headings was assumed to be 0. A mining recovery of 100% was applied to all development activities.

16.5.4 Run-of-Mine Material for Mine Plan

The run-of-mine (ROM) material represents the total tonnage of mineralized material planned for extraction and delivery to the processing plant over the life of mine. ROM material comprises both stope tonnes and sill development tonnes, which together account for all mineralized material meeting the diluted economic cut-off criteria described in 16.5.1.

A summary of the ROM tonnages and corresponding NSR values is presented in Table 16-18. The table outlines contributions from stoping and sill development, as well as the ROM that forms the basis for subsequent mine scheduling. Total LOM ROM material is estimated at 21.34 Mt with an average NSR value of $179/t. ROM material estimate by resource class is shown in Table 16-19 .

Table 16-18: ROM Material by Source

Item	Unit	Total	Camp	Los Cuyes
Stope Tonnes	Mt	17.91	12.13	5.78
NSR - Stope	$/t	183	175	201
Ore Development Tonnes	Mt	3.43	1.69	1.74
NSR - Ore Development	$/t	154	147	161
ROM Tonnes	**Mt**	**21.34**	**13.82**	**7.51**
NSR - ROM	**$/t**	**179**	**171**	**192**

Table 16-19: ROM Material by Resource Class

Mine	Category	Run-of-Mine Plant Feed Tonnes (Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	NSR ($/t)
Camp	Measured	-	-	-	-	-	-
	Indicated	3.07	2.14	15.09	0.06	0.60	181
	Measured + Indicated	**3.07**	**2.14**	**15.09**	**0.06**	**0.60**	**181**
	Inferred	**10.75**	**1.99**	**14.78**	**0.05**	**0.65**	**169**
Los Cuyes	Measured	-	-	-	-	-	-
	Indicated	1.79	2.09	12.16	0.05	0.38	169
	Measured + Indicated	**1.79**	**2.09**	**12.16**	**0.05**	**0.38**	**169**
	Inferred	**5.73**	**2.48**	**13.27**	**0.07**	**0.35**	**199**
Total	Measured	-	-	-	-	-	-
	Indicated	4.86	2.12	14.01	0.06	0.52	176
	Measured + Indicated	**4.86**	**2.12**	**14.01**	**0.06**	**0.52**	**176**
	Inferred	**16.48**	**2.16**	**14.26**	**0.06**	**0.54**	**179**
Total ROM	**Measured + Indicated + Inferred**	**21.33**	**2.15**	**14.20**	**0.06**	**0.54**	**179**

* Totals may not sum due to rounding.

** **The estimated run-of-mine is partly based on Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment based on these mineral resources will be realized.**

*** The reader is cautioned that the mineralized material should not be misconstrued as a mineral resource or a mineral reserve. The quantities and grade estimates are derived from the block model and include mining dilution and losses.

16.6 Underground Mine Model

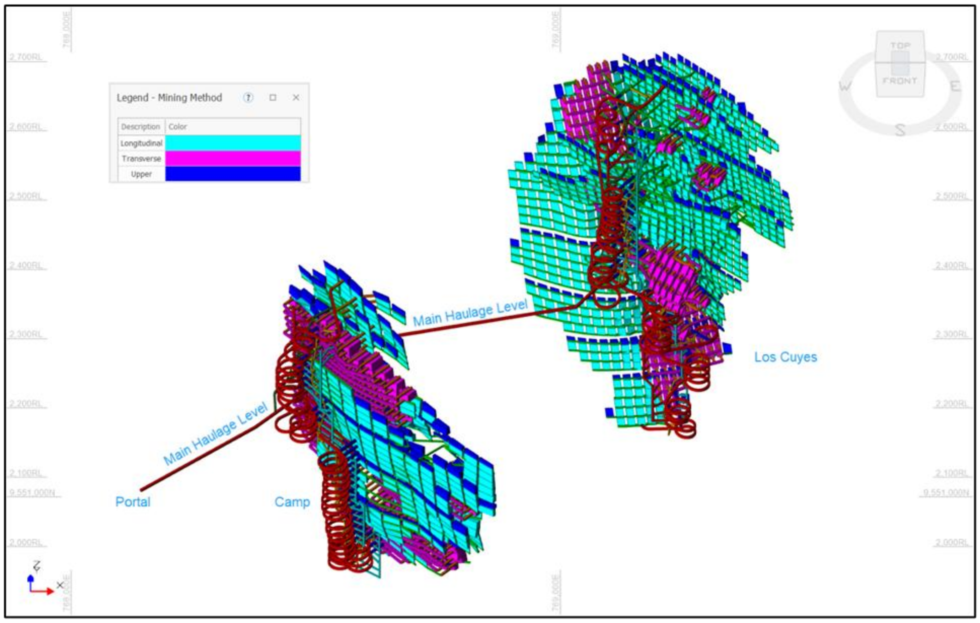
16.6.1 Underground Mine Layout

The underground mine will be accessed via a single adit, which also serves as the main haulage level connecting the Camp and Los Cuyes deposits. Above this level, mill feed will be loaded by load-haul-dump (LHD) units and discharged into rock passes, from where it will be hauled to surface mill plant by trucks. Below the adit level, the ROM will be hauled to surface mill plant through the ramp and haulage drift. Waste material generated from development will be first used for stope backfill, while the excessive portion will be transported to surface waste pile via the ramp and haulage drift, and be backhauled as backfill when needed.

Each deposit is subdivided into five mining fronts, designed to enable independent production sequencing and enhance ventilation efficiency and material handling capacity. Figure 16-12 illustrates the overall underground mine layout, showing the distribution of mining fronts as defined in the accompanying legend. This configuration allows for simultaneous development and stoping across multiple fronts, thereby optimizing production rates and minimizing overall cycle times. Figure 16-13 presents the underground mine layout color-coded by mining method.

Figure 16-12: Condor Underground Mine Layout Showing Mining Front (Looking Northwest)

Figure 16-13: Condor Underground Mine Layout Showing Mining Method (Top-Front Orientation)



Source: SRK, 2025

16.6.2 Lateral Development

The lateral development has been classified as either capital or operating expenditure. Sill drives and crosscuts are considered operating development, while all other lateral headings are classified as capital development.

To minimize internal dilution in SLOS stopes, a smaller sill drive dimension is assigned to stopes with widths less than 4.5 m, while a larger dimension is used for wider stopes.

Table 16-20 summarizes the dimensions and cost classifications for the lateral development headings

Table 16-20: Condor Lateral Development Dimensions and Cost Classification

Description	Cost Classification	Dimension (mW × mH)
Adit	Capital	A_5.1 × 5.4
Ramp	Capital	A_5.0 × 5.0
Level Access	Capital	A_5.0 × 5.0
Footwall Drive	Capital	A_5.0 × 5.0
Return Air Drive	Capital	A_3.5 × 3.5
Fresh Air Drive	Capital	A_3.5 × 3.5
Escapeway Drive	Capital	A_3.5 × 3.5
Ore Pass Drive	Capital	A_4.5 × 4.5
Ore Drive (for stope width >=4.5m)	Operating	F_4.5 × 4.5
Ore Drive (for stope width <4.5m)	Operating	F_3.0 × 3.5
Cross Cut	Operating	F_4.5 × 4.5

* "A" under Dimension indicates an arched back profile, while "F" indicates a flat back profile

All lateral capital development includes a 15% growth factor in Deswik Sched to account for additional minor infrastructure development that may be required over the LOM period.

16.6.3 Vertical Development

The vertical development includes fresh air raises, return air raises, rock passes, and escapeways, each serving distinct operational functions within the underground mine.

The fresh air and return air raises will be developed using raise bore and will connect directly to the underground ventilation network, providing intake and exhaust air pathways. Rock passes will be constructed using the Alimak to facilitate the efficient transfer of blasted ROM from the production levels above the main haulage level.

The escapeway raises will be developed using the drop raise method. These openings will function as secondary egress routes for personnel safety and will also contribute to supplemental ventilation during decline and level development.

Table 16-21 summarizes the dimensions and cost classifications for the vertical development.

Table 16-21: Condor Vertical Development Dimensions and Cost Classification

Description	Cost Classification	Dimension
Return Air Raise	Capital	D_5.0m
Fresh Air Raise	Capital	D_4.0m
Escapeway Raise	Capital	S_2.4m × 2.4m
Ore Pass	Capital	S_2.4m × 2.4m

Notes: “D” under Dimension indicates a circular profile, while “S” indicates a square profile
Source: SRK, 2025

16.7 Underground Mine Production Schedule

The collaring of the main adit is scheduled to commence in September of Year-2, marking the start of underground development activities. Total ROM material mined from Year-1 to Year 1 is estimated at approximately 1.37 Mt, aligning with the planned mill construction and commissioning schedule.

Year 1 represents the first year of commercial production, with mine output continuing to ramp up until steady-state ROM throughput of approximately 1.8 Mtpa is achieved in Year 2. Consistent ROM production is maintained from Year 2 through Year 12, with Year 13 representing the final year of production. The mine schedule assumes 360 operating days per year to account for regular maintenance and operational downtime.

The LOM production and development summary are summarized in Table 16-22 to Table 16-26 and Figure 16-14 to Figure 16-18.

Table 16-22: Condor LOM – Mined Material Summary (Millon Tonnes)

Item	LOM Total	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Development ROM	3.43	-	0.15	0.36	0.49	0.32	0.34	0.41	0.29	0.27	0.32	0.15	0.19	0.09	0.04	-
Stope ROM	17.91	-	0.30	0.57	1.31	1.48	1.46	1.39	1.51	1.53	1.48	1.65	1.61	1.71	1.62	0.30
Total ROM	21.34	-	0.45	0.92	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.67	0.30
Development Waste	3.70	0.02	0.55	0.49	0.27	0.37	0.37	0.25	0.43	0.44	0.18	0.11	0.09	0.07	0.06	-
Total Mined Material	25.03	0.02	1.00	1.41	2.07	2.17	2.17	2.06	2.23	2.24	1.98	1.91	1.89	1.87	1.72	0.30

* Some values may not sum exactly due to rounding

Source: SRK, 2025

Table 16-23: Condor LOM – Total ROM Summary

Item	Unit	LOM Total	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
ROM Tonnage	Mt	21.34	-	0.45	0.92	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.67	0.30
NSR	$/t	179	-	205	175	204	182	193	181	192	159	169	184	166	170	162	178
Gold Grade	g/t	2.15	-	2.49	2.19	2.51	2.22	2.33	2.22	2.32	1.89	2.01	2.18	1.99	2.00	1.87	2.04
Silver Grade	g/t	14.20	-	13.01	9.17	11.30	13.41	15.48	12.53	15.63	14.63	15.15	15.38	12.60	14.44	17.20	23.71
Lead Grade	%	0.06	-	0.04	0.04	0.05	0.06	0.07	0.05	0.07	0.06	0.06	0.06	0.05	0.05	0.05	0.06
Zinc Grade	%	0.54	-	0.47	0.25	0.41	0.49	0.52	0.44	0.52	0.54	0.57	0.59	0.58	0.69	0.74	0.63

* Some values may not sum exactly due to rounding

Source: SRK, 2025

Table 16-24: Condor LOM – Stope ROM Summary

Item	Unit	LOM Total	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
ROM Tonnage	Mt	17.91	-	0.30	0.57	1.31	1.48	1.46	1.39	1.51	1.53	1.48	1.65	1.61	1.71	1.62	0.30
NSR	$/t	183	-	235	194	218	188	200	187	197	160	172	187	170	171	163	177
Gold Grade	g/t	2.21	-	2.86	2.44	2.70	2.30	2.42	2.30	2.38	1.90	2.05	2.23	2.03	2.02	1.89	2.04
Silver Grade	g/t	14.46	-	14.15	8.83	11.12	12.90	16.21	12.68	15.54	14.69	15.26	15.54	12.79	14.56	17.39	23.71
Lead Grade	%	0.06	-	0.05	0.04	0.05	0.06	0.07	0.05	0.07	0.06	0.06	0.06	0.05	0.05	0.05	0.06
Zinc Grade	%	0.55	-	0.48	0.24	0.39	0.49	0.54	0.44	0.52	0.55	0.56	0.59	0.58	0.69	0.74	0.63

* Some values may not sum exactly due to rounding

Source: SRK, 2025

Table 16-25: Condor LOM – Development ROM Summary

Item	Unit	LOM Total	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
ROM Tonnage	Mt	3.43	-	0.15	0.36	0.49	0.32	0.34	0.41	0.29	0.27	0.32	0.15	0.19	0.09	0.04	-
NSR	$/t	154	-	146	144	164	153	158	159	166	154	154	147	142	149	111	-
Gold Grade	g/t	1.86	-	1.77	1.78	2.01	1.82	1.93	1.94	1.99	1.83	1.80	1.74	1.69	1.75	1.29	-
Silver Grade	g/t	12.84	-	10.73	9.71	11.79	15.75	12.32	12.00	16.11	14.29	14.67	13.60	10.98	12.21	10.03	-
Lead Grade	%	0.06	-	0.04	0.04	0.06	0.07	0.06	0.05	0.07	0.05	0.06	0.06	0.04	0.03	0.03	-
Zinc Grade	%	0.47	-	0.44	0.26	0.45	0.49	0.41	0.46	0.51	0.50	0.63	0.54	0.53	0.66	0.61	-

* Some values may not sum exactly due to rounding

Source: SRK, 2025

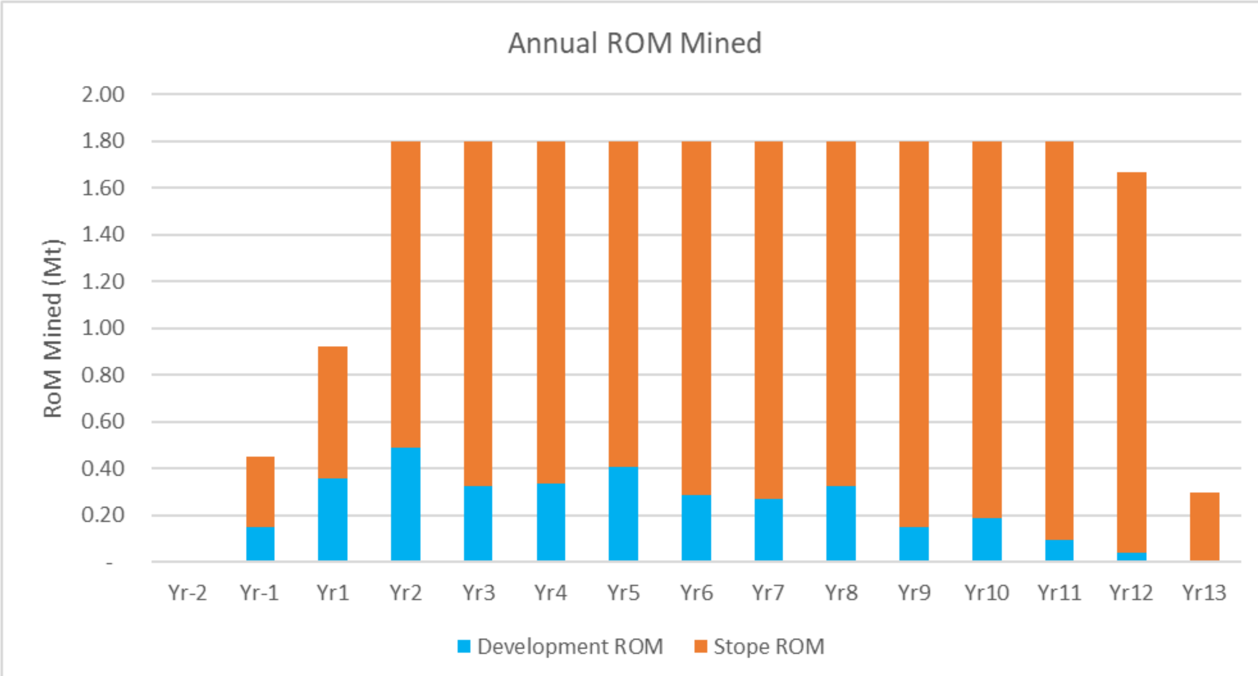
Table 16-26: Condor LOM – Development Metres Summary (km)

Item	LOM Total	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Lateral (OPEX)	94.1	-	2.9	8.4	12.4	9.3	9.8	11.2	9.1	8.3	9.5	5.0	5.0	2.3	1.0	-
Lateral (CAPEX)	42.6	0.3	7.4	7.2	2.9	4.3	4.1	2.9	5.0	4.9	0.9	0.8	0.7	0.6	0.5	-
Vertical (CAPEX)	4.8	-	1.4	0.7	0.4	0.6	0.3	0.4	0.4	0.3	0.2	-	-	-	-	-
Total OPEX	94.1	-	2.9	8.4	12.4	9.3	9.8	11.2	9.1	8.3	9.5	5.0	5.0	2.3	1.0	-
Total CAPEX	47.4	0.3	8.8	8.0	3.4	4.9	4.5	3.3	5.3	5.2	1.1	0.8	0.7	0.6	0.5	-
Total Lateral	136.7	0.3	10.3	15.6	15.3	13.6	13.9	14.1	14.0	13.1	10.4	5.8	5.7	3.0	1.6	-
Total Vertical	4.8	-	1.4	0.7	0.4	0.6	0.3	0.4	0.4	0.3	0.2	-	-	-	-	-
Total Development	141.5	0.3	11.7	16.4	15.8	14.2	14.2	14.5	14.4	13.4	10.6	5.8	5.7	3.0	1.6	-

* Some values may not sum exactly due to rounding

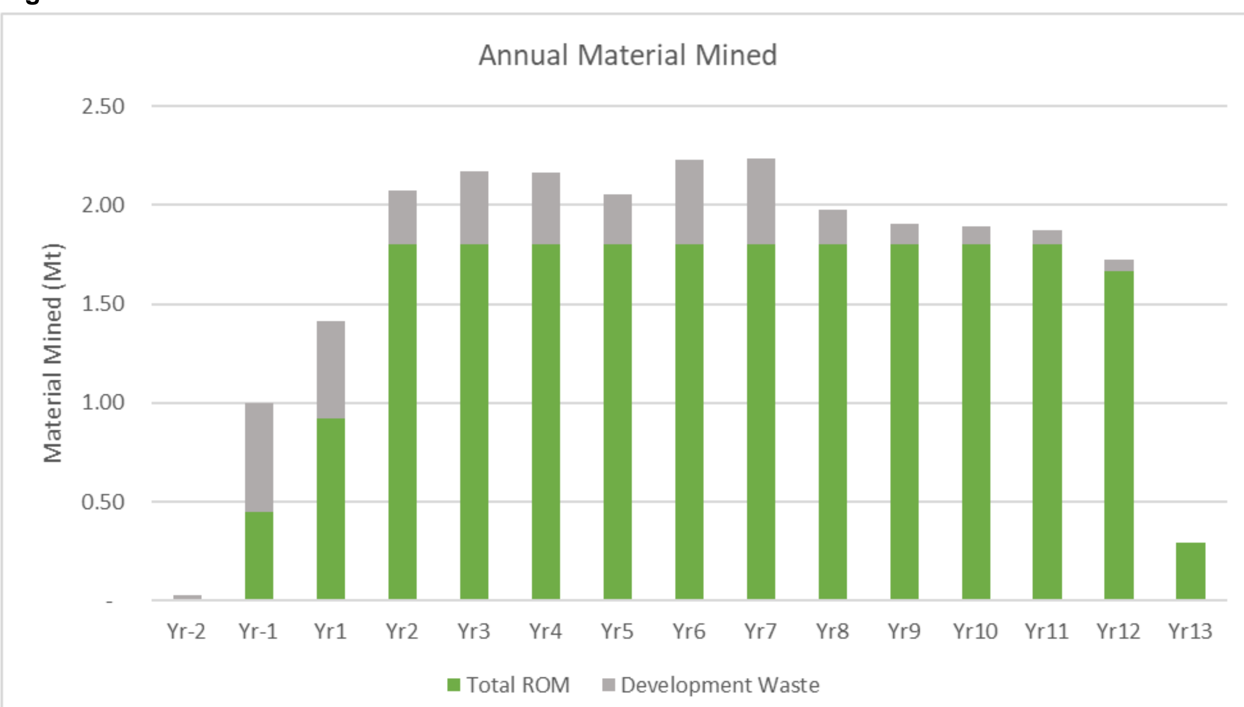
Source: SRK, 2025

Figure 16-14: Condor Annual ROM Profile by Source



Source: SRK, 2025

Figure 16-15: Condor Annual Material Mined



Source: SRK, 2025

Figure 16-16: Condor ROM Production Profile

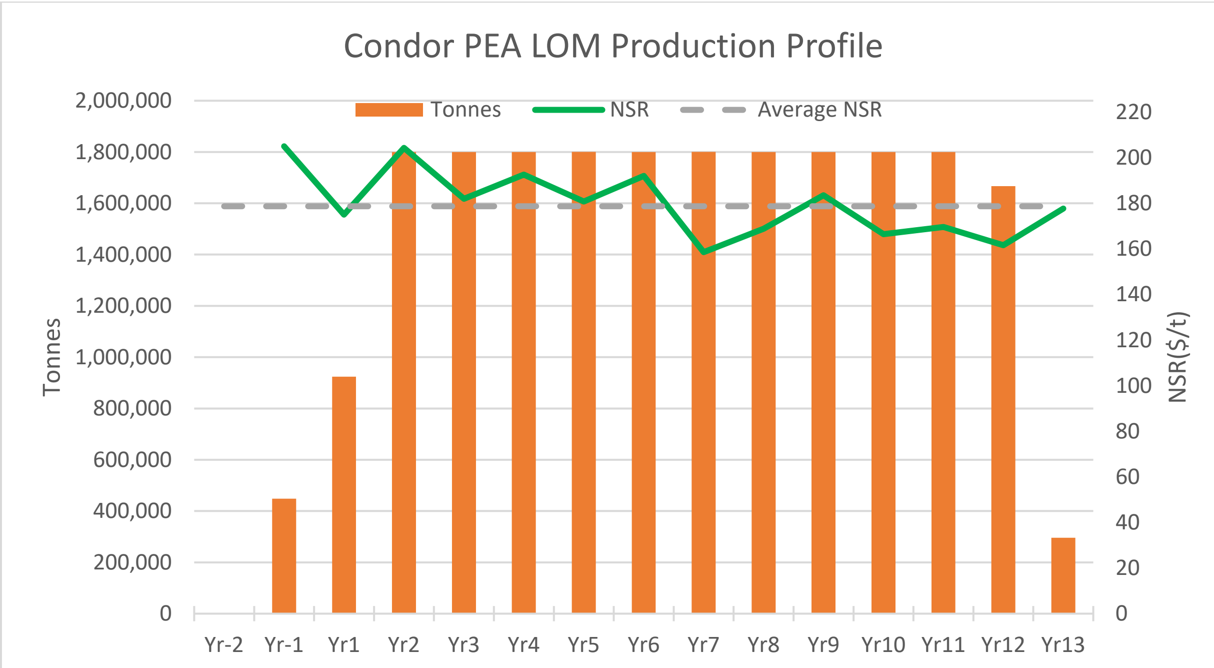


Figure 16-17: Condor Annual Mine Development by Cost Classification

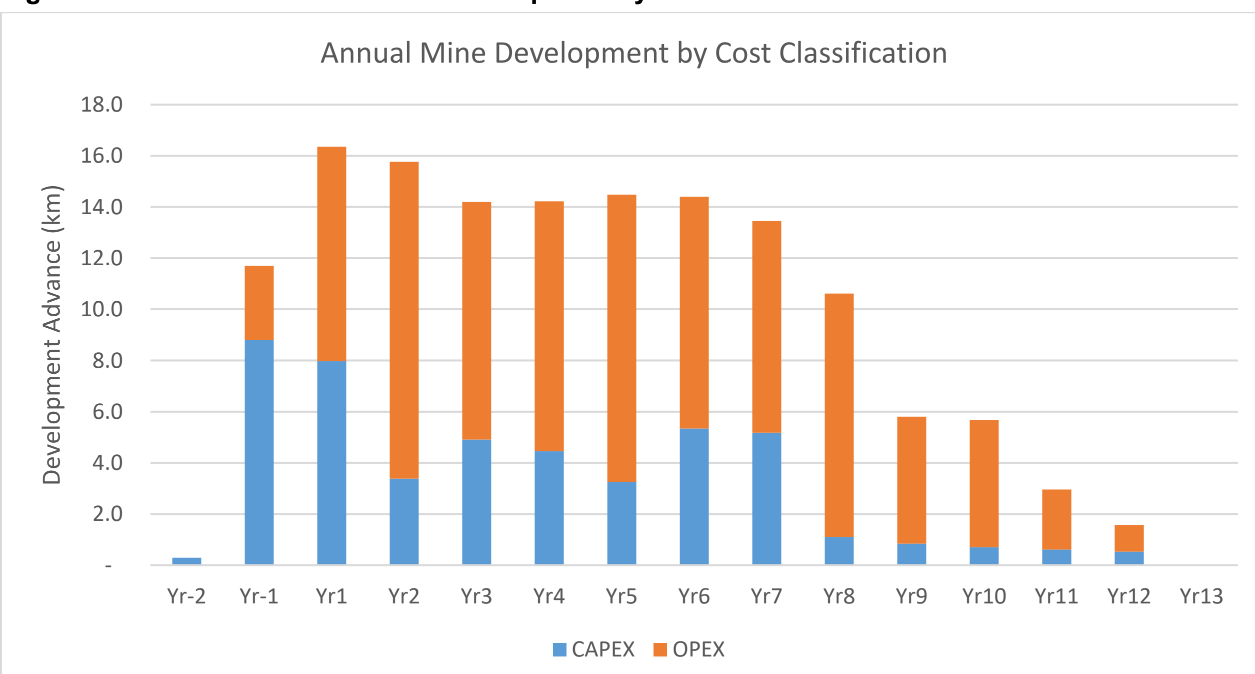
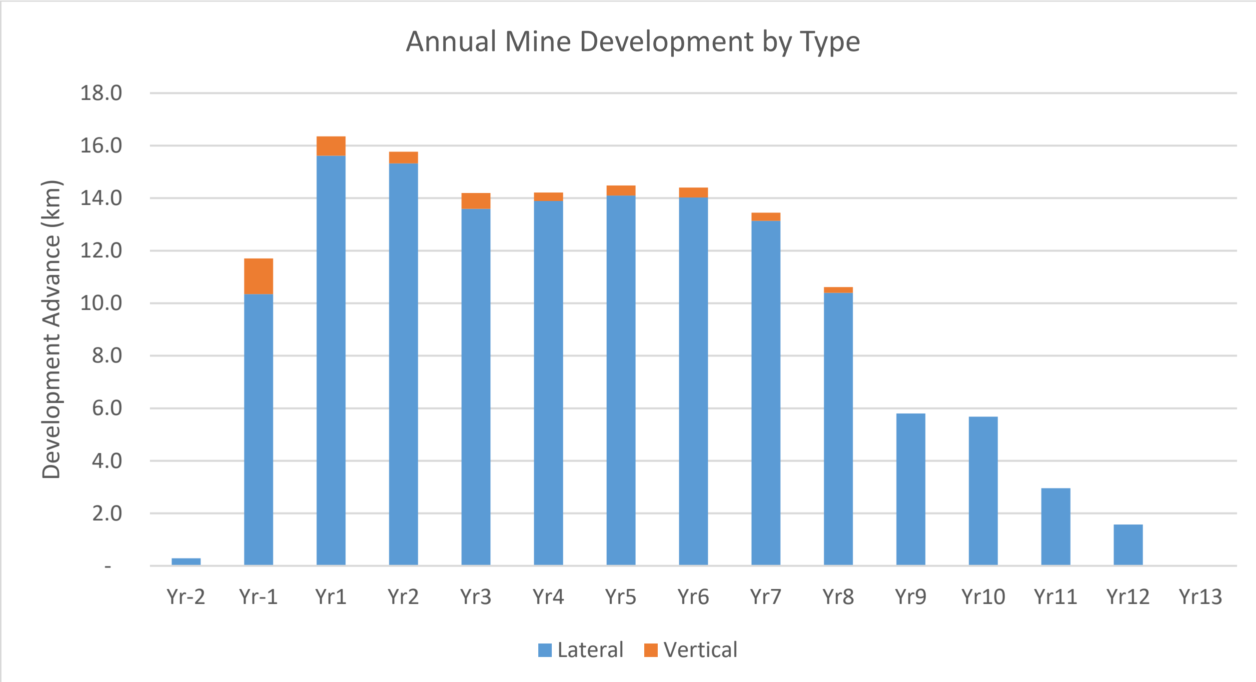


Figure 16-18: Condor Annual Mine Development by Mine



16.8 Mobile Equipment

The mobile equipment fleet has been estimated based on the planned mine production schedule, development rates, and operating requirements throughout the LOM. The fleet is divided into primary underground production units, secondary support equipment, and surface equipment to ensure continuous ROM and waste handling, development, and logistics. The fleet will be shared between the Camp and Los Cuyes zones, with allocation adjusted according to production and development requirements in each area.

Table 16-27 summarizes the required equipment fleet throughout the LOM.

Table 16-27: Mobile Equipment Fleet

Equipment Type	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Primary Underground Equipment															
10t LHD	1	5	7	7	6	7	7	7	7	6	5	5	5	2	1
17t LHD	0	1	2	3	2	3	3	2	3	3	3	3	3	3	1
30t Haul Truck	1	2	2	2	2	2	3	2	2	3	3	3	4	4	0
50t Haul Truck	0	2	2	3	3	3	3	3	3	4	4	4	4	4	1
Production Drill	0	1	2	3	3	3	3	3	3	3	3	3	3	3	1
Jumbo	1	5	6	6	5	6	6	6	6	5	3	3	3	1	0
Mechanized Bolter	1	5	6	6	5	6	6	6	6	5	3	3	3	1	0
Secondary Underground Equipment															
Service/Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cable Reeler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor Lift	1	3	3	3	3	3	3	3	3	3	2	2	2	1	0
Personnel Carrier	2	2	4	4	4	4	4	4	4	4	4	4	4	2	2
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Transmixer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Land Cruiser	1	2	4	4	4	4	4	4	4	4	4	4	4	4	4
Surface Equipment															
Dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
20t Surface Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Source: SRK, 2025

16.9 Labour Requirement

The labor requirements were estimated based on the planned production and development activities throughout the LOM. The underground operation will be staffed on a two-shift schedule, with two 12-hour shifts per day, resulting in four rotation crews to provide continuous coverage. Workforce estimates include both operations and support personnel required to sustain production, development, and maintenance activities. The projected headcount over the LOM is summarized in Table 16-28.

Table 16-28: LOM Labor Requirement

	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Mine Development	44	88	88	88	88	88	88	88	66	66	66	66	44	44	22
Mine Production	16	31	62	62	62	62	66	62	62	70	70	70	74	74	31
Mine Services	19	38	38	38	38	38	38	38	38	38	38	38	38	38	19
Mine Maintenance	35	65	65	65	65	65	65	65	65	65	65	65	65	65	35
Technical Staff	23	26	26	26	26	26	26	26	26	26	26	26	26	26	15
Mine Management Staff	10	19	19	19	19	19	19	19	19	19	19	19	19	19	10
Site General Staff	12	22	22	22	22	22	22	22	22	22	22	22	22	22	12
Total Headcount	**159**	**289**	**320**	**320**	**320**	**320**	**324**	**320**	**298**	**306**	**306**	**306**	**288**	**288**	**144**

Source: SRK, 2025

16.10 Material Handling

The mineralized material produced from production and development processes will be trammed to level re-mucks for short-term storage or side-loaded directly into haulage trucks at the ramp intersection. This material will then be hauled either to the surface ROM pad for temporary stockpiling or directly to the mill.

All of the waste generated during the development process will be consumed as part of the backfilling process. During the initial stage of mine development, the waste shall be brought to surface and stockpiled until required for backfill purposes. When possible, development waste will remain underground to be used as backfill to minimize operating costs related to re-handling of material.

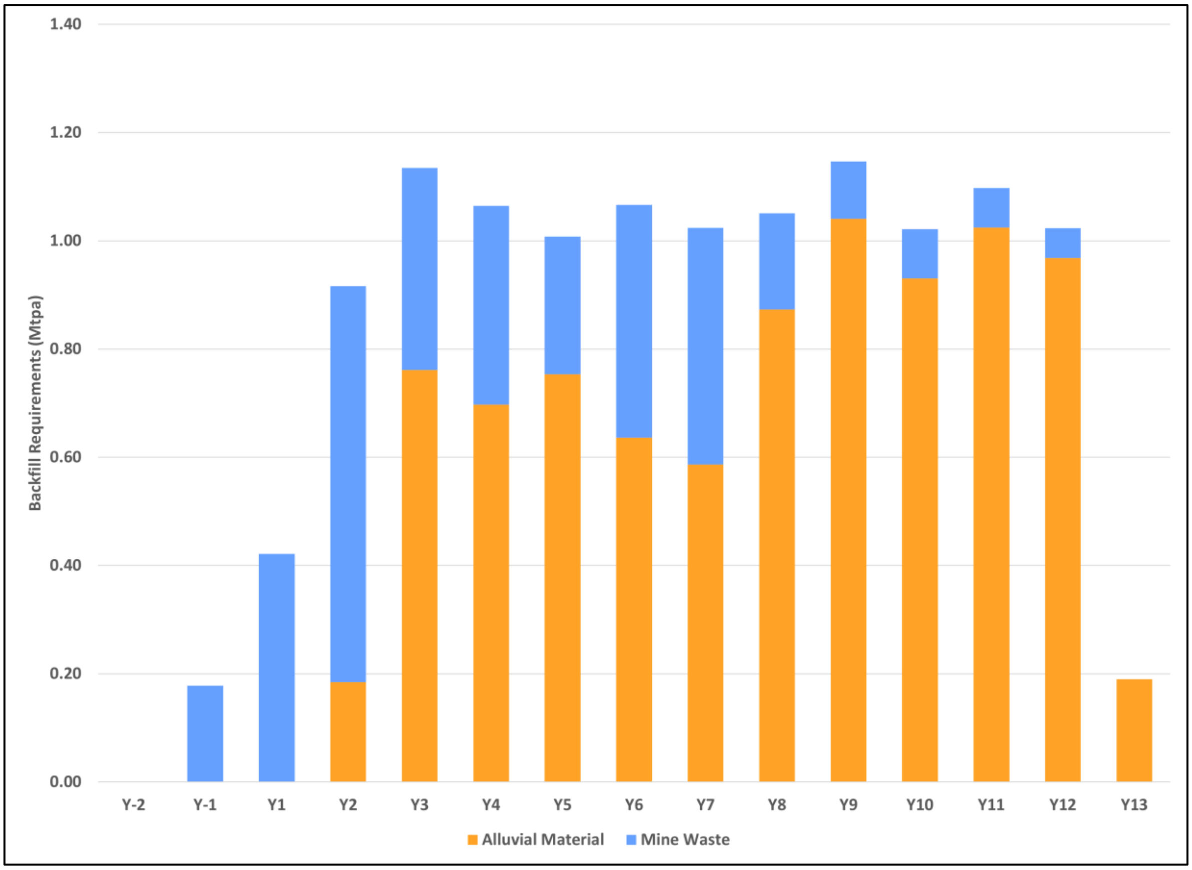
All supplies and materials necessary for underground mining will be stored in designated locations on the levels. As the mining fronts progress, these storages should be evaluated to ensure they are still required for operations as originally intended. If they are not, they may be re-purposed to minimize further costs related to infrastructure.

16.11 Backfill

Once a stope has been completely mucked and declared empty by the technical services personnel, it will be backfilled in order for mining activities to continue in sequence and allow for a greater extraction of resources. Following a trade-off study of backfill methods, the decision was made to proceed with uncemented rock fill (URF or RF) as the methodology for the project.

The media utilized in the URF will be a combination of waste rock generated through the development process as well as alluvial material sourced from off the property, as there is an insufficient amount of waste rock generated to fulfill the life of mine requirements. Figure 16-19 depicts the annual backfill requirements by material source over the mine life.

Figure 16-19: LOM Backfill Requirements



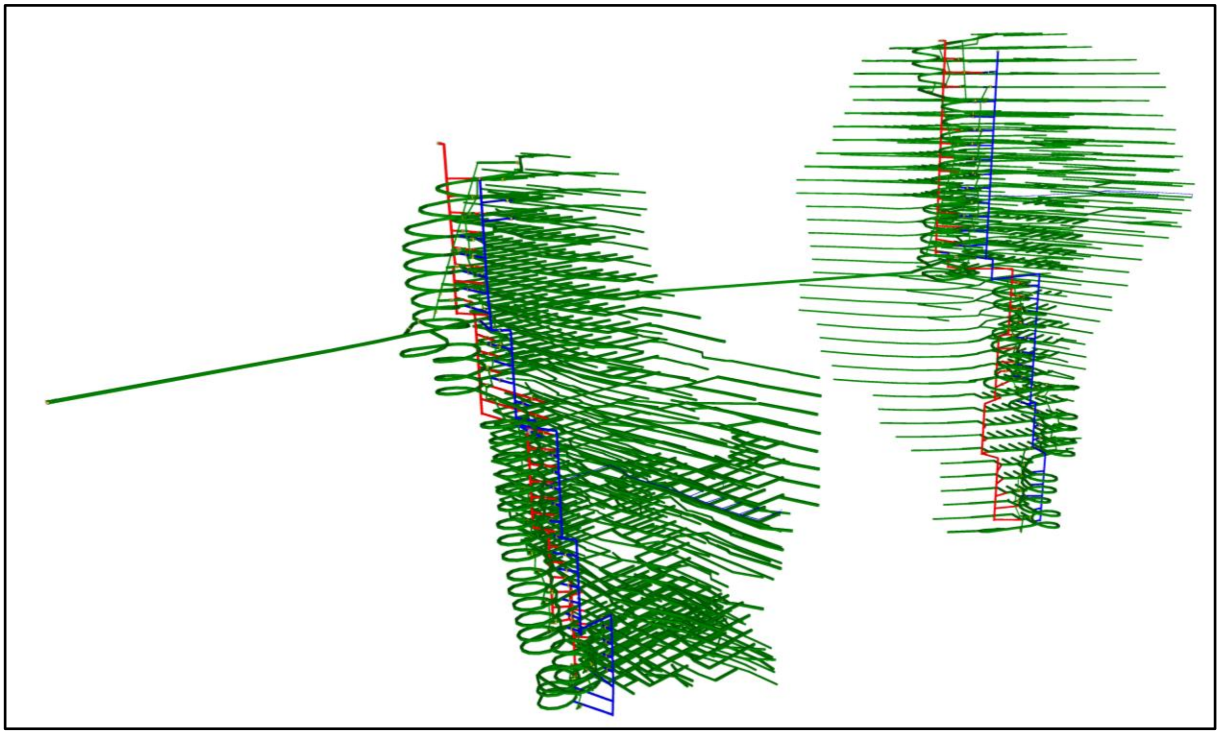
16.12 Mine Ventilation

The ventilation system has been developed based upon best practices and traditional ventilation techniques.

16.12.1 Ventilation System Layout

The mine design incorporates two exhaust raises to surface, two fresh air raises to surface, and a single adit providing both haulage and service access. The fresh air and exhaust air raises are developed by raisebore in segments but could be developed by Alimak if the dimensions were slightly increased. An isometric view of the mine is provided in Figure 16-20.

Figure 16-20: Layout Projections (Isometric View)



16.12.2 Required Airflow for Mining Criteria Establishment

Several factors must be considered when determining the airflow requirements for the mine such as diesel gas dilution, diesel particulates, heat, maintaining minimum air velocities, and meeting government regulations. These factors need to be applied to target areas to determine the actual total mine airflow requirement. Any fixed facilities underground (e.g., fuel and lubricant storage) will also demand dedicated airflow splits directed to the exhaust system.

Gases

Harmful strata gases are not expected to be encountered at this site. The configuration of the system as an exhausting ventilation system minimizes the blast clearance time/possibility of exposure to blast-generated gases by maintaining the ramp clear of blasting fumes. Each level will have access to an exhaust connection point and a fresh air connection point which will provide a limited compartmentalization of the ventilation system. No specific airflow requirements were established based on this criterion, though these hazards factor into direction of airflow and general ventilation configuration.

Diesel Particulates

General best practices require a minimum factor of 0.06 m3/s per kW of engine power (for modern diesel equipment supplied with 50ppm diesel fuel) to ensure gaseous and aerosol contaminants from diesel equipment are sufficiently diluted which is a typical minimum design value for many ventilation systems. This is the recommended minimum airflow to ensure sufficient dilution of contaminants with new equipment. If used equipment is purchased, or diesel equipment is poorly-maintained, this value may be insufficient.

Ventilation Raises

The fresh air and exhaust air raises have been identified as being developed by the raisebore method as shown in Table 16-29. If the method is modified and either short drop raises or Alimak raises are developed then the friction factor of the raise will be increased, this can be mitigated or offset by increasing the development dimension of the raises.

Table 16-29: Friction Factors

Raise Type	Method	Dimension (m)	Friction Factor (Ns^2/m^4)
Ventilation Raise (Fresh Air)	Raise Bore	4.5	0.005
Ventilation Raise (Exhaust Air)	Raise Bore	5.0	0.005
Duct		1.0 to 1.4	0.004

Source: SRK 2025

Horizontal Airways

Horizontal airways in the ventilation system were designed based off the Deswik output mine designs which were imported into VentSimTM ventilation modeling software. Modeled airway dimensions and friction factors are shown in Table 16-30.

Table 16-30: General airway dimensions

Airway Type	Dimension (m)	Friction Factor (Ns^2/m^4)
Primary Main Access	5.1 x 5.4	0.012 (developed with long straight stretches)
Level Access	5 x 5	0.012 (developed with long straight stretches)
Spiral Ramp	5 × 5	0.012 (tight spiral)
Footwall	5 × 5	0.012 (assumed clutter)
Stope Access	4.5 × 4.5	0.012 (assumed free of clutter)

Source: SRK, 2025

Air Velocities

The air velocity through the main access adit will be in the range of 8 m/s to 9 m/s if left unmitigated. The installation of a booster fan in the Camp fresh air raise will control the air velocity in the main adit by balancing the fresh air system. This will reduce the air velocity below 6 m/s. General air velocity limitations are shown in Table 16-31. Air velocity limits and recommended values for travelways are established to accommodate work and travel by people and equipment.

Table 16-31: Recommended Maximum Air Velocities for Various Airway Types (Design)

Airway Type	Maximum Air Velocity (m/s)
Travelways	6
Primary dedicated ventilation intake and exhaust accesses	8-10
Primary ventilation shaft	20
Ventilation shaft with conveyance or escape (may be temporarily reduced during an emergency)	10
Minimum air velocity	0.3

Source: SRK 2025

In general, the minimum air velocity in a heading (without diesel equipment in operation) is based on the perceptible movement of airflow which, based on best practice, is between 0.3 m/s and 0.5 m/s. The higher value of 0.5 m/s is used in areas with possible diesel equipment operation to both ensure compliance and air mixing, the value of 0.3 m/s is used as a minimum air velocity for areas with only electrical equipment.

Air velocities in long upcasting shafts should be maintained outside of the range of 7 m/s to 12 m/s to avoid water blanketing. Variability of the number of equipment and mining locations throughout the mine life makes this hard to plan for in advance by manipulating the size of raises. A solution to the problem may be to slightly increase or decrease flow in problematic shafts. This may require some shifting of mining activities.

Heat

Detailed rock and water temperature data was not available for the proposed mining zone. However, when this data becomes available it should be incorporated into the ventilation design to ensure that heat will not be a significant issue. Heat produced by equipment (diesel or electric) may not dissipate quickly in areas of minimal velocity, and could result in high air temperatures which could pose a hazard to workers. This will require the proper design and implementation of auxiliary ventilation systems.

Specific Area Ventilation Requirements

The basic ventilation model was developed with the following general area ventilation requirements;

Main Development Ventilation

The main adit will be ventilated during development with twin 1.4m diameter flexible duct lines. A slightly larger duct is selected to keep the pressure/power requirements on the auxiliary fans lowers. Both a truck and an LHD can be operated simultaneously in this development area.

Transverse Stoping

The transverse stoping areas will be ventilated by 120 kW auxiliary fans and 1.4m duct, each fan/duct system will service three stopes (one with an LHD, two with support equipment). The upper levels will require a roll up curtain to be installed to control by-pass circulation.

Longitudinal Stoping

The longitudinal stoping areas will be ventilated by 75 kW auxiliary fans and 1.2m duct, each fan/duct system will service a single stope with a long access and is designed to support one LHD. The upper levels will require a roll up curtain to be installed to control by-pass circulation.

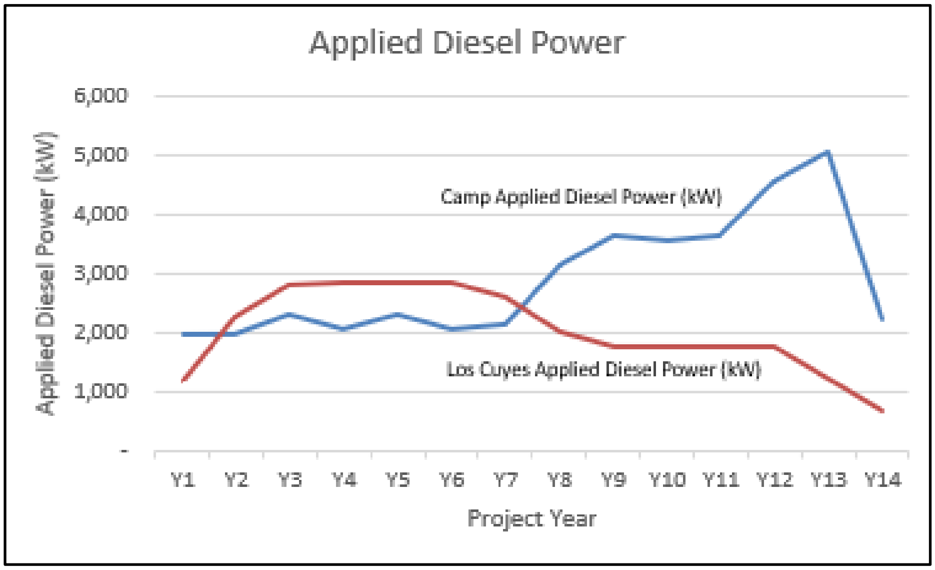
Fixed Facilities Airflow Allocation

Each mining zone (Camp and Los Cuyes) has a 50 m3/s airflow allocation to support minor shops, fuel bays, etc.

16.12.3 Airflow Calculations and Equipment

SRK engineers developed an equipment schedule from which the overall applied diesel power for the mine could be estimated. The applied diesel equipment load power used in the airflow calculation is shown in Figure 16-21.

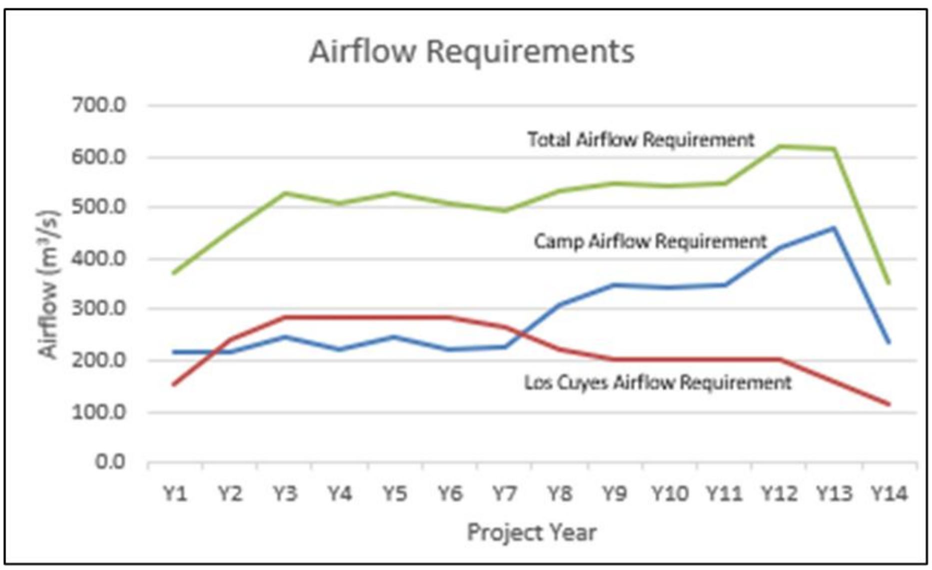
Figure 16-21: Applied Diesel Power



Source: SRK, 2025

The overall airflow requirement utilizes the diesel dilution airflow as a base value then adds additional airflow for leakage, point of use applications, and development areas. The overall airflow requirement staged during the life of mine is shown in Figure 16-22.

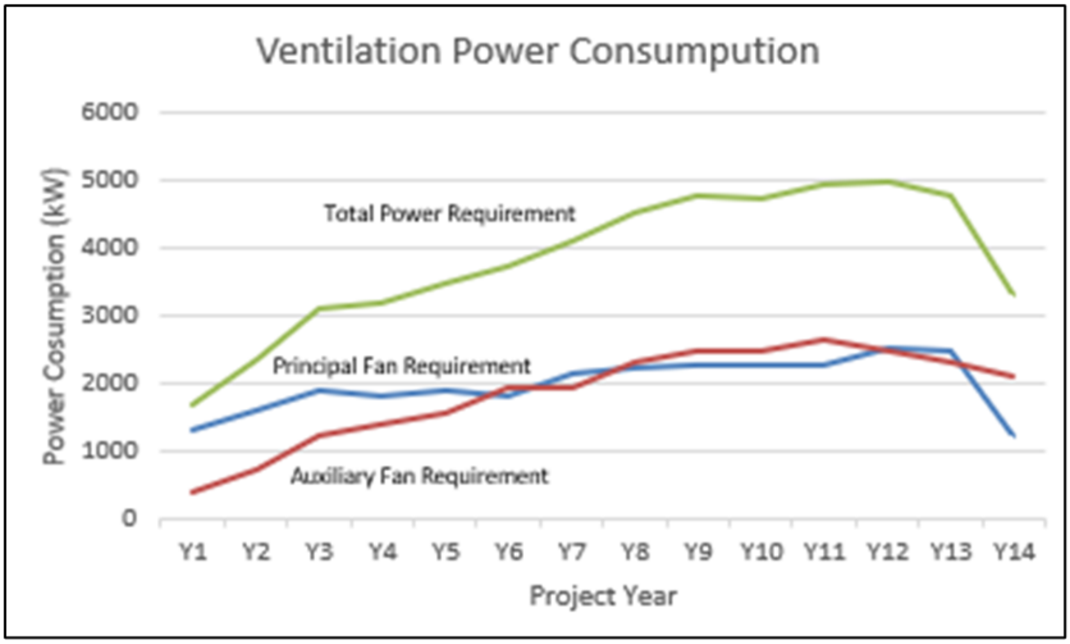
Figure 16-22: General Airflow Requirement Based on Diesel Dilution, Leakage, and Factors



Source: SRK 2025

The ventilation model was developed with the VentSimTM software package to estimate the overall fan power for the decline development, and for a life of time frame as shown in Figure 16-23. The fan power was then calculated for the maximum life of mine time phase. The yearly power requirements are based on scaling the ventilation power with the applied diesel power. The auxiliary ventilation fan power was determined by the placement of assumed operating fan locations and applying a 60% capacity.

Figure 16-23: Ventilation Power Requirements (kW)



Source: SRK 2025

The required fan operating characteristics were developed based on the life of mine time frame during which both production and development activities are taking place. These fan operating points are identified in Table 16-32.

Table 16-32: Ventilation System Fan Requirements

Fan List	Airflow (m^3/s)	Pressure (kW)	Installation Losses (15%) (kW)	Fan Total Pressure (kW)	Fan Power (85% Efficiency) (kW)
Primary Exhaust Fans (Camp)	375	2.93	0.44	3.37	1490
Primary Exhaust Fan (Los Cuyes)	300	2.69	0.40	3.10	1100
Booster Fan (Camp)	250	1.07	0.16	1.23	385
Longitudinal Stope Fan	25	2.12	0.32	2.44	75
Transverse Stope Fan	40	2.00	0.30	2.30	120
Main Development Fans (Two Systems)	80	2.00	0.30	2.30	240
Small Alcove Fan	n/a	n/a	n/a		20

The principal surface exhaust fans and underground booster fan are assumed to be developed with parallel fans installed for each system. As an example, the primary exhaust fan system for the Camp exhaust would be provided by two fans operating in parallel (each fan providing approximately 188 m^3/s, 745 kW) to achieve the required system operation.

16.12.4 Secondary Egress

Prior to the mine entering into production, a secondary means of egress must be developed. Although there will be a small raise developed parallel to the spiral ramp system providing secondary egress in the mining and development zones, the mine will require a means of egress to the surface. Because the mine will be in production prior to the completion of the ramp close to surface and the extension of the small segmented parallel raise system, a portable/relocatable bullet hoist can be used to provide egress through either the Camp or Los Cuyes fresh air raise.

16.13 Mine Services and Infrastructure

16.13.1 Contractor Involvement

The Condor underground project is proposed to be a contractor-operated mining operation. The mining contractor will be fully responsible for all underground mining activities except owner's small technical and contractor supervision team. The contractor will also bring its own primary and secondary mining equipment.

16.13.2 Mine Services and Infrastructure

As the mines utilize similar mining methods and a common mine access, the underground infrastructure for both is essentially the same between them, with certain aspects differing in size or amount depending on the mine and mining front. This section summarizes the key items of underground infrastructure that will be constructed.

Primary Infrastructure

The amount and location of the underground infrastructure are dictated by the development and production schedules for each of the two mines. The infrastructure contemplated in this study is typical of other modern open stoping mining operations, and the primary components consist of the following:

- Electrical and communications systems
- Explosive magazines
- Fuel bays
- Maintenance facilities
- Mine services
- Refuge stations (both permanent and portable)
- Re-muck bays
- Storages (e.g., development, shotcrete)
- Sumps and dewatering equipment

Further detail regarding these items is presented below.

Power and Fuel Distribution Systems

Underground power will be delivered via 13.8 kV main feeder cables from the portal to the sub-stations located in each of the Camp and Los Cuyes mines. From there, power will be distributed to mine power centers (MPC) located adjacent to the point of utilization (e.g., dewatering sumps, level substations, etc.).

Fuel bays will be located within each mining front, with portable fueling systems capable of being added as mining progresses. The bays will feature safety mechanisms such as concrete containment (in the event of a leak) and automatic fire suppression equipment.

Explosive Magazines

Multiple sets of explosive and initiator (cap) magazines will be constructed in each mine, with each set of magazines sited to allow them to facilitate development and production activities on multiple levels. It is assumed that bulk emulsion will be used for development and production blasting, as it can provide superior overbreak and dilution control when compared to packaged products or ANFO. The magazines shall be constructed in a manner to enable safe and efficient transfer of product to/from the magazine and explosive trucks.

Maintenance Facilities

Each mine will feature a primary mobile equipment maintenance shop, centrally located to allow for efficient access during the mine’s entire operating life. These facilities shall include wash, lubricant, and repair bays, warehouse facilities, and administrative offices. Smaller repair bays will also be constructed, with one typically located in each mining front. These supplementary facilities are intended for the management of smaller repairs to ease the burden on the primary shops.

Refuge Stations & Latrines

A combination of portable and standard refuge stations will be used throughout the mines. A permanent refuge will be sited in each mining front with portable refuge stations utilized in active mining areas that are removed from the vicinity of the permanent refuges or from the routes of emergency egress.

Latrines shall also be located near both the permanent and portable refuge stations.

Re-Muck Bays and Storages

Allowance for two re-muck bays per level (one for ROM, one for waste) has been made in the infrastructure estimate. These bays can be re-purposed depending on the amount of production activity on the level.

Construction facilities will include one shotcrete storage and one development storage per every three levels. As the mine development progresses the frequency of these storages will be continually evaluated to ensure they align with local development requirements.

Sumps and Dewatering

The dewatering system will be primarily managed via gravity methods both on level and from level to level. Each production level will feature a small sump designed to manage the localized water introduced by mining processes and infiltrating as groundwater. Water will then be directed to a larger sump facility, with one of these main sumps located in each mining front. Slimes will be decanted in the main sumps, with water then discharged to surface for treatment and/or recycling.

17 Recovery Methods

17.1 Introduction

There are several distinct mineralization zones within the Condor property. The study focuses on the Camp Zone and the Los Cuyes Zone. The Condor process flowsheet is developed based on the metallurgical test work results described in Section 13.0 and the mine plan presented in Section 16. The primary metal values are gold and silver, with minor associated values from lead and zinc. The metallurgical test results indicate that the Condor mineralization is amenable to gold and silver recovery through a combination of gravity concentration and cyanidation. Although the lead and zinc grades of the mineralization are relatively low, the lead and zinc minerals also respond well to conventional flotation process.

The proposed process plant will treat the mineralized material at a milling rate of 5,000 t/d with an average LOM head grade of 2.15 g/t gold and 14.2 g/t silver. The overall gold and silver recoveries to doré in cyanidation circuit are estimated to be approximately 93% and 46%, respectively. A two-stage grinding circuit, integrated with a gravity concentration, is proposed to grind the cyanide leach feed to 80% passing (P80) approximately 74 µm. The ground mill feed is processed in a carbon-in-pulp (CIP) circuit. The loaded carbon is washed and stripped, and the pregnant solution is treated by an electrowinning unit to recover the gold and silver, producing gold-silver doré. The leach residue is treated to destroy residual weak acid dissociable (WAD) cyanide. Subsequently, the leach residue is further processed by conventional differential flotation to produce marketable silver-lead and zinc concentrates separately. The flotation tailings is thickened and pumped to tailings storage facility (TSF) for storage.

The processing plant will operate 24 hours per day (h/d) and 365 days per annum (d/a) with an operational availability of 92%.

The proposed process plant will include the following unit operations:

- Primary Crushing: Run-of-mine (ROM) from the Camp and Los Cuyes underground mine is trucked to a ROM pad within the primary jaw crushing area. A truck dump hopper with a fixed grizzly and a jaw crusher in open circuit will reduce the ROM particle size to 80% passing approximately 100 to 120 mm.
- Primary and Secondary Grinding: The grinding circuit consists of a SAG mill and a ball mill in closed circuit with hydrocyclones, reducing the crushed materials to a product of 80% passing approximately 74 µm.
- Gravity Separation: Integrated with the secondary grinding circuit, a gravity separation circuit receives approximately 25% of the hydrocyclone feed to recover the coarse-free gold grains using one centrifugal concentrator. The gravity concentrate is leached in an intensive leach reactor to extract the recovered gold and silver.
- Cyanide Leaching: The ground leach feed is thickened and directed to a CIP circuit for cyanide leaching.
- Acid Washing, Desorption, and Refining: The loaded carbon from the CIP circuit is treated by acid washing and elution to produce a gold- and silver-rich solution for electrowinning, which yields a final gold-silver doré product for sale. The stripped carbon is reused in the CIP circuit, either after acid washing to remove inorganic contaminants or following thermal regeneration to remove organic foulants. Fresh carbon is added as required after attrition and sizing treatment.

- Cyanide Detoxification: The cyanide leach residue is detoxified using the sulfur dioxide (SO2)/air process, reducing WAD cyanide to less than 1 ppm level prior to further processing by silver-leach and zinc flotation.
- Flotation: Lead, zinc and the residual gold and silver are recovered from the detoxified leach residue by differential flotation, producing a silver-lead concentrate and a zinc concentrate.
- Final Flotation Tailings Disposal: The flotation tailings slurry is thickened and the thickener underflow is pumped to the TSF. A reclaim water system pumps supernatant water from the TSF back to the mill site for process reuse.

17.2 Plant Design Criteria

17.2.1 Process Design Criteria

The key process design criteria is presented in Table 17-1. The design life of the Condor process plant is approximately 13 years. Where applicable, design factors are incorporated into equipment sizing and circuit design.

Table 17-1: Major Process Design Criteria

Criteria	Unit	Nominal Value
Mill Feed Characteristics		
Specific Gravity	-	2.7
ROM Moisture	%	3
Bulk Density	t/m^3	1.7
Abrasion Index (Ai) – Los Cuyes	kWh/t	0.088
Average Bond Ball Mill Work Index (BWi) – Camp	kWh/t	14.9
Los Cuyes	kWh/t	12.5
JK SMC – Camp (Average) – Camp		
Los Cuyes	A × b	56.9
LOM Average Mill Feed Grade	g/t Au	2.15
	g/t Ag	14.2
	% Pb	0.06
	% Zn	0.54
Operation Schedule		
Operating Days Per Year	d/a	365
Daily Shifts		
Crushing	shift/d	2
Grinding/Flotation/Leaching	shift/d	2
Operating Hours Per Shift		
Crushing	h/shift	12
Grinding/ Leaching/Flotation	h/shift	12
Process Plant Throughput	t/d	5,000
Process Plant Throughput – Crushing	t/h	298
Grinding/Flotation/Leaching	t/h	226
Main Process Plant Availability	%	92
Process Method & Metal Recovery		
Gold and Silver Recovery Method	-	Gravity + Cyanidation + Flotation

Criteria	Unit	Nominal Value
Lead and Zinc Recovery Method	-	Flotation
Primary Grind Size, 80% Passing	µm	74
Leach Residue Treatment (Cyanide Destruction)	-	SO_2/Air
Tailings Storage	-	Conventional Wet Storage
Metal Recovery – Overall	% Au	94
	% Ag	64
	% Pb	36
	% Zn	54

Note: ROM = Run of Mine

17.2.2 Operating Schedule and Availability

The simplified process plant operates on two 12-hour shifts per day, 365 d/a. The overall availability of the primary crushing circuit is 70%. The grinding, gravity concentration leaching, and flotation circuits operate with an availability of 92%.

17.2.3 Plant Design

The proposed process flowsheet is presented in Figure 17-1. The general process plant arrangement is illustrated in Figure 17-2 and Figure 17-3.

Figure 17-1: Simplified Process Flow Diagram

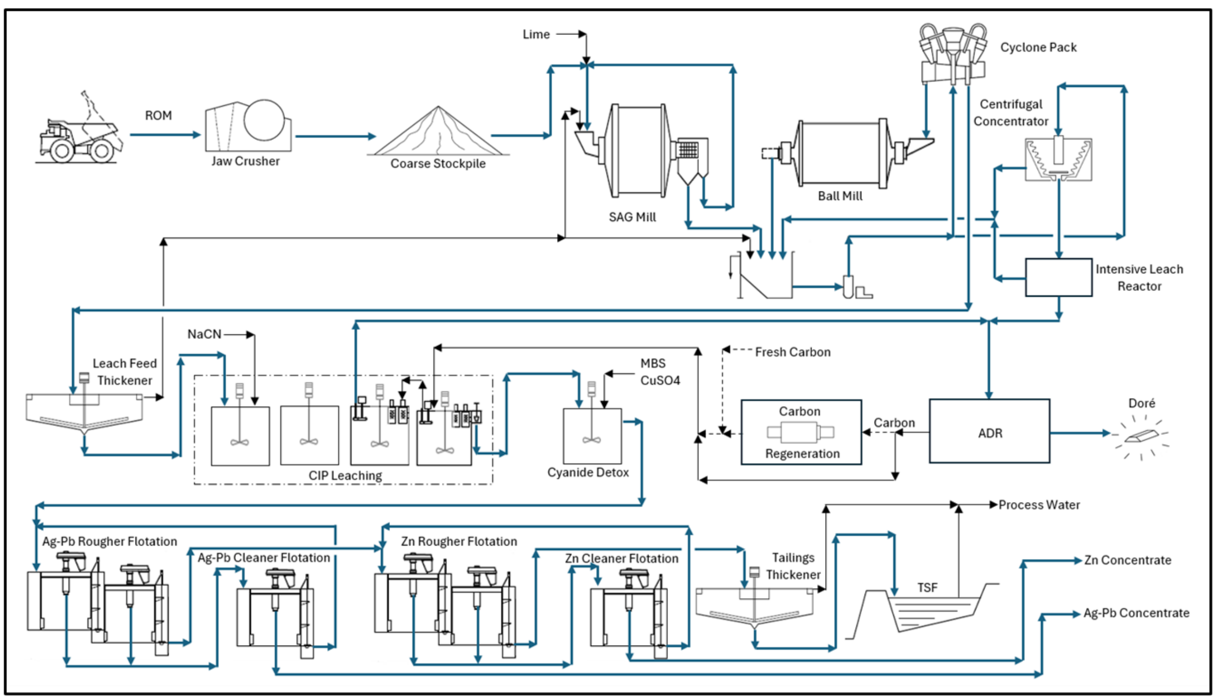


Figure 17-2: Mill Layout – Overall Mill Site

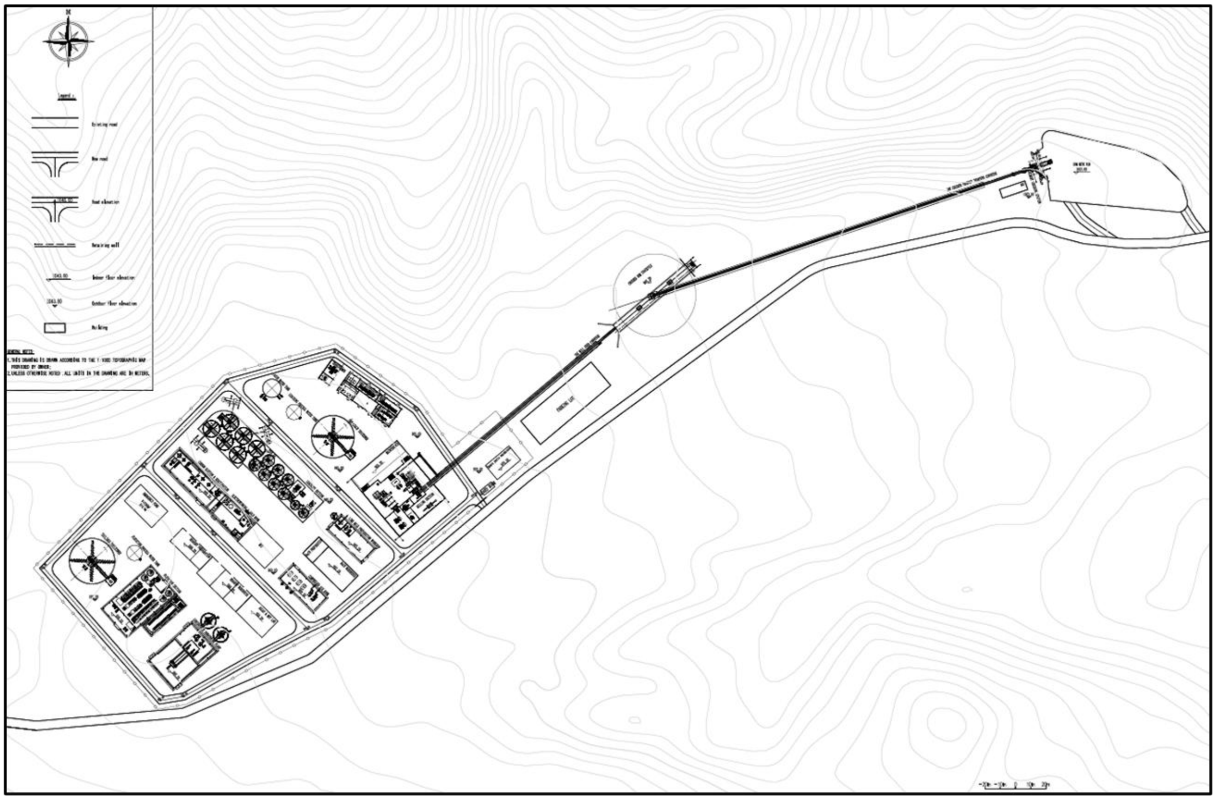
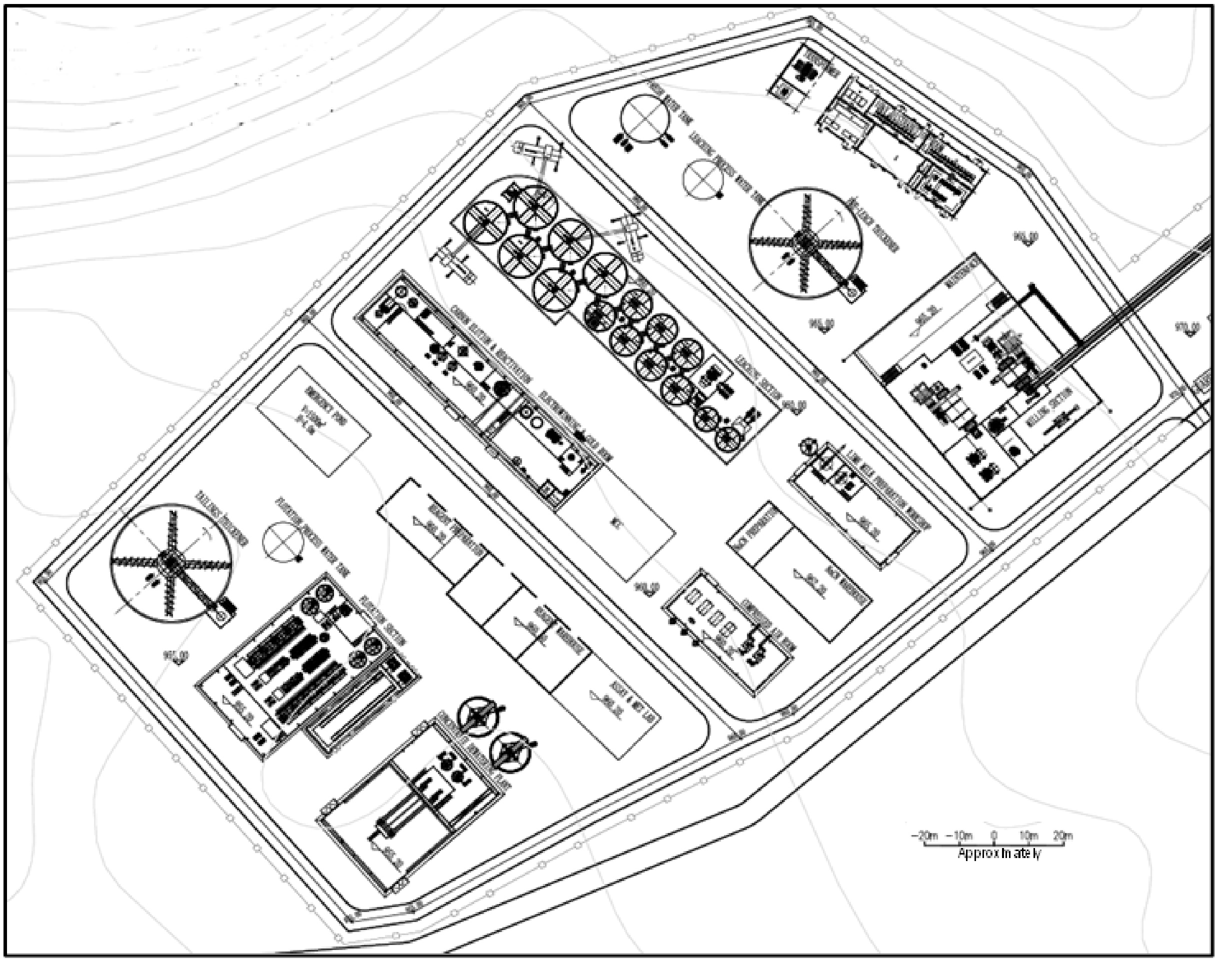


Figure 17-3: Mill Layout – Grinding/Cyanidation/Flotation Areas



17.3 Process Plant Description

17.3.1 Primary Crushing

ROM from the underground mine is hauled by trucks to the primary crushing area consisting of a ROM receiving/storage pad and a primary crusher. The crushing facility has an average processing capacity of 5000 t/d. ROM materials are either directly dumped into the jaw crusher feed dump hopper or stockpiled on the ROM receiving pad and reclaimed by a front-end loader to the jaw crusher feed hopper. The jaw crusher feed hopper is equipped with a static grizzly with 600 mm spacing bars. Oversize material from the static grizzly is removed for particle size reduction using a rock breaker. The undersize material reporting to the dump hopper is reclaimed by a vibrating grizzly feeder at a rate of 298 t/h. The vibrating grizzly oversize material feeds directly into the jaw crusher with a feed opening of approximately 1,150 mm × 760 mm or equivalent, with an installed power of 160 kW. The jaw crusher discharge, together with vibrating grizzly feeder undersize, reports onto a transfer conveyor and then onto an overland conveyor to a coarse mill feed stockpile. The primary crushing reduces the feed particle size to 80% passing approximately 100 to 120 mm.

Mill Feed Stockpile

The SAG mill feed surge stockpile is designed to have a live capacity of 5,000 t. The crushed product from the primary crushing facility is transported to the stockpile via a conveying system, consisting of one transfer conveyor and one overland conveyor.

The SAG mill feed from the mill feed stockpile is reclaimed by three apron feeders, each 1,370 mm wide and 8,200 mm long, onto the SAG mill feed conveyor at a nominal rate of 226 t/h.

The crushed mill feed conveyor transfer points at the SAG mill feed surge stockpile are equipped with water-spray dust suppression systems to control fugitive dust generated while transporting the crushed materials.

17.3.2 Grinding, Classification, and Gravity Concentration

A SAB (SAG mill + Ball Mill) grinding circuit is installed to grind the coarse mill feed to a target product size of 80% passing approximately 74 µm. It is assumed that the mill feed is relatively soft and there is no need for a pebble crusher in the grinding circuit. However, space for potential installation, if required in the future, is allotted. Further studies are required to verify whether a pebble crushing circuit is needed, including additional test work for the grinding circuit and simulations.

One centrifugal gravity concentrator, with a nominal unit capacity of 226 t/h, is installed to recover gold/silver nugget grains that are liberated or partially liberated from their host minerals.

The main grinding/gravity concentration circuit includes:

- One SAG mill, 6,700 mm diameter by 3,350 mm long (22 ft by 11 ft) (EGL), driven by a 2,500 kW variable frequency drive (VFD)
- One ball mill, 4,900 mm diameter by 8,500 mm long (16 ft by 27.9 ft) (EGL), powered by a 3,000 kW motor
- One 1.8 m wide by 4.9 m long vibrating screen for SAG mill discharge
- Two 300 mm × 250 mm hydrocyclone feed slurry pumps, each with an installed power of 432 kW
- One hydrocyclone pack with eight 350 mm hydrocyclones, six in operation and two on standby
- One centrifugal gravity concentrator and ancillary screens
- One intensive cyanide leach unit to process the concentrate from the centrifugal gravity concentrator; the washed leach residue is recycled backed to the ball mill grinding circuit while the leach solution is sent to the gold room to recover the leached gold and silver
- One particle size analyzer and one online sampler for sampling and analyzing the particle size of the hydrocyclone overflow

The crushed mill feed from the stockpile is reclaimed onto a belt conveyor that transports the mill feed to the SAG mill, equipped with 65-mm pebble grates to discharge the fine fraction from the SAG mill. The SAG mill discharge is screened by a vibrating screen with 10-mm wide slots. The oversize from the screen is transported by conveyors back to the SAG mill feed conveyor for further grinding. The screen undersize flows by gravity to the hydrocyclone feed pump box, where the ball mill discharge and the gravity concentrator tailings report as well.

The ball mill is operated in closed circuit with hydrocyclones and a centrifugal gravity concentrator. The product from the SAG mill, the ball mill, and the gravity concentrator is discharged into the ball mill discharge pumpbox. Approximately 25% of the discharge from the pumpbox is pumped to the gravity concentrator while the remaining slurry is pumped to the hydrocyclone pack for classification. The hydrocyclone underflow returns by gravity to the ball mill. The circulating load to the ball mill is approximately 300%. The particle size of the hydrocyclone overflow, or the product of the primary grind circuit, is 80% passing approximately 74 µm. The pulp density of the hydrocyclone overflow slurry is approximately 32% w/w solids.

Steel balls are manually added into the SAG and ball mills on a batch basis as grinding media.

The concentrate from the gravity concentrator is processed in an intensive cyanide leach reactor on a batch basis. The leach residue is recycled back to the ball mill grinding circuit, while the leach solution is sent to the gold room to recover gold and silver from the intensive leach solution.

Dilution water is added to the grinding circuit as required. A particle size analyzer is installed to monitor and optimize the operating efficiency, in conjunction with an automatic sampling system and the required instrumentation such as solid density, pressure, and flow rate meters.

17.3.3 Cyanide Leaching and Carbon Adsorption

The hydrocyclone overflow from the primary grinding circuit is screened to remove any oversize material. The trash screen undersize flows by gravity to a 28-m diameter leach feed high-rate thickener for the optimum solid density control in the downstream cyanidation. Diluted flocculant solution is added to the thickener to assist the thickening process. The thickener overflow reports to one process water tank, which services the grinding and cyanidation circuits.

The thickener underflow is pumped to the head of a cyanide leach bank and adjusted by adding process water to achieve the optimum cyanidation slurry solid density of 40 to 45% w/w. The cyanidation is performed in a CIP circuit consisting of six 10.7-m diameter by 13.4-m high direct leach tanks and seven 7.5-m diameter by 9.4-m high CIP tanks. The leach tanks provide a total retention time of more than 28 hours. The tanks are aerated with compressed air from two oil-free compressors (one in operation and one on standby). The CIP tanks are equipped with air-lift carbon transferring and inter-stage screen systems to advance the loaded carbon to the preceding leach tank. Activated carbon is added into the last CIP leach tank and the loaded carbon leaves the CIP circuit from the first CIP tank. Activated carbon concentration is maintained approximately 15 g/L slurry within the CIP tanks.

Sodium cyanide is added to the leach tanks to extract gold. If required, hydrated lime slurry is added to maintain the slurry pH at approximately 10.5.

The loaded carbon leaving the first CIP tank is transferred to the carbon stripping circuit, while the leach residue is sent to a carbon safety screen to recover any coarse carbon grains. The screen undersize reports to a cyanide destruction circuit prior to being pumped to the downstream silver/lead and zinc flotation circuit.

The key equipment in the leach circuit includes:

- One 28-m diameter thickener
- One 2.0 m wide by 4.0 m long thickener feed trash screen
- Six 10.7 m diameter by 13.4 m high leach tanks
- Seven 7.5 m diameter by 9.4 m high CIP leach tanks equipped with carbon transferring and screen systems
- One 1.2 m wide by 2.4 m long loaded carbon screen
- One 2.0 m wide by 4.0 m long carbon safety screen with 0.6 mm apertures
- Two dedicated oil-free type air compressors.

Cyanide detection/alarm systems, safety showers, and emergency medical stations are provided in the area to protect operators. Automatic samplers are installed to collect samples for metallurgical balance and to monitor operational performance.

17.3.4 Loaded Carbon Stripping and Gold-Silver Refining

The loaded carbon is harvested from the CIP circuit. The harvested carbon is then pumped to a gold and silver elution and electrowinning system. The main equipment includes a desorption column, electric heaters, filters, circulating pumps, desorption solution tank, electrowinning cell, and related rectifier cabinet and control system.

The operation temperature for this system is high, up to 150°C, which is higher than the typical conventional operation temperature. This system operates at a relatively high pressure, up to 0.5 MPa. Under the conditions of high temperature and high pressure, gold desorption from the loaded carbon and gold and silver electrodeposition is expected to be completed in a relatively short cycle of 12 hours.

Pregnant solution from the loaded carbon elution circuit is pumped through filters and then to the electrowinning cell, where the gold and silver are electrochemically deposited onto stainless steel woven wool cathodes. Periodically, the stainless-steel cathodes are cleaned to remove precious metals in the form of sludge. The gold-silver sludge is filtered by a vacuum filter for dewatering on a batch basis.

The temperature-control system automatically regulates the electric heaters to maintain the system temperature. The elution and electrowinning system is equipped with pressure-relief valves to protect the system. Sodium hydroxide (NaOH) is used for the elution of the loaded carbon.

The depleted solution from the electrowinning circuit is circulated within the heating units and stripping vessel or sent to the cyanide leach circuit for reuse.

The filtered gold sludge cake is mixed with flux and melted at approximately 1,150°C in an induction furnace to produce gold-silver bullion containing mostly gold and silver with some impurities.

The loaded carbon elution and electrowinning system and gold melting are housed in a dedicated building. The areas are provided with sufficient ventilation. The gold room is located in a secure facility with restricted access to personnel and monitored by 24-hour CCTV surveillance.

17.3.5 Carbon Reactivation

The stripped carbon is transferred by a recessed impeller pump to a stationary dewatering screen for dewatering, and then to a carbon regeneration kiln feed hopper, prior to reactivation. The carbon regeneration kiln is capable of regenerating barren carbon at a rate of approximately 200 kg per hour. The kiln is heated by electricity and operated at approximately 650°C in an inert atmosphere. The regeneration is carried out to remove or burn away any passivating organic foulants such as oils or greases accumulated during the CIP process. The hot, reactivated carbon then leaves the kiln and is quenched in a conical bottomed quench tank flooded with water. The regenerated carbon is subsequently sized and circulated back into the CIP circuit. As required, make-up fresh carbon is added. The fresh carbon is treated by attrition and sizing prior to being introduced into the CIP circuit.

The main equipment used for the carbon reactivation process and make-up carbon addition includes:

- One carbon reactivation kiln feed hopper
- One electrically fired carbon reactivation kiln
- One carbon quench tank
- One carbon sizing screen (1.0 m wide by 1.5 m long)
- One carbon abrasion tank equipped with an attrition agitator
- Fine carbon handling associated equipment

17.3.6 Treatment of Leach Residue

The residue slurry from the carbon safety screen in the cyanide leach circuit is pumped to a cyanide detoxification circuit consisting of two reactors, each 6.0 m in diameter and 7.0 m in high and equipped with an agitator. The WAD residual cyanide in the slurry is decomposed through a sulphur dioxide/air oxidation process. The reagents used include sodium bisulphite and copper sulphate, as needed. The treated leach residue slurry is expected to contain less than 1 ppm of WAD cyanide. The reagent storage, preparation, and dosing systems for these reagents are provided. An emergency discharge pond, servicing the processing plant, is available for any emergency discharge of the leach slurry.

Following detoxification, the residue slurry is pumped to the downstream flotation circuit for further recovery of lead, zinc, and residual silver.

17.3.7 Silver-Lead and Zinc Flotation

Silver-Lead Flotation

The cyanide detoxified slurry is repulped by adding dilution water, if required, to provide the optimum solid density for the silver-lead flotation. The slurry is conditioned in two stages with lime, zinc sulfate ($ZnSO_4$), and collectors (3418A and A208 or equivalent) prior to silver-lead rougher and scavenger flotation. Methyl isobutyl carbinol (MIBC) is added into the flotation circuits as a frother to assist in recovering the target minerals. The resulting flotation rougher and scavenger concentrates are pumped to a silver-lead cleaner flotation circuit for further upgrading. The cleaner circuit consists of one additional conditioning tank and three stages of cleaner flotation. The tailings from the first cleaner flotation cells is recycled back to the first flotation cell in the preceding lead rougher flotation bank. The silver-lead rougher scavenger flotation tailings are pumped to the zinc flotation circuit

The final silver-lead flotation concentrate, mainly containing silver, is sent to the silver-lead flotation concentrate dewatering circuit to produce the concentrate filter cakes.

Zinc Flotation

The tailings from the silver-lead flotation circuit report to the zinc flotation. The tailings slurry is conditioned in two stages with copper sulfate (CuSO4) and collectors (isopropyl xanthate (SIPX) and A208 or equivalent) prior to zinc rougher and scavenger flotation. MIBC is added to the zinc flotation circuits as a frother to assist in recovering the target minerals. The zinc rougher scavenger flotation tailings leave the flotation circuit and are pumped to a 30-m diameter high-rate tailings thickener.

The resulting flotation rougher and scavenger concentrates are pumped to a regrinding circuit to further liberate zinc-bearing minerals from the gangue. Whether the regrinding is required should be further verified in future studies. The reground zinc rougher scavenger concentrate is further upgraded in the zinc cleaner flotation circuit. The cleaner circuit consists of one additional conditioning tank and three stages of cleaner flotation. The tailings from the first cleaner flotation cell bank are recycled back to the first flotation cell in the preceding zinc rougher flotation bank.

The final zinc flotation concentrate, mainly containing zinc value, will be sent to the zinc flotation concentrate dewatering circuit.

The main equipment used for the flotation circuits includes:

- Five 38-m^3 flotation tank cells for lead rougher and scavenger flotation
- Three 8-m^3 flotation tank cells for first lead cleaner flotation
- Three 3-m^3 flotation tank cells for second and third lead cleaner flotation
- Four 38-m^3 flotation tank cells for zinc rougher and scavenger flotation
- Three 8-m^3 flotation tank cells for first zinc cleaner flotation
- Three 3-m^3 flotation tank cells for second and third zinc cleaner flotation
- One regrinding tower mill for zinc rougher flotation concentrate
- Conditioning-related mixing tanks

17.3.8 Flotation Concentrate Dewatering

Both the flotation concentrates from the silver-lead and zinc differential flotation circuits are pumped to two separate 4-m diameter thickeners separately. The concentrates are thickened to approximately 60% w/w solid density. The slurries are then sent two separate surge tanks before being pumped to two separate pressure filters to produce filter cakes with a moisture content of approximately 9% w/w.

The filtered lead and zinc concentrates are conveyed and stored in two separate concentrate stockpiles. The concentrates are loaded and shipped in bulk to a seaport for delivery to oversea smelters.

17.3.9 Tailings Management

The zinc rougher scavenger flotation tailings are pumped to a 30-m diameter high-rate thickener. The tailings is thickened to approximately 60% w/w solids. Diluted flocculant solution is added to the thickener feed well to assist the thickening process. The thickener underflow is pumped to the TSF for subaqueous storage via a HDPE overland pipeline. The thickener overflow is pumped to a separate process water tank servicing the flotation circuits. The supernatant from the TSF is reclaimed by reclaim water pumps to the two process tanks located at the plant site as process make-up water. The tailings storage facility, including tailings dam construction and water management, is described in Section 18.

17.3.10 Reagents Handling

The main reagents used in the process include:

- Cyanide Leach and Gold Recovery: Pebble lime (CaO), sodium cyanide (NaCN), activated carbon, sodium hydroxide (NaOH), hydrochloric acid (HCl), and flux
- Cyanide Destruction: Sodium bisulphite (NaHSO3), copper sulphate (CuSO4), and pebble lime (CaO)
- Silver-Lead and Zinc Flotation: 3418A, A208, isopropyl xanthate (SIPX), copper sulfate (CuSO4), zinc sulfate (ZnSO4), and Methyl Isobutyl Carbinol (MIBC)
- Other Reagents: Flocculant and antiscalant

All the reagents are prepared in a separate reagent preparation and storage facility within a containment area. The reagent storage tanks are equipped with level indicators and instrumentation to ensure that spills do not occur during operation.

Hydrochloride acid is diluted to approximately 10 to 20% prior to being added to the required process circuits via a metering pump, while 208A, 3418A, antiscalant, and MIBC are added in undiluted form.

Solid reagents (sodium cyanide, copper sulphate, zinc sulfate, SIPX, and sodium bisulphite) are mixed with fresh water to 10 to 25% solution strength in respective mixing tanks and are stored in separate holding tanks before being added to various addition points by metering pumps. Pebble lime is directly added onto the SAG mill feed conveyor at a controlled rate. The lime is also slaked, diluted to approximately 15% w/w solids, and added to the circuits via a pressurized loop.

Cyanide monitoring/alarm systems are installed at the cyanide preparation and leaching areas. Emergency medical stations and emergency cyanide detoxification chemicals are provided at the related areas.

Flocculant is received in solid form and prepared in a packaged preparation system, which includes a screw feeder, a flocculant eductor, and mixing devices. The prepared solution is transferred and stored in an agitated flocculant holding tank. Flocculant is then further diluted and added via metering pumps to the leach feed thickener and the final tailings thickener.

17.3.11 Water Supply

Two separate water supply systems support the process operations: one freshwater system and one process water system for various process circuits.

Fresh Water Supply System

Fresh water is used primarily for the following:

- Fire water for emergency use
- Cooling water for mill motors and mill lubrication systems
- Carbon elution/intensive leach/dust suppression
- Reagent preparation

By design, the firewater tank remains full at all times for any emergency. Potable water is supplied by bottled water.

Process Water Supply System

Two separate process water tanks supply the process water:

- One for the grinding/gravity, cyanide leach, and gold recovery circuits. The overflows from the cyanide leach feed thickener report to this process water tank.
- One for the flotation and related concentrate and tailings handling circuits. The overflows from the flotation tailings thickener report to this process water tank.

Any additional make-up process water is reclaimed from the TSF and pumped to the process water tanks. The process water is distributed to the various service points. A separate high-pressure system provides pump gland seal water, which is pumped from the process water tank to the various service points. The gland seal water system includes a filter to remove suspended solids from the water.

17.3.12 Air Supply

Plant air service systems supply air to the following areas:

- Leach/cyanide detoxification circuits: High-pressure air is supplied by dedicated oil-free type air compressors
- Crushing circuit: High-pressure air is provided for the dust suppression system and other services by an air compressor
- Plant services: High-pressure air is delivered for various services by two dedicated air compressors
- Instrumentation services: Instrument air is supplied from the plant air compressors, dried and stored in a dedicated air receiver prior to distribution to various service points
- Flotation circuits: Low pressure air is provided by two dedicated air blowers.

17.3.13 Assay and Metallurgical Laboratory

An assay laboratory, equipped with necessary sample preparation equipment and analytical instruments to provide routine assays for the mine, process, and environmental departments, is provided. The assay laboratory performs various assays, including gold fire assay. The data obtained from the assay is used for routine process optimization and metallurgical balance accounting.

A metallurgical laboratory located is also located in the process plant. The laboratory is equipped with metallurgical test equipment and conducts metallurgical tests to optimize the process flowsheet and improve metallurgical performance.

17.3.14 Process Control and Instrumentation

The plant control system consists of a distributed control system (DCS) with PC-based operator interface stations (OIS) located in the plant site control room. The plant control room is staffed by trained personnel 24 h/d.

The DCS, in conjunction with the OIS, performs all equipment and process interlocking, control, monitoring, event logging, and report generation.

Programmable logic controllers (PLCs) or other third-party control systems supplied as part of mechanical packages are interfaced to the plant control system via Ethernet network interfaces when possible.

Operator workstations are capable of monitoring the entire plant site process operations and provide alarm viewing and equipment control within the plant.

In addition to the plant control system, a CCTV system is installed at various locations throughout the plant, especially at the crushing facility, grinding facility, gravity concentration area, and gold recovery facilities. The cameras are monitored from the central control room.

The primary grinding control is enhanced with the installation of an automatic sampling system. The system collects samples from the hydrocyclone overflow for on-line particle size analysis and the daily metallurgical balance. Samples from various process streams are automatically collected and assayed for daily metallurgical balance and process optimization. An on-stream analyzer is provided for automatic process control, especially for the flotation circuits.

17.4 Yearly Metallurgical Performance Projection

According to the test work results described in Section 13.0 and the proposed mine production schedule, the plant metallurgical performance is projected annually and presented in Table 17-2. Further metallurgical test work is recommended to improve projections of the metallurgical performance.

Table 17-2: Yearly Gold Production Projections

Year	Feed					Dore		Silver Lead Conc				Zinc Conc			
	Tonnage	Grade				Recovery		Grade		Recovery		Grade		Recovery	
	kt	Au, g/t	Ag, g/t	Pb, %	Zn, %	Au, %	Ag, %	Ag, g/t	Pb, %	Ag, %	Pb, %	Ag, g/t	Zn, %	Ag, %	Zn, %
Year 1	1,372	2.29	10.4	0.04	0.32	92.7	47.5	2,400	28.6	11.8	35.3	140	45.0	4.5	47.5
Year 2	1,800	2.51	11.3	0.05	0.41	93.2	47.5	2,030	28.2	11.8	35.2	117	45.0	4.5	48.0
Year 3	1,800	2.22	13.4	0.06	0.49	92.6	47.7	2,100	29.2	11.7	35.4	115	45.0	4.3	46.6
Year 4	1,800	2.33	15.5	0.07	0.52	92.9	46.5	2,370	32.6	11.9	36.1	153	45.0	5.7	50.1
Year 5	1,801	2.22	12.5	0.05	0.44	92.3	48.3	2,170	27.6	11.7	35.0	97	45.0	3.6	47.5
Year 6	1,800	2.32	15.6	0.07	0.52	92.9	47.2	2,120	29.9	11.8	35.6	135	45.0	5.0	49.9
Year 7	1,800	1.89	14.6	0.06	0.54	92.8	45.9	2,760	34.6	12.0	36.5	144	45.0	6.5	55.0
Year 8	1,800	2.01	15.2	0.06	0.57	93.2	45.1	3,400	40.0	12.1	37.4	153	45.0	7.4	58.1
Year 9	1,800	2.18	15.4	0.06	0.59	93.5	44.7	3,250	40.3	12.2	37.4	160	45.0	7.9	58.0
Year 10	1,800	1.99	12.6	0.05	0.58	93.3	44.6	3,370	40.8	12.2	37.5	135	45.0	8.0	58.5
Year 11	1,800	2.00	14.4	0.05	0.69	93.6	44.2	4,220	41.6	12.3	37.6	135	45.0	8.4	59.2
Year 12	1667	1.87	17.2	0.05	0.74	93.7	44.0	5,070	44.9	12.3	38.0	151	45.0	8.7	59.8
Year 13	296	2.04	23.7	0.06	0.63	93.9	44.0	5,460	45.0	12.3	38.0	245	45.0	8.7	60.0
Total Average	**21,336**	**2.15**	**14.2**	**0.06**	**0.54**	**93.1**	**45.7**	**2,800**	**34.0**	**12.0**	**36.4**	**141**	**45.0**	**6.4**	**54.2**

18 Project Infrastructure

There are four primary areas on-site requiring infrastructure, namely, the underground mines, the processing plant, the tailings storage facility, and the general site area. Figure 18-1 provides a view of the conceptualized site plan of surface infrastructure, along with the underground workings for reference.

The underground infrastructure is covered in Section 16 while the processing plant is discussed in Section 17. The remaining surface infrastructure is summarized below.

18.1 Tailing Storage Facility

The conventional slurry tailings TSF design was developed in consultation with the study team to suit the project setting, regional precedent, and economics. The TSF location is approximately 3 km southwest of the proposed processing plant and was selected after a comparison of several options based on the embankment volume to storage capacity ratio and the TSF catchment area. The TSF was designed to accommodate 21.3 Mt of tailings over the life of the mine. At an assumed average settled dry density of 1.3 t/m³, the required tailings solids storage volume is 16.2 Mm³ plus water management storage.

18.2 TSF Design Criteria

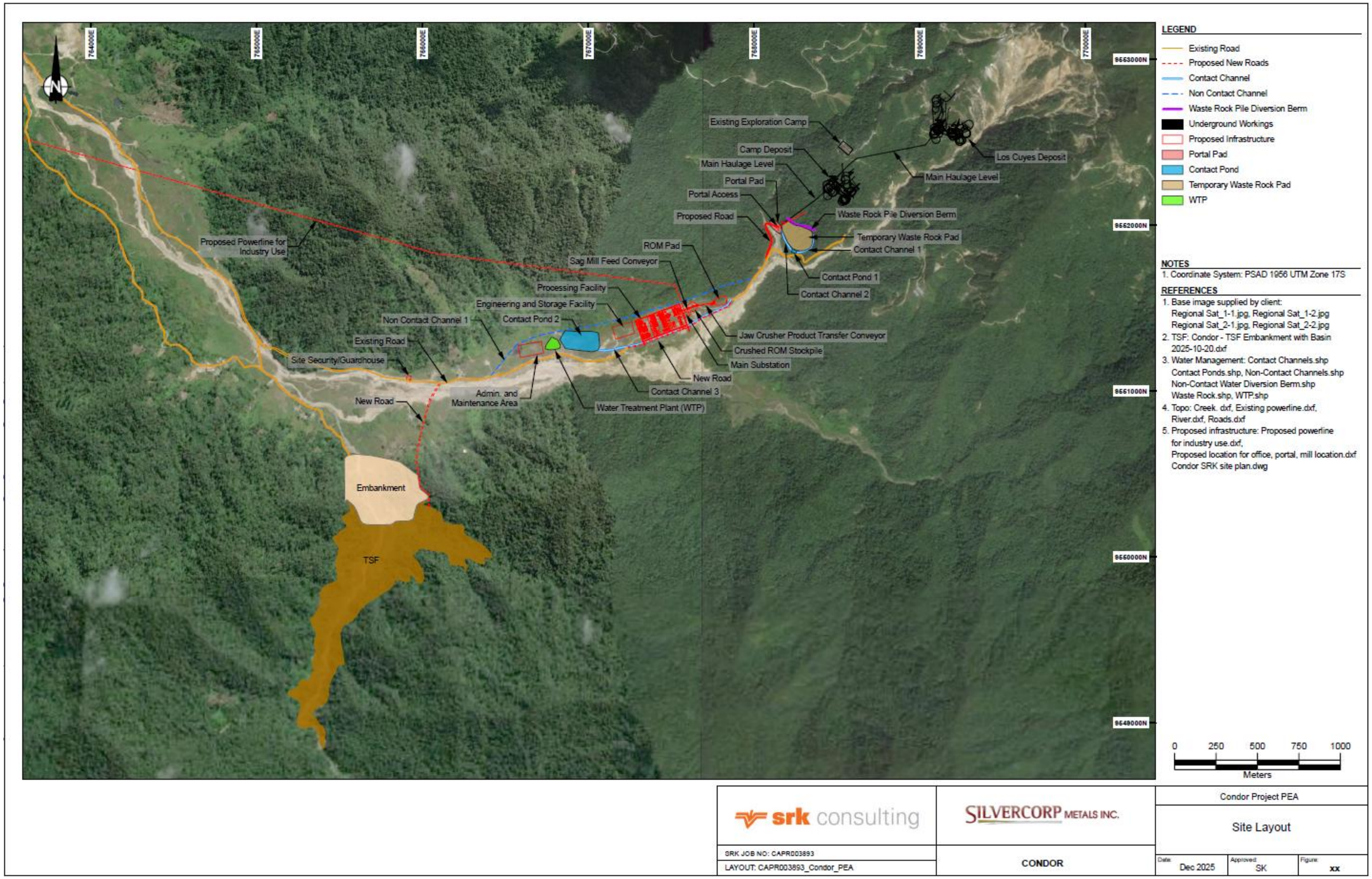
The geotechnical embankment design concept was prepared with reference to:

- Canadian Dam Association (CDA) Dam Safety Guidelines 2007 (2013 Edition) (CDA, 2013)
- CDA Technical Bulletin: Application of Dam Safety Guidelines to Mining Dams (CDA, 2019)
- Global Industry Standard on Tailings Management (GISTM) (GISTM, 2020)

Tailings will be retained by a zoned cross-valley embankment constructed of rockfill, compacted saprolite, and a granular filter. Embankment seepage will be mitigated by an LLDPE geomembrane liner on the upstream slope and a low permeability compacted saprolite layer in the embankment. The LLDPE geomembrane liner will extend 100 m upstream into the TSF basin to mitigate foundation seepage. A 3 m thick granular filter zone will be constructed between the rockfill and compacted saprolite to mitigate fines migration through the embankment.

The embankment geometry will consist of a 15 m wide crest, a 2.75H:1V downstream slope, and a 2H:1V upstream slope. The 15 m crest width was selected to allow room for tailings distribution and water reclaim pipelines as well as traffic and safety berm requirements. At final elevation, the dam will be 540 m long at the crest and 84 m high from the crest to the lowest point of the downstream toe. This geometry will allow for the storage of tailings solids plus 2 m freeboard.

Figure 18-1: Site Plan



Seepage from the embankment will be captured in a seepage collection pond near the downstream toe of the embankment. The seepage collection pond will also intercept sediment runoff.

Surface water management will consist of a diversion ditch from the catchment area around the TSF and a spillway to manage excess water from significant storm events. The spillway will be constructed in bedrock at the embankment abutment for each embankment raise.

18.3 TSF Construction, Operation, and Closure

The zoned rockfill embankment will be constructed in stages using downstream construction methods over the mine life to suit tailings and water storage requirements. A 40 m high starter dam will be constructed to accommodate the first two years of production followed by five raises throughout the 13 year mine life. Each raise will have detailed design specifications prepared and will be constructed to accommodate the maximum tailings pond level required before the next raise.

Preparation of the starter dam embankment foundation and abutments will be required prior to embankment construction. Foundation preparation will consist of clearing and grubbing of the vegetation as well as scarification, moisture control, and compaction of the in-situ soils. Future raises will require the same foundation preparation as the embankment is expanded downstream. Rockfill, saprolite, and the granular filter construction materials will be sourced from borrow pits within 2 km of the embankment. The granular filter material will require additional screening to ensure the correct grain size criteria is achieved based on filter criteria for the saprolite and rockfill materials. Fill placement will occur in lifts of a predetermined thickness to achieve density requirements and quality assurance and quality control (QA/QC) construction monitoring will be on-site during lift placement. QA/QC activities will include in-situ density tests, moisture-density and grain size tests, and visual inspections of placement areas.

The TSF basin will be cleared of vegetation and grubbed prior to deposition. Tailings will be pumped as a conventional slurry to the TSF and deposited subaerially from a network of spigots on the embankment and basin perimeter. The deposition strategy will primarily focus on covering the geomembrane liner while maintaining a supernatant pond away from the embankment and developing a wide tailings beach. The deposition points will be accessed by an access track constructed around the perimeter of TSF basin area. Supernatant water in the TSF will be pumped back to the process plant for re-use via a reclaim water pump system.

At closure, a dry soil cover will be placed over the accessible tailings beach above water to mitigate erosion. Natural runoff from the catchment area will ingress into the pond and a closure spillway will be constructed to manage water from storm events. Water quality monitoring and treatment will proceed until discharge criteria are met.

18.4 Instrumentation Monitoring

Geotechnical and environmental instrumentation will be installed as part of the tailings management and monitoring system. The instrumentation monitoring program will be designed to measure and record key performance indicators including displacement, pore pressure, and seepage flow. The following geotechnical instrumentation will be incorporated in the facility:

- Surface survey monuments

- Vibrating Wire and Standpipe Piezometers
- Ground water sampling wells
- Change detection LiDAR monitoring
- V-notch weir flow monitoring devices at the outlets of the surface water diversion ditches and seepage locations

18.5 TSF Cost Estimate

The life of mine cost estimate for the TSF conceptual design was prepared based on the following assumptions:

- The entire footprint of the embankment and basin will be stripped and grubbed, with greater preparation effort invested in the embankment foundation than the tailing pond footprint.
- The embankment will be constructed of 90% rock fill and 10% compacted saprolite fill with a 3 m granular filter between the two zones.
- An LLDPE geomembrane liner will be installed at the upstream embankment slope that extends 100 m into the tailings storage basin.
- The design geometry requires 15 m wide crest, a 2.75H:1V downstream slope, and a 2H:1V upstream slope.
- Allow for storage of 16.2 Mm^3 tailings solids with a nominal 2 m freeboard during operational life.
- Starter embankment raised in stages and borrow material sourced within 2 km.
- Seepage collection pond and return pump system will be constructed at the toe of the embankment.
- Surface water diversion ditch and an access track will be constructed around the TSF basin perimeter.
- There will be operational and closure spillways to manage flows from significant storm events.
- QA/QC construction monitoring will be present during earthworks.
- An instrumentation monitoring system will be designed to measure and record key performance indicators.
- Closure will include staged construction of a nominal 1 m thick cover over the tailings beach.

The construction and closure capital cost estimate of the TSF conceptual design is included in Appendix B. Quantities were estimated using CAD and the digital terrain model provided by Silvercorp. Unit rates estimated based on experience on other projects and input from Silvercorp based on recent construction costs for other work in Ecuador. Operating costs are included with the process engineering cost estimate.

18.6 Electrical Supply and Distribution

18.6.1 Current Power Source

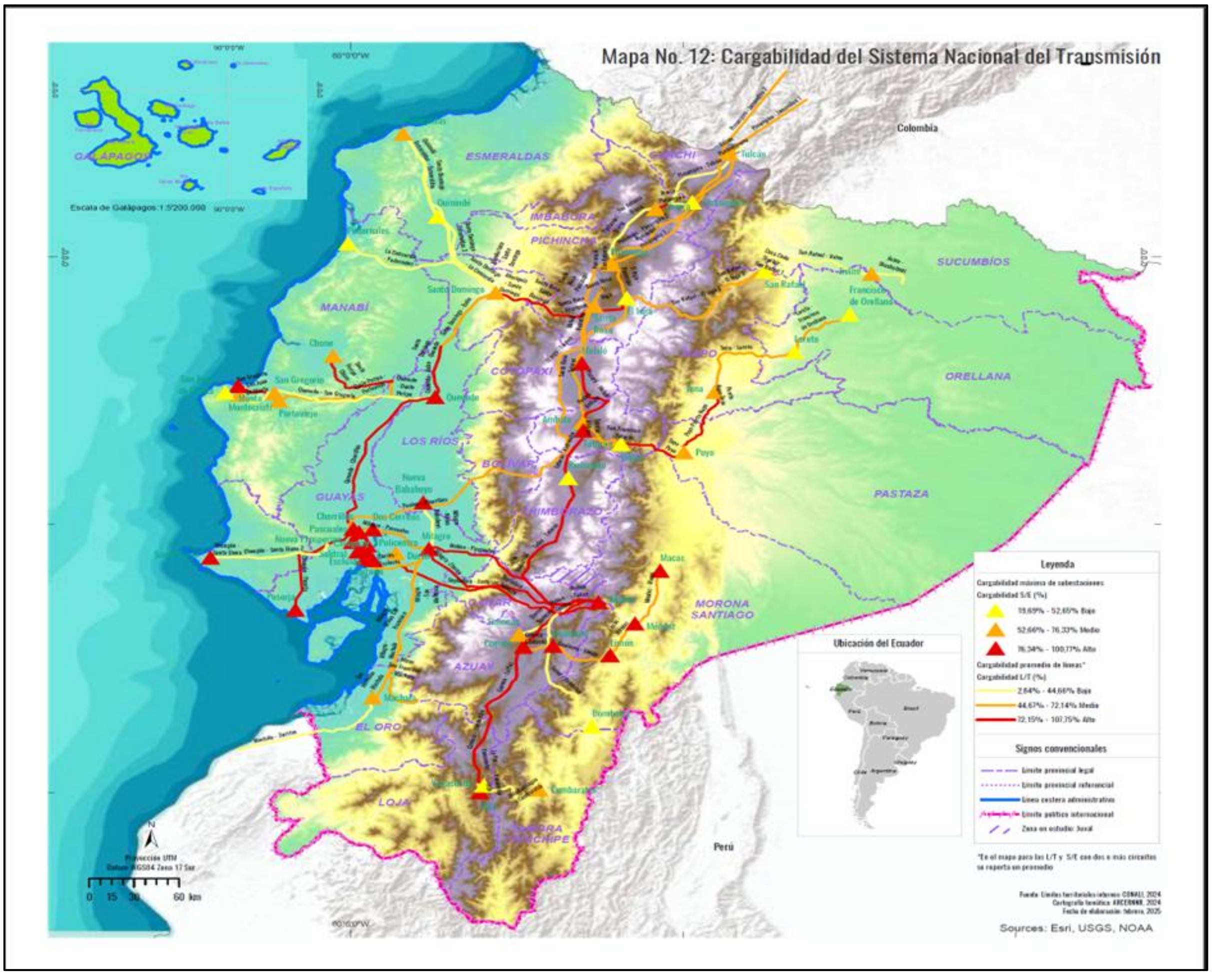
The nearest power source with enough capacity to supply the power demand of the project (18MW at 5000 TPD) is the Cumbaratza substation of 138/69 KV, which is part of the Ecuadorian SIN (National Interconnected System) and is located in the same Zamora Chinchipe province (Figure 18-2).

The owner of the Cumbaratza substation is CELEC EP (Corporación Eléctrica del Ecuador), a strategic public company in Ecuador responsible for the generation, transmission, marketing and export of electrical energy.

As per the ARCONEL (Electricity Regulation and Control Agency of Ecuador) last report of February 2025 the incoming 138 kV power line has currently a Load ability under 45%. Furthermore, the 180 MW Delsitanisagua hydroelectric plant is connected 20 km upstream from this substation, which will provide stability to the operation of the mills.

The existing power transformer 138/69 kV - 33.3 MVA is over 50% of its capacity and is not able to supply the mine demand. A new bay a substation of 138/69 kV will be required.

Figure 18-2: National Transmission System Capacity

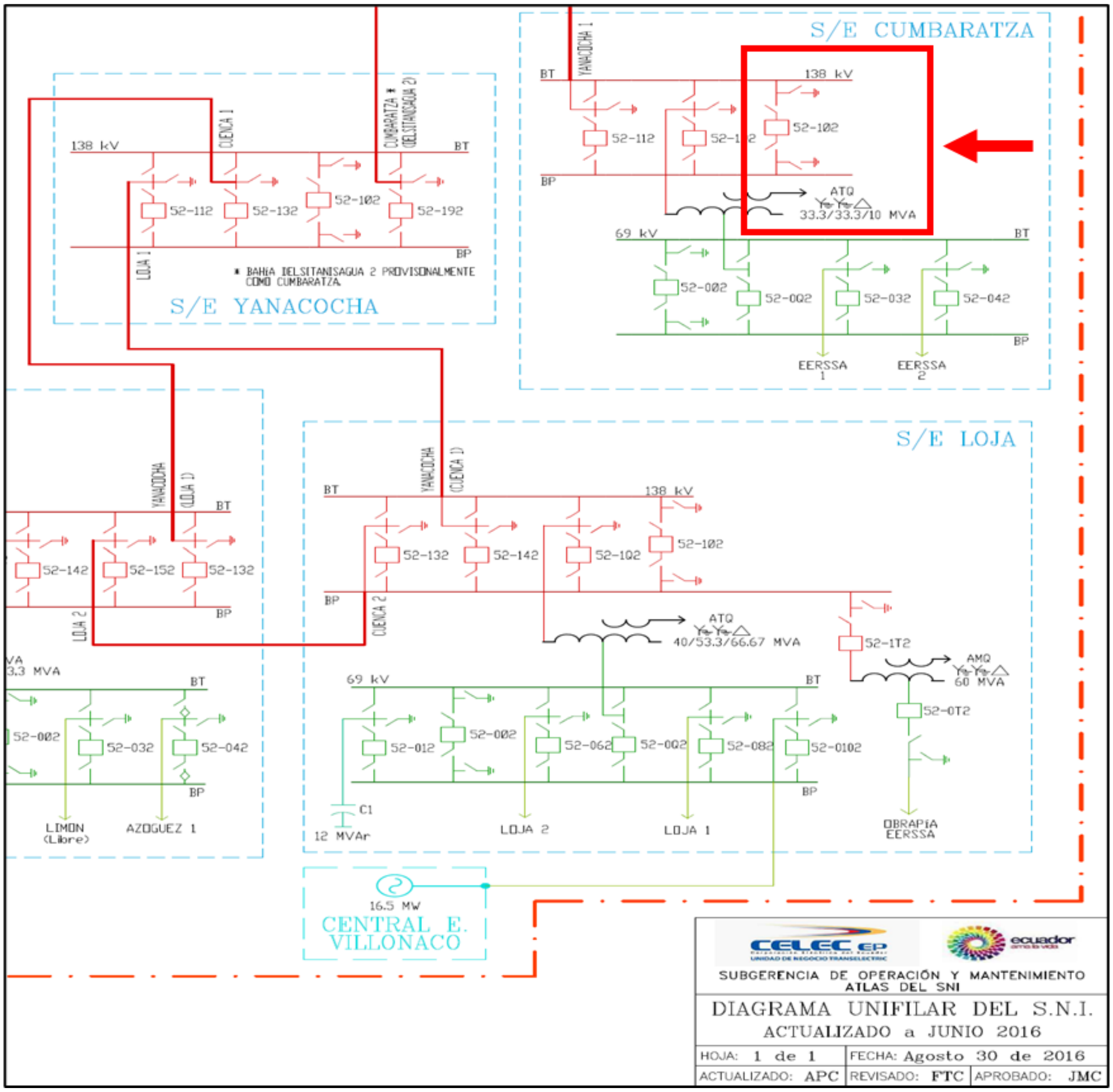


Source: ARCONEL Feb 2025

18.6.2 Aerial Power Line and Mine Site Substation

The Cumbaratza substation is a double-busbar 138kV substation and needs to be expanded with an additional bay to provide room for a substation 138/69 kV with a transformer of 25 MVA to supply power to the mine in 69 kV. This new bay must have a double disconnect switch with grounding, a circuit breaker, power transformer, instrument transformers, surge arresters, protection relays, grounding mesh and a dedicated control room with backup batteries. (Figure 18-3).

Figure 18-3: Annotated Cumbaratza Substation Diagram



Source: CELEC "Corporación Eléctrica del Ecuador", August 2016

The proposed overhead power transmission line, spanning 46 km from the Cumbaratza substation to the project site, will traverse varying terrain with elevations that range between 1,000 and 1,500 meters, inclusive of regions of dense vegetation. A single-circuit configuration is necessary, based on required capacity, with the conductor material specified as Aluminum Conductor Alloy Reinforced (ACAR) featuring a cross-sectional area of 500 MCM (253 mm²) to meet mechanical and electrical performance criteria.The line must incorporate an Optical Ground Wire (OPGW) to provide both lightning protection and integrated communication capabilities.

A critical consideration for the project timeline and budget is the acquisition of necessary permits and the negotiation of easements, as these processes are anticipated to represent the primary constraints and could significantly influence the overall economic feasibility and schedule of the construction phase.

The 69/13.8 incoming substation at the mine will have one 25 MVA oil transformers, without redundancy.

For the 13.8kV distribution system, a switchgear with 10 cubicles has been planned, each with its own dedicated circuit breaker for a specific component of the project. This ensures operational independence in case of electrical failures in individual circuits. For example, each cubicle will be dedicated to a specific process such as grinding, flotation, tailings, the mine Camp, the Los Cuyes mine, camps and services, etc.

18.6.3 Power Distribution to Camp and Los Cuyes Mines

For operational reliability, each mine will have an independent 13.8 kV feeder from the main switchgear. A short section will be cable, and the longer section will be overhead line to reach each mine portal. Near each portal, there will be a main 13.8/4.16 kV substation with its switchgear to initiate the 4.16 kV feeder cables up to the portable underground substations. These substations will be located at different levels according to the mining progress to serve loads like pumps, secondary fans, jumbos, etc.

Due to environmental and safety criteria, dry-type transformers are being considered for underground portable substations. These substations incorporate a 4.16kV incoming circuit breaker, pluggable outputs with 0.48kV protection, ground fault relays, a neutral resistor, and lighting transformers.

18.6.4 Power Distribution to Plant and Tailings Management Facilities (TMF)

The plant's feeders are 13.8kV cables that originate at the main switchgear and terminate at dry-type transformers within the plant. There is a single 13.8/4.16kV transformer to power the mill motors, which, due to their sizes, must operate at medium voltage. All other transformers are 13.8/0.48kV for the majority of low-voltage motors.

Two portable electrical rooms, or e-shelters, have been considered for the plant. One will house the switchgear and MCCs for Primary Crushing, Grinding, CIL & Cyanide Detox, Carbon Elution, Reactivation, and Electrowinning. The other will house the switchgear and MCCs for Flotation, Concentrate Dewatering, Process Water, and Compressed Water.

Dry transformers have been considered in the plant and tailings facilities, for environmental, safety and ease of installation criteria.

The feeder for Tailings Management Facilities (TMF) is also 13.8KV from the main switchgear, runs a short distance in cable and then in overhead line ending in a 13.8/0.48 KV dry transformer substation for reclaim water pumps, seepage and others.

18.7 Surface Water Management

18.7.1 Introduction

A conceptual level plan, design and arrangement for a water management system was developed for the project. The objectives for the Condor Project water management system are to:

- Ensure there is sufficient water for mineral processing
- Separate contact and non-contact water
- Collect and treat contact water prior to discharge to the environment

The project is located on the western edge of the Amazonian Basin. There is not a baseline climate or hydrologic characterization for the site. Site specific data were not provided. To develop hydrologic and climatic parameters for preparing a scoping level assessment of the water management system for the project, climate and hydrologic parameters from a nearby mine site (<50km distant) were used.

There are three sources of contact water that will need to be managed:

- Groundwater inflows to the underground mine
- Runoff and seepage from the waste rock pile
- Excess process water that accumulates in the TSF

18.7.2 Conceptual Level Site Water Balance

An annual water balance was prepared. For the purposes of developing a robust water management system the annual precipitation from a 1:20 wet year was used. The annual precipitation depth in a 1:20 wet year is approximately 4,200mm/year.

The TSF is estimated to gain water for a year with mean annual precipitation (approximately 3,400mm). Even during a 1:20 dry year, the TSF should still accumulate water, therefore meeting the water supply objective for the site is unlikely to be a risk.

The average daily flows for a 1:20 wet year were estimated for runoff from areas impacted by mining and processing and in the TSF (Table 18-1). Estimates of groundwater inflows to the underground workings are described in Section 15.2. Runoff from impacted areas using the rationale method. A runoff coefficient of 0.9 was conservatively used because rain is so frequent that the ground is likely to be saturated and most of the rainfall will runoff. The volume of excess water that would accumulate in the TSF was estimated. This scoping level annual water balance accounted for

- Tailings slurry water discharged to the facility
- Direct precipitation on entire footprint of the facility
- Evaporation from the entire footprint of the facility
- Water lost to burial in tailings voids

The volume of excess water was expressed as average daily rate for the 1:20 wet year.

Table 18-1: Summary of Average Daily Contact Water Flows for a 1:20 Wet Year

Source	Average Daily Flows (m^3/day)
Groundwater Inflows to the Underground Mine	1,000
Runoff from Areas Impacted by Mining or Processing	2,420
Excess Water Accumulating in the TSF	2,530
Total	**5,950**

Expressing the annual flow to a daily flow, the average daily flow rate in a 1:20 wet year is approximately 250 m^3/hr.

18.7.3 Geochemistry and Water Quality

There currently is no geochemical investigation describing the metal leaching and acid generating potential of any of the mine waste, overburden or quarry rock. For the purposes of this study, the mine waste (waste rock and tailings) and underground mine wall rock are assumed to be metal leaching and acid generating. It is recommended that a full geochemical investigation is undertaken at future stages of project development. For the purposes of this study the following assumptions regarding water chemistry were made:

- The tailings water has circum-neutral pH with elevated metals concentrations that require treatment.
- The underground mine water will be slightly acidic with elevated metal concentrations.
- Runoff and seepage from the waste rock pile will be acidic and have elevated metals concentrations.

Additionally, there are no background water quality data. The water course adjacent to the proposed location of the mine infrastructure appears to be heavily impacted by artisanal mining. Future studies should characterize the existing water quality of this water course.

18.7.4 Proposed Water Management System

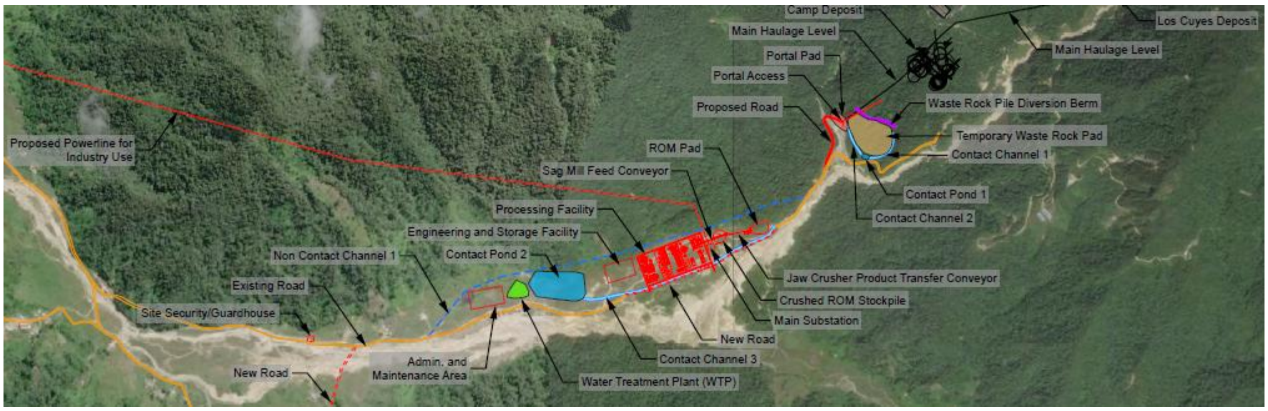
Figure 18-4 shows the site layout including water management infrastructure. Mine facilities and infrastructure are located to the north of a sediment choked watercourse. The sediment in the watercourse is from the erosion at the head of the watershed caused by artisanal mining activities.

Water management infrastructure includes:

- Contact Channels 1 and 2 – These channels collect runoff and seepage from the waste rock pile and divert it to Contact Water Pond 1.
- Waste Rock Pile Diversion Berm – This diversion berm diverts runoff from upgradient around the waste rock pile.
- Non-Contact Channel 1 – This channel intercepts non-contact runoff from upgradient of the mine infrastructure pad and discharges into the watercourse south of the site.
- Contact Pond 1 – This pond receives runoff and seepage from the waste rock pile near the portal that is collected by Contact Channels 1 and 2, runoff from the portal pad and groundwater inflows to the underground workings.
- Contact Channel 3 – This channel begins immediately upgradient of the ROM Pad and extends to Contact Pond 2. This channel collects contact water from mining and mineral processing infrastructure.
- Contact Pond 2 – Receives contact water from Contact Channel 3 and pumped water from Contact Pond 1. This pond acts as a storage and surge pond for influent to the Water Treatment Plant.
- Water Treatment Plant – The water treatment plant will draw influent from Contact Water Pond 2 and discharge treated water to the water course.

The tailings storage facility is located to the south of the watercourse. Water management around the tailings storage facility is described in Section 17.1.

Figure 18-4: Site Layout with Water Management System



Channels and ponds are designed to convey or contain the 1:100 year, 24 hour precipitation event. Local unit costs for construction were used when available. In the absence of local rates, rates for Canadian projects were used.

18.7.5 Conceptual Level Description of the Water Treatment Plant

The design objective of the water treatment plant is to neutralize acidity and precipitate and settle metals. It is expected that the discharge from the water treatment plant will need to meet Ecuadorian water chemistry limits for mining.

The capacity for the conceptual level water treatment plant design was 450 m3/hr. This differs from the average daily flow rates for a 1:20 wet year (i.e. 250 m3/hr). The capacity was increased to account shorter higher intensity shorter duration rain events. Once a site specific climate and hydrologic study has been completed, and the water management system (i.e. channels and ponds) are sized to route a more appropriate design storm through the system, the design capacity of the water treatment plant can be optimized.

The treatment process will be a lime neutralization high density sludge process. Major water treatment equipment will include:

- Lime silo
- Lime addition system
- Lime slurry makeup tank
- Lime slurry and recycled sludge mix tank
- Agitated influent holding tank
- Two agitated reactor tanks that can be aerated
- A flocculant addition system
- A clarifier

- A multi-media filter
- An agitated sludge recycle tank
- An effluent holding tank

Influent will be drawn from Contact Pond 2. Influent will be pumped to the first reactor tank. Combined lime slurry and recycled sludge will be added to the first reactor. The combined lime slurry recycled sludge dose will be metered by monitoring the pH in the second reactor tank. The pH set point will be based on testing to be conducted as part of future geochemistry and water chemistry prediction investigations. Precipitated metals will be settling from the water by a clarifier. Flocculant will be added to enhance settling.

Overflow from the clarifier may be directed to a multimedia filter if needed. The filtrate will be sent to a holding tank from where it will be discharged into the adjacent water course. The underflow from the clarifier will be split with a portion directed to a lime slurry-sludge mix tank and the other portion will be pumped to a drop box in the tailings line. The sludge will ultimately be deposited in the tailings storage facility.

The water treatment plant capital and operation and maintenance costs were based on a recent quote for a similar plant at another project in Ecuador provided Silvercorp.

18.7.6 Recommendations

The following studies should be completed as the project advances:

- A geochemical evaluation of the wasterock, tailings, quarry rock and existing water quality.
- A climatic and hydrologic investigation including installing a meteorologic station and hydrologic gauging stations.
- A more complete and detailed site wide water balance that more fulsomely integrates the management of excess water collected in the TSF with the site water management system.

18.8 Roadways

Access to the site is made using Via Chinapintza, which leads through the small town of Congüime located just west of Condor. Allowance has been made to upgrade portions of the roadway both internally and externally to the site to accommodate the heavy vehicle traffic necessary as part of the mining operations.

Internal roadways linking the primary infrastructure (e.g., mine portal, TSF, mill, stockpiles, administration and maintenance areas) shall be constructed. Parking areas will be sized and built to accommodate site personnel, contractors, and visitors. The primary parking area will be located in the office and administrative area, with smaller lots built adjacent to surface infrastructure to facilitate ease of use by light equipment.

18.9 Office and Administrative Area

In addition to the structures associated with the mill and TSF, several other permanent facilities will be constructed. These include an office building equipped with dry facilities, which will feature meeting and assembly spaces for mine operations and maintenance teams, offices for mine management and technical staff, as well as storage and maintenance areas for mine rescue equipment. The camp building, which shall include lodging and cafeteria facilities, will also be built in this area.

Additionally, a warehouse will be built, incorporating attached surface workshops to support the electrical and mobile maintenance teams. Additional laydown area for larger supplies will be located adjacent to the maintenance area.

19 Market Studies and Contracts

19.1 Market Study

The principal commodities produced at Condor are gold-silver dore and two saleable concentrates: silver-lead concentrate and zinc-silver concentrate. No specific market study has been commissioned because gold, silver, lead, and zinc are globally traded commodities. The commodity prices are sourced from an independent analyst, Consensus Market Forecast (CMF), and the projected outlook (in real USD) was issued by CMF in November 2025. The long-term consensus prices were used for economic analysis.

19.2 Product Specifications and Terms

The economic model applies industry-standard sales terms for gold-silver doré and saleable concentrates, based on comparable operations and published industry benchmarks. Table 19-1 summarises the sales terms used in the economic model.

Table 19-1: Sales Terms

Terms	Unit	TCRC Charge	Payabilities
Gold-Silver Dore			
Au Dore Refining Cost	$/oz	8.5	99.80%
Ag in Au Dore Refining Cost	$/oz	0.5	90.00%
Silver-Lead Concentrate			
Pb Conc			95.00%
Pb Conc TC	$/t	100	
Pb Conc Freight/Insurance	$/t	85	
Pb Conc Deduct	percentage	3.00%	
Au in Pb Conc			
Au in Pb Conc Deduct	g/t (dmt)	1	
Au in Pb Conc RC	US$/oz	8.5	
Payable Rate for Au in Pb Conc	%		95.0%
Ag in Pb Conc			96.50%
Ag in Pb Conc Deduct	g/t (dmt)	50	
Ag in Pb Conc RC	$/oz	0.5	
Payable Rate for Ag in Pb Conc			95.0%
Zinc-Silver Concentrate			
Zn Conc			85.00%
Zn Conc TC	$/t	175	
Zn Conc Freight/Insurance	$/t	85	
Zn Conc Deduct	percentage	8.00%	

Terms	Unit	TCRC Charge	Payabilities
Au in Zn Conc			
Au in Zn Conc Deduct	g/t (dmt)	1	
Au in Zn Conc RC	US$/oz	8.5	
Payable Rate for Au in Zn Conc	%		70.0%
Ag in Zn Conc			96.50%
Ag in Zn Conc Deduct	g/t (dmt)		
Ag in Zn Conc RC	$/oz	0.5	
Payable Rate for Ag in Zn Conc	%		70.0%

Additionally, a 5% royalty has been applied.

19.3 Contracts

At the PEA stage, no marketing contracts exist.

19.4 Conclusions

The Qualified Person is of the opinion that there are well-established markets for gold and the other commodities and that the lack of formal sales contracts at the PEA stage does not materially affect the economic viability of the Project.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environmental Regulatory Setting

The regulatory framework for mining in Ecuador is governed primarily by the Mining Law (2009) and recent reforms, such as those introduced by the Organic Law of Social Transparency (2025).

The 2008 Constitution establishes that Mineral Resources are the property of the State and are regulated by the Mining Regulation and Control Agency (ARCOM), which is responsible for monitoring and controlling mining activity.

In Ecuador, the environmental assessment process generally includes an environmental impact assessment (EIA), which begins with a baseline study to diagnose pre-existing conditions. Subsequently, the potential effects of the project on the environment are identified, predicted, and evaluated, considering biotic, abiotic, and sociocultural aspects. If the effects are significant, mitigation measures and an environmental management plan are designed and compiled in an Environmental Impact Statement (EIS). The EIS is reviewed by the Ministry of Environment and Energy through the Single Environmental Information System (SUIA).

The authorities review the EIA. If they deem it acceptable, an environmental license is granted, allowing the project to proceed under the established conditions.

An estimation of the length of time the EIA process requires is 36 months, this includes: the collection baseline data (14 months), preparation and elaboration of reports (10 months), review period (12 months).

The time required to obtain mining permits in Ecuador varies, but a process for a mining concession can take between three and four months, or up to a year due to administrative delays. The process includes applications, technical and legal reports, and newspaper publications. The main delays are due to inter-institutional consultations; responses from other government entities may take longer. The citizen participation process for environmental consultation (PPC-CA) which follows the approval process of the EIS and involves seeking approval or consent from local communities. This process generally requires approximately three months to complete. Assuming this process is successful the Ministry can issue the environmental permit.

20.2 Environmental Assessment Requirements of the Project

The Project currently holds all necessary environmental permits for the advanced exploration phase and complies with all applicable legal and regulatory obligations.

In 2015, the Condor project obtained EIA certification, granting them an environmental license for advanced exploration (Resolution 267, dated April 22nd, 2013, MAATE).

A new EIS report was submitted to the Ministry of Environment in March 2025, to obtain a new environmental permit for exploitation (the current one is for exploration only), under the regime of Small Mining. With this new environmental permit, underground development can occur to provide access for underground resource definition drill programs. Currently, the report has been reviewed and approved by various functional departments of the ministry, with a final statement of approval to be announced by the regularization directory of the ministry. Once the approval of the EIS is announced, the PPC process can be initiated to get the approval or consent of the local communities. Silvercorp project team has been working together with the local communities to get their social consent. Once the PPC is completed and assuming it is in favour of the proposed project, the ministry can issue the new environmental permit.

The results of the EIS review was expected in November 2025. However, the announcement has been delayed due to the restructuring of government ministries. An announcement is expected prior to the end of 2025.

An administrative resolution from the Ministry of Mines (MINEM) has categorized the project concessions as small-scale mining regime, this provides benefits to the project for legal & administrative processes.

A framework for the permitting process of the project is shown in Table 20-1.

Table 20-1: Framework for Permitting Process for Exploitation, Underground Mine

Legal Compliance According to Mining Scale								
						Mining Scale Condor Project Case		
N.	**Legal Framework**	**Date of Publication**	**Article**	**Description**	**Responsible Authority**	**Small UG < 300 t/d**	**Medium UG: 301 – 1,000 t/d**	**Large UG > 1,001 t/d**
1	Constitution of the Republic of Ecuador	Official Registry No. 449 October 20, 2008	Art. 57 Prior, free and informed consultation	**Prior Consultation** As long as the operative area intersects within the Indigenous people and communities. It applies in any mining scale.	MAE - Mines	N/A	Apply	Apply
2	Executive Decree N. 754 CRE Art. 398	Amendment of RCODA. 51-23-IN/23 November 9, 2023.	Art. 462 Citizen Participation Process for the Environmental Consultation	**Environmental Consultation** CPP - Applies for all mining scale projects in any stage	MAE - Env.	Apply	Apply	Apply
3	Mining Law	Official Register No. 602 December 21, 2021	Art. 26. Prior administrative acts before starting any mining activity	a) Environmental License	MAE - Env.	Apply	Apply	Apply
				b) Certificate of Non-Affectation to water sources	MAE - Water	Apply	Apply	Apply
				c) Water Permits: Industrial and Human Consumption	MAE - Water	Apply	Apply	Apply
4	Environmental Regulations for Mining Activities	Official Register No. 507 June 12, 2019	Art. 83 Background values above the standard	Register of water, soil, and air monitoring values that exceed the limits established by the standard	MAE - Env.	Apply	Apply	Apply
			Art. 93 *Waste Rock Facility* site preparation	It´s not a permit or authorization, the company needs to identify the location inside the EIS	MAE - Mines	Apply	Apply	Apply
5	Regulation of the Organic Environmental Code (RCODA)	Official Register No. 171 October 18, 2022	Ministerial Agreement N. 26	Use and management of chemical substances and hazardous waste	MAE - Env.	Apply	Apply	Apply
6	Agreement N. MERNNR-2020-0043-AM	July 15, 2020	Art. 32 Application for review and approval of the *Tailings Facility Design*	Technical Facilities for the installation of tailings dams	MAE & ARCOM	NA	Apply	Apply
7	Ministerial Agreement N. 18	Official Register 554 July 29, 2015	Art. 3 Requirements for the Authorization of *Process Plant* for beneficiation, smelting and refining.	Requirements for granting a permit to install a processing plant	MAE & ARCOM	Apply	Apply	Apply
8	Ministerial Agreement N. 145. Ministry of National Defense	April 14, 2023	Art. 32-33 Authorization of Explosives consumer	Purchase, transportation and use of explosives	FFAA	Apply	Apply	Apply

Source: Silvercorp, 2025

Notes: Glossary: **MAE** – Ministry of Energy and Environment, **SENAGUA** – National Water Secretariate of MAATE, **ARCOM** – Mining Regulation and Control Agency, **FFAA –** Armed Forces of Ecuador, **CPP** – Citizen Participation Process, **SUIA** – Unique Environmental Information System, **EIS** – Environmental Impact Study, **CODA** – Organic Code of the Environment, **RAAM** – Environmental Regulations for Mining Activities.

20.2.1 Environmental Considerations

The project is located in southern Ecuador, in the Province of Zamora Chinchipe, near the Ecuador – Peru border and the southern end of the Cordillera del Condor, approximately 400 kilometers southeast of Quito, 149 kilometers east of the city of Loja, and 76 kilometers southeast of the canton of Zamora.

Physiography of the area is steep terrain drained by several seasonally energetic streams. It is in the Congüime River sub-basin, which flows to the Nangaritza River, a main tributary of the Zamora River.

The Project is surrounded by secondary tropical forest, which has been heavily impacted by illegal mining and other intrusive anthropic activities for at least the last 30-40 years. The climate in the Project area is highland tropical, with high rainfall and a distinct annual rainy season.

Concession areas are dominated by naturally mineralized soils with high background metals concentrations that are considered unsuitable for agriculture.

Stream water quality sampling upgradient of Project exploration areas indicate generally acidic conditions, with pH values below the effluent limits established by Ecuadorian discharge regulations.

Other transitory contaminants have been observed that are likely due to anthropic influences, including human habitation and sporadic illegal mineral processing.

The nearest biological reserve established under Ecuador's national system of protected areas (Sistema Nacional de Areas Protegidas or SNAP) is the Podocarpus National Park, about 20 km from the eastern boundary of the Project's environmental area of influence (AOI).

In this stage of the Condor project, details of design and mine lay out are under preparation. However, as stated in Chapter 16, the project will be a conventional underground mine accessed via ramp. The project will require the construction of several waste management facilities, such as a Tailings Management Area, and waste and ore stockpiles, as well as contact and non-contact water management provisions (Chapter 18).

In support of the project's EIS and in accordance with Ecuador's Ministry of Environment and Energy legislation and regulations, Silvercorp has initiated and/or completed a variety of baseline studies. These studies include:

- Meteorological studies
- Biodiversity studies
- Vegetation studies
- Hydrological studies
- Biological studies

Environmental studies for current exploration activities were completed as part of the exploration licensing process.

20.3 Social Considerations

Morona Santiago and Zamora Chinchipe, are both provinces in southeastern Ecuador which are rich in Mineral Resources, making them focal points for both legal and illegal mining activities. This has led to significant security challenges, environmental degradation, social unrest, and conflicts involving indigenous communities.

The Cordillera del Condor Mountain range, spanning both provinces, is an area of high biodiversity and ancestral territory of the Shuar indigenous people. Large-scale mining projects in this ecologically sensitive zone have led to disputes over land rights and environmental concerns.

Silvercorp is aware of the Shuar community concerns about mining, their approach to address these concerns is noted below under Social Risks.

In response to the challenges posed by illegal mining, Ecuador has intensified military operations in affected provinces, including Morona Santiago and Zamora Chinchipe. These efforts aim to dismantle clandestine mining camps and seize equipment used in illegal activities in the support of legal activities such as the proposed Condor Project.

20.4 Areas of Direct Influence of the Project Concessions (AODI)

It should be noted that the populations of both cantons are directly and indirectly linked to informal mining, engaging in minimal agricultural activities for family subsistence.

The Indigenous and mestizo communities included in the cantons, parishes, and neighborhoods are:

- Paquisha Canton (district):
  - Nuevo Quito Parish
  - Indigenous communities: Shuar communities of San Luis, San Francisco de Ikiam, and Conguime.
  - Mestizo communities: Puerto Minero and Chinapintza are considered areas of direct influence, while La Herradura, La Pangui, and Conguime Alto are areas of indirect influence.
- Nangaritza Canton (district):
  - Nankais Parish
  - Native communities: Naichap (Wankuis), Centro Shuar Tsarunts with its neighborhoods (San Andrés, San Manuel, Nuevo Amanecer, Santa Elena), Los Achos, Warints – Diamantes, San Pedro, and Pachikutza (Pachkius).

20.4.1 Social risks

The main social risk for the project is the strong presence of informal and illegal mining within and near to the concession. In response to this, Silvercorp has been developing engagement strategies to help address this risk and achieve the projected outcomes. Properly enforced Ecuadorian regulations by government officials can support this effort.

This risk, Illegal and informal mining, is currently active and at risk of expansion. This is in part due to direct and effective payments made to Shuar communities and the use of their labor by the illegal operations.

Silvercorp has proposed two strategies for two distinct groups: one for informal and illegal artisanal miners and one for, the indigenous communities.

The Informal and illegal artisanal miners, which have been established in the area since the 1980s, claim ownership over the territories they exploit. Suggesting any negotiation will be highly complex, as the profits and interests involved are significant.

Silvercorp seeks to recognize these miners and grant them titles as small-scale miners to ensure formalization and production within their concession areas. However, this must be accompanied by a truthful, timely, and transparent communication strategy to support decision-making and prevent misinformation and conflicts. This recognition creates a contingency zone against the expansion of informal and illegal mining.

The Indigenous communities have been exposed to formal, informal, and illegal mining—from artisanal to small and medium scale. These communities are also arguably neglected by the Ecuadorian State, which has led them, due to their needs and subsistence conditions, to find alternate sources of revenue such as payments from informal mining operations for access to their territories and for hiring localsas laborers. The community members are also aware that illegal mining severely degrades the environment.

Silvercorp is developing social programs focusing on training for labor inclusion in mining activities and promoting entrepreneurship or self-employment that generates a local economy. Social programs of this nature will support Silvercorp's efforts to obtain a social license from these communities in support of the Condor project. Silvercorp is aware social investment of this nature is key to the sustainability of both the indigenous communities and the Condor project.

In addition, other social risks must be considered. These include concerns with deforestation and potential environmental impacts to the rivers and streams.

A significant portion of the mestizo mining communities in the social influence area (AOI) of La Pangui and Chinapintza communities, especially the Indigenous ones, are beginning to perceive the negative impacts of illegal mining, such as deforestation, the loss of rivers and streams, and river flooding during the rainy season. It will be important that Silvercorp develops appropriate management plans to address these potential impacts and ensures past activities are not inappropriately assigned to the Condor project.

Silvercorp will need to develop an effective employment strategy in order to meet the expected Ecuadorian employment percentages as per Ecuadorian law.

As the proposed project moves into the next phase of engineering studies a robust Community Engagement Plan will be developed to help address the social risks.

20.5 Closure

In this stage of the Condor mine project, details of design and mine lay out are under preparation, closure of current exploration activities is included in the exploration license permit.

A brief discussion and summary of current regulations are shown in the following paragraphs.

In Ecuador, mine closure is not governed by a specific law, but rather falls under the general framework of the Mining Law (2009) (last amended August 21, 2018) and the Environmental Regulations for Mining Activities (RAAM), currently administered by the Ministry of Environment, Water and Ecological Transition (MAATE). Both instruments establish the obligations that mining concession holders must fulfill to ensure responsible, environmentally sound, and financially sound closure.

The Mining Law recognizes mine closure as a further stage in the mining activity lifecycle. Therefore, it requires mining operators to develop a Mine Closure Plan that includes the dismantling of facilities, the rehabilitation of affected areas, and the prevention of subsequent environmental risks. This plan must be approved by the Ministry of Environment and Spatial Planning (MAATE) and reviewed periodically, incorporating implementation costs into the annual environmental management programs.

The Environmental Regulations for Mining Activities outline the technical and administrative procedures for implementing these obligations. While their provisions cover all phases of mining activity—from exploration to abandonment—in practice they constitute the operational framework for closure, as they regulate aspects such as physical and geochemical stabilization, revegetation, environmental monitoring, and compliance verification. These regulations were updated in 2025 by Ministerial Agreement MAATE-MAATE-2025-0045-A, which introduced a new fundamental requirement: the obligation to establish a financial guarantee for the liabilities arising from environmental closure.

The main obligations established in the regulations for the closure phase are presented below:

1. Preparation and approval of the Closure Plan: Every mining company must submit a technical, budgeted, and verifiable closure plan to the Ministry of Environment and Territorial Planning (MAATE). This plan must include timelines, objectives, and environmental rehabilitation activities. It must be approved by the environmental authority and updated in the event of any changes in project conditions or execution costs.
2. Implementation of remediation and post-closure measures: During and after closure, the mining company remains responsible for preventing, mitigating, and repairing environmental impacts. This includes dismantling facilities, stabilizing tailings and waste rock, controlling acid drainage, and implementing revegetation and long-term monitoring programs.
3. Environmental Declaration: The MAATE issues a formal declaration on mine closure and abandonment plans after reviewing the submitted documentation and verifying compliance with technical and environmental requirements.
4. Mandatory Financial Guarantee: Agreement MAATE-MAATE-2025-0045-A consolidates one of the most important requirements: the submission of a specific financial guarantee for mine closure, ensuring the availability of resources to implement remediation measures, even if the

operator ceases operations. This guarantee must be submitted to MAATE every six months and is calculated based on the estimated actual closure costs. Its purpose is to ensure that environmental obligations are not neglected and that restoration is carried out effectively and verifiably.

5. Audits and Monitoring: Both the Law and the Regulations require periodic updating of the closure plan and associated costs, as well as the performance of environmental audits to verify compliance with the measures and the consistency between financial planning and its actual implementation.

21 Capital and Operating Costs

21.1 Introduction

This section includes the results of a conceptual capital and operating cost estimation which follows the requirements of a PEA level of study and is based on Indicated and Inferred Mineral Resources, as there is not sufficient confidence to declare Mineral Reserves. The PEA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.The estimates presented in this PEA are based on underground mining of the Camp and Los Cuyes deposits, and the construction and operation of a processing plant, tailings storage facility, and supporting on- and off-site infrastructure.

Two categories of expenditures were estimated for the Condor Project's LOM period, namely:

- Capital Expenditures
- Operating Expenditures

These are detailed in sections 21.2 and 21.3, respectively. These costs were incorporated into the economic analysis discussed in Section 22.

21.2 Capital Expenditures

21.2.1 Summary

Capital expenditures are defined as costs associated with major equipment and facilities, whereas operating expenditures relate to the resources required to support ongoing production activities. Within the scope of this study, Project expenditures are incurred during three distinct phases of the Project life:

1. the Pre-production Period
2. the Production Period
3. the Closure Period

The Pre-production Period follows the exploration stage of the mine life cycle. During this phase, expenditures are incurred before the mine achieves production in reasonable commercial quantities. These costs are considered capital in nature and are eligible for annual depreciation. For the Project, the Pre-production Period is comprised of Years -2 and Year -1. Any expenditures incurred prior to this period are classified as sunk costs and are excluded from this report and the associated financial analysis. During the Pre-production Period, underground mine development is initialized and significant portions of the surface infrastructure (including aspects related to the mill, TSF, electrical network, water management system, and civil works) are constructed.

Production in reasonable commercial quantities is defined by output levels rather than profitability. A mine is generally considered to have reached this milestone on the first day of the first month of a consecutive three-month period during which the processing plant operates at a minimum of 60% of its rated capacity, provided that the mine is the sole source of mineralized material for the processing plant.

The Production Period begins once the mine reaches a condition in which it can be operated and managed as an ongoing mining operation, rather than by a capital project team. As defined in this study, the Production Period continues for 13 years following the completion of the Initial Period. Capital expenditures during this time are largely associated with continuance of underground mine development, expansion of the TSF, and initialization of site remediation activities.

The final phase is the Closure Period, which is solely comprised of TSF-related costs incurred once mining has concluded. The Closure Period runs for 5 years, resulting in an overall project life of 20 years. All costs incurred during the Pre-production Period are expensed as capital, with the costs in the Production and Closure periods (Years 1 through 18) classified either as operating costs or sustaining capital, depending on their nature.

The estimate of capital expenditures was developed in Q4 2025 using a combination of supplier quotes, internal databases, and benchmarks from similar operations. The estimate has been prepared to fall within the Association for the Advancement of Cost Engineering (AACE) International Class 5 level of estimate. All currency units are presented in U.S. dollars (US$ or USD) unless otherwise noted. An average contingency of 20% has been to applied to all items during both the Initial and Sustaining periods.

It is estimated that the Initial Capital amounts to US$292M with the Sustaining Capital estimated at US$382M, for a total capital expenditure over the LOM period of US$674M. The capital expenditure is summarized in Table 21-1.

Table 21-1: Summary of Capital Expenditures

Category	UOM	Initial Capital	Sustaining Capital	Total LOM Capital
Mine	US$M	71	238	309
Mill	US$M	118	6	124
TSF	US$M	18	74	92
Water Management	US$M	4	0	4
Other On- & Off-site Infrastructure	US$M	70	1	71
G&A	US$M	10	0	10
Other Sustaining	US$M	0	21	21
Closure	US$M	0	42	42
Total Capital Expenditures	**US$M**	**292**	**382**	**674**

21.2.2 Mining

The underground mining capital costs are predominantly related to the development of underground excavations to facilitate the extraction of ore. This development, a combination of lateral, ramp, and vertical development, is envisioned to be executed by a contractor. The estimated unit costs are sourced from a mining contractor with regional expertise and have been based on the planned drift dimensions and expected ground conditions. It is assumed that the same contractor will carry out the mining activities during the Production Period.

During the Pre-production Period, the mining costs capitalized will include some expenditures that are expensed as operating costs during the Production Period (such as development within mineralized material). Once the Production Period begins this will revert to only included true capital expenditures.

The mining capital totals US$309 million over the LOM and is summarized in Table 21-2.

Table 21-2: Summary of Capital Expenditures - Mining

Category	UOM	Initial Capital	Sustaining Capital	Total LOM Capital
Underground Development	US$k	42,286	170,900	213,186
Underground Infrastructure	US$k	8,425	66,770	75,195
Capitalized Operating Costs	US$k	20,614	0	20,614
Total Capital Expenditures	**US$k**	**71,325**	**237,670**	**308,995**

Note: Values may not sum due to rounding.

21.2.3 Processing Plant

Total plant and related infrastructure capital cost was completed by Tetra Tech, and is estimated at US$124.2 million over the LOM, with $6.0 million of that amount apportioned to sustaining capital. The cost breakdown is provided in Table 21-3.

Table 21-3: Plant Capital Cost, 000'US$

Major Description	Total Cost	Contingency Cost	Total Cost with Contingency
Primary Crushing	3,702	740	4,442
General	4,772	954	5,726
Grinding and Gravity Concentration	22,234	4,447	26,680
CIL and Cyanide Detox	7,364	1,473	8,837
Carbon Handling, Gold Elution and EW	7,312	1,462	8,774
Lead and Zinc Flotation	9,949	1,990	11,939
Product Concentrate Dewatering	2,657	531	3,188
Process Water Distribution	2,088	418	2,506
Compress Air Distribution	3,487	697	4,184
Tailings Delivery and Reclaim Water Pumping	1,334	267	1,600
Reagents	3,823	765	4,588
Total Initial Project Directs	**68,721**	**13,744**	**82,465**
Project Indirects	26,239	5,248	31,487
Provisions	3,536	707	4,243
Total Initial Capital Cost	**98,496**	**19,699**	**118,196**
Sustaining Capital	5,000	1,000	6,000
Total LOM Capital Cost	**103,496**	**20,669**	**124,195**

Source: Tetra Tech, 2025

Note: Values may not sum due to rounding

Major mechanical costs were based on the proposed equipment list provided by Tetra Tech and equipment budget quotations provided by qualified process equipment manufacturers in China and Tetra Tech's database. All equipment and material costs included as free-on-board marine (FOB) manufacturer plants are exclusive of spare parts, taxes, freight, and packaging which, if appropriate, are covered in the indirect cost section of the estimate. General area costs include plant site civil works, including plant site pad preparation and roads.

Class of Estimate

This Class 5 cost estimate has been prepared in accordance with the standards of AACE International. The expected accuracy of this estimate is within ±35%.

Estimate Base Date and Validity Period

This estimate was prepared with a base date of Q4 2025 and does not include any escalation beyond this date. Vendor quotations used for this PEA estimate were obtained in Q4 2025 and have a validity period of 90 calendar days or less.

Estimate Approach

Currency and Foreign Exchange

The capital cost estimate uses US Dollars (USD or US$) as the base currency. Where applicable, quotations received from vendors were converted to US dollars using a currency exchange rate of CAD1.00:USD0.72. The equipment budgetary prices provided by Chinese manufacturers are in USD, which were converted to US dollar at an exchange rate of RMB1.00:USD0.14. There are no provisions for foreign exchange fluctuations.

Duties and Taxes

Duties and taxes are not included in the estimate.

Measurement System

The International System of Units (SI) is used in this estimate.

Work Breakdown Structure

The estimate is organized according to the following hierarchical work breakdown structure (WBS):

- Level 1 = Major Area
- Level 2 = Area
- Level 3 = Sub-Area

Elements of Cost

This capital cost estimate consists of the four main parts: direct costs, indirect costs, Owner's costs, and contingency.

Direct Costs

The costs are directly attributable to equipment performance and are necessary for its completion. In construction, direct costs are considered to be the cost of installed equipment, material, labour, and supervision directly or immediately involved in the physical construction of the permanent facility.

Examples of the direct costs include processing equipment, related piping and electrical systems, process control system, equipment foundations, operation platforms, and permanent buildings that are based on preliminary workshop and overall plant layouts.

The total direct cost for the Project is estimated to be $68.7 million.

Indirect Costs

In construction, field indirects are the costs that do not become a final part of the installation, but which are required for the orderly completion of the installation and include, but are not limited to, EPCM (engineering, procurement, and construction management), expats, camp and catering, worker construction equipment and supports, start-up costs, vendor assistances, initial fills (allowances for reagents, steel balls, lubricants, fuel).

The total indirect cost for the Project is estimated to be $26.2 million.

Owner's Costs

Owner's costs are the costs related to owner team to support and execute the Project. The Project execution strategy, particularly for construction management, involves the Owner working with engineering, EPCM organization(s) and supervising the general contractor(s). The Owner's costs include home office staffing, home office travel, home office general expenses, field staffing, field travel, general field expenses, community relations, and Owner's contingency.

The total Owner's cost allowance for the Project is estimated to be US$3.5M.

Contingency

Tetra Tech estimated a contingency for each activity or discipline based on the level of engineering effort as well as experience on past projects. The average contingency for the Project is 20% of the total direct and indirect costs resulting in a total contingency allowance of $19.7 million.

Sustaining Capital Cost

The sustaining capital costs are all required from Year 1 of operations to sustain the processing operation for the LOM and are estimated to be $6 million for the project.

21.2.4 Tailings Storage Facility

The LOM capital cost estimate for the TSF conceptual design is estimated to be $91.8 million. The total initial capital cost is estimated to be $18.1 million, including $0.5 million for owner costs and $2.9 million for contingency. The overall LOM sustaining capital cost for the tailings storage facility is estimated to be $73.7 million, including $2.1 million for owner costs and $11.9 million for contingency. The sustaining capital cost also includes $2.9 million for closure earth works.

The initial and sustaining capital cost breakdown is summarized in Table 21-4.

Table 21-4: Tailings Management Capital Cost, 000'US$

Major Description	Initial Cost	Sustaining Cost	Total Cost
Startup/Overhead	2,164	8,775	10,939
Earth Works	10,820	40,986	51,806
TSF Capital Items	29	371	400
Closure Earth Works		2,888	2,888
Design, Construction QC/QA	1,627	6,637	8,264
Total Project Directs	**14,640**	**59,658**	**74,297**
Owner's Cost	512	2,088	2,600
Contingency	2,928	11,932	14,859
Overall Total Cost	**18,080**	**73,677**	**91,757**

Source: Tetra Tech 2025

Note:Values may not sum due to rounding.

The cost estimates were prepared based on the following assumptions:

- The entire footprint of the embankment and basin will be stripped and grubbed, with greater preparation effort invested in the embankment foundation than the tailing pond footprint.
- The embankment will be constructed of 90% rock fill and 10% compacted saprolite fill with a 3 m granular filter between the two zones.
- An LLDPE geomembrane liner will be installed at the upstream embankment slope that extends 100 m into the tailings storage basin.
- The design geometry requires 15 m wide crest, a 2.75H:1V downstream slope, and a 2H:1V upstream slope.
- Allow for storage of 16.2 million m^3 tailings solids with a nominal 2 m freeboard during operational life.
- Starter embankment raised in stages and borrow material sourced within 2 km.
- Seepage collection pond and return pump system will be constructed at the toe of the embankment.
- Surface water diversion ditch and an access track will be constructed around the TSF basin perimeter.
- There will be operational and closure spillways to manage flows from significant storm events.
- QA/QC construction monitoring will be present during earthworks.
- An instrumentation monitoring system will be designed to measure and record key performance indicators.
- Closure will include staged construction of a nominal 1 m thick cover over the tailings beach.

Quantities were estimated using ACAD and the digital terrain model provided by Silvercorp. Unit rates estimated based on experience on other projects and input from Silvercorp based on recent construction costs for other work in Ecuador. Operating costs related to tailings delivery and reclaim water pumping are included with the process cost estimate.

21.2.5 Other Capital Expenditures

Additional capital is required over the LOM in order to facilitate the mining activities. Significant components in this category include:

- The water management and treatment system (e.g., channels, ponds, pumping station, and water treatment plant).
- Construction of supporting infrastructure, including the electrical supply and distribution system, water management facilities, equipment maintenance buildings and warehouses, the camp and administrative complex, and access roads.
- Pre-production General and Administrative (G&A) costs that have been expensed as capital.
- Sustaining capital for the maintenance of the aforementioned items.
- Site closure expenses.

The capital expenditure on these items totals $149 million over the LOM and is summarized in Table 21-5.

Table 21-5: Summary of Additional Capital Expenditures

Category	Unit	Initial Capital	Sustaining Capital	Total LOM Capital
Water Management	US$k	4,355	0	0
Other On- and Off-Site Infrastructure	US$k	69,739	1,224	70,963
G&A	US$k	10,200	0	10,200
Other Sustaining	US$k	0	21,336	21,336
Closure	US$k	0	42,000	42,000
Total Capital Expenditures	**US$k**	**84,294**	**64,560**	**148,854**

Notes: Values may not sum due to rounding.

21.2.6 Capital Cost Exclusions

The cost estimate presented herein is for information only and does not indicate the future capital cost estimate produced for subsequent studies.

The following items are not included in the capital cost estimate:

- Force majeure
- Schedule delays, such as those caused by:
  - Major scope changes
  - Unidentified ground conditions
  - Uncertainties in geotechnical or hydrogeological conditions
  - Labour disputes
  - Environmental permitting activities
  - Abnormally adverse weather conditions
- Schedule acceleration costs
- Cost of financing (including interests incurred during construction)

- Corporate expenses
- Receipt of information beyond the control of the EPCM contractors
- Salvage value for assets only used during construction
- Taxes (PST, GST, HST, and VAT) and duties
- Land acquisition if required
- Project sunk costs (exploration programs, etc.)
- Vendor price fixing/gouging
- Macroeconomic factors
- Currency fluctuations
- Geopolitical tensions or war
- Disruptions of global supply and logistical services
- Pandemics or other natural disasters
- Royalties, which are included in financial analysis, or permitting costs, except as expressly defined
- Forward inflation
- Escalations beyond the effective date of this study

21.3 Operating Expenditure Estimate

21.3.1 Summary

The operating costs are comprised of all expenditures required to produce and deliver a saleable product to the customer during the Production Period, exclusive of the costs noted in Section 21.2. These include direct site operating costs such as mining, surface haulage, milling, site G&A, management of water and tailings, mining supervision, and conservation. Expenditures related to refining, freight, royalty payments, and profit sharing complete the items classified as operating costs.

The total operating cost for Condor is estimated to be $2037.7 million, with the costs on a total and unit basis detailed further in Table 21-6 .

Table 21-6: Total Site Operating Expenditures

Category	Total LOM ($M)	Total Unit Cost ($/t milled)
Underground Mining (incl. Surface Haulage)	875.0	41.01
Processing	391.7	18.36
Site G&A	288.0	13.50
Water Management	14.6	0.68
Mining Supervision Fees	17.4	0.82
Conservation Fees	1.8	0.08
Direct Site Operating Expenditures	**1,588.5**	**74.45**
Refining and Freight	53.6	2.51
Royalties	190.7	8.94
Profit Sharing (State and Employee)	204.9	9.60
Total Operating Expenditures	**2,037.7**	**95.51**

21.3.2 Mining

The mining cost assumes that this work will be carried out by a contract mining firm, and thus the estimate is based on a quotation sourced from a contractor with experience in the region. It is inclusive of personnel, equipment, and supply costs required to undertake underground mining and development activities. Table 21-7 summarizes the key components of the mining operating expenditures, which totals $875.0 million over the LOM period (an an average unit cost of $41.01 per milled tonne).

Table 21-7: Total Mining Operating Expenditures

Category	Total LOM ($M)	Total Unit Cost ($/t milled)
Development	172.9	8.10
Stope Production	390.8	18.32
Surface Haulage to Mill	20.9	0.98
Backfill	142.8	6.69
Mining and Haulage of Alluvial Gravel	17.3	0.81
Total Contractor Mining Expenditures	**744.7**	**34.90**
Owner Operating Expenses	130.3	6.11
Total Mining Operating Expenditures	**875.0**	**41.01**

21.3.3 Water Management

The water management costs are inclusive of those related to the movement and treatment of water on site, including the operation of the pumping stations and the water treatment plant. This is expected to total $14.6 million over the mine’s life, or $0.68/milled tonne.

21.3.4 Mining Supervision

These expenditures are the supervision and control fees that mining rights holders must pay to ARCOM, as of June 2025, to ensure regulatory compliance with the mining agency. The fees are determined by 1) the size of the operation, 2) the project phase, and 3) the amount of hectarge distrubed. During the Pre-production Period these costs are calculated using the rate for the Advanced Exploration phase for a Large-scale Mining operation; once the Production Peeriod has commenced the rate for the Exploitation phase will be utilized for the calculaiton. These fees are expected to total $17.4 million over the LOM, with the average amounting to $0.82/milled tonne.

21.3.5 Conservation Fees

The conservation fees for mining concessions are based on a percentage of a Unified Basic Salary per hectare mined. Similar to the mining supervision fees, these are partially determined by size of the mining operation and the project phase, resulting in varying percentages applied in the Pre-production and Production phases. These are expected to total $1.8 million over the mine’s life, or $0.08 per milled tonne.

21.3.6 Refining and Freight

These costs are inclusive of the treatment, refining, shipping, insurance, and marketing of doré and concentrate. This is expected to total $53.6 million over the mine's life, which amounts to a unit cost of $2.51/tonne milled.

21.3.7 Royalties

The total royalties payable to receiving parties are 5% of NSR. The royalties are expected to total $190.7 million over the LOM, equivalent to $8.94 per milled tonne.

21.3.8 Profit Sharing

According to Ecuadorian law, a large-scale mining concessionaire is responsible for sharing a portion of its profit with both its employees (3% of profits annually) and the state (12%). The profit sharing payments are expected to total $204.9 million over the LOM, at an average of $9.60/tonne milled.

21.3.9 Processing

The LOM processing operating cost is estimated to be $33.05 million per year, or $18.36/t processed for the 5,000 t/d or 1.825 Mt/a nominal operation rate, excluding the costs associated with off-site shipment. The main processing includes:

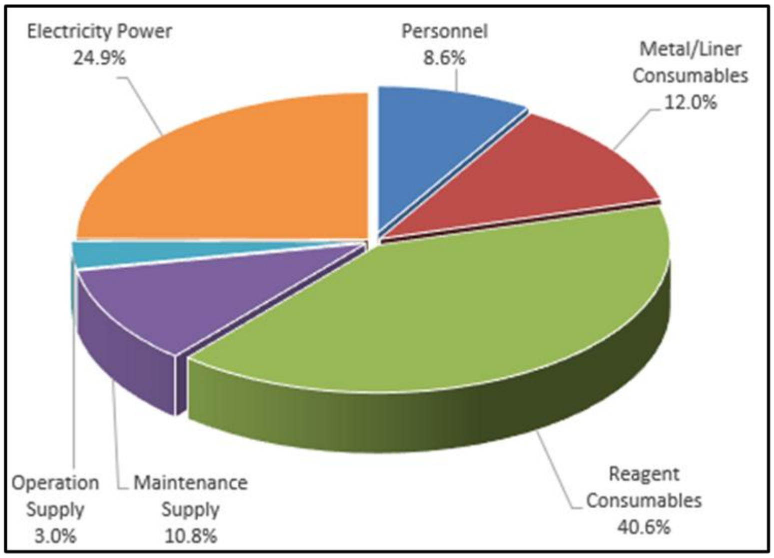
- ROM plant feed pad and crushing facility
- SAG and ball mill grinding with a gravity concentration to recover coarse free gold and cyanide intensive leaching
- Gold and silver recovery by cyanidation (CIP) and related elution and melting unit operation
- Residual cyanide detoxification
- Residual silver and lead and zinc recovery by differential flotation
- Tailings slurry delivery to TSF and reclaim water pumping back from TSF to process plant

The breakdown for the estimated process operating cost is summarized in Table 21-8 and Figure 21-1. The process operating cost presented in the table is slightly lower than the LOM average processing operating cost values. All the estimated costs are in US dollar (US$) fund, unless specified.

Table 21-8: Process Operating Cost Summary

Description	Unit Cost ($/t Proceed)
Personnel	1.51
Steel Balls/Mill Liner Consumables	2.09
Reagents	7.10
Maintenance Supplies	1.82
Operating Supplies including Fuel	0.60
Electricity Power	4.36
Others	0.88
Total Process Operating Cost	**18.36**

Figure 21-1: Nominal Process Operating Cost Distribution



The process operating cost estimate includes the following:

- Hourly and salaried personnel requirements and costs, including expats, supervision, operation and maintenance, and salary/wage levels, including burdens
- Crusher and mill liners estimated from the in-house experience
- Steel ball consumptions for primary and secondary grinding based on steel consumptions from grindability parameters and steel ball prices from similar projects or Tetra Tech's database
- Maintenance supplies, based on major equipment capital costs
- Reagent consumption based on test results and reagent prices estimated according to similar projects or Tetra Tech's database
- Operation consumables, including laboratory and service vehicles consumables
- Electricity power consumption for the processing plant and tailings delivery and water reclaim from TSF is based on preliminary plant equipment load estimates and a power unit cost of $0.102/kWh, estimated based on the power supply from local electricity grid network

Tailings dam management related construction costs are included in sustaining capital costs.

Personnel

At a nominal processing rate of 5,000 t/d, the estimated average personnel cost is $1.51/t processed. The projected process personnel requirement is 121 persons, including:

- 17 staff for expats, management, and technical support, including personnel at laboratories for quality control and process optimization, but excluding personnel for sample collection and preparation
- 60 operators servicing overall operations from crushing, grinding, cyanide leaching, cyanide detoxification, residual silver, lead and zinc flotation, and tailings delivery to TSF and reclaim water extracted from TSF to processing plant, including personnel for sample collection and preparation
- 44 personnel for equipment maintenance, including the maintenance management team.

Consumable Supplies

The operating costs for major consumables, including mill liners, steel balls and reagents, were estimated at $9.19/t processed. The unit prices of consumables were based on similar local similar projects and operations or from Tetra Tech's database and industry experience. The major consumable costs are those related to grinding and cyanide leach, especially steel ball and sodium cyanide consumptions.

Maintenance/Operation Supplies

The cost for maintenance/operation supplies was estimated at $1.82/t processed. Maintenance supplies were estimated based on the information from Tetra Tech's database/experience. Operating supplies were estimated to be approximately $0.60/t, including vehicle fuel consumption.

Electricity Power

The total process electrical power cost was estimated to be $4.36/t processed. The estimated power unit cost was approximately $0.105/kWh which is assumed to be supplied from local grid power supply network.

The power consumption was estimated from the preliminary power loads estimated from the process equipment load list. The average annual power consumption was estimated to be approximately 77.86 GWh.

Others

The other miscellaneous costs that have not been clearly identified were estimated to be 0.88/t processed based on 5% of the operating costs listed above.

21.3.10 Site General and Administration Cost

The G&A operating cost was estimated based on similar operations in Ecuador and other nearby South American countries. It is estimated $24M per annum or $13.50/t-processed G&A expenditure will be needed. The LOM G&A operating costs amount of $288M. the site G&A operating costs exclude aforementioned water management cost, mining supervision fees, mining conservation fees, and State and employees profit sharing expenses.

Pre-production period G&A is included in capital cost estimates.

22 Economic Analysis

This section summarizes the economic analysis completed for the Condor PEA. A PEA is a conceptual study of the potential viability of Mineral Resources. A PEA should not be considered a prefeasibility or feasibility study, and the economics and technical viability of the Condor project have not been demonstrated at this time. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

22.1 Valuation Methodology

Condor project has been evaluated using a discounted cash flow (DCF) approach. This method of valuation requires projecting yearly cash inflows, or revenues, and subtracting yearly cash outflows such as operating costs, capital costs, royalties, state and employee profit sharing, and provincial and federal taxes. Cash flows are taken to occur at the middle of each period. The resulting net annual cash flows are discounted back to the date of valuation, January 1 of Year-2 (nominal as 2027), and totalled to determine net present values (NPVs) at the selected discount rates. The internal rate of return (IRR) is calculated as the discount rate that yields a zero NPV. The payback period is calculated as the time needed to recover the initial capital spent from initial investment start and the start of commercial production.

The economic analysis includes capital costs that are forecast to be incurred after the start of a two-year construction period. Condor expenditures that will be incurred prior to this point, such as costs for further exploration drilling, field investigations and analysis, more detailed technical and environmental studies, and additional surface rights land acquisition, are excluded from the PEA economic analysis.

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

All monetary amounts are presented in US dollars (US$ or $), unless otherwise specified, and financial results are reported on both post-tax and pre-tax basis.

22.2 Assumptions

The key assumptions used in the economic analysis are shown in Table 22-1, which metal prices are based on consensus average long-term metal prices, and also compared with the similar project published in public domains.

Table 22-1: Economic Analysis Assumptions

Assumption	Units	Value	Comment
Au Price	US$/oz	2,600.00	
Ag Price	US$/oz	31.00	

Pb Price	US$/lb	0.91	
Zn Price	US$/lb	1.27	
Royalty	%	5%	
Profit Sharing - State		12%	
Profit Sharing - Employee		3%	
Mining Supervision and Control Fee	$/ha/a	470.00	Exploitation
Conservation Fees for Mining Concessions*	$/ha/a	47.00	Exploitation
Income Tax		25%	

Notes:

* Varied from $94.00/ha/a to $470.00/ha/a, dependent on mining concession

** Varied from $9.4/0ha/a to $47.00/ha/a, dependent on mining concession

Source: SRK 2025

22.3 Production and Mill Feed

The proposed mining schedule and plant feed schedule is presented in Table 16-23 in Section 16 of this report, where Year-1 ROM material will be processed in Year 1 with Year 1 ROM together in Year 1. The mining schedule is based on a two-year plant construction period followed by approximately 13 years of underground operation. Underground pre-production development is scheduled in 1.25 years (9 months later than plant construction start) prior to plant commercial production. The plant feed schedule is based on processing 5,000 tonnes per day (1.8 Mta) for approximately 13 years.

The mill plant will produce gold-silver dore and two saleable concentrates: silver-lead concentrate and zinc-silver concentrate. Process recoveries based on the preliminary metallurgical test work and flowsheet are shown in Table 17-1 and Figure 17-1 in Section 17. Overall, the weighted average process recoveries over the life of mine life are estimated as shown in Table 22-2.

Table 22-2: Summary of Mine Physical and Metal Recovery

Item	Units	Value
Physicals (Mill Feed)		
Mill Feed	Mt	21.34
Au Feed Grade	g/t	2.15
Ag Feed Grade	g/t	14.20
Pb Feed Grade	%	0.06
Zn Feed Grade	%	0.54
NSR	$/t	178.74
Total Salable Metal Recovery		
Au Recovery		93.9%
Ag Recovery		64.4%
Pb Recovery		36.4%
Zn Recovery		54.2%
Overall Payability		
Au Payable		99.3%
Ag Payable		84.0%
Pb Payable		86.6%
Zn Payable		69.9%

22.4 Capital and Operating Costs

Capital and operating cost estimates are presented in Section 21 of this report and are summarized in Table 21-1. Initial capital over a two-year construction period is estimated at $292 million. Sustaining capital, principally for mining development and TMF expansion, is estimated at $382 million.

As presented in Table 21-6 in Section 21 total operating costs for mining, processing and G&A average are $95.51/t processed and total $2,037.7 million over the 13 years of mine life.

22.5 Working Capital

Working capital includes the requirements of saleable metals in mill circuit and concentrates, which will delay the receipt of saleable product revenue. Working capital is also required to maintain an operating supplies inventory on site. Accounts payable, estimated at one month on site operating cost, partially offsets these working capital requirements. In this preliminary economic assessment, working capital requirement is assumed to be two-month of total annual operating costs.

22.6 Mine Closure and Salvage Value

The mine closure cost as presented in Section 21.2 is estimated at $42 million excluding TSF closure cost which is included in TSF sustaining capital. This includes pre-production as government bond or security and during production and post closure period. For the purposes of economic evaluation, the remaining closure cost is assumed incurred in Years 11, 12, and 13.

It is assumed that there is no salvage value for mine mobile equipment and process machinery and equipment in the project economic model.

22.7 Taxation, Profit Sharing, Mining Concession Management Fees, and Royalty

Condor project is located in Ecuador, the applicable taxes and fees for the project are listed below:

- 25% state income tax.
- 12% profit sharing state.
- 3% profit sharing employee.
- 5% NSR based royalty.
- Up to $470/ha/yr mining supervision and control fee during exploitation stage (100% SBU/ha, or 100% Unified Base Salary). This fee varies from $94.00/ha/a to $470.00/ha/a, dependent on the status of Condor mining concession.
- Up to $47.00/ha/yr conservation fees for mining concession during exploitation stage (10% of SBU/ha-mined). This fee varies from $9.40/ha/a to $47.00/ha/a, dependent on the status of Condor mining concession.
- Value added taxes (VAT) are excluded from the economic analysis. The Ecuador standard VAT rate is 15% but it is understood from public domain information that gold exporters such as Condor, which will export doré or refined gold, are in a 0% VAT category, and that there are mechanisms in place for input credits. It is assumed for the PEA that any VAT payments made during the preproduction and production period will be recoverable.

22.8 Indicative Economic Results

The base case indicative economic results, at a discount rate of 5%, are summarized in Table 22-3 and are favourable for Condor project.

At the base case metal prices as shown in Table 22-3, the potential pre-tax net present value (NPV) at the start of the projected two-year construction period using a 5% discount rate (NPV5%) is estimated at $720 million, and potential project post-tax NPV5% is estimated at $522 million. Potential internal rates of return (IRR) are respectively 36% pre-tax and 29% post-tax.

At the base case metal prices and project cost estimates payback of the initial capital is forecast to occur in the third year of the 13-year operating mine life. The payback period is defined as the time after process plant start-up that is required to recover the initial expenditures incurred developing Condor project. At the effective date of this report, the project's cumulative undiscounted net cash flow is zero.

Table 22-3: Economic Analysis Summary

	Unit	Total
Plant Feed	**Mt**	**21.34**
Payable Gold	Oz (000)	1,375
Payable Silver	Oz (000)	5,266
Payable Lead	lbs (000)	8,448
Payable Zinc	lbs (000)	95,656
Equiv. Payable Gold	**Oz (000)**	**1,487**
Net Smelter Return	$/t	179
Operating Costs		
Mining	$M	875
Processing	$M	392
Water Management	$M	15
Mining Supervision Fees	$M	17
Conservation Fees	$M	2
Refining and Freight	$M	54
Royalties	$M	191
Profit Sharing State	$M	164
Profit Sharing Employee	$M	41
All Other G&A	$M	288
Total Operating Cost	**$M**	**2,038**
Total Operating Cost	**$/t-milled**	**95.51**
Capital Costs		
Initial Capital	$M	292
Sustaining Capital	$M	382
LOM Total Capital	**$M**	**674**
Project All-In Cost	$M	3,002
Cash Cost	$/EqOz-Payable	1,118
All-in Sustaining Cost (AISC)*	**$/EqOz-Payable**	**1,359**
Project All-in Cost	$/EqOz-Payable	2,018
Economic Indicators		
Project Pre-tax Cash Flow	$M	1,156
Pretax NPV 5%	$M	720
Pre-tax IRR		36%
Payback from Mill Start	Yr	3.0
Post-tax Cash Flow	$M	865
Post-tax NPV 5%	$M	**522**
Post-tax IRR		**29%**

Notes:
*Based on World Gold Council June 27,2013 Press Release: “Guidance Note on Non-GAAP Metrics - All-In Sustaining Costs and All-in Costs”

22.9 Sensitivity Analysis

The results of the base case sensitivity and other sensitivity analyses are summarized in Table 22-4 to Table 22-5 and Figure 22-1 to Figure 22-2.

Table 22-4: Project NPV Sensitivity to Key Input Parameters

Percent to Base Case		70%	80%	90%	**100%**	110%	120%	130%
Gold Price	$/oz	1,820	2,080	2,340	2,600	2,860	3,120	3,380
NPV5% vs Au Price	M$	75	224	373	522	671	821	970
NPV5% vs Site Opex	M$	735	664	593	522	452	381	310
NPV5% vs Capex	M$	657	612	567	522	477	432	387

Table 22-5: Project IRR Sensitivity to Key Input Parameters

Percent to Base Case		70%	80%	90%	**100%**	110%	120%	130%
Gold Price	$/oz	1,820	2,080	2,340	2,600	2,860	3,120	3,380
IRR vs Au Price		9%	17%	23%	29%	34%	39%	44%
IRR vs Opex		36%	34%	31%	29%	26%	23%	20%
IRR vs Capex		44%	38%	33%	29%	25%	22%	20%

Figure 22-1: Condor Project NPV5% Sensitivity to Key Input Parameters

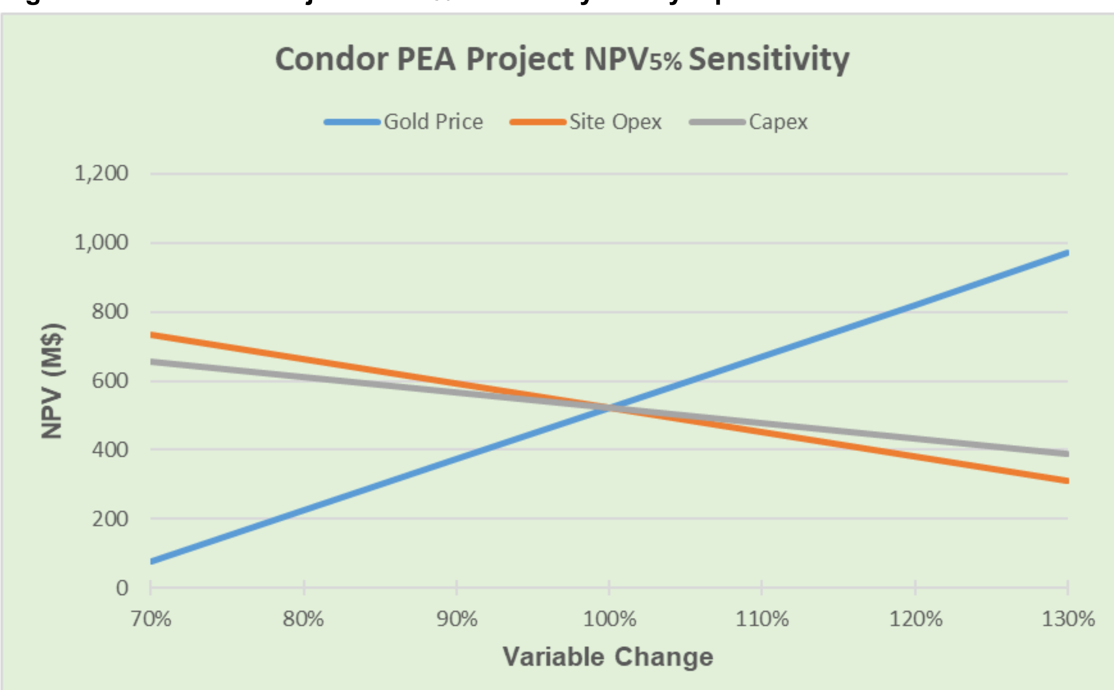
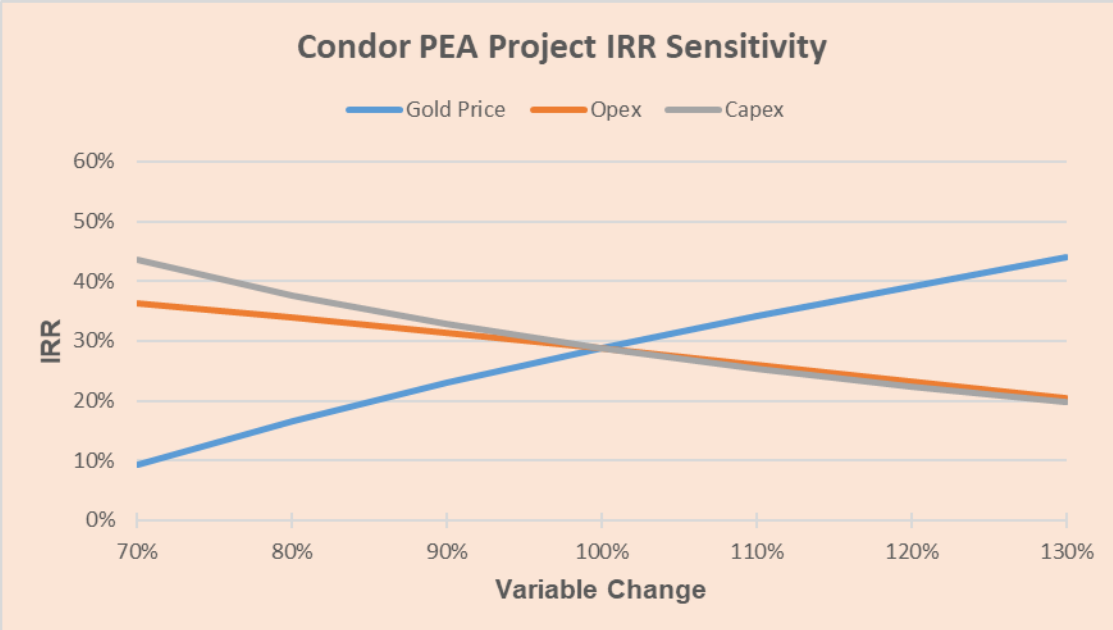


Figure 22-2: Condor Project IRR Sensitivity to Key Input Parameters



The sensitivity analysis was performed on the base case taking into account variations in gold price, operating cost, and capital cost.

Like most greenfield mining projects, the key economic indicators of NPV5% and IRR are most sensitive to change in gold price (i.e. revenue), as they affect directly the revenue stream. A 10% reduction from the US$2,600/oz base case gold price reduces Condor's post-tax NPV5% and IRR by 28.5% and 6 percentage, respectively. A 10% increase from the US$2,600/oz base case gold price increases Condor's post-tax NPV5% and IRR by 28.5% and 5 percentage, respectively. The sensitivity analysis shows that the project is less sensitive to operating cost and capital expenditure.

22.10 Capital Cost Exclusions

The cost estimate presented herein is for information only and does not indicate the future capital cost estimate produced for subsequent studies.

The following items are not included in the capital cost estimate:

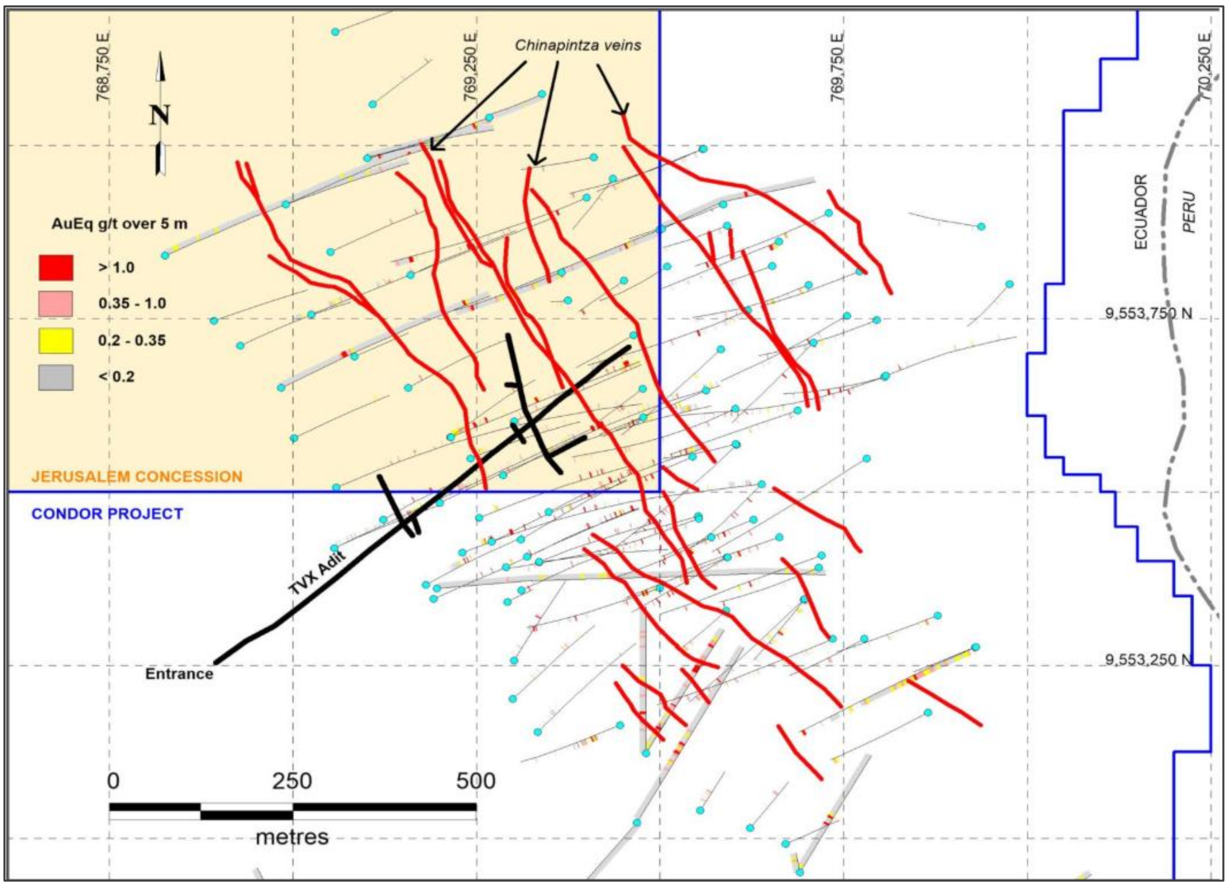
- Force majeure
- Schedule delays, such as those caused by:
  - Major scope changes
  - Unidentified ground conditions
  - Uncertainties in geotechnical or hydrogeological conditions
  - Labour disputes

  - Environmental permitting activities
  - Abnormally adverse weather conditions
- Schedule acceleration costs
- Cost of financing (including interests incurred during construction)
- Corporate expenses
- Working or deferred capital (included in the financial model)
- Receipt of information beyond the control of the EPCM contractors
- Salvage value for assets only used during construction
- Taxes (PST, GST, and HST) and duties
- Land acquisition if required
- Project sunk costs (exploration programs, etc.)
- Cost of this study and future studies, including feasibility studies
- Closure and reclamation, which is included in financial analysis, excluding closure related earth works for TSF
- Vendor price fixing/gouging
- Macroeconomic factors
- Currency fluctuations
- Geopolitical tensions or war
- Disruptions of global supply and logistical services
- Pandemics or other natural disasters
- Royalties, which are included in financial analysis, or permitting costs, except as expressly defined
- Forward inflation
- Escalations beyond the effective date of this study

23 Adjacent Properties

There are a number of other mineral occurrences in the Zamora copper-gold metallogenic belt, including deposits in the Condor Central and Condor South areas owned by Silvercorp. The notable Silvercorp deposits are shown in Figure 7-2 and include the Chinapintza epithermal gold veins immediately to the north of Los Cuyes, which extends beyond the Condor project mining concessions onto the adjacent Jerusalem Concession (Figure 23-1). To the south on the Silvercorp concessions are known occurrences at Prometedor, El Hito, Santa Barbara and Nayumbi.

Figure 23-1: Plan Map - Chinapintza Veins - Jerusalem Concession-



Sources: Ronning, 2003; Luminex, 2018, Luminex 2021

Luminex 2021 reports that TVX did an extensive amount of exploration work on the Jerusalem claim, including diamond drilling (35 holes; 9,338.1 m), trenching and underground development and sampling. In 1996, it calculated a historical Mineral Resource for this zone of 535,828 tonnes grading 12.5 g/t Au, 66.4 g/t Ag, 0.07% Cu, 0.76% Pb, 3.57% Zn (Ronning, 2003). This historical Mineral Resource estimate is detailed in the NI 43-101 Technical Report entitled “Review of the Jerusalem Project, Ecuador” with an effective date of May 30, 2003, and is available on SEDAR. The QP responsible for the Mineral Resource estimates in this technical report have not done sufficient work to classify these historical estimates as current mineral resources and Silvercorp is not treating these historical estimates as current Mineral Resources.

In 2004, Maynard (2004) provided an updated historical Mineral Resource estimate for the veins on the Jerusalem concession (Table 23-1). This historical Mineral Resource estimate is detailed in the NI 43-101 Technical Report entitled “Independent Geological Evaluation, Jerusalem Project, Zamora Chinchipe, Ecuador for Dynasty Metals & Mining Inc.” with an effective date of October 29, 2004 and is available on SEDAR. The QP responsible for the Mineral Resource estimates in this technical report have not done sufficient work to classify these historical estimates as current Mineral Resources and Silvercorp is not treating these historical estimates as current Mineral Resources. The QP has been unable to verify this Mineral Resource estimate, and it is not necessarily indicative of mineralization on the Condor North Project.

Table 23-1: Maynard (2004) Jerusalem Concession Mineral Resources

Category	Tonnes	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
Measured	298,900	13.9	102	576	563	26,859
Indicated	722,500	12.8	98	360	3,560	17,660
Inferred	1,785,200	11.6	103	424	3,887	18,397

Source: Maynard, 2004

Notes: These have not been reviewed by SRK.

The authors of this report have not completed sufficient work to verify the historical Mineral Resource on the Jerusalem concession and this information is not necessarily indicative of mineralization on the Condor North area.

In 2021, Luminex reported a Mineral Resource for the Santa Barbara deposit. Santa Barbara is a gold-copper porphyry hosted in alkali basalts of unknown age. These are intruded by diorite and surrounded by the Zamora Batholith. These host units are capped by a veneer of conglomerates of the Chapiza Formation and in turn overlain by quartz arenites of the Hollín Formation. The Luminex Mineral Resource estimate for Santa Barbara is shown in Table 23-2. The QP has not reviewed the Santa Barabara Mineral Resources. The QP responsible for the Mineral Resource estimates in this technical report have not done sufficient work to classify these historical estimates as current Mineral Resources and Silvercorp is not treating these historical estimates as current Mineral Resources.

Table 23-2: Luminex (2021) Mineral Resource estimate for the Santa Barbara Deposit

Class	Tonnes (Mt)	Average Grade			Contained Metal		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (Moz)
Indicated	39.8	0.83	0.67	0.8	1,057	859	1.0
Inferred	166.7	0.66	0.52	0.9	3,534	2,768	4.9

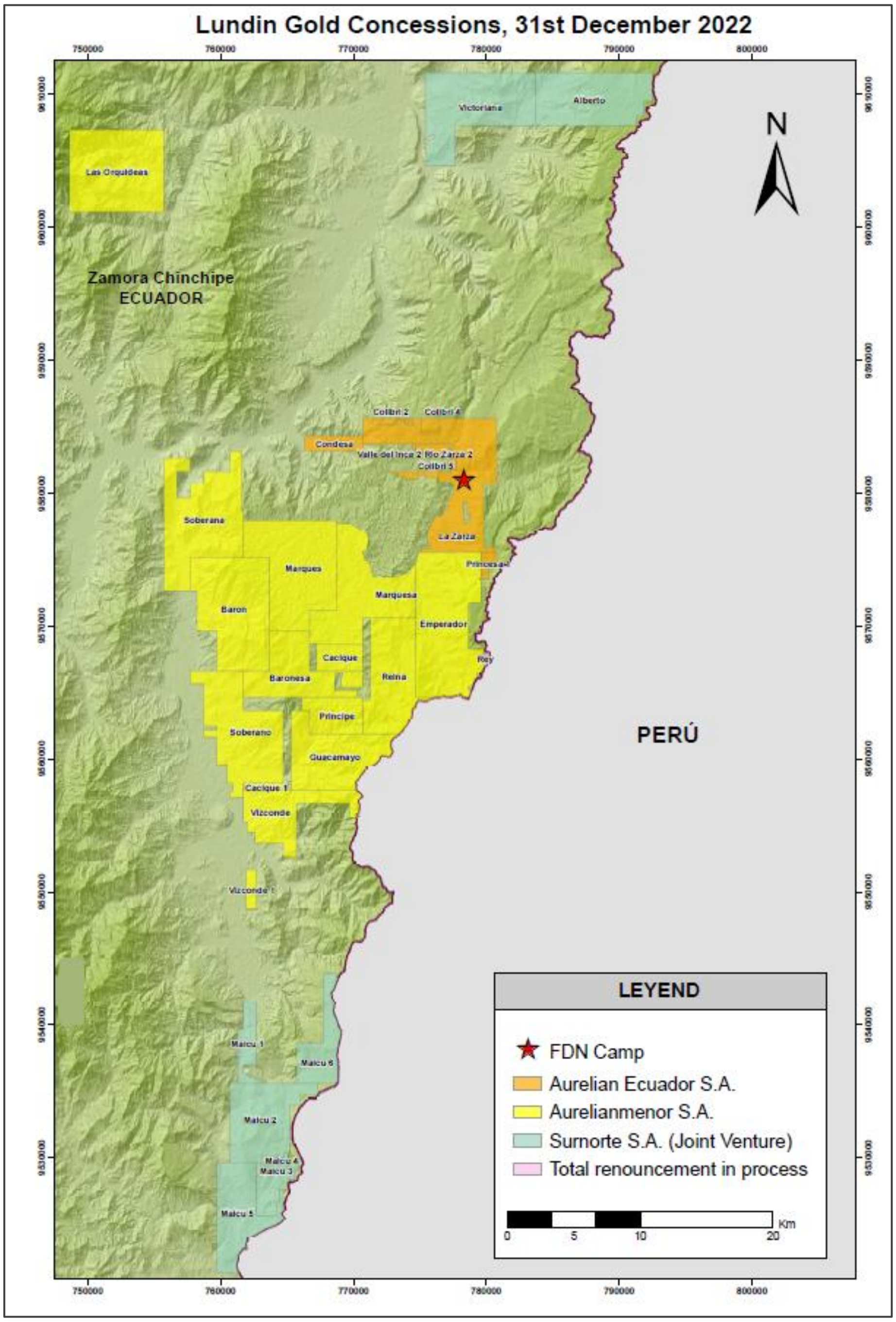
Source: Luminex, 2021

Notes: Mineral resources exhibit reasonable prospects of eventual economic extraction using open pit extraction methods. The base case cut-off grade is 0.37 g/t AuEq where: AuEq = Au g/t + (Ag g/t × 0.012) + (Cu% × 1.371).

23.1 Fruta del Norte

The Fruta del Norte mine is an operating gold-silver deposit located within the Zamora-Chinchipe province in southeast Ecuador, approximately 32 km north from the Condor project. The deposit is owned and operated by Lundin Gold Inc. through its Ecuadorian subsidiary, Aurelian Ecuador S.A. (Figure 23-2).

Figure 23-2: Location and Tenure Plan – Fruta del Norte Mine



Source: Lundin Gold, 2023

Fruta del Norte is one of the largest gold projects in South America. The deposit comprises epithermal gold-silver mineralization hosted within Jurassic volcanic and sedimentary rock units. According to Lundin Gold's public disclosures (Lundin Gold, 2023), the Proven and Probable Mineral Reserves as of December 31, 2022 are reported at approximately 5.0 million ounces of gold (18.0 Mt at 8.7 g/t Au) and 6.6 million ounces of silver (18.0 Mt at 11.4 g/t Ag). The QPs have not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves and Silvercorp is not treating these historical estimates as current Mineral Reserves.

The mine commenced commercial production in February 2020, and is developed utilizing underground mining methods, principally long-hole stoping, with ore processed via gravity separation and flotation followed by leaching of the gravity concentrate and flotation concentrates. In 2024, Lundin Gold reported gold production of approximately 502,029 ounces of gold (Lundin Gold Inc., News Release, January 8, 2025. "Lundin Gold exceeds 2024 production guidance and achieves record annual production of 502,029 ounces of gold.").

The inclusion of the Fruta del Norte deposit in this report is for information purposes only. The qualified person(s) responsible for this report have not independently verified the mineralization, reserves, resources, or geological information related to the Fruta del Norte mine. Information regarding Fruta del Norte has been derived from publicly available sources, including Lundin Gold's annual information forms and technical reports filed on SEDAR.

24 Other Relevant Data and Information

The QP is not aware of any other relevant data available about the Condor project.

25 Interpretation and Conclusions

Geology and Mineral Resources

Silvercorp has undertaken a review, re-logging, and remodelling of the mineralization at the Condor Project. At the Los Cuyes and Camp deposits the updated model of mineralization has included identification of several high-grade tabular domains which are potentially amenable to extraction using underground mining methods. At Soledad, Enma and outside of the high-grade domains at Los Cuyes Silvercorp have modelled a lower grade disseminated mineralization which has the potential for extraction using an open pit mining method.

This mineralization interpretation at Los Cuyes is a change from the previous model which only considered a disseminated mineralization style and did not isolate the high-grade zones separately. For some domains at Los Cuyes (such as the LCW domain) the data strongly support the revised interpretation, with good continuity in the mineralization observed over the project area. While for other domains, the continuity is less clear, and the quantity of data supporting these is less. Resulting in lower confidence in these interpretations. The lateral extents of some of the domains are based on wider spaced drilling which naturally carries some additional risk to the confidence in the interpretation of the domain continuity.

At Camp, the previous models relied on interpolated domain definition using indicators, and the current interpretation is supported by a more geologically rigorous interpretation using a combination of the grade and geological logs to link up intersections between drill holes into more coherent and continuous domains.

The geological interpretation at Soledad and Enma is not as well developed as that of Los Cuyes and Camp, relying on grade shells to constrain the mineralization. At Soledad, there is sufficient dense sampling in several locations to confirm the continuity of the mineralization despite the lower understanding of the mineralization controls, and SRK considers this sufficient to support an Indicated Mineral Resource classification.

For all the deposits, the metallurgical test work indicated that there are reasonable prospects for achieving the recoveries applied to the economic assessment. However, further work is required to be able to confirm the optimal processing configuration for each style of mineralization. As such, there is a risk that these recovery factors may change with additional test work and depending on the ultimate processing flow sheet that is selected if the project is developed.

The additional drilling that was completed by Silvercorp during 2025 has generally confirmed the previous mineralization interpretation. Mineralization is intersected close to where the models predicted, and the grades intersected are generally consistent with the estimated grades, which supports that the interpretation is reasonably robust. The mineralization is not closed on some sides or at depth, and there is potential to expand the currently defined Mineral Resources with additional drilling.

Mineral Processing and Metallurgical Testing

Based on the available metallurgical testwork data, a few important observations are summarized below:

- The gold is generally free milling. The whole-ore cyanide leach achieves gold recovery on the order of 96% for the Camp domain, 89% for the Los Cuyes domain, 87% for the Soledad domain and 75% for the Enma domain.
- The whole-ore cyanide leach results in poor silver recovery on the order of 45% for the Camp domain, 46% for the Los Cuyes domain, 75% for the Soledad domain and 69% for the Enma domain. Most of the unrecovered silver is probably associated with galena.
- The results of gravity concentration testwork show a significant amount of gravity recoverable gold around 34% for the Camp domain, but a less amount of gravity recoverable gold (23%) for the Los Cuyes domain and a further less amount of gravity recoverable gold (5%) for the Enma domain. Because gold and silver account for about 94% of total in-situ value in the mill feed, the flowsheet of gravity concentration followed by cyanide leach is preferred so that the final will be the gold/silver dore. Subject to the metal prices and operating cost, the remaining gold, silver, lead and zinc in the cyanide leached residue may be recovered by selective flotation. Although further flotation testwork is needed, the completed testwork has indicated that the marketable lead/silver concentrate and zinc/silver concentrate can be produced. The mineralized materials have a moderate hardness and a low abrasion property./As with the high gold recovery achieved from the whole-ore cyanide leach, the bulk flotation also resulted in the high gold recovery.
  - For the Camp domain, average gold recovery was 97.5% at 14.2% concentrate mass pull. Average silver recovery was 95.9%.
  - For the Los Cuyes West domain, average gold recovery was 94.5% at 12.5% concentrate mass pull. Average silver recovery was 89.3%.
  - Because of the high sulfide (pyrite) content, the bulk flotation will not generate a high-grade gold concentrate attractive to sell. Nevertheless, the bulk flotation concentrate is amenable to cyanide leach with gold recovery on the order of 94% for the Camp domain and 93% for the Los Cuyes West domain. Thus, the net gold recovery is 97.5% x 94% = 91.7% for the Camp domain, and 94.5% x 93% = 87.9% for the Los Cuyes West domain. If the bulk flotation concentrate is reground prior to cyanide leach, the gold
  - ecovery from cyanide leach may be higher. For the Camp domain, this net gold recovery is about 4% lower than the whole-ore cyanide leach.
  - The cyanide leached residue was tested for selective flotation to generate the lead/silver concentrate and zinc/silver concentrate. Although further flotation testwork is needed, it appears that the marketable lead/silver concentrate and zinc/silver concentrate can be produced by floating the cyanide leached flotation concentrate.
- The flowsheet of “bulk flotation followed by cyanide leach of the flotation concentrate” is an alternative to the whole-ore cyanide leach. This alternative would be attractive in a situation where it is problematic and expensive to dispose of the cyanide leached tailing.

Mine Geotechnical

- A reasonable drillhole based geotechnical data set has been established for both the Camp and Los Cuyes areas, including an extensive point load test data set for each area. The majority but not all veins have sufficient geotechnical data as noted below.
- 3D geological models have been established for both Camp and Los Cuyes with sufficient detail for use in current and future studies; the structural model is considered more limited and includes three major faults at the extents of the vein systems, oriented approximately perpendicular to the vein strike direction.
- Geotechnical conditions at Camp are generally good with little variability observed between mineralized zones, hangingwall, and footwall; occasional altered and damaged zones are observed which are occasionally concurrent with the NW veins and may locally affect HW and FW stability.
- Stope design at Camp includes options for both longitudinal and transverse orientations with a maximum long-wall hydraulic radius of 6.3m (20mH x 35mL).
- Geotechnical conditions at Los Cuyes are generally fair to good with evidence of adverse matrix alteration or matrix weakening associated with some geological contacts which results in the presence of poor ground conditions.
- Stope design at Los Cuyes includes options for both longitudinal and transverse orientations with a maximum long-wall hydraulic radii of 4.2m (20mH x 15mL) and 4.7m (20mH x 18mL).
- Ground support has been designed for all permanent excavations using resin grouted rebar, and temporary areas using inflatable (Swellex or Omega type) anchors; walls and back require welded wire mesh and an allocation of shotcrete has been included for areas of broken or lower quality ground.
- Ground support for stoping assumes cable bolting is required for all transverse stope backs, and for longitudinal stopes over 6.0mW at Camp and 5.0mW at Los Cuyes (hangingwall to footwall distance).

Mining Methods

- The updated block models for the Camp and Los Cuyes zones form a sound foundation for mine planning.
- NSR values were derived from metallurgical recoveries, processing costs, and metal prices, providing realistic economic guidance for stope design and cut-off determination.
- The selected longhole stoping method is technically appropriate for the steeply dipping vein systems.
- Preliminary designs demonstrate that the geometry and rock conditions can support stable stopes with manageable dilution and acceptable recovery assumptions.
- The main portal located at approximately 1,100 m elevation provides efficient access for haulage, ventilation, and services.
- The mine plan projects a production life of approximately 13 years, excluding the pre-production construction period.
- Annual throughput targets a steady-state rate of 1.8 Mtpa, or 5,000 tpd, balancing production between the Camp and Los Cuyes zones.
- The proposed mobile fleet and underground infrastructure are consistent with industry practice for a mechanized longhole stoping operation of this scale.
- Equipment sizing, haulage profiles, and development dimensions align with the projected production rate.

SRK is not aware of any significant risks and uncertainties that could be expected to affect the reliability or confidence in the early stage exploration information discussed herein.

Environmental

The social context surrounding the project is marked by the expansion of informal and illegal mining, the vulnerability of Shuar communities, and the growing local demand for employment and services. Silvercorp is aware of this risk and are developing a strategy to address the social risks associated with development in this region in order to secure a social license to operate the Condor Project. Compliance with national labor regulations and proactive management of population dynamics will be essential to ensure the project’s long-term social viability and positive community relations.

Water Management

There are no identified fatal flaws in terms of water management affecting the project.

There is very little site to base the design of the water management system.

Additional studies are needed to better characterize site climate, hydrology, geochemistry and water quality to advance the project and under.

Recovery Methods and Processing

According to the tests results and the proposed mine plan, one gold-silver doré product, together with marketable silver-lead and zinc concentrates, are expected to be produced from the proposed 5000-t/d plant. The plant feed will be ground in a two-stage grinding circuit (a SAG mill with pebble recycling + a ball mill), integrated with a gravity concentration. The ground mill feed slurry will be processed by a conventional carbon-in-pulp (CIP) cyanidation circuit integrated with loaded carbon washing, elltion and gold electrowinning on pregnant solution to recover the gold and silver, producing gold-silver doré. The leach residue is treated to destroy residual WAD cyanide. Subsequently, the leach residue is further processed by conventional differential flotation to produce marketable silver-lead and zinc concentrates separately. The flotation tailings is thickened and pumped to tailings storage facility (TSF) for storage.lThe LOM overall gold and silver recoveries to the gold-silver doré are estimated to be approximately 93% and 46%, respectively. The differential flotation is expected to further recover approximately 12% silver and 36% lead to a silver-lead concentrate and approximately 6.4% silver and 54.2% zinc to a zinc concentrate.

Tailings Management

The TSF design was developed with reference to Canadian Dam Association guidelines and to suit the project setting, regional precedent, and economics. Risks associated with the proposed design include uncertainty with respect to regulatory approval requirements, and the challenges associated with earthwork construction and water management in this project setting.

Economics

The results of the analysis show that Condor project to be potentially favourable. Sensitivity analysis shows that the project economic indicators are most sensitive to gold price, capital costs, then to operating cost and capital.

There is a risk that metal prices especially gold price will be lower than the long-term price assumed in this study.

There is a risk that the project will incur additional unforeseen taxation charges. The assumption for this study that as a gold mining project and exporter, the project would not be subject to Ecuador VAT or allowing for VAT fully recovery or customs duties during the preproduction period or during mine operation, is supported by current Ecuador mining regulations. This assumption is generally supported by public domain information but should be confirmed with Ecuador tax authorities.

26 Recommendations

Exploration

To confirm the interpretation of the high-grade domains at Camp and Los Cuyes SRK recommends an exploration program should be undertaken. Silvercorp has planned an initial exploration program of underground drilling as summarised in Table 26-1. Silvercorp has planned to develop an exploration drive (pending the approval of an environmental permit which is in progress at present) to intersect the mineralization, and to provide platforms for drilling which will allow for better targeted drilling of shorter holes from the underground development Table 26-1.

Table 26-1: Proposed Initial Exploration Program for the Condor Project

Activity	Meters	Cost (US$ Million))
UG tunnel development	1,500	$4.5
UG diamond drilling	30,000	$6.0
Total	**3,500**	**$10.5**

Source: Silvercorp

The drilling should be targeted to improve the drilling density and improve the classification of the Mineral Resources, particularly on the western margins of Los Cuyes LCW domain and the eastern margin of the Camp domains. The infill drilling should also aim to resolve the improve the confidence in the interpreted fault at Camp and investigate its impact, if any, on the mineralization.

SRK is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform the exploration work recommended for the Condor Gold project.

Mineral Processing and Metallurgical Testing

Although the completed metallurgical testwork has demonstrated that the mineralized materials from the Condor project are amenable to the whole-ore cyanide leach and to the bulk flotation, further investigations are needed to maximize the gold/silver recoveries and to generate a series of process parameters necessary for engineering design of the process plant. The costs for these metallurgical test programs are estimated to be approximately US$350k.

- More representative samples from each domain should be selected for the comminution testing, including the crusher work index, SMC or drop weight test, rod mill work index, ball mill work index and abrasion index.
- The single-stage gravity concentration at grind size of 80% passing 210 μm was exclusively carried out. The multi-stage gravity concentration testwork is strongly recommended for each domain. The gravity recoverable gold will likely increase with the multi-stage gravity concentration. After the testwork data from the multi-stage gravity concentration are obtained, a series of simulations are recommended to forecast the expected gold recovery from the future commercial operations. For the whole-ore cyanide leach, optimization testwork is recommended to fine tune the operating conditions, including the pulp density, grind size, cyanide concentration, pH, dissolved oxygen and retention time. Previous cyanide leach testwork showed a weak preg-robbing with some materials, and this should be verified by carrying out the parallel CIL cyanide leach and CIP cyanide leach, and then compare their gold recoveries. The adsorption behaviour of dissolved gold and silver on the activated carbon should be

determined using the actual pregnant leachate. The dissolved silver does not adsorb strongly on the activated carbon and thus some of the dissolved silver may be lost to the CIP tail in the future commercial operation. Also, some dissolved copper and zinc are present in the pregnant leachate, and they may adversely impact the loading of gold and silver on the activated carbon. When the process water is recycled, the dissolved copper and zinc will build up in the process water.

- After representative CIP tail samples become available, the continuous cyanide destruction testwork is recommended.
- The cyanide leach tail after cyanide destruction has never been tested for the sequential selective flotation to generate the lead/silver concentrate and zinc/silver concentrate. The oxidizing nature during cyanide destruction may deteriorate the subsequent flotation performance. Although the economic contribution by these two flotation concentrates will be marginal, a series of flotation tests are needed to verify the marketable lead/silver concentrate and zinc/silver concentrate can indeed be produced consistently. Previous testwork data showed the concentrates produced from the Los Cuyes West domain contained high levels of arsenic, cadmium and antimony. The assays of these penalty elements plus mercury, chloride and fluoride, etc should be repeated when the representative flotation concentrates become available.
- Because of the high sulfide (pyrite) content, the cyanide leached tail may generate acid in the tailing pond when the sulfide minerals are oxidized over time. This potential acid generation may remain even after the cyanide leached tail is floated again to produce the lead/silver and zinc/silver concentrate. Therefore, several environmental tests, including ABA, SPLP, TCLP, column leach and humidity cell, are recommended for the representative tail samples.
- The mineralized material from the Soledad (San Jose) domain seems acidic in-situ. As a result, the in-situ pH of all future mineralized samples should be measured. The in-situ acidity will cause some corrosion issue to the mining equipment and process equipment.
- As for the cyanide leach of the bulk flotation concentrate, gold and silver recoveries will likely increase if the bulk flotation concentrate is reground. Such testing is recommended. Also, the addition of lead nitrate to the cyanide leach of flotation concentrate may be beneficial to gold recovery, and thus, some testing is recommended.
- The thickening and filtration testwork for the final tail, lead/silver concentrate and zinc/silver concentrate are required for the engineering design of the process plant.

Mine Geotechnical

- Complete a geotechnical re-logging or machine-learning program of pre-2020 Los Cuyes drill core to provide geotechnical data coverage over all veins.
- In addition to ongoing geotechnical data collection from exploration drill holes, a series of dedicated geotechnical drill holes (using triple tube and oriented core methods) should be completed targeting hangingwall, vein, footwall, and critical infrastructure areas.
- Completed a laboratory testing program (unconfined compressive strength, Brazilian tensile strength) to confirm intact rock strength for dominant lithologies, and determine correlations for point load testing.
- Undertake an update to the major structures model (all areas) including a structure description matrix and confirmation of small-scale joint orientations for input to kinematic analyses.

- Review correlations between weakening alteration types and rock quality to establish geotechnical domains of lower quality ground at Los Cuyes.
- Review pillar stability (particularly in transverse areas) to determine the effectiveness of the temporary narrow pillars designed to contain uncemented waste rock.
- Update stopes designs considering revised geotechnical model inputs including structure, alteration, and rock strength.
- Complete numerical modeling of the stope and local pillar geometries, sill and crown pillars, and global mine extraction sequence.

The cost of these geotechnical studies is estimated to be between US$150-200k excluding orientation core drilling.

Mining Methods

- Re-benchmark metal prices, treatment charges, and recoveries prior to the PFS to ensure that the NSR values accurately reflect current market and operating conditions.
- Conduct a sensitivity analysis to test the impact of different cost and recovery assumptions on the NSR cut-off used for stope optimization.
- Refine longhole stope dimensions, spacing, and strike lengths based on updated geotechnical domains and structural data.
- Assess dilution and recovery factors through numerical modeling or trial stope simulations to reduce uncertainty in mineable tonnage estimates.
- Evaluate alternative backfill options (e.g., cemented rockfill, paste fill) to increase recovery and improve ground control where required.
- Confirm pillar stability through additional geotechnical analysis and, where appropriate, empirical or numerical modeling.
- Review ramp and level spacing to balance development capital against production flexibility.
- Conduct a detailed mine scheduling study using updated stope geometries and equipment cycle times to confirm annual production targets and ore delivery consistency.
- Review mobile equipment fleet size and utilization rates to ensure compatibility with production requirements and ventilation constraints.
- Investigate the potential use of battery-electric or low-emission equipment to reduce ventilation demand and operating costs.
- Advance the design of key underground infrastructure including power distribution, pumping, dewatering, and material handling systems to a PFS level of definition.

SRK estimates that the costs of the mining and associated studies recommended to be undertaken as part of a PFS will be between US$1.2-1.5M.

Water Management

The following studies should be completed as the project advances:

- A geochemical investigation of the wasterock, tailings, quarry rock and existing water quality. There is the potential the artisanal miners may be using mercury and the background concentrations for mercury and other constituents should be established. US$200k
- A climatic investigation including installing a meteorologic station(s) and hydrologic gauging stations.
- A hydrologic investigation to characterize flow in the nearby river and creeks. This must include installing hydrologic gauging stations. Also the flood plain of the river adjacent to the proposed location of water management infrastructure, the process plant and mine support buildings should be delineated. US$100k
- Ground conditions (e.g. depth to groundwater, soil characteristics, and geotechnical properties) in the footprint of water management infrastructure should be characterized. US$250k
- A more complete and detailed site wide water balance that more fulsomely integrates the management of excess water collected in the TSF with the site water management system. US$250k
- Field data for assessing the hydrogeological system should be collected as part of either exploration or mine geotechnical drilling. This work could consist of packer based injection test to determine hydraulic conductivity (K) and/or installation of monitoring wells for K testing and water quality sampling. Hydraulic testing of geological structures should be carried out when possible as these are likely to be the dominant inflow paths/features. US$500k.

Recovery Methods and Processing

Further process optimization should be conducted, including equipment sizing and type and general layout and site selection. The costs related to the process optimization is part of future studies.

Plant site geotechnical condition investigations should be conducted. The cost related to geotechnical condition investigations are estimated to be approximately US$100k.

Plant design related parameters should be determined and collected, such as crushed material bulk density, leach feed and residue settling rates, slurry rheological property. The costs for these property determinations are estimated to be approximately US$80k.

A further study to optimize the flotation flowsheet and determine this circuit economics should be conducted with an integration of concentrate market study. The cost for this study is estimated to be US$30k, excluding metallurgical test work.

Tailings Management

The following studies and investigations are recommended to support TSF design for the pre-feasibility phase of the project:

- Geotechnical investigation and assessment. The scope of this work will involve characterization of the TSF embankment foundations, basin, and potential borrow areas. Assessment will involve materials characterization, preliminary geological model development, and preliminary stability and seepage modelling for the TSF. US$100k.
- Hydrology and water management design. This scope of work includes catchment and runoff determination, establishment of design storm events, and preliminary TSF water balance. US$50k.
- Geochemical assessment of the tailings, mine process water, and TSF construction materials. US$25k.

- TSF risk assessment. This work should be undertaken to support identification of risks and controls, and determination of dam consequence class. US$20k.
- Advance TSF design detail to prefeasibility level, including construction sequence and cost estimate update. US$100k.

Economic Analysis

It is recommended that the following project taxation issues be clarified with the Ecuador tax authorities:

- Applicability of VAT and customs duties on equipment, consumables, and services during the project's preproduction and operational phases.
- Appropriate asset classes and depreciation rates.
- Withholding taxes on interest payments and dividends.;.
- Property taxes, if any.

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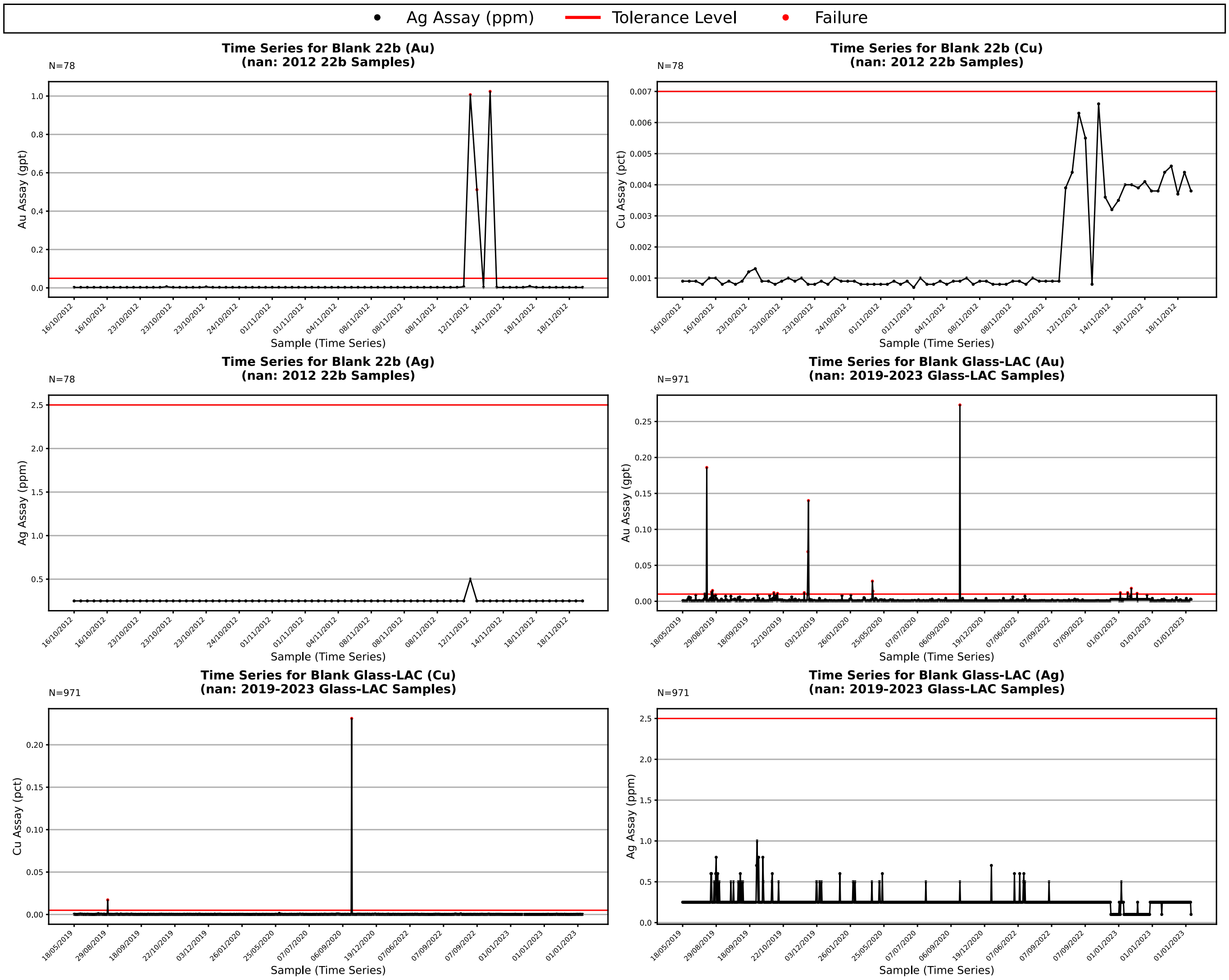
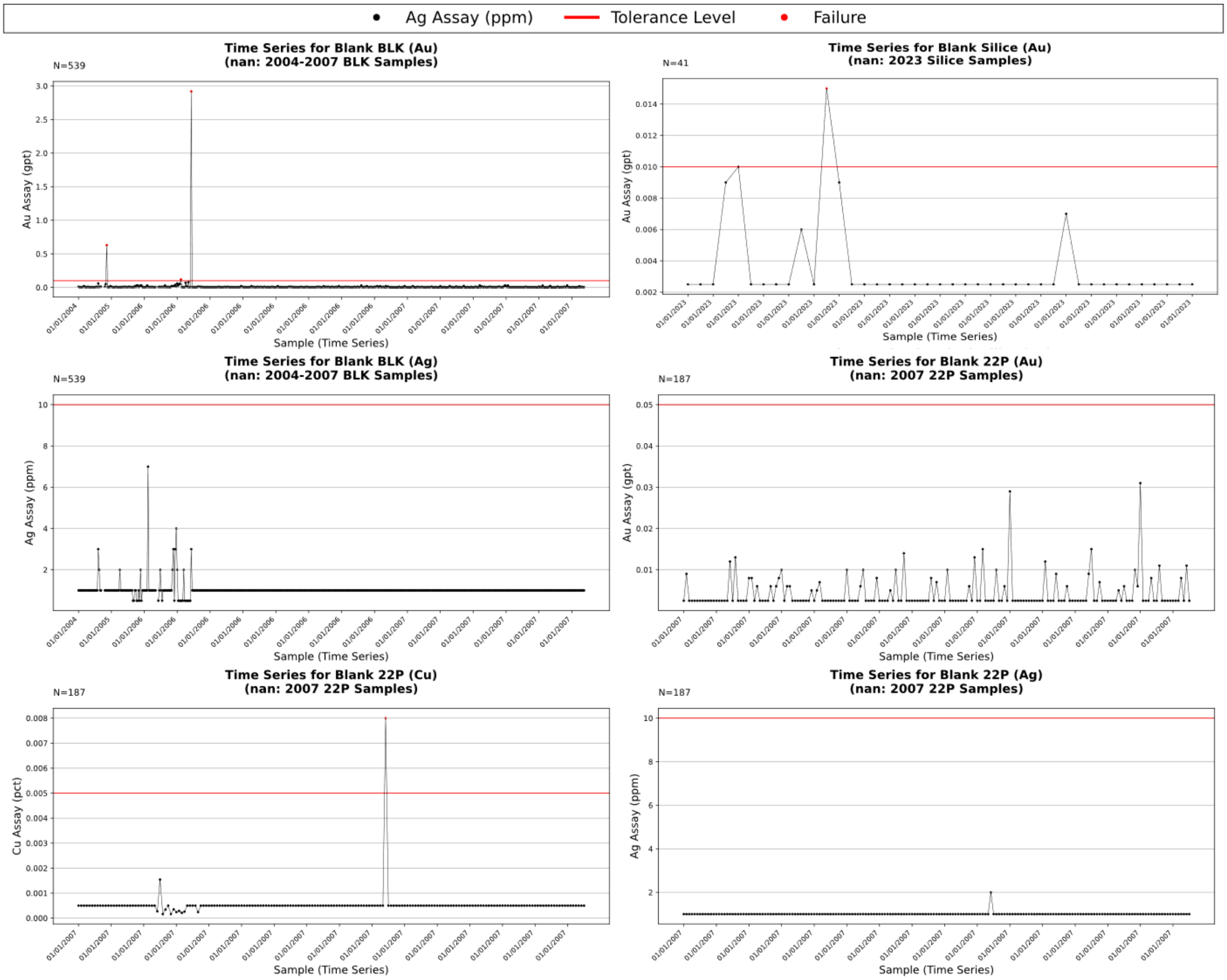
APPENDIX A Analytical Quality Control Data and Relative Precision Charts

Time Series Plots for Blank Material Samples Assayed Between 2012 and 2023



Time Series Plots for Blank Material Samples Assayed Between 2007 and 2012



Time Series Plots Certified Reference Material Samples Assayed for the Condor Project Between 2019 and 2021

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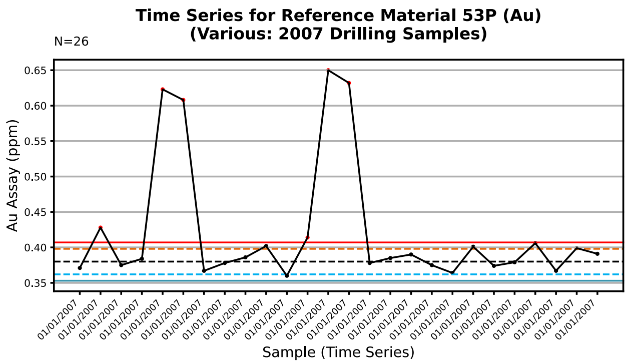
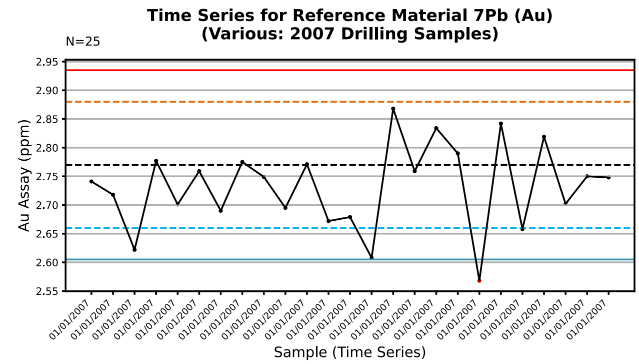
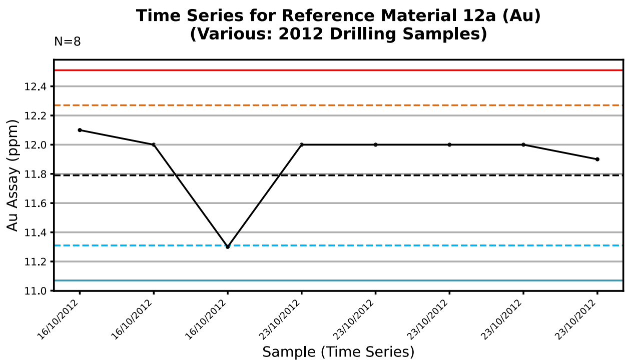
Project	Camp Condor, Enma, Los Cuyes, Soledad
Data Series	2007-2012 Standards
Data Type	Drilling Samples
Commodity	Gold (g/t)
Laboratory	Various
Analytical Method	Fire Assay
Detection Limit	Various

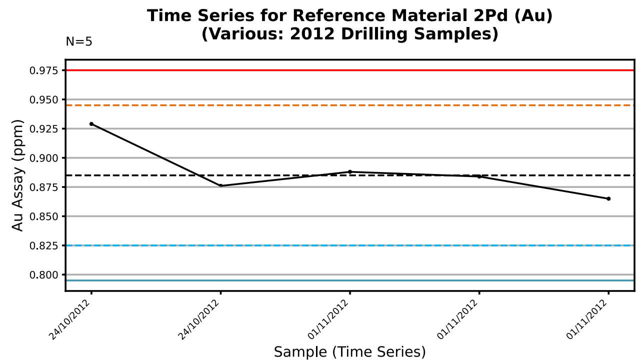
Statistics	15Pa	15Pc	17Pb	18Pb	61Pa	62Pa
Sample Count	35	28	29	25	25	20
Expected Value	1.02	1.61	2.56	3.63	4.46	9.64
Standard Deviation	0.026	0.05	0.17	0.07	0.06	0.14
Data Mean	0.98	1.57	2.53	3.54	4.50	9.47
Outside 3StdDev	11.4%	3.6%	3.4%	16.0%	20.0%	15.0%
Below 3StdDev	4	1	1	3	1	1
Above 3StdDev	0	0	0	1	4	2

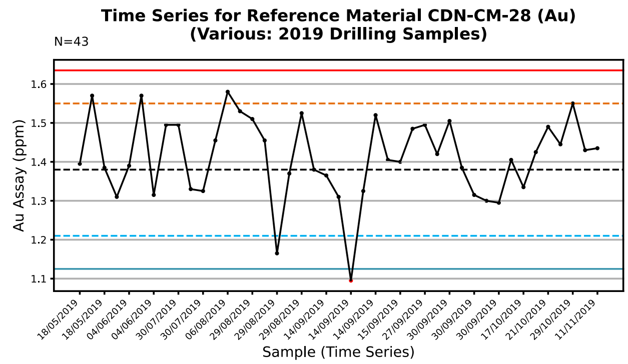
● Au Assay (ppm) — Expected Value — +3SD --- +2SD --- -2SD — -3SD ● Failure

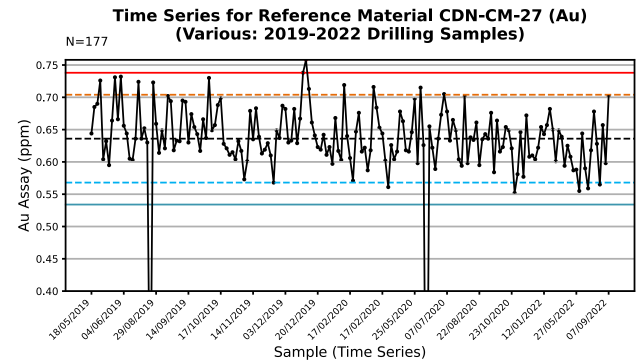








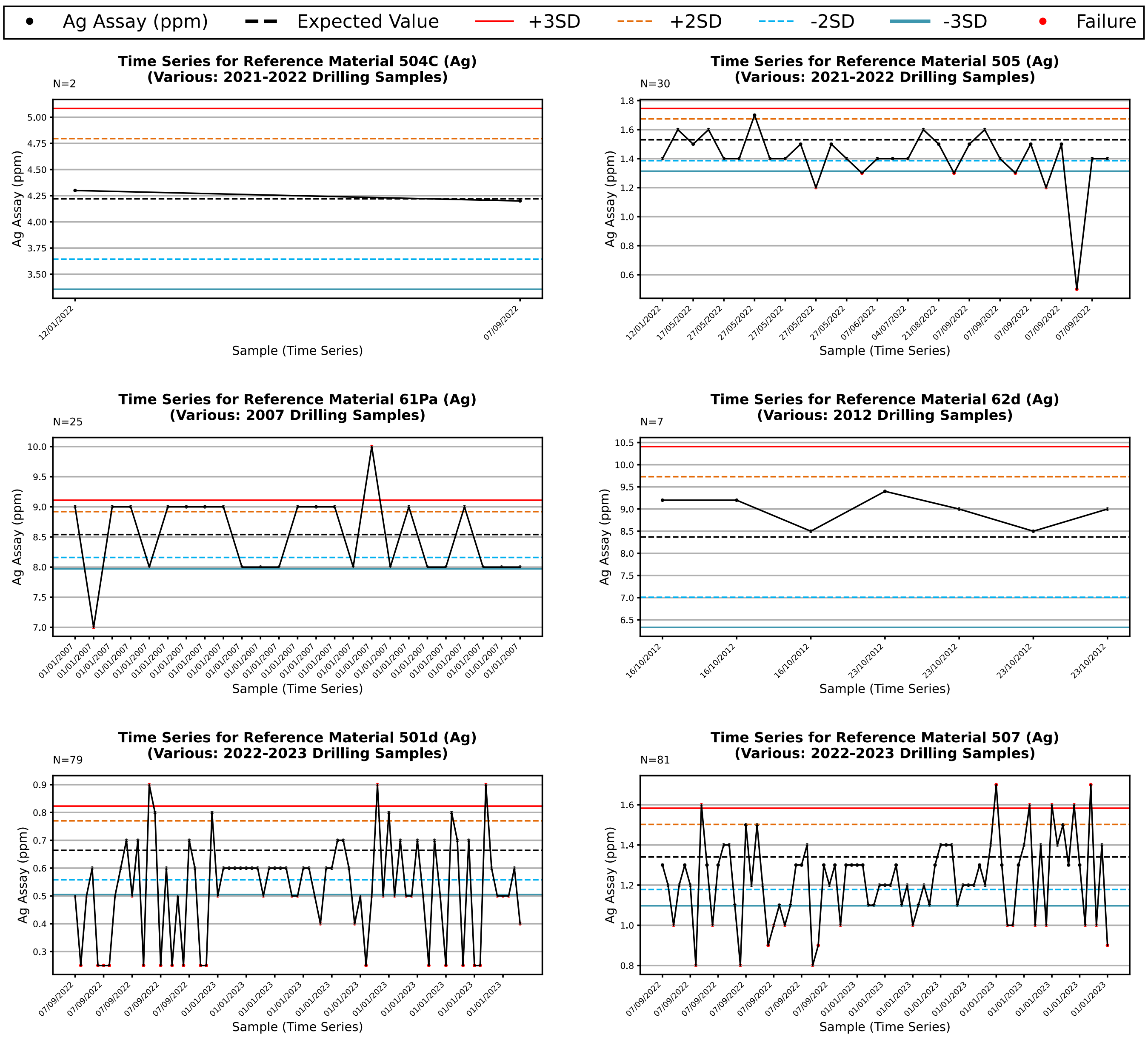




Time Series Plots Certified Reference Material Samples Assayed for the Condor Project Between 2019 and 2021

Project	Camp Condor, Enma, Los Cuyes, Soledac
Data Series	2007-2022
Data Type	Drilling Samples
Commodity	Gold (g/t)
Laboratory	Various
Analytical Method	Fire Assay
Detection Limit	Various

Statistics	12a	2Pd	53P	7Pb	CM-27	CM-28
Sample Count	8	5	26	25	177	43
Expected Value	11.79	0.89	0.38	2.77	0.64	1.38
Standard Deviation	0.24	0.03	0.01	0.06	0.03	0.09
Data Mean	11.91	0.89	0.42	2.73	0.63	1.41
Outside 3StdDev	0.0%	0.0%	23.1%	4.0%	1.7%	2.3%
Below 3StdDev	0	0	0	1	2	1
Above 3StdDev	0	0	6	0	1	0



Time Series Plots Certified Reference Material Samples Assayed for the Condor Project Between 2019 and 2021

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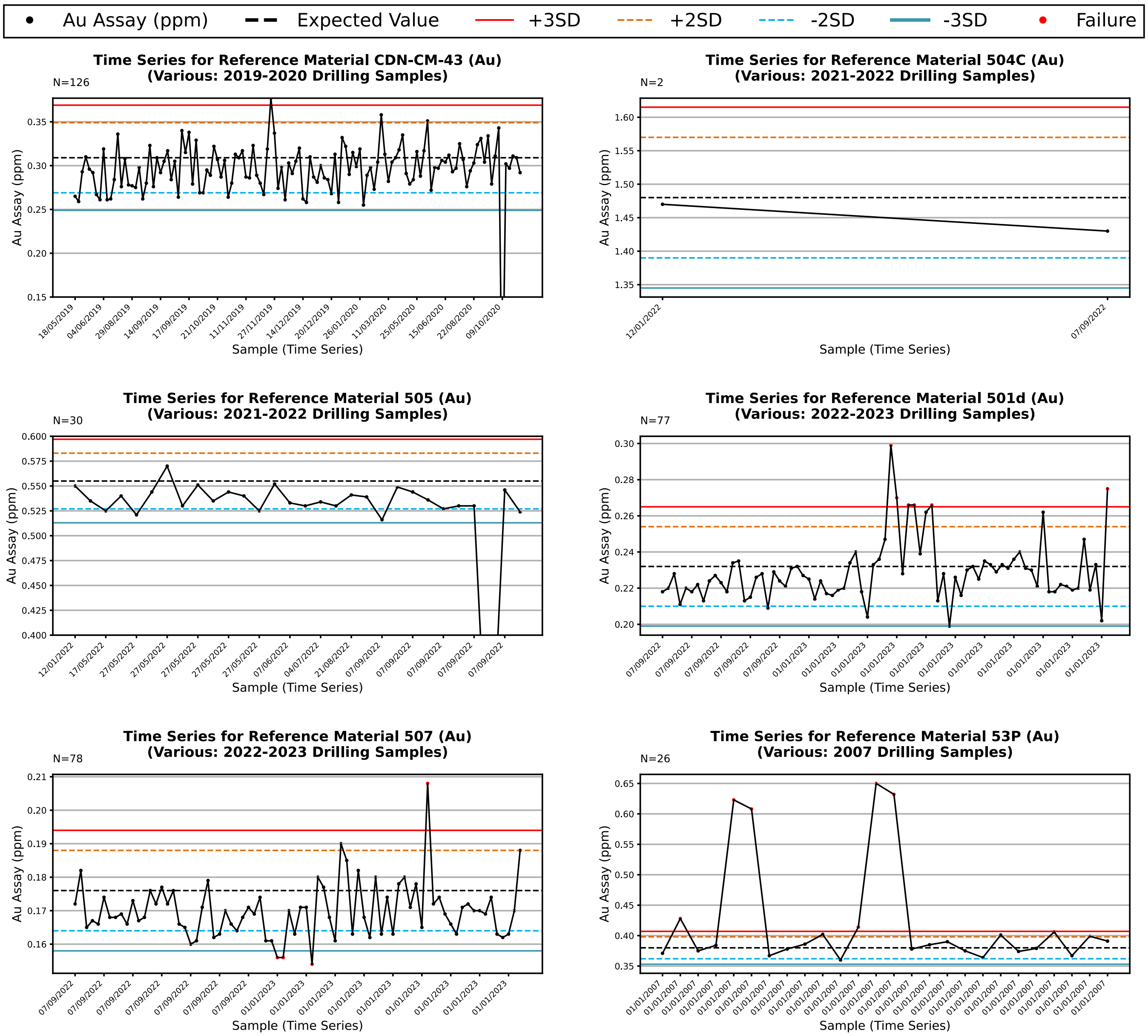
Project	Camp Condor, Enma, Los Cuyes, Soledac
Data Series	2007-2023 Standards
Data Type	Drilling Samples
Commodity	Gold (g/t) and Copper (%)
Laboratory	Various
Analytical Method	Fire Assay & ICP
Detection Limit	Various

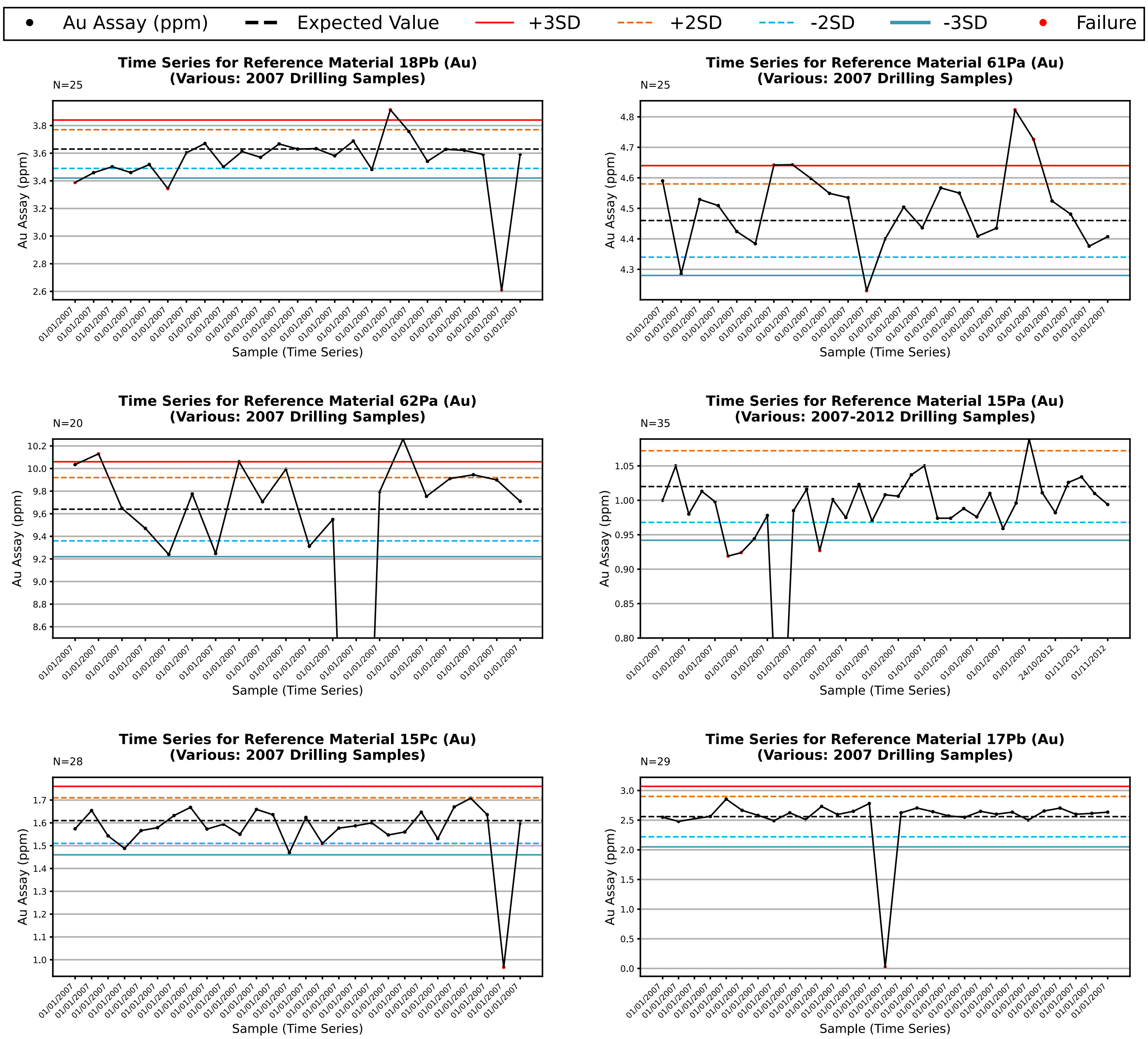
Statistics	501d	504c	505	507	53P	CM-43
Sample Count	77	2	30	78	26	126
Expected Value	0.23	1.48	0.56	0.18	0.41	0.31
Standard Deviation	0.01	0.05	0.01	0.01	0.01	0.02
Data Mean	0.23	1.45	0.53	0.17	0.43	0.30
Outside 3StdDev	7.8%	0.0%	3.3%	5.1%	15.4%	1.6%
Below 3StdDev	0	0	1	3	0	1
Above 3StdDev	6	0	0	1	4	1



Time Series Plots Certified Reference Material Samples Assayed for the Condor Project Between 2019 and 2021

Project	Camp Condor, Enma, Los Cuyes, Soledad
Data Series	2019-2023 Standards
Data Type	Drilling Samples
Commodity	Copper (%)
Laboratory	Various
Analytical Method	ICP
Detection Limit	Various

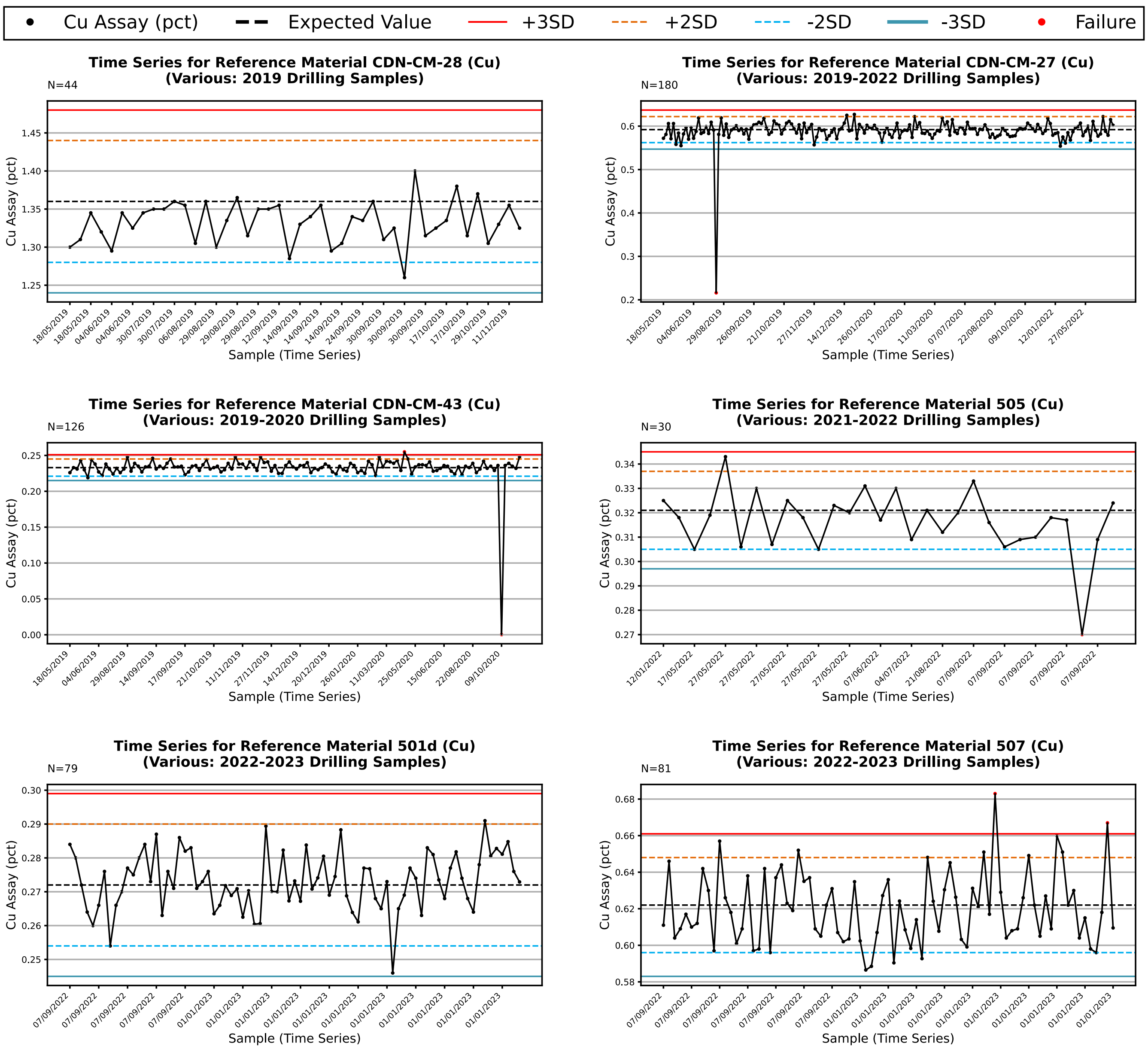
Statistics	501d	505	507	CM-27	CM-28	CM-43
Sample Count	79	30	81	180	44	126
Expected Value	0.27	0.32	0.62	0.59	1.36	0.23
Standard Deviation	0.01	0.01	0.01	0.02	0.04	0.01
Data Mean	0.27	0.32	0.62	0.59	1.33	0.23
Outside 3StdDev	0.0%	3.3%	2.5%	0.6%	0.0%	1.6%
Below 3StdDev	0	1	0	1	0	1
Above 3StdDev	0	0	2	0	0	1



Time Series Plots Certified Reference Material Samples Assayed for the Condor Project Between 2019 and 2021

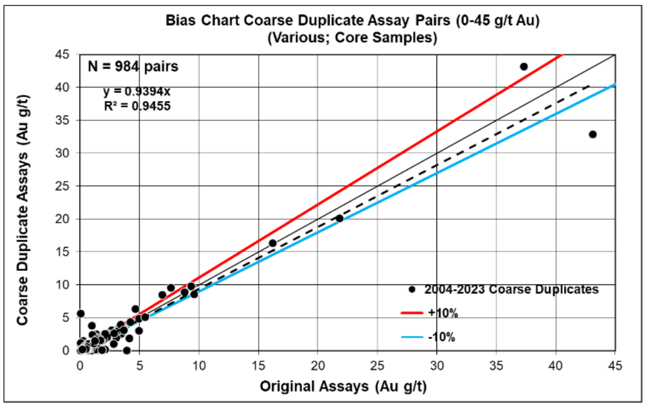
Project	Camp Condor, Enma, Los Cuyes, Soledac
Data Series	2019-2021 Standards
Data Type	Drilling Samples
Commodity	Silver (ppm)
Laboratory	Various
Analytical Method	Fire Assay
Detection Limit	Various

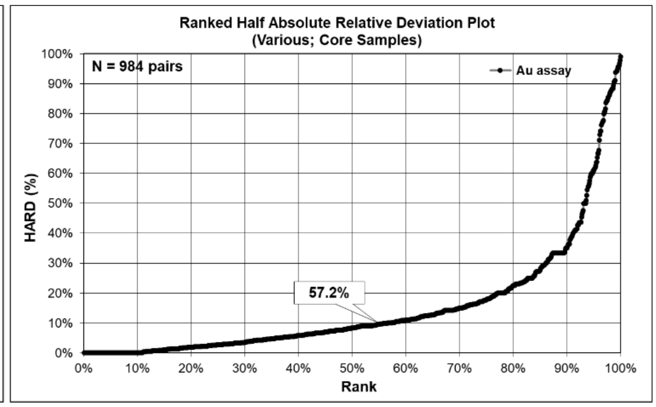
Statistics	**501d**	**504c**	**505**	**507**	**61Pa**	**62d**
Sample Count	79	163	30	81	25	7
Expected Value	0.66	4.22	1.53	1.34	8.54	8.37
Standard Deviation	0.05	0.29	0.07	0.08	0.19	0.68
Data Mean	0.53	4.19	1.41	1.23	8.52	8.97
Outside 3StdDev	54%	6%	20%	30%	8%	0%
Below 3StdDev	40	8	6	18	1	0
Above 3StdDev	3	1	0	6	1	0

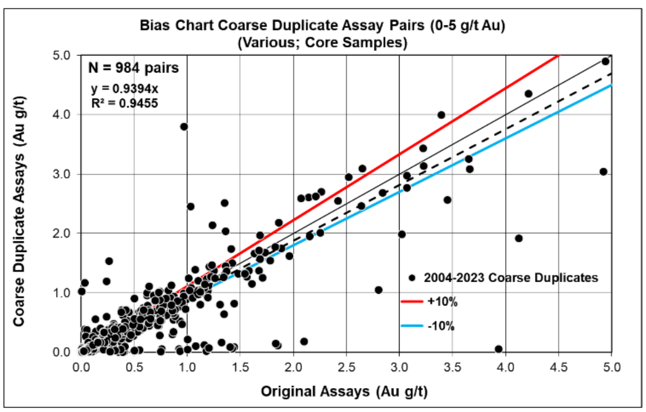


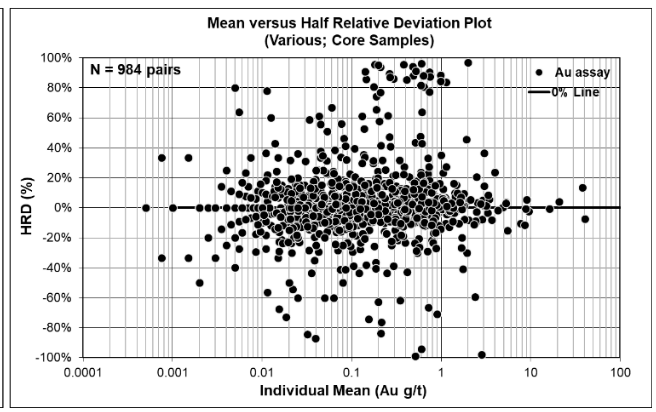
Bias Charts and Precision Plots for Coarse Reject Samples Analyzed Between 2004 and 2023 for Camp, Soledad, Los Cuyes and Enma Deposits

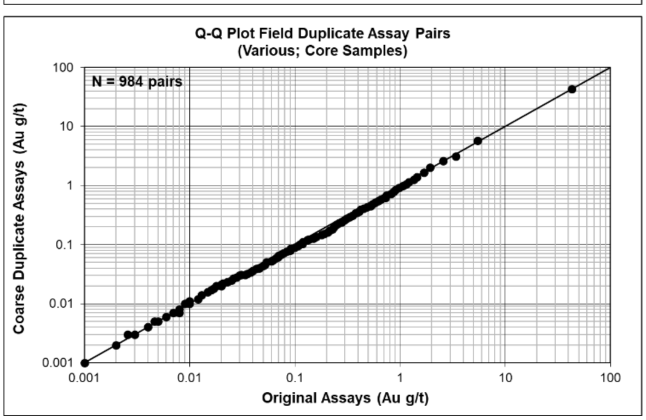
srk consulting		Statistics	Original	Coarse Duplicate
		Sample Count	984	984
Project	Condor	Minimum Value	0.001	0.001
Data Series	2004-2023 Coarse Duplicates	Maximum Value	43.10	43.20
Data Type	Core Samples	Mean	0.519	0.495
Commodity	Au in g/t	Median	0.092	0.086
Analytical Method	Fire Assay AAS Finish	Standard Error	0.069	0.067
Detection Limit	Various	Standard Deviation	2.173	2.101
Original Dataset	Original Assays	Correlation Coefficient	0.9708	
Paired Dataset	Coarse Duplicate Assays	Pairs ≤ 10% HARD	57.2%	

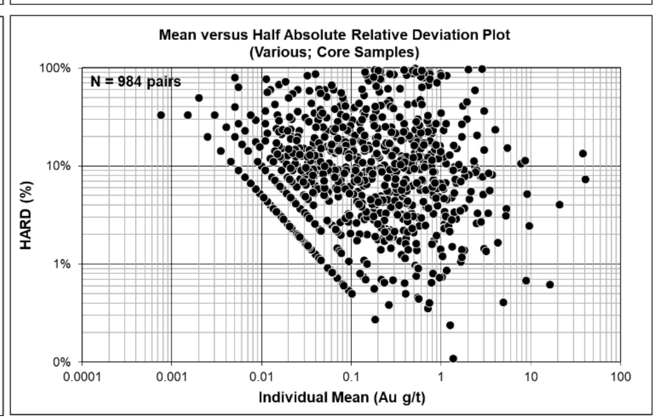










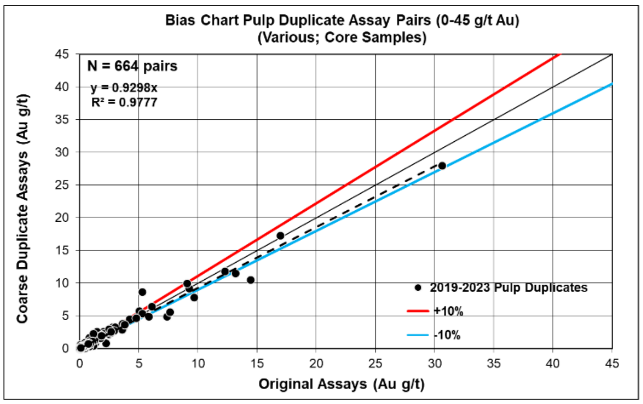


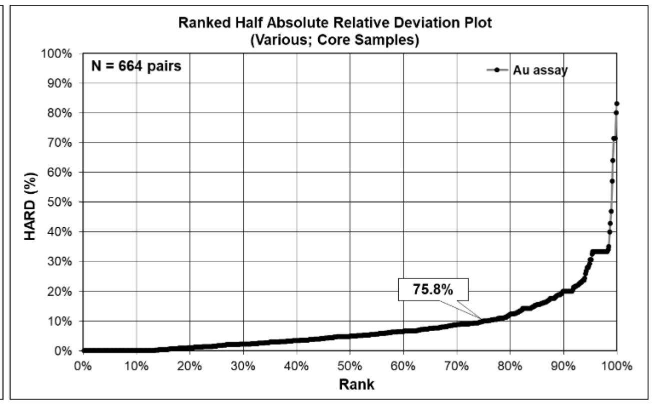
Bias Charts and Precision Plots for Pulp Duplicate Samples Analyzed Between 2019 and 2023 for Camp, Soledad, Los Cuyes and Enma Deposits

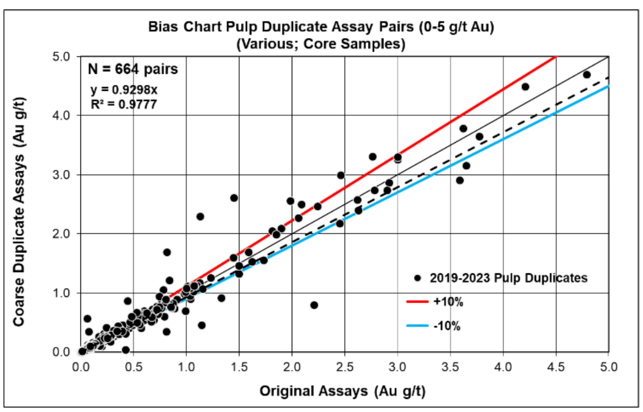
srk consulting

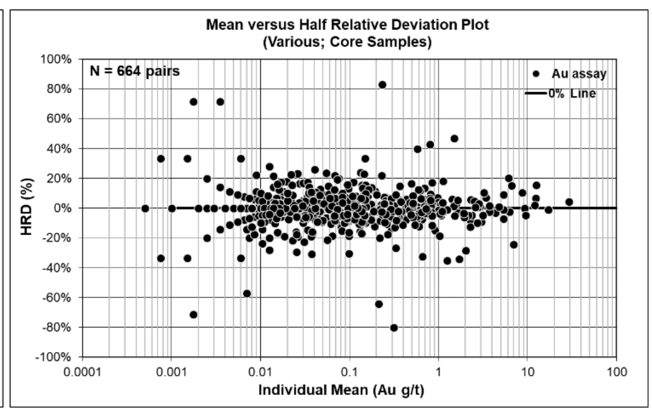
Project	Condor
Data Series	2019-2023 Pulp Duplicates
Data Type	Core Samples
Commodity	Au in g/t
Analytical Method	Fire Assay AAS Finish
Detection Limit	Various
Original Dataset	Original Assays
Paired Dataset	Coarse Duplicate Assays

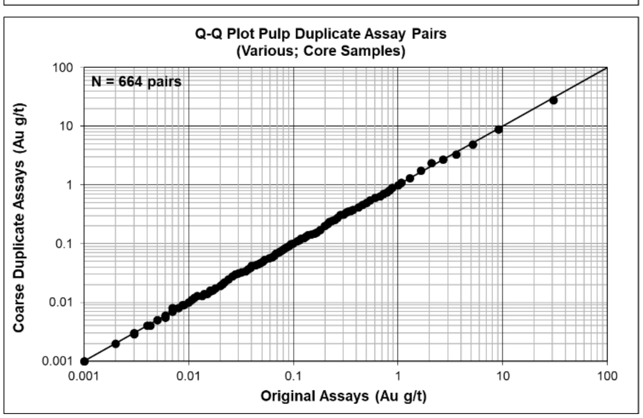
Statistics	**Original**	**Pulp Duplicate**
Sample Count	664	664
Minimum Value	0.001	0.001
Maximum Value	30.60	28.00
Mean	0.513	0.503
Median	0.068	0.068
Standard Error	0.074	0.069
Standard Deviation	1.896	1.777
Correlation Coefficient	0.9880	
Pairs ≤ 10% HARD	75.8%	

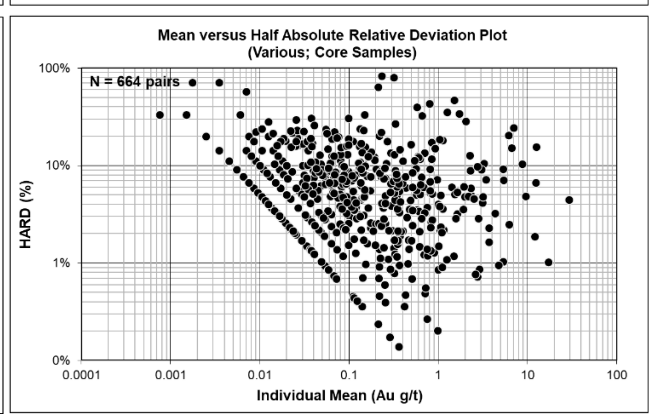












APPENDIX B TSF Concept Cost Estimate

Condor Tailings Storage Facility PEA Estimate

Silvercorp Metals inc.
Preliminary TSF Construction Cost Estimate
Wednesday, November 19, 2025

TETRA TECH

Item No.	Description	Unit	Rate (USD)	Quantity	Cost	Comments
1.00	**Startup/Overhead**					
1.01	Mobilisation/Demobilisation, Insurance, Accommodation, Overheads, Supervision etc.	Factor	---	20%	$ 10,938,888	20% earthworks cost for contractor constructed items (2.00 and 4.00).
	Total				**$ 10,938,888**	
2.00	**Earth Works**					
2.01	Clearing and Grubbing/Stripping of Site	m^2	0.30	1,070,133	$ 321,040	Clearing and grubbing of TSF foundation, basin, and quarry.
2.02	Foundation Preparation	m^2	1.00	217,131	$ 217,131	Scarify, moisture condition, and compact in place.
2.03	Embankment Rock Fill	m^3	9.00	4,779,126	$ 43,012,134	Quarry, haul, place, compact, and grade.
2.04	Embankment Compacted Saprolite - low permeability zone.	m^3	5.00	551,700	$ 2,758,500	Compacted saprolite low permeability zone comprises 10 % of total fill required. Borrow within 2 km, haul, place, and compact.
2.05	60 mil LLDPE Geomembrane Liner Supply and Install.	m^2	7.00	122,058	$ 854,406	Supply and install on upstream embankment slope plus 100 m into TSF basin.
2.06	Embankment Filter Layers	m^3	6.00	186,174	$ 1,117,044	Assumes 3.0 m thick gravelly sand filter zone placed between saprolite and rockfill zones. Screen from local source, haul, place, and compact.
2.07	Surface Water Diversion Ditch	m^3	4.00	6,500	$ 26,000	Assumes 1m deep x 1m base width, cut to fill diversion ditch/berm, includes local sourced rip rap armor on less than 10% footprint and clearing/grubbing costs.
2.08	Access track around TSF basin	m	300.00	6,000	$ 1,800,000	6000 m access road from around TSF basin perimeter.
2.09	Operational Spillway	Lump Sum	---	5	$ 1,000,000	Operational spillway on each dam raise, spillway cut into abutment bedrock. Assumed 5 raises over mine operation phase.
2.10	Seepage collection/polishing pond, return pump system.	Lump Sum	---	1	$ 700,000	
	Total				**$ 51,806,255**	
3.00	**TSF Capital Items**					
3.01	Slurry Pipeline for Discharge	Lump Sum	---	1	-	By Process
3.02	Water Return System: Pipework, Pump	Lump Sum	---	1	-	By Process
3.03	Geotechnical Instrumentation	Lump Sum	---	1	$ 400,000	Geotechnical monitoring instrumentation: survey pins and VWPs, change detection topo monitoring.
	Total				**$ 400,000**	
4.00	**Closure Earth Works**					
4.01	Cover Soil	m^3	3.50	396,624	$ 1,388,184	Assume 1 m thick soil cover for erosion and dust protection over 80% of tailngs area. Load, haul, place, compact. Staged Construction.
4.02	Closure Spillway	Lump Sum	---	1	$ 1,500,000	Closure spillway in abutment bedrock, minimal rip rap required.
	Total				**$ 2,888,184**	
5.00	**Miscellaneous**					
5.01	Design, Construction QC/QA	Factor	---	15%	$ 8,264,166	0
	Total				**$ 8,264,166**	
	Subtotal				**$ 74,297,493**	
	Owner's Cost			3.5%	$ 2,600,412	
	Contingency			20%	$ 14,859,499	
	Cost Estimate Total				**$ 91,757,403**	

Condor Tailings Storage Facility PEA Life of Mine Capital Cost Estimate

Silvercorp Metals inc.

Preliminary TSF Construction Life of Mine Cost Estimate

Wednesday, November 19, 2025

TMF Cost Schedule

Item	Pre Production: Year -1	Production →: Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Startup/Overhead	$ 2,164,019	$ -	$ 1,468,061	$ -	$ 1,468,061	$ -	$ 1,468,061	$ -	$ 2,375,704	$ -
Earthworks	$ 10,820,093	$ -	$ 7,340,305	$ -	$ 7,340,305	$ -	$ 7,340,305	$ -	$ 11,878,518	$ -
TSF Capital Items	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ 28,571
Closure	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Design, Construction QA/QC	$ 1,627,300	$ 4,286	$ 1,105,332	$ 4,286	$ 1,105,332	$ 4,286	$ 1,105,332	$ 4,286	$ 1,786,063	$ 4,286
Owner Cost	$ 512,399	$ 1,150	$ 347,979	$ 1,150	$ 347,979	$ 1,150	$ 347,979	$ 1,150	$ 562,410	$ 1,150
Contigency	$ 2,927,996	$ 6,571	$ 1,988,454	$ 6,571	$ 1,988,454	$ 6,571	$ 1,988,454	$ 6,571	$ 3,213,771	$ 6,571
Total Cost	**$ 18,080,378**	**$ 40,579**	**$ 12,278,703**	**$ 40,579**	**$ 12,278,703**	**$ 40,579**	**$ 12,278,703**	**$ 40,579**	**$ 19,845,037**	**$ 40,579**

Item	Year 10	Year 11	Year 12	← Production: Year 13	Closure →: Year 14	Year 15	Year 16	Year 17	Year 18	Total
Startup/Overhead	$ 1,492,346	$ 75,000	$ 75,000	$ 75,000	$ -	$ 69,409	$ 69,409	$ 69,409	$ 69,409	$ 10,938,888
Earthworks	$ 7,086,729	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 51,806,255
TSF Capital Items	$ 28,571	$ 28,571	$ 28,571	$ 28,571	$ -	$ -	$ -	$ -	$ -	$ 400,000
Closure	$ 375,000	$ 375,000	$ 375,000	$ 375,000	$ -	$ 347,046	$ 347,046	$ 347,046	$ 347,046	$ 2,888,184
Design, Construction QA/QC	$ 1,123,545	$ 60,536	$ 60,536	$ 60,536	$ -	$ 52,057	$ 52,057	$ 52,057	$ 52,057	$ 8,264,166
Owner Cost	$ 353,717	$ 18,869	$ 18,869	$ 18,869	$ -	$ 16,398	$ 16,398	$ 16,398	$ 16,398	$ 2,600,412
Contigency	$ 2,021,238	$ 107,821	$ 107,821	$ 107,821	$ -	$ 93,702	$ 93,702	$ 93,702	$ 93,702	$ 14,859,499
Total Cost	**$ 12,481,145**	**$ 665,797**	**$ 665,797**	**$ 665,797**	**$ -**	**$ 578,612**	**$ 578,612**	**$ 578,612**	**$ 578,612**	**$ 91,757,403**

Ongoing Activities

Year -1
- Site Stripping
- Starter Dam (944 m)
- Diversion Ditch
- Acces track
- Operation Spillway
- Seepage Collection Pond
- Geotech Instrumentation

Year 2
- Raise 1 (958 m)
- Operational Spillway
- Geotech Instrumentation

Year 4
- Raise 2 (968 m)
- Operational Spillway
- Geotech Instrumentation

Year 6
- Raise 3 (976 m)
- Operational Spillway
- Geotech Instrumentation

Year 8
- Raise 4 (984 m)
- Operational Spillway
- Geotech Instrumentation

Year 10
- Raise 4 (989 m)
- Closure Spillway
- Geotech Instrumentation

Year 11
- Topsoil replacement
- Geotech Instrumentation
- Closure Spillway

Year 12
- Geotech Instrumentation
- Closure Spillway

Year 13
- Geotech Instrumentation
- Closure Spillway

Year 14
- N/A

Year 15
- Tailings cover

Year 16
- Tailings cover

Year 17
- Tailings cover

Year 18
- Tailings cover